Public comment for Dec 13th meeting (5 min.)

Hello my Name is Ted Stout a resident of Plumas County and Indian Valley,

I would like to thank the County for the opportunity to address this issue.

I would also like to recognize the county's staff for the hard work they are doing under difficult conditions. Our county has been hit by several natural disasters and is being stretched very thin.

Note A

~~While recognizing the matter at hand is a finding of facts, I must note that the decision made will have profound long lasting effects for our county and the environment.~~

It is important to note that a finding of Vested Rights will allow US Copper Corp. to extend these rights over a 13 sq. mile area. It is my understanding that once a determination of Vested Rights is made it can not be revoked at a later date, so there is no putting the genii back in the bottle.

Let me state that I am not against mining categorically, however I am against mining done with reduced oversight and regulation.

US Copper Corp. very much wants Vested Rights to be granted because it will exempt them from many reporting, permitting and environmental requirements. There will still be some oversight, however the County of Plumas will effectively have their ability to enforce any controls removed. The community will have lost their opportunity to have input into this project and make no mistake this project as laid out in the company's report is vast.
(Exhibit A: https://miningdataonline.com/property/3292/Moonlight-Superior-Project.aspx#Documents)

Now to the meat of the matter:

The original request was submitted to the county on April 26th 2023.
The county Staff report was publicly published on October 11th 2023, but the document was not complete in it's published format.

The report appears to have no external resources referenced other then what was provided by U.S. Copper Corp, a Canadian company. A corporation with obvious interests in a positive finding should not be the sole source from which to obtain the facts to determine this issue.

Removed because of ZA Scope limitation Note A:

~~U.S. Copper Corp asserted that there are no plans for this site in the October 11th 2023 Board of Supervisors meeting, which is not what they are telling their investors. They generated a PEA(Exhibit A) document that includes, buildings to be built, processing methods, staffing requirements, staffing count, where staff will be sourced, a detailed list of expenses, operating costs, tax projections, facilities to process over 60,000 tons of material a day(Let that sink for a second, if that was cars it would be a pile of 20,000 cars a day 24/7/365 for 17 years) and expected profits. Also of interest in this document there are no expected fees, permits or taxes paid to Plumas County.~~

~~The workforce they hire will be from Canada and Nevada, which means the limited housing stock in our county will be stretched thinner and rents will rise, with no direct local jobs created.~~

With that in mind I think it's critical to verify and validate the facts that are being asserted to justify the granting of Vested Rights.

There are several legal questions about Vested Rights that should be reviewed by legal council that specializes in mining and possibly environmental law.

All Vested Rights regulations I can find specifically use the term surface mining and historically the sites in question appear to have been underground works.

Given the information above, I would respectfully request the items spelled out in Addendum A, attached.

Note A

~~This project has the potential to effect water rights for our ranchers, our farmers, our wells, First Nations Sacred Sites, property values in Indian Valley and Lake Almanor, the Feather River watershed, as well as other impacts I am not even aware of at this time.~~

Given that it took the county months to generate that staff report, I think it is reasonable for the county to give the citizens a similar amount of time to review and comment on this issue.

Note A

~~These are thousands of examples of mine issues that cause long term negative effects, for the sake of brevity, I will only cite two. Currently there is a mine in Spain that was mined 4500 years ago and is still leaking contaminates into the environment.~~
~~(See Exhibit B:~~
~~(https://www.researchgate.net/publication/270457097_4500-year-old_mining_pollution_in_Southwestern_Spain_Long-Term_implications_for_modern_mining_pollution/link/561e73a408ae50795afefc10/download?_tp=eyJjb250ZXh0Ijp7ImZpcnN0UGFnZSI6InB1YmxpY2F0aW9uIiwicGFnZSI6InB1YmxpY2F0aW9uIn19~~)

Note A

~~An example closer to home in neighboring Shasta county is Iron Mountain. This site was designated a Superfund site in 1983 and is polluting to this day. Billions have been spent and the pollution still reaches the S.F. Bay. There is a multi-million dollar water treatment plant, costing millions to run per year. This plant will need to run for the next 3000 years. See Exhibit C:~~
~~(https://www.sfgate.com/news/article/Iron-Mountain-a-study-in-perpetual-pollution-3092377.php~~)

In summary I do not believe we have sufficient verified facts to support a finding of fact in this matter.

Thank you for this opportunity to address the County and the community.

Ted Stout

Example of Legal Quesions:
Does selling aggregates from a tailing pile constitute surface mining?

Addendum A:

1) That the county acquire appropriate legal council with experience in mining law/rights/regulations in California and at the Federal level.
2) Aquire and provide to the public a complete list of all APNs/claims and other property identifiers that US Copper owns or controls in Plumas County and surrounding counties.
3) Aquire and provide to the public information collected from the county museum, county records, and relevant agencies such as USGS, US Forest Service, BLM, etc. about the above properties.
4) Documentation that these APNs/Claims are current on all permits, fees and/or other requirements.

Link regarding Vested Rights and extension past private property

southbayquarrylibrary.org/Catalog/PH_2011_01_10_Vested_Mining_Rights_11-1.pdf

Exhibit A

https://miningdataonline.com/property/3292/Moonlight-Superior-Project.aspx#Documents

Report to:

Crown Mining Corp.

CROWN MINING
CORP

Technical Report and Preliminary Economic Assessment for the Moonlight Deposit, Moonlight-Superior Copper Project, California, USA

704-MIN-VMIN03123-02

TETRA TECH

Report to:

Crown Mining Corp.

Technical Report and Preliminary Economic Assessment for the Moonlight Deposit, Moonlight-Superior Copper Project, California, USA

Effective Date: March 2, 2018

Prepared by Hassan Ghaffari, P.Eng.
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HG/vc

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GLOSSARY

Units of Measure

acre	ac
arcminute	'
arcsecond	"
Canadian dollars	Cdn$
coefficient of variation	CV
cubic foot	cu ft
cubic metre	m^3
cubic yard	cu yd
day	d
days per week	d/wk
days per year	d/a
degrees Celsius	°C
degrees Fahrenheit	°F
degrees	°
foot	ft
gigawatt hour	GWh
gram per tonne	g/t
grams	g
horse power	hp
hour	h
hours per day	h/d
inch	in
kilograms	kg
kilowatt hour	kWh
kilowatt	kW
litre	L
megawatt	MW
metres	m
microgram	µg
micron	µm
mile	mi
millimetre	mm
million troy ounces	Moz
million years	Ma
million	M
parts per million	ppm
percentage	%
pound	lb
short ton (2,000 lb)	st

short tons per day	st/d
short tons per year	st/a
square mile	sq mi
tonne (1,000 kg) (metric ton)	t
troy ounce	oz
troy ounces per short ton	oz/st
US dollars	US$
volt	V
yard	yd
year (annum)	a

Abbreviations and Acronyms

acid generation potential	MPA
Acid-base Accounting	ABA
Alternative Minimum Tax	AMT
American Exploration and Mining Co.	Placer -Amex
ammonium nitrate fuel oil	ANFO
arsenic	As
atomic absorption spectrophotometer	AAS
atomic absorption	AA
below detection	BDL
Biotic Ligand Model	BLM
California Environmental Quality Act	CEQA
California-Engels Mining Company	California-Engles
Cameron Resources Consulting LLC	CRC
Canadian Institute for Mining, Metallurgy and Petroleum	CIM
Canyon Copper Corp.	Canyon Copper
change-of-support	COS
closed-circuit television	CCTV
cobalt	Co
Controlled-source Audio-frequency Magnetotellurics	CSAMT
copper	Cu
Crown Mining Corp	Crown Mining
detection level	DL
digital terrain model	DTM
distributed control system	DCS
east	E
electromagnetics	EM
end of hole	EOH
Engineering, procurement and construction management	EPCM
Environmental Impact Reports	EIR
Environmental Impact Statement	EIS
Environmental Protection Agency	EPA
exploratory data analysis	EDA

free board marine	FOB
free carrier	FCA
Fugro Airborne Surveys	Furgo
general and administrative	G&A
global positioning system	GPS
gold	Au
graphite furnace atomic adsorption	GFAA
heating, ventilation, and air conditioning	HVAC
Heinrichs Geoexploration Company	HGC
high-pressure grinding roll	HPGR
hydrofluoric acid	HF
Induced Polarization	IP
inductively coupled plasma atomic emission spectrometry	ICP-AES
inductively coupled plasma	ICP
inductively coupled plasma-mass spectrometry	ICP-MS
inlet/outlet	I/O
internal rate of return	IRR
Internal Revenue Code	IRC
inverse distance weighting	ID3
Iron Oxide Copper Gold	IOCG
Kappes, Cassidy & Associates	KCA
life-of-mine	LOM
light rare earth element	LREE
Lights Creek District	LCD
Lights Creek Stock	LCS
Material Safety Data Sheet	MSDS
memorandum of understanding	MOU
methyl isobutyl carbinol	MIBC
molybdenum	Mo
motor control centre	MCC
National Ambient Air Quality Standards	NAAQS
National Environmental Protection Act	NEPA
National Instrument 43-101	NI 43-101
net operating losses	NOL
net present value	NPV
net smelter return	NSR
neutralization potential	NP
Nevoro Copper Inc.	Nevoro Copper
Nevoro Inc.	Nevoro
nickel	Ni
nitric acid	HNO_3
North American Datum	NAD
north	N
OreQuest Consultants Ltd.	OreQuest
perchloric acid	$HClO_4$

potassium amyl xanthate	PAX
Preliminary Economic Assessment	PEA
Prevention of Significant Deterioration	PSD
PricewaterhouseCoopers	PwC
programmable computer	PC
programmable logic controller	PLC
Qualified Person	QP
quality assurance	QA
quality control	QC
quartz monzonite	QM
Quatreface Consulting LLC	Quatreface
Regional Water Quality Control Board	RWQCB
run-of-mine	ROM
Sheffield Resources Inc.	Sheffield
Society of Mining Engineers	SME
south	S
specific gravity	SG
Starfield Resources Inc.	Starfield
State Mining & Geology Board under the Surface Mining and Reclamation Act	SMARA
tailings management facility	TMF
Tax Cuts and Jobs Act	TCJA
Tetra Tech Canada Inc.	Tetra Tech
the Moonlight-Superior Project	the Project or the Property
The US Army Corps of Engineers	USACE
Toronto Stock Exchange	TSX
total dissolved solids	TDS
Union Assay Laboratory	Union Assay
United States Geological Survey	USGS
Universal Transverse Mercator	UTM
US Forest Service	USFS
Voice-over Internet Protocol	VoIP
waste rock storage facility	WRSF
west	W
work breakdown structure	WBS
World Goedetic System	WGS
x-ray fluorescence spectrometer	XRF

1.0 SUMMARY

1.1 INTRODUCTION

Crown Mining Corp. (Crown Mining) retained Tetra Tech Canada Inc. (Tetra Tech) to complete a Technical Report and Preliminary Economic Assessment (PEA) for the Moonlight deposit, which forms part of the Moonlight-Superior Project (the Project or the Property) located in Plumas County, California. This PEA has been written in accordance with National Instrument 43-101 (NI 43-101) Standard of Disclosure for Mineral Projects, including the NI 43-101 Companion Policy and Form NI 43-101F1.

The key outcomes for the Moonlight deposit are summarized in Table 1.1.

Table 1.1 Moonlight Deposit Project Summary

Description	Unit	Value
Commodity	-	Copper/Silver
Mining Area	-	Moonlight Deposit
Mining Method	-	Open Pit
Estimated Average Mill Feed Grade (LOM)	% Cu	0.25
LOM	years	17
Processing Method	-	Conventional Flotation
Production Rate	st/d	60,000
Metallurgical Copper Recovery	%	86
Metallurgical Silver Recovery	%	70
Total Initial Capital, excluding Leasing Cost	US$ million	513
Total LOM Capital	US$ million	818
Operating Cost	US$/st processed	7.77
Copper Price	US$/lb	3.15
Silver Price	US$/oz	18.00
Pre-tax IRR	%	16.4
Pre-tax NPV (8%)	US$ million	237
Pre-tax Payback (undiscounted)	years	4.8
Post-tax IRR	%	14.6
Post-tax NPV (8%)	US$ million	179

Notes: LOM = life-of-mine; IRR = internal rate of return; NPV = net present value
A PEA should not be considered a prefeasibility or feasibility study, as the economics and technical viability of the Project have not been demonstrated at this time. A PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results as reported in the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Tetra Tech led a group of qualified consultants, commissioned by Crown Mining, to complete portions of this PEA:

- Tetra Tech: overall project management, project description and location, accessibility, mining, metallurgy, process, project infrastructure, tailings and waste rock management, water management, capital and operating cost estimates, and economic analysis.
- Cameron Resources Consulting LLC (CRC): history, geological setting, deposit types, exploration, drilling, sample preparation, data verification, Mineral Resource estimate, and adjacent properties.
- Quatreface Consulting LLC (Quatreface): environmental studies, permitting, and social or community impact.

The effective date of this PEA is March 2, 2018 and the effective date of the Moonlight deposit Mineral Resource estimate is December 15, 2017.

All measurements are reported in US imperial units, unless otherwise noted.

All currency is reported in US dollars (US$), unless otherwise noted.

1.2 Property Description and Location

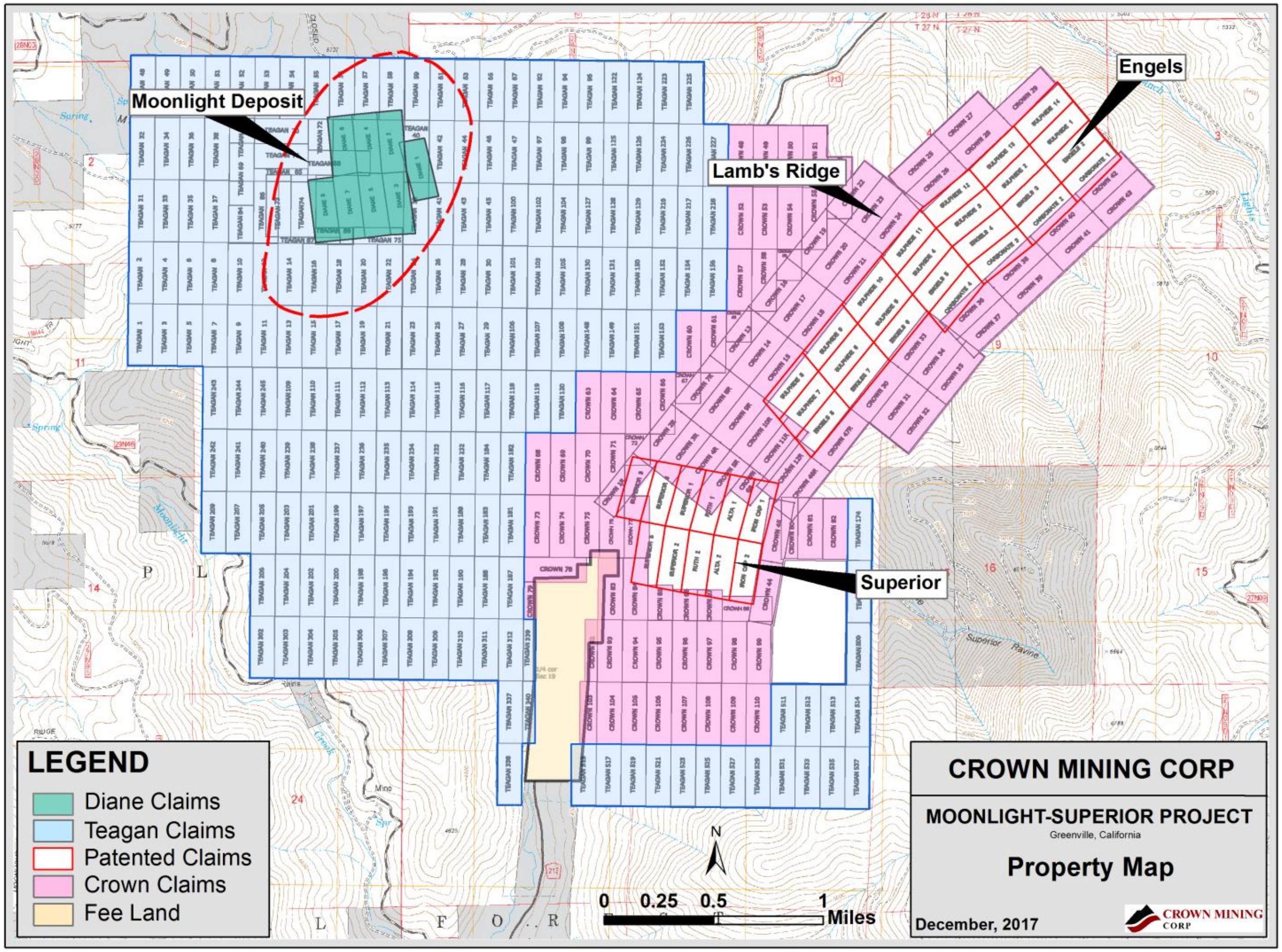
The Moonlight-Superior Project is located approximately 12 air miles southwest of the town of Susanville in Plumas County, California, which is approximately 85 mi northwest of Reno, Nevada (Figure 1.1). The Property consists of eight unsurveyed, unpatented, contiguous optioned mining lode claims, 110 surveyed, unpatented lode claims, 36 patented lode mineral claims, and 204 unsurveyed, unpatented, optioned lode claims covering an area of approximately 6,822 ac (Figure 1.2).

Crown Mining, through a lease agreement in 2013 with California-Engels Mining Company (California-Engles) and an option agreement in 2016 with Canyon Copper Corp. (Canyon Copper), both subject to small underlying production royalties, controls the majority of the historic Lights Creek District (LCD) located in Plumas County in northern California.

Crown completed the purchase of a 100% undivided title interest in the Moonlight Property located in Plumas County on March 13, 2018.

Figure 1.1 Regional Location Map

Figure 1.2 Moonlight-Superior Project Property Map



1.3 HISTORY

Gold was discovered in Plumas County in 1850. Copper deposits were noted but were not exploited until the American Civil War (1861-1865) when a smelter was built in Genesee Valley. Copper was mined and shipped from the LCD during this period. Henry A. Engels and sons acquired the Superior Mine in 1880 and discovered the Engels Mine in 1883. The Engels Mine is located approximately 3 mi east of the Moonlight deposit. The Superior Mine is approximately 2.2 mi southeast of the Moonlight deposit. Both now form part of the Crown Mining claim holdings. Both mines shut down in 1930. From 1930 through 1961 activity in the LCD was largely limited to exploration.

From 1961 through 1981 American Exploration and Mining Co. (Placer-Amex) conducted exploration in the LCD. Reconnaissance surveys were completed in 1962 and 1963. Stream sediment and soil sampling surveys were conducted in 1964 and 1965. In addition to the Superior and Engels mine sites, Lambs Ridge (formerly Sulfide Ridge) and the Moonlight area showed significant copper anomalies in soils. Beginning in 1964 and continuing through 1970, Placer-Amex conducted an extensive drilling program covering much of the LCD.

Despite the success of the Placer-Amex program, which included the discovery of the Moonlight deposit, the low price of copper and refocused priorities led the company to abandon the Property in 1994.

Subsequently from 2004 to 2012, a succession of Canadian junior companies (Sheffield Resources Inc. [Sheffield], Nevoro Inc. [Nevoro], and Starfield Resources Inc. [Starfield]) reassembled the Property and completed some focused, but limited work, including drilling. Between 2004 and 2008, Sheffield staked an additional 410 unpatented lode claims in the district. In April 2006, Sheffield optioned the California-Engels land consisting of approximately 894 ac of deeded land covering the historic Engels and Superior mines. Additional unpatented lode claims were staked in 2007 (33 total), 2008 (23 total) and 2011 (12 total). Sheffield was acquired by Nevoro Copper Inc. (Nevoro Copper) in July 2008.

In 2009, Starfield acquired Nevoro, the parent company of Nevoro Copper. In 2012, following limited drilling at the Engels Mine and additional district-wide exploration including an airborne electromagnetics (EM) program, Starfield dropped the unpatented claims encompassing the Moonlight deposit.

By 2013 the LCD was again split with the Moonlight deposit controlled by Canyon Copper, by virtue of an assignment from Starfield, while the Superior-Engels lease was acquired out of bankruptcy court by Crown Gold Corp. The Superior-Engels acquisition included the complete database held by Nevoro, which comprehensively documented all known exploration activity on the Property from 1960 through 2013. In February 2016, Crown Mining (re-named from Crown Gold Corp. in 2014) optioned Canyon Copper's position and the LCD again became a unified property.

1.4 GEOLOGICAL SETTING AND MINERALIZATION

The Project area covers most of the historic LCD, located at the northern end of the Sierra Nevada physiographic province at the juncture with the late-Tertiary-to-Recent Cascade volcanic province to the north, and the Basin and Range province immediately to the east. The LCD lies at the northern end of the 25 mi long, 5 mi wide, N20W trending Plumas Copper Belt, interpreted to represent an extension of the north-northwest trending Walker Lane structural lineament and at the eastern terminus of the Mendocino Fracture Zone.

The LCD copper deposits are primarily hosted in the early Jurassic (178 Ma), multi-phased, quartz monzonite Lights Creek Stock (LCS), which intrudes slightly older meta-volcanic rocks and is itself intruded by the younger Sierra Nevada Batholith. The LCS is a roughly circular fine- to medium-grained quartz monzonite to granodioritic tourmaline-rich intrusive, with an exposure of approximately 7 sq mi. Structural preparation has been important in localizing mineralization in the LCD. Multiple structurally distinct sets of fracture zones appear to control much of the copper mineralization in the LCD.

Placer-Amex, Sheffield, and Sheffield's successors recognized that there are at least two styles of mineralization at the Moonlight deposit. Structurally controlled mineralization is preferentially located in stockwork zones, with fractures of multiple orientations, or at the intersection of structures and lithologic contacts. Disseminated copper mineralization associated with rosettes of tourmaline is also a significant component of mineralization at Moonlight. This mineralization usually consists of fine-grained chalcopyrite, but zones of disseminated bornite are also common.

At the Moonlight deposit the primary copper-bearing minerals are bornite and chalcopyrite, with lesser amounts of covellite and chalcocite. The dominant iron species found within the deposit are magnetite and hematite (especially specularite). The Moonlight deposit also contains minor amounts of pyrite. The copper sulfides show a vertical zonation, with chalcocite dominating in the upper levels of the deposit. With increasing depth, bornite dominates and chalcopyrite appears. At the deeper levels, chalcopyrite typically dominates in fracture hosted mineralization, but bornite is locally still abundant. Limited oxidation and supergene products of copper minerals are observed in surface outcroppings and in the tops of some drillholes. Minor amounts of precious metals are associated with the copper mineralization, but their paragenesis has not been studied in detail.

Mineralization at the Moonlight deposit also includes an acid soluble component that overlies the sulfide deposit in three areas: North, Central and South Oxide Zones. In the 1970s, Placer-Amex estimated an oxide resource of 12.2 million st at an average grade of 0.54% copper. Sheffield drilled 15 shallow reverse circulation holes at Moonlight in 2007, which appear to support the deposit's potential for economic copper oxide mineralization.

1.5 DEPOSIT TYPES

Copper deposits of the LCD were historically classified as porphyry copper deposits with associated gold and silver credits. Nevertheless, Placer-Amex geologists recognized that the deposits of the LCD copper deposits had many characteristics that were not typical of porphyry copper deposits. L.O. Storey (1978) noted, "Typical porphyry copper-type alteration zonation as illustrated by Lowell and Guilbert is nonexistent." Recent work, noting the lack of porphyry style veining, the ubiquitous presence of magnetite (Superior), and specularite (Moonlight), and the relative scarcity of pyrite suggest an Iron Oxide Copper Gold (IOCG) affinity for much of the mineralization in the LCD (Stephens 2011).

Regarding IOCG deposits, Sillitoe (2003) noted, "The deposits…reveal evidence of an upward and outward zonation from magnetite-actinolite-apatite to specularite-chlorite-sericite and possess a Cu-Au-Co-Ni-As-Mo-LREE (light rare earth element) signature…" The high-grade mineralization at Superior is associated with magnetite-actinolite-tourmaline-apatite. At Moonlight, copper mineralization is associated with tourmaline-specularite-chlorite-sericite. During an April 2015 field visit to the LCD, Sillitoe categorized Engels, Lambs Ridge, Superior, and Moonlight as IOCG deposits (Crown Mining April 2015).

1.6 EXPLORATION

In 1961, Placer-Amex initiated modern exploration in the LCD with reconnaissance sampling, a magnetometer survey, geologic mapping and, in 1964 and 1965, an extensive stream sediment, rock and soil sampling program that covered approximately 10 sq mi of the LCD. Soil sampling produced six >1,000 ppm copper-in-soil anomalies and several other anomalies of lower magnitude. This work identified a number of exploration targets in the district, including what would become the Moonlight deposit.

Placer-Amex began exploration drilling in 1964 and carried on through November of 1970. They drilled 198,916 ft in 409 drillholes. More than 90% of the footage drilled tested the six >1,000 ppm copper-in-soil anomalies from the geochemical sampling program, and 85% of that was concentrated at Moonlight, and at the Superior and Engels mines. The Placer-Amex drilling program discovered and defined the Moonlight deposit and outlined a substantial Mineral Resource at Superior; however, several other anomalies in the district have probably not been fully tested. Subsequent drilling by Sheffield and its successors was confined to the Moonlight deposit, Superior and Engels.

In 1965 and 1966, Placer-Amex followed up their soil sampling program with several Induced Polarization (IP)-Resistivity surveys over the most promising soil anomalies. The survey was conducted by Heinrichs Geoexploration Company (HGC) of Tucson, Arizona. HGC's conclusions recommended follow-up drilling at several targets including Moonlight. In 1969, Placer-Amex contracted an airborne magnetic and gamma ray survey over the LSC. Placer-Amex regarded the results as inconclusive. Finally, in 1970, Placer-Amex contracted McPhar Geophysics to run IP-Resistivity surveys on Gossan Ridge, southwest of Moonlight.

In 2009, Garry Carlson of Gradient Geophysics reviewed the existing geophysical data and recommended an airborne EM survey, a Deep IP-Resistivity survey and a Controlled-source Audio-frequency Magnetotellurics (CSAMT) survey. The Deep IP-Resistivity and the CSAMT surveys were never done, but in 2010 Starfield contracted Fugro Airborne Surveys (Furgo) to conduct a property-wide airborne EM-Magnetics survey. It is this author's understanding that, to date, the results of the Fugro airborne survey have not been applied in a systematic way to exploration of the LCD.

1.7 DRILLING

Between 1964 and 1970 Placer-Amex drilled 198,916 ft in 409 drillholes. Most were diamond drillholes using a combination of NX and BX core. However, 3,550 ft of reverse circulation drilling at the Superior deposit is included in the total. For the most part, drilling was focused on six areas containing anomalous copper in soils. Only 49 drillholes totaling 17,034 ft tested other areas of the LCD. Drilling included 133 holes at the Superior deposit, 28 holes on Lamb's Ridge, and 10 holes at the Engels Mine. Beginning in 1966, and continuing through 1970, Placer-Amex drilled 99,436 ft of NX and BX core in 199 holes at the Moonlight deposit (Placer-Amex 1972).

In 2005 and 2006 Sheffield drilled 11,135 ft of HQ core in 14 holes on the Moonlight deposit, all but two of which were angle holes. Sheffield's drilling was designed primarily to confirm the reliability of Placer-Amex copper grades and to test the lateral continuity of mineralization. In addition, Sheffield hoped to understand controls on mineralization, derive an accurate tonnage factor, and expand the limits of the deposit. In 2007, Sheffield concentrated their drilling program at the Engels Mine, drilling 32 holes totaling 7,613 ft; however, they also drilled 1,390 ft in 15 reverse circulation holes at the Moonlight deposit to test the copper oxide potential of the deposit.

Sheffield was acquired by Nevoro Copper in July 2008. In the fall of 2008, besides drilling 4,071 ft in 12 holes at Engels, Nevoro Copper completed 2,603 ft in 7 vertical core holes at the Moonlight deposit. The Nevoro Copper holes were designed to twin selected Placer-Amex holes and were the last holes drilled at Moonlight. They are not included in the current Mineral Resource estimate but the results were reviewed as part of Placer-Amex data validation. Starfield, the successor to Nevoro, drilled an additional seven holes at Engels in 2009 and 2010.

1.8 SAMPLE PREPARATION, ANALYSIS AND SECURITY

The copper deposits of the LCD have seen two major exploration campaigns separated by a 35-year hiatus. Placer-Amex explored the district from 1964 to 1970 drilling nearly 200,000 ft of core. From 2005 through 2010, Sheffield and its successors, Nevoro Copper and Starfield, drilled 28,884 ft in 87 holes, including 15,128 ft in 36 holes at the Moonlight deposit. Placer-Amex results comprise the majority of the Moonlight deposit data upon which the current Mineral Resource estimate is based.

Placer-Amex initially assayed drill core for copper at their facility at the Golden Sunlight Mine in Montana. In mid-1967, Placer-Amex geologists realized that assay results from the Golden Sunlight Mine were unreliable and instituted a re-assay program using Union Assay Laboratory (Union Assay) in Salt Lake City. Gold and silver were also routinely assayed using 100 ft long composites. Union Assay ceased operations in the late 1990s and, in the intervening years, supporting information such as assay certificates for drill results reported by Placer-Amex appear to have been lost. Neither the Placer-Amex Summary Report from 1972 or Robert Wetzel in his 2009 report discuss the details of sample handling, sample preparation, quality assurance (QA)/quality control (QC) procedures or analytical methods for the Placer-Amex LCD drilling program. Although these procedures are not available for review, the authors assume that work done by employees of Placer-Amex, a well-known international mining company at the time, was done in accordance with best practices of the time.

The Sheffield/Nevoro/Starfield programs were designed in large part to support the credibility of the assay results reported by Placer-Amex. Sheffield's 2005-2006 program appears to have been conducted according to current industry best practices; QA/QC results for copper in this drilling campaign are acceptable. Results from Nevoro's 2008 drilling are nearly identical with the twinned Placer-Amex holes.

1.9 DATA VERIFICATION

Donald E. Cameron (CRC), an independent Qualified Person (QP) for this report, undertook steps to verify and validate the Project information. These steps included an extensive review of the historical drillhole database, drill logs, QA/QC information and information gaps, a two-day site visit, and selection of outcrops and drill core for detailed inspections.

CRC was able to verify the locations of several drillhole collars from both historic Placer-Amex and Sheffield campaigns, the presence of copper, and controls of copper mineralization in surface outcrops and prospect pits. CRC personally supervised sawing of two intervals from remaining Sheffield core and submitted the quartered core for assay at Bureau Veritas's Sparks, Nevada laboratory. Assays from the Sheffield drillhole intervals returned results very close to the ones in the database. Specific gravity (SG) determinations conducted at the lab for these two samples also fell within the range of historic Sheffield results.

CRC checked copper, gold, and silver assays from approximately 7% of the Placer-Amex portion of the database against scans of paper, originally handwritten drillhole logs, and determined error rates of 4% for copper and silver, with most errors minor. No certificates for the Placer-Amex drilling, nor any collar survey information could be recovered from existing records. CRC notes that gold and silver were generally assayed by Placer-Amex as 100 ft composites of ten individual 10 ft sample intervals. In those days, these were normally prepared by removing a set amount of material from each pulp envelope to make the composite pulp. At some point these composites were decomposed in the electronic database to 10 ft intervals, the nominal copper assay support, and combined into a single table with copper. Neither practice is considered Best Practice today, and additionally, the fire assay with gravimetric finish used by Placer-Amex for gold was generally not sensitive enough for the low levels of this metal in the deposit.

Sheffield assaying is supported by stored core, certificates showing appropriate methods for copper, silver, and gold, and by a QA/QC program. CRC's opinion is that the Sheffield program was largely in compliance with current Exploration Best Practices recommended by the Canadian Institute for Mining, Metallurgy and Petroleum (CIM) and suitable for estimation of Mineral Resources under Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines (2003). Based on this finding, copper and silver assays from Placer-Amex drill campaigns are also suitable for use in the estimations since findings summarized in Section 11.0 of this report, which include a review of Sheffield twin hole copper assay results and comparisons of Sheffield angled and Placer-Amex vertical holes, show that assays from Sheffield drilling confirm Placer-Amex results for copper and gold.

1.10 Mineral Processing and Metallurgical Testing

Several metallurgical test programs were conducted before 2017. The early work was mainly focused on heap leach processing although some test work had been conducted using flotation to concentrate the copper minerals. In 2017, Crown Mining undertook a metallurgical test work program for the Project to confirm previously completed test work and to confirm effective flotation reagent conditions and demonstrate the recoveries and concentrate quality that can be achieved with the tested material. Further tests on the samples that are better representative to the mineralization should be conducted.

Crown Mining provided material identified as Moonlight Sulfide, Moonlight Oxide, and Superior Sulfide. Baseline conditions were developed based on previously completed test work so the results would be comparable. The scope of the test work program included sample characterization, grinding tests, and batch flotation work that included both rougher and cleaner testing.

The test work results identified that a good copper concentrate grade containing potential precious metal credits can be expected. The results appear to suggest the potential need for a regrind mill. As chalcopyrite tends to be harder and floats at a coarser size with associated gangues, the regrind is anticipated to improve the target mineral liberation and remove any entrained particles. The grindability test results show that the Bond ball work index for the three samples ranged from 18.1 to 21.3 kWh/st, indicating that these materials should be very resistant to ball mill grinding.

1.11 Mineral Resource Estimates

An updated Mineral Resource estimate of copper, silver, and gold for the Moonlight copper deposit is presented here with an effective date of December 15, 2017. The Mineral Resource estimate incorporates updated geologic interpretations and compilation of a drillhole database from historical diamond drilling campaigns by Placer-Amex from 1966–1970 and by Sheffield from 2005–2006.

Drill spacing over the deposit is 300 ft by 300 ft in a fairly regular grid of vertical holes drilled by Placer-Amex. The Sheffield drilling comprises angled holes from platforms situated in key areas of mineralization that were apparently drilled to confirm results from the earlier Placer-Amex campaigns.

The principal deposit host is a quartz-monzonite stock that is mineralized with fracture-controlled quartz-magnetite and quartz-tourmaline veinlets carrying copper sulfide minerals. Sulfide percentages are low; surface oxidation has affected only the top of the mineralized zone and locally is not strongly apparent. Copper is concentrated in north-westerly-trending lenses that are aligned north-northeast. Low values of silver and traces of gold occur in the zones of strong copper mineralization.

CRC prepared geostatistical estimates for copper, the metal with most potential economic value, silver and gold and a block model comprising 100 ft by 100 ft blocks with block heights of 50 ft corresponding to the proposed bench height. CRC prepared 25 ft-long downhole composites for copper. Copper estimates employed lithology and redox domains and a 0.1% copper shell constraint. Raw metal values were capped by domain, as determined by decile analysis and probability plots. Directional and omnidirectional correlograms determined for copper and silver were used in ordinary kriging estimates of grade in two passes for each domain, except for the metavolcanics where the silver estimate method was by inverse distance methods. Gold estimates were generated in a single pass by anisotropic inverse distance methods using Sheffield data only; non-estimated blocks were assigned a low default value. A single specific gravity of 2.67 (approximately equal to 12.0 cu ft/st) was assigned to each block.

Checks of estimate statistics, graphical comparison of drillhole composites with block grades and domain coding, checks of global and local bias, and for appropriate change-of-support validate the block estimates.

Classification of Mineral Resources conforms to CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM 2014). Blocks are pre-classified based on estimation confidence criteria, primarily estimation pass and average composite distance to each block. The pre-class values are contoured to smooth edges and eliminate inliers and outliers of small groups of blocks with different pre-class values than their surroundings. The mining method for the Moonlight deposit will be open pit, for which pit optimizations discussed in Section 16.0, form the basis for demonstration of “reasonable prospects for economic extraction” of the Mineral Resources. Classified Mineral Resources, presented in Table 1.2 are contained within the PEA pit.

Table 1.2 Moonlight Mineral Resources as of December 15, 2017[1,2,3,4,5]

Class	Tons ('000 st)	Cu (%)	Au (oz/st)	Ag (oz/st)	Cu ('000 st)	Au ('000 oz)	Ag ('000 oz)
Indicated	252,000	0.25	0.0001	0.07	636	18	18,400
Inferred	109,000	0.24	0.0001	0.08	267	9	9,000

Notes: [1]The QP for the Mineral Resource estimate is Donald E. Cameron, Registered Geologist, Society of Mining Engineers (SME).
[2]Rounding as required by reporting guidelines may result in apparent differences between tons, grade, and contained metal content.
[3]Mineral Resources are reported above a US$6.25 net smelter return (NSR) cut-off (NSR = 44.08*Cu + .348*31.10348*Ag) and within a conceptual pit shell using copper, gold, and silver prices of US$3.00/lb, US$1,275/oz, and US$17.50/oz, respectively, and preliminary operating costs as of the effective date of this Mineral Resource.
[4]Effective date of the Mineral Resource estimate is December 15, 2017.
[5]There is no assurance that Mineral Resources will be converted into Mineral Reserves. Mineral Resources are subject to Modifying Factors and inclusion in a mine plan that demonstrates economics and feasibility of extraction in order to be considered Mineral Reserves.

There are no Measured Resources or Mineral Reserves for the Moonlight deposit. All Mineral Resources are fresh material; oxidized material is treated as waste in the pit optimization and has been excluded from Mineral Resources. Moonlight Mineral Resources are moderately sensitive to the selection of the reporting cut-off grade.

There is no assurance that Mineral Resources will be converted into Mineral Reserves. Mineral Resources are subject to Modifying Factors and inclusion in a mine plan that demonstrates economics and feasibility of extraction in order to be considered Mineral Reserves.

Estimates for some Mineral Resources rely on historical data which cannot be verified without re-sampling. Certain weaknesses and deficiencies have been identified in the drillhole database that should be addressed with the work program presented in this report. No production records with which to validate the estimates are available since the Moonlight deposit has no mining history.

The current estimates of Mineral Resources differ from historical estimates with respect to the drillhole database, estimation plan and methods, and the methods used to classify mineralization. The current estimate includes a test for reasonable prospects of economic extraction by constraint with an optimized pit. Partially as a result of this, Mineral Resources are reduced from the most recent historical estimate (Cavey and Giroux 2007).

The QP is of the opinion that estimation of Mineral Resources for the Moonlight copper deposit has been performed to Estimation of Mineral Resources and Mineral Reserves Best Practices (CIM 2003), and reporting conforms to the requirements of CIM Definition Standards (CIM 2014).

Discussion of environmental, permitting, legal, title, taxation, socio-economic, marketing, political and other relevant factors that could materially affect the Mineral Resource estimates are included in Sections 18 to 20, and 22.

1.12 MINING METHODS

All mining for the Moonlight deposit will be conducted utilizing conventional open pit mining methods with drill and blast, followed by load and haul with large diesel truck and shovel equipment.

Limited information on rock quality is available for mine planning, and as such Tetra Tech has assumed the bulk of the mining will be hard rock excavation, which requires drill and blast.

The preliminary mine plan includes inferred Mineral Resources.

An open pit optimization was completed prior to developing an open pit mine design. The optimization used the Lerchs-Grossman algorithm in the GEOVIA Whittle™ software package. Pit optimization was based on economic parameters including but not limited to:

- process throughput of 60,000 st/d
- copper price of US$3.00/lb.
- silver price of US$17.50/oz
- copper recovery of 86%
- mining cost of US$1.25/st mined
- processing cost of US$6.00/st milled
- general and administrative (G&A) cost of US$0.25/st milled
- maximum slope angle of 45°.

Based on the results of the pit optimization, interim and final pit shells were selected and subjected to pit design. Pit design parameters included 45° slopes and the use of 100 ft wide haul roads to allow for due lane traffic.

The ultimate pit and contained mineralization are summarized Figure 1.3 and Table 1.3.

The detailed mine design final pit result is shown in Figure 1.3

Figure 1.3 LOM Pit Design Result (Plan Review)

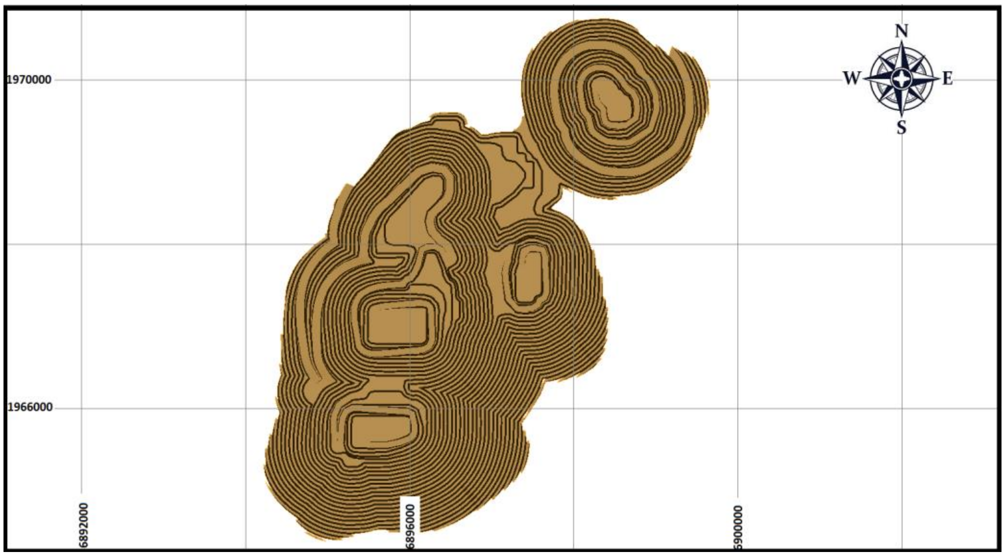


Table 1.3 highlights the tons and grades of the material extracted from the engineered designed pit.

Table 1.3 Designed Pit Results

Item	Units	Results
Mineralized Material	million st	365
Diluted Copper Grade	%	0.25
Contained Copper	'000 st	912
Diluted Silver Grade	oz/st	0.08
Contained Silver	million oz	27
Waste	million st	286
Total Material	million st	651
Strip Ratio	st:st	0.78

Preliminary estimates of drilling and blasting requirements were completed, including mining equipment fleet and support equipment fleet. Cycle times for haul fleet were estimated using Runge TALPAC™, which were subsequently used to estimate mining costs.

A preliminary mining schedule was prepared using the Geovia Whittle™ Milawa algorithm. This produced a schedule with mill feed of 365 million st will be mined, along with 286 million st of waste rock.

For the PEA, waste rock piles were placed as close to the pit as possible to facilitate subsequent backfilling of the pit at the end of the mine life.

Tetra Tech has estimated that 177 people will be require for staffing the mining operation.

1.13 RECOVERY METHODS

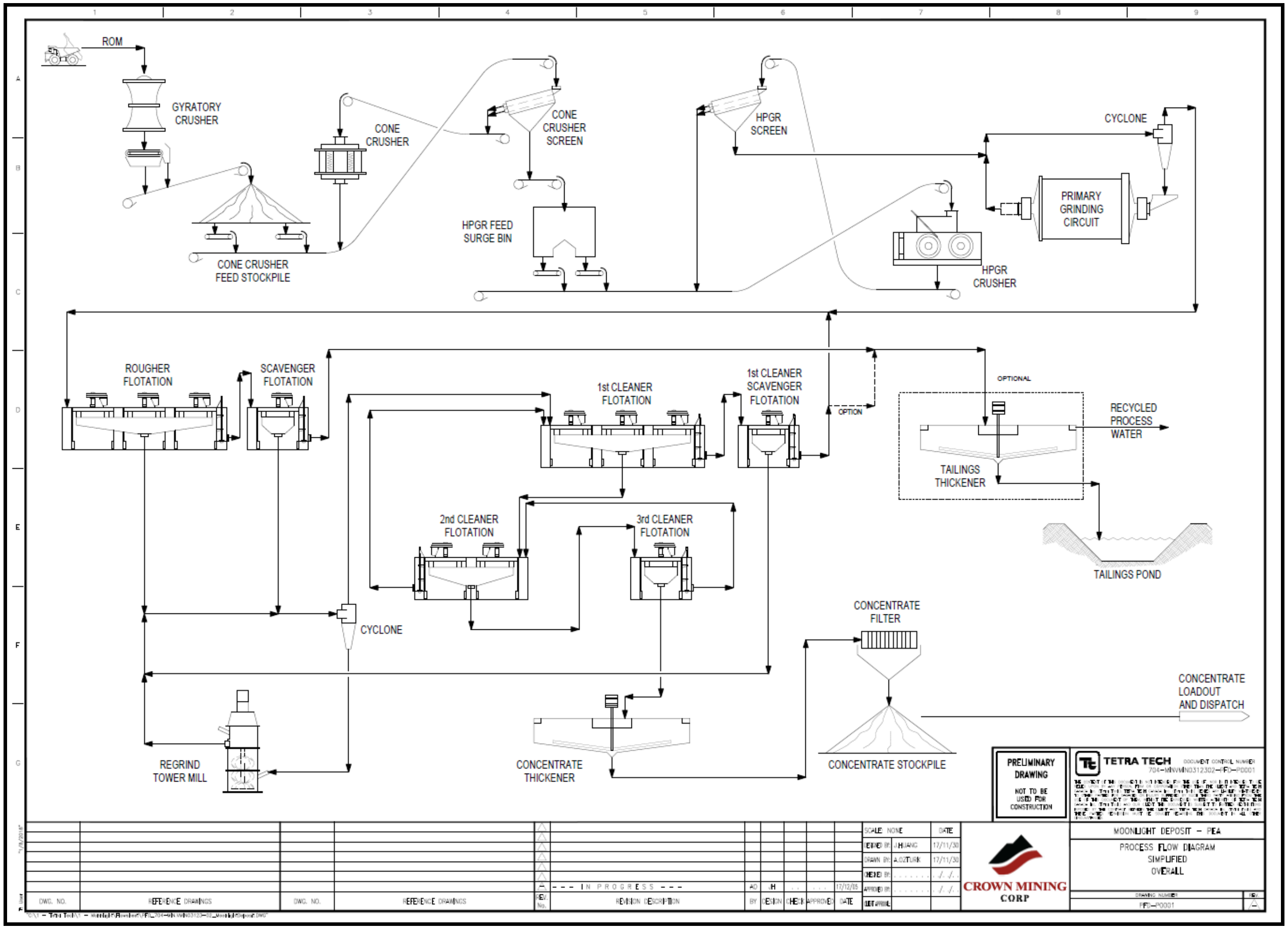
Tetra Tech designed a 60,000 st/d processing plant for the Project, to process the porphyry mineralization containing copper and associated precious metal credits. The processing plant will operate in two, 12 h shifts per day, 365 d/a and will process mineralized material at an annual rate of 21.9 million st. The primary crushing plant availability will be 70%. Secondary crushing (two in operation and one stand by), tertiary crushing, grinding, and flotation plant availability will be 92%.

The mill feed will be crushed by gyratory crusher to 80% passing 150 mm, followed by a cone crusher to 80% passing 45 mm, and then by high-pressure grinding roll (HPGR) to 80% passing 3.5 mm.

The crushed mineralized material will be conveyed to the grinding area and ground to 80% passing 110 µm in a ball mill grinding circuit. The ground material will be processed using copper rougher/scavenger flotation followed by copper rougher/scavenger concentrate regrinding. The reground copper rougher/scavenger flotation concentrate will then be further upgraded by three stages of cleaner flotation. The cleaner scavenger tailings will be recycled backed to the rougher flotation circuit for reprocessing. Copper rougher/scavenger flotation tailings will be delivered to the tailings management facility (TMF). The third cleaner flotation concentrate, which will on average contain approximately 28% copper, will be thickened and then pressure-filtered before it is shipped to smelters.

Figure 1.4 illustrates the simplified process flowsheet for the Project.

Figure 1.4 Simplified Process Flowsheet



1.14 PROJECT INFRASTRUCTURE

The Property is approximately 1.5 mi from Diamond Mountain Road, a two-lane paved all-weather highway and is accessible through a network of existing forestry service roads. The site access road will require minimal upgrading to service the mine.

Figure 1.5 illustrates the overall Project site layout.

The process plant will house the ball mills, rougher flotation and cleaner flotation cells, regrind area, reagents area, concentrate surge tank, concentrate filter press, and laydown areas. There will be a mezzanine level above for the control room, offices, and electrical room. The concentrate thickener and water services will be located inside the process plant. A structure housing the concentrate stockpile and loadout will be located adjacent to the west side of the process plant. A fibre-optic backbone will be included throughout the plant in order to provide an ethernet-type system for voice, data, and control systems bandwidth requirements.

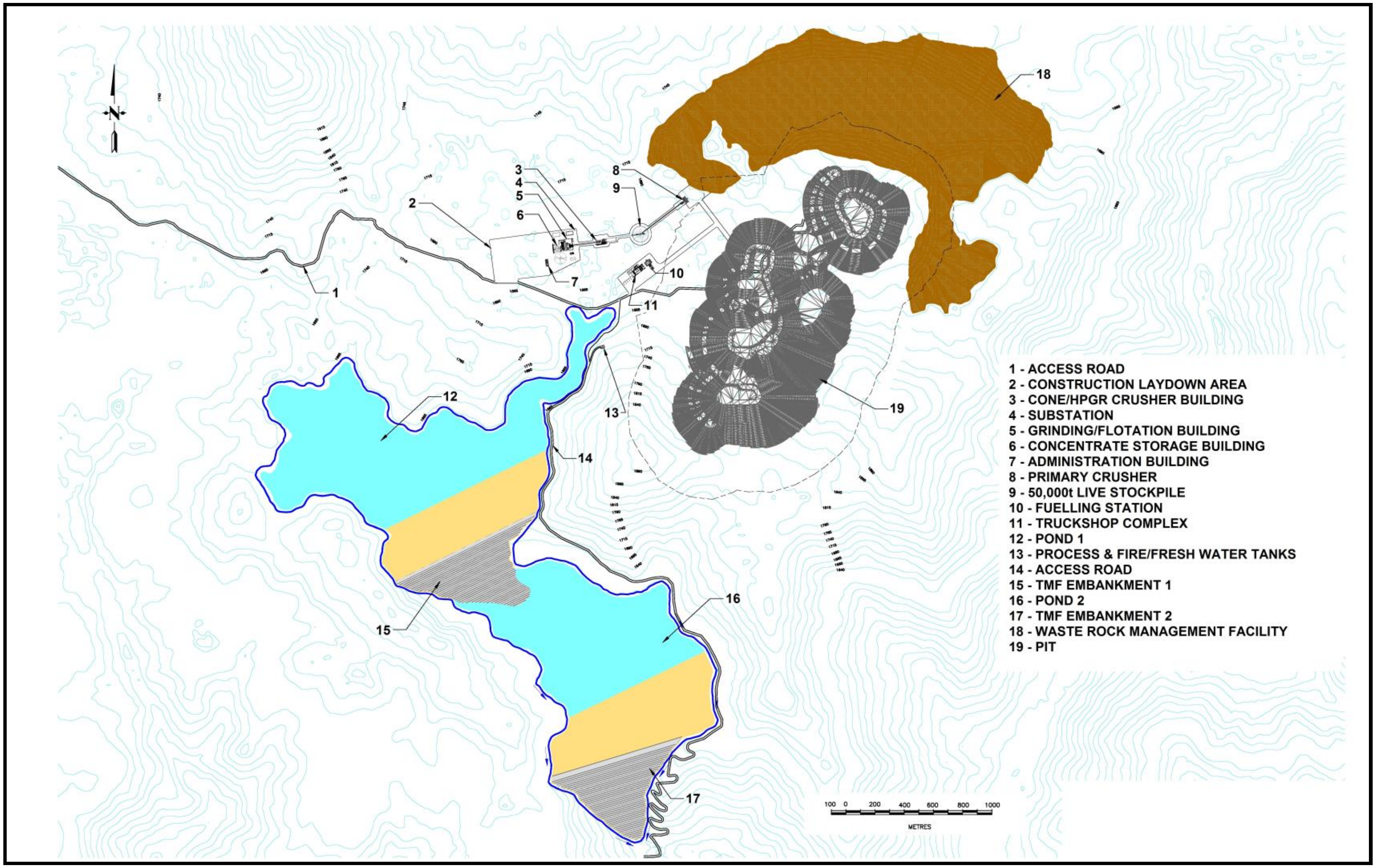
The administrative building will be a single-storey steel structure and will house the mine dry, lockers, shower facilities, first-aid, and emergency vehicle parking, in addition to office areas for management, administration, engineering, and geology personnel.

The maintenance/truck shop and warehouse (cold/warm) will house a wash bay, repair bays, parts storage areas, welding area, machines shop, electrical room, mechanical room, compressor room, and lube storage room. The facility will also house the cold/warm storage warehouse and support warehouse and maintenance personnel. The facility is designed to support both the mining haul fleet and the process plant fleet.

Fuel storage requirements for the mining equipment, process equipment, and ancillary facilities will be supplied from above-ground diesel fuel tanks located near the truck shop. A dedicated service truck will transport the fuel to the mining equipment and the process plant fleet.

The assay laboratory will be a single-story modular building complete with the required laboratory equipment for grade assaying and control. The laboratory will be equipped with all the required heating, ventilation, and air conditioning (HVAC) systems and chemical disposal equipment.

Figure 1.5 Moonlight-Superior Project Site Layout



1.14.1 Tailings Management Facility

The TMF is designed as a cross-valley type TMF concept, with embankments constructed of cyclone sand. Over the LOM, two dams (TMF1 and TMF2) will be constructed in the same valley to store tailings and process water. The proposed TMF dams will be located to the south and at a lower elevation than the proposed process plant location. The design will permit storage of approximately 315 million cu yd of tailings. The adopted embankment design geometry is 2.5H:1V downstream to suit typical stability and closure requirements.

Coarse tailings from the cyclone underflow will be deposited in cells and compacted as part of embankment raising construction by centreline methods. Fine tailings from the cyclone overflow will be deposited into the basin upstream of the embankment. This approach will optimize tailings storage capacity while reducing the risks associated with embankment stability and seepage. Tailings deposition will be undertaken to maintain the decant water return pond adjacent to the water return intake. Decant water will be returned to the process plant for re-use.

1.15 Environmental Studies, Permitting, and Social or Community Impact

There are no significant issues identified in this study pertaining to environmental conditions, permitting or social/community impact. Limited baseline sampling shows that applicable standards for water quality are not exceeded in the main stems of water bodies draining pre-existing historical mining impacts. Acid-base Accounting (ABA) indicates that neither the tailings produced from flotation testing nor existing tailings are acid producing.

Permitting a new mine in the area is not precluded by existing legislation or land use regulations. A full environmental review under both the state and federal regulations is anticipated as land management and ownership is divided between private and federal entities. As with most mines, transportation, air quality, waste disposal, and water quality will likely constitute the primary areas of focus. California has a strict mining law that mandates pit backfilling of new open-pit mines if wastes are available.

The local economy is still dominated by resource use and extraction industries such as logging and ranching although recreation is a growing presence in the county. Community support for the Project will likely be mixed, with support from those residents interested in economic development and opposition from those who see no direct benefits. No serious outreach to the communities of interest has been undertaken so the full extent of support or opposition is not quantified.

1.16 CAPITAL AND OPERATING COST ESTIMATES

1.16.1 CAPITAL COST ESTIMATE

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is US$512.9 million. A summary breakdown of the initial capital cost is provided in Table 1.4. This total includes all direct costs, indirect costs, Owner's costs, and contingency. All costs are shown in US dollars and the accuracy range of the estimate is ±35%.

This estimate has been prepared with a base date of Q4 2017 and does not include any escalation past this date. Where applicable, the quotations used in this estimate were obtained in Q4 2017 and are budgetary and non-binding.

Table 1.4 Initial Capital Cost Summary

Area		Cost (US$ million)
Direct Costs		
10	Overall Site	40.4
30	Mining (excludes leased equipment)	15.6
40	Processing	234.0
50	TMF	12.2
70	On-site Infrastructure	22.6
Direct Cost Subtotal		**324.8**
Indirect Costs		
X	Project Indirect Costs	105.0
Y	Owner's Costs	12.4
Z	Contingency	70.7
Indirect Cost Subtotal		**188.1**
Total Initial Capital Cost		**512.9**

1.16.2 OPERATING COST ESTIMATE

On average, the LOM on-site operating cost for the Project are estimated to be $7.77/st of material processed. The operating costs are defined as the direct operating costs including mining, processing, surface services and G&A costs (Table 1.5). The expected accuracy range of the operating cost estimate is +35%/-25%.

Table 1.5 LOM Average Operating Cost Summary

Area	Cost (US$/st milled)
Mining	2.35
Process and TMF	4.77
G&A and Site Services	0.65
Total Operating Cost	**7.77**

1.17 ECONOMIC ANALYSIS

A PEA should not be considered a prefeasibility or feasibility study, as the economics and technical viability of the Project have not been demonstrated at this time. A PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results as reported in the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

An economic evaluation was prepared for the Project based on a pre-tax financial model. The following pre-tax financial parameters were calculated using the base case copper price of US$3.15/lb and silver credits of US$18.00/troy oz (revenue from gold was not included in the economic analysis):

- 16.4% IRR
- approximately 4.8-year payback on US$512.9 million initial capital
- US$237 million NPV at an 8% discount rate.

Crown Mining commissioned PricewaterhouseCoopers (PwC) in Vancouver, British Columbia to prepare a tax model for the post-tax economic evaluation of the Project, with the inclusion of applicable income and mining taxes (see Section 22.3 for further details).

The following post-tax financial results were calculated as:

- 14.6% IRR
- US$179 million NPV at an 8% discount rate
- The project has a negative net present value at a copper price of less than US$2.85/lb.

Sensitivity analyses were conducted to analyze the sensitivity of the Project merit measures (NPV, IRR, and payback periods) to changes in copper price, silver price, operating costs, and capital costs. The Project's pre-tax NPV, calculated at an 8% discount rate, is most sensitive to copper price followed by on-site operating costs and least sensitive to capital costs.

1.18 RECOMMENDATIONS

Tetra Tech recommends that Crown Mining proceed with the next phase of work to identify potential cost savings and additional revenue generating opportunities.

Table 1.6 outlines the estimated implementation costs of the recommendations described in this section.

Table 1.6 Estimated Implementation Costs of Selected Recommendations

Recommended Test Work or Study	Estimated Cost (US$)
Resource Development Drilling and Modeling	
Exploration Drilling	3,900,000
Resource Model Updates	200,000
Mining	
Preliminary Geotechnical Study	300,000
Pit Designs	150,000
Metallurgical Test Work	
Mineralogical Study and Flotation Optimization Tests	250,000
Crushability and Grindability	60,000
Bacterial Oxidation Heap Leaching	80,000
Tailings Sample Characterization	20,000
Recovery Methods	
Plant Design & Layout Optimization	200,000
Project Infrastructure	
Geotechnical Drilling Investigation	300,000
Construction Schedule Optimization and Equipment Modularization	50,000
Tailings Management Facility	
Site Investigation and Testing Programs	500,000
Environmental	
Air	150,000
Hydrology (surface water)	50,000
Water Quality	25,000
Groundwater Hydrology and Chemistry	150,000
Waste characterization	100,000
Meteorology	50,000
Biologic Baseline	50,000

A full list of recommendations is can be found in Section 26.0.

2.0 INTRODUCTION

Crown Mining commissioned a team of engineering consultants to complete this PEA in accordance with NI 43-101 Standards of Disclosure for Mineral Projects, NI 43-101 Companion Policy, and Form NI 43-101F1.

The following consultants contributed to this report:

- Tetra Tech: overall project management, project description and location, accessibility, mining, metallurgy, process, project infrastructure, tailings and waste rock management, water management, capital and operating cost estimates, and economic analysis.
- CRC: history, geological setting, deposit types, exploration, drilling, sample preparation, data verification, Mineral Resource estimate, and adjacent properties.
- Quatreface: environmental studies, permitting, and social or community impact.

The effective date of this study is March 2, 2018 and the effective date of the Moonlight deposit Mineral Resource estimate is December 15, 2017.

2.1 QUALIFIED PERSONS

A summary of the QPs responsible for this report is provided in Table 2.1. The following QPs conducted site visits of the Property:

- Hassan Ghaffari, P.Eng., completed a site visit on November 16, 2017.
- Mark Horan, P.Eng., completed a site visit on November 16, 2017.
- Donald E. Cameron, M.Sc., SME completed a site visit on September 26-27, 2017.

Table 2.1 Summary of QPs

Report Section		Company	QP
1.0	Summary	All	Sign-off by Section
2.0	Introduction	Tetra Tech	Hassan Ghaffari, P.Eng.
3.0	Reliance on Other Experts	Tetra Tech	Hassan Ghaffari, P.Eng.
4.0	Property Description and Location	Tetra Tech	Hassan Ghaffari, P.Eng.
5.0	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Tetra Tech	Hassan Ghaffari, P.Eng.
6.0	History	CRC	Donald E. Cameron, SME
7.0	Geological Setting and Mineralization	CRC	Donald E. Cameron, SME
8.0	Deposit Types	CRC	Donald E. Cameron, SME
9.0	Exploration	CRC	Donald E. Cameron, SME
10.0	Drilling	CRC	Donald E. Cameron, SME
11.0	Sample Preparation, Analyses and Security	CRC	Donald E. Cameron, SME
12.0	Data Verification	CRC	Donald E. Cameron, SME
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	Hassan Ghaffari, P.Eng.
14.0	Mineral Resource Estimates	CRC	Donald E. Cameron, SME
15.0	Mineral Reserve Estimates	Tetra Tech	Mark Horan, P.Eng.
16.0	Mining Methods	Tetra Tech	Mark Horan, P.Eng.
17.0	Recovery Methods	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
18.0	Infrastructure	-	-
	18.1 Introduction	Tetra Tech	Hassan Ghaffari, P.Eng.
	18.2 Access Roads	Tetra Tech	Hassan Ghaffari, P.Eng.
	18.3 Buildings	Tetra Tech	Hassan Ghaffari, P.Eng.
	18.4 Power Supply and Distribution	Tetra Tech	Hassan Ghaffari, P.Eng.
	18.5 Water Management	Tetra Tech	Hassan Ghaffari, P.Eng.
	18.6 Tailings Storage Facility	Tetra Tech	Chris Johns, P.Eng.
19.0	Market Studies and Contracts	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
20.0	Environmental Studies, Permitting and Social or Community Impact	Tetra Tech	Hassan Ghaffari, P.Eng.
21.0	Capital and Operating Costs	-	-
	21.1 Capital Cost Estimate	Tetra Tech	Hassan Ghaffari, P.Eng.
	21.2 Operating Cost Estimate	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng. Mark Horan, P.Eng.
22.0	Economic Analysis	Tetra Tech	Mark Horan, P.Eng.
23.0	Adjacent Properties	CRC	Donald E. Cameron, SME
24.0	Other Relevant Data and Information	Tetra Tech	Hassan Ghaffari, P.Eng.
25.0	Interpretation and Conclusions	All	Sign-off by Section
26.0	Recommendations	All	Sign-off by Section
27.0	References	All	Sign-off by Section

2.2 Sources of Information

All sources of information for this study are in Section 27.0.

2.3 Units of Measurement and Currency

All measurements are reported in US imperial units, unless otherwise noted.

All currency is reported in US dollars, unless otherwise noted.

3.0 RELIANCE ON OTHER EXPERTS

Tetra Tech followed standard professional procedures in preparing the contents of this report. Data used in this report has been verified where possible and Tetra Tech has no reason to believe that the data was not collected in a professional manner.

Technical data provided by Crown Mining for use by Tetra Tech in this PEA is the result of work conducted, supervised, and/or verified by Crown Mining consultants.

Tetra Tech has not independently verified the legal status or title of the claims or exploration permits, and has not investigated the legality of any of the underlying agreement(s) that may exist concerning the Property.

Mr. Hassan Ghaffari, P.Eng., relied on Mr. Dave Godlewski of Quatreface, for matters relating to the environmental permitting plan and social or community impact in Section 20.0.

Mark Horan, P.Eng., relied on PwC, who are experts that are not QPs, concerning tax matters relevant to this report. The reliance is based on a letter to Crown entitled "Preparation and review of the US Federal and California state corporate income tax portion of the Pro-Forma Conceptual Cash Model ("the Model") of Crown Mining Corp.'s ("Crown Mining") Moonlight Copper/Silver Project mine plan ("the Project")" and dated March 5, 2018. Tetra Tech has relied entirely on this letter for disclosure contained in Section 22.3. Tetra Tech believes that it is reasonable to rely on PwC, based on the assumption that PwC staff have the necessary education, professional designations and relevant experience in tax matters relevant to this study.

4.0 PROPERTY DESCRIPTION AND LOCATION

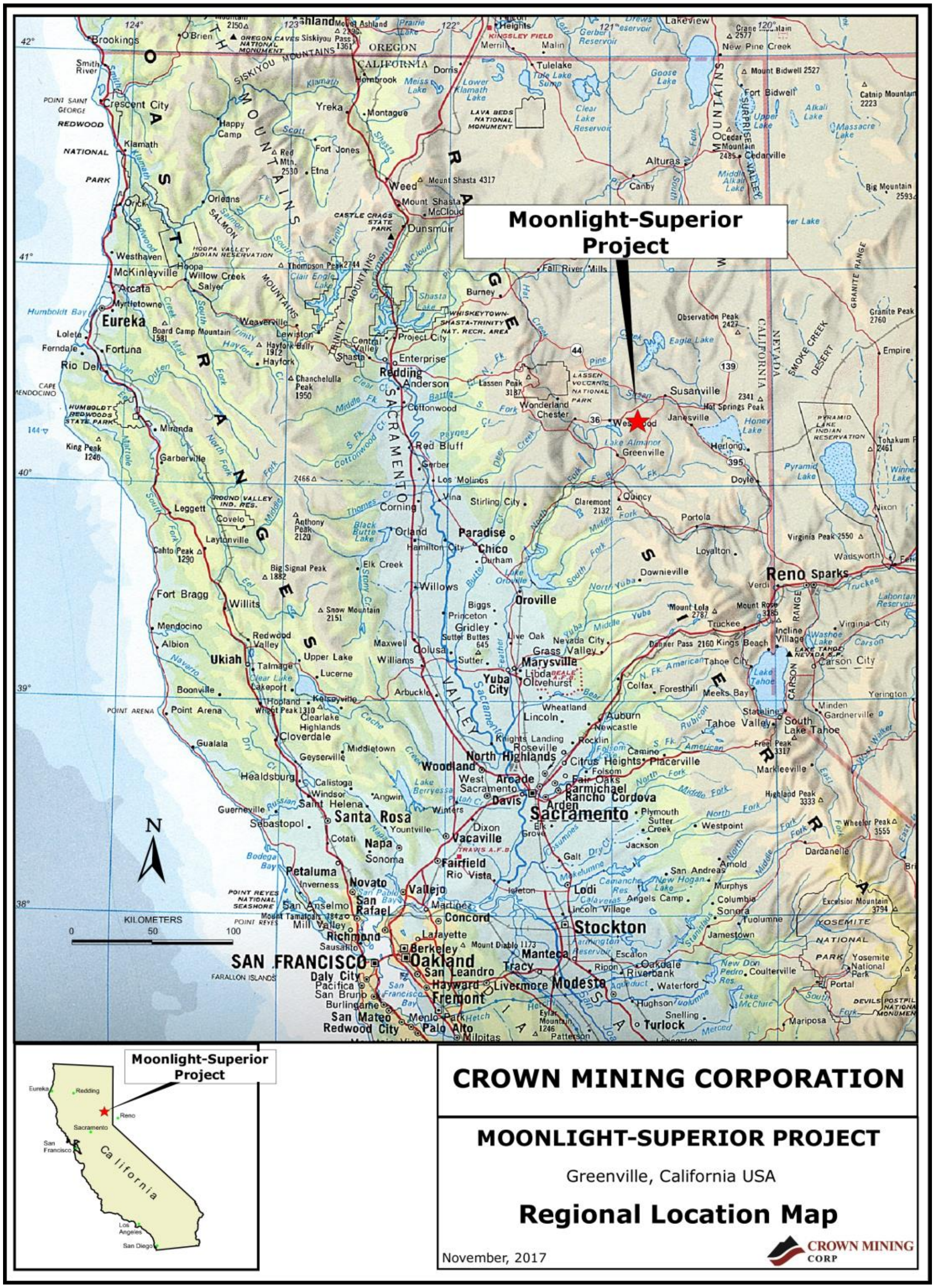
The bulk of the following property description was taken from the 2007 NI 43-101 report prepared by OreQuest Consultants Ltd. (OreQuest) for Sheffield (Cavey and Giroux 2007). Modifications consist of updates to reflect additional work done within the Property boundary, including claims staked and claims dropped, between 2007 and 2017.

The Moonlight deposit, the subject of this report, forms part of Crown Mining's Moonlight Superior Project, which covers the historic LCD and contains multiple areas of copper mineralization, including the Engels and Superior Mines.

Crown completed the purchase of a 100% undivided title interest in its Moonlight property located in Plumas county on March 13, 2018.

The Property is located approximately 10 mi northeast of Greenville, California and approximately 100 mi northwest of Reno, Nevada via Highways 395 and 70 (Figure 4.1). The Project location is shown on the Moonlight Peak and Kettle Rock 7.5' United States Geological Survey (USGS) topographic maps. The latitude at the approximate center of the Moonlight deposit is 40°13'36" N and the longitude is 120°48'11" W or State Plane Coordinates of 6,896,600 E, 1,967,700 N (NAD_1983_2011_StatePlane_California_I_FIPS_0401_Ft_US). The Property lies within Sections 1, 2, 11 12, 13, 14, and 24 T27N R10E; Sections 4, 5, 6, 7, ,8, 9, 17, and 18 T27N, R11E; Sections 35 and 36 T28N, R10E; and Sections 31 and 32 T28N, R11E in Plumas County, California.

Figure 4.1 Regional Location Map



The Project covers an area of approximately 6,822 ac, and consists of the following mineral claims:

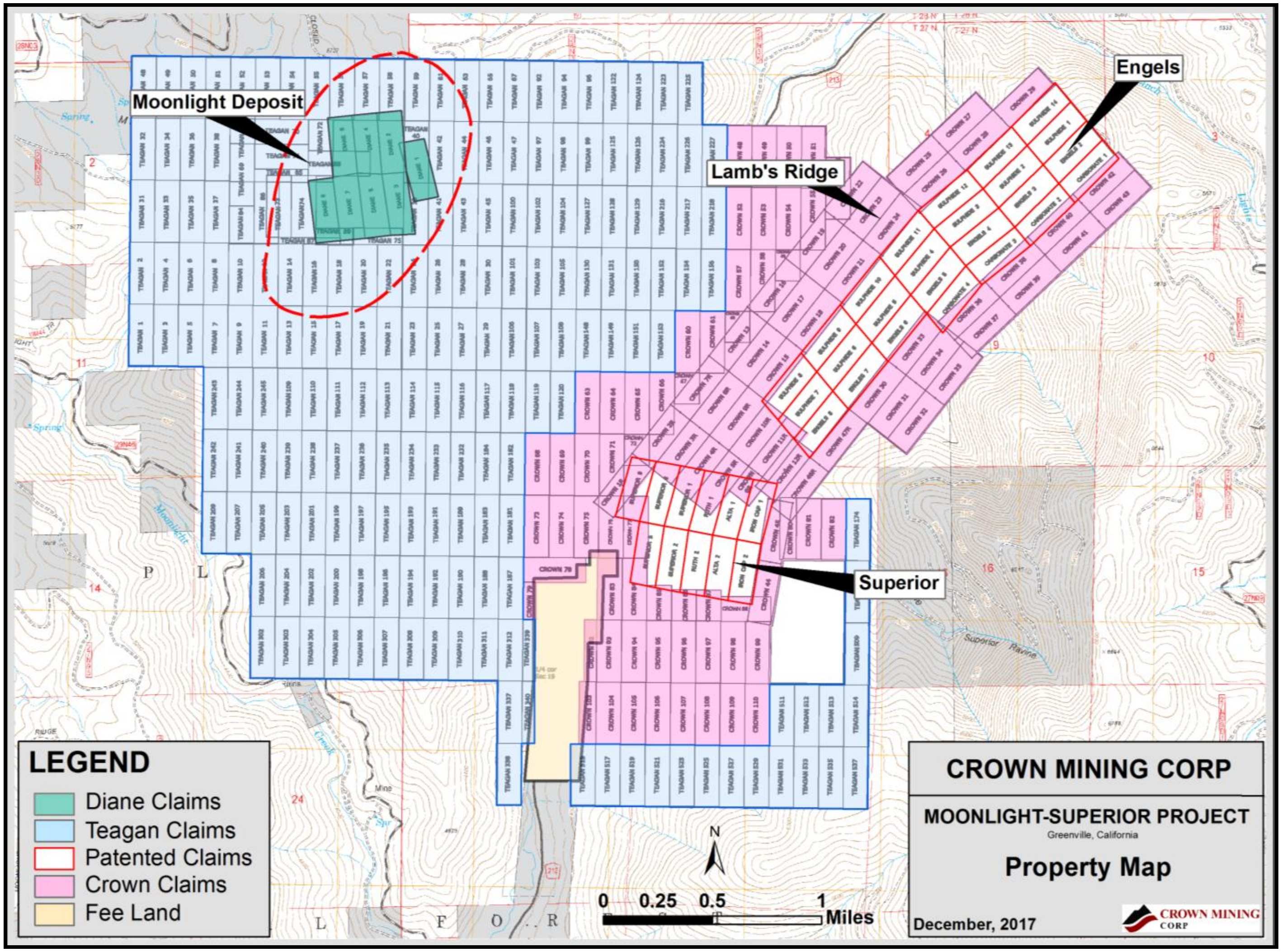
- the Diane claims: eight unsurveyed, unpatented, contiguous optioned mining lode claims
- the Crown claims: 110 surveyed, unpatented lode claims
- the Teagan claims: 204 unsurveyed, unpatented, optioned lode claims.
- patented claims: 36 patented lode mineral claims

The Moonlight deposit claim block, as optioned in 2016 from Canyon Copper, originally consisted of 289 Teagan claims and 8 Diane claims. In accordance with the option agreement, Crown Mining dropped 85 Teagan claims. A summary of claims groups and associated acreage follows in Table 4.1. Claims are shown on Figure 4.2.

Table 4.1 Claims Summary and Acreages

Mineral Claims and Mineral Leases	Acreage
Diane 1-8	164.09
Teagan 1-75, 83-89, 92, 94, 96-120, 122, 124-131, 148-156, 181-184,207,209, 216-218, 223-227, 232-245, 302-312, 337-340 accounting for overlap with Diane claims and fee land and reduced size claims.	3,604.92
Teagan 174, 508, 509, 511-515, 517, 519, 521, 523, 525, 527, 529, 531, 533, 535, 537	392.28
Patented claims 36	734.58
Fee land	162.12
Crown 1-110 accounting for overlap with patents, fee land and contiguous Crown claims	1,764.05
Total	**6,822.04**

Figure 4.2 Moonlight-Superior Project Property Map



There are no old mine waste dumps known to exist on the Property that may present a potential environmental liability. A flooded shaft of unknown depth and an old exploration adit exist on the claims. The adit is reported to have collapsed and does not present a problem at this time. The QP did not observe the old adit. The adjacent California-Engel option (Engels and Superior mine sites) does contain old mine waste dumps, trenches, small open pits, shafts, abandoned mill foundations, and other mining openings.

Exploration and mining can be conducted year-round, due to the established road and its proximity to infrastructure. Existing roads and drill sites that date from exploration conducted in the 1960s and 1970s are present. They were used in the first Sheffield drilling program and most are still passable. In addition, at least two currently maintained US Forest Service (USFS) roads traverse the Property.

The Property is large enough to support all future exploration or mining operations, including facilities and potential waste disposal areas. Potential process plant sites may have to be located closer to water. Controlling the mineral rights under valid lode claims will not fully entitle Crown Mining to develop a mine. Permitting will need to be carefully planned and executed to be sustainable in the community and this area of California. Proper communication and public relations with local communities, USFS personnel, and county and state officials can minimize the impact of environmental groups on development at the Property.

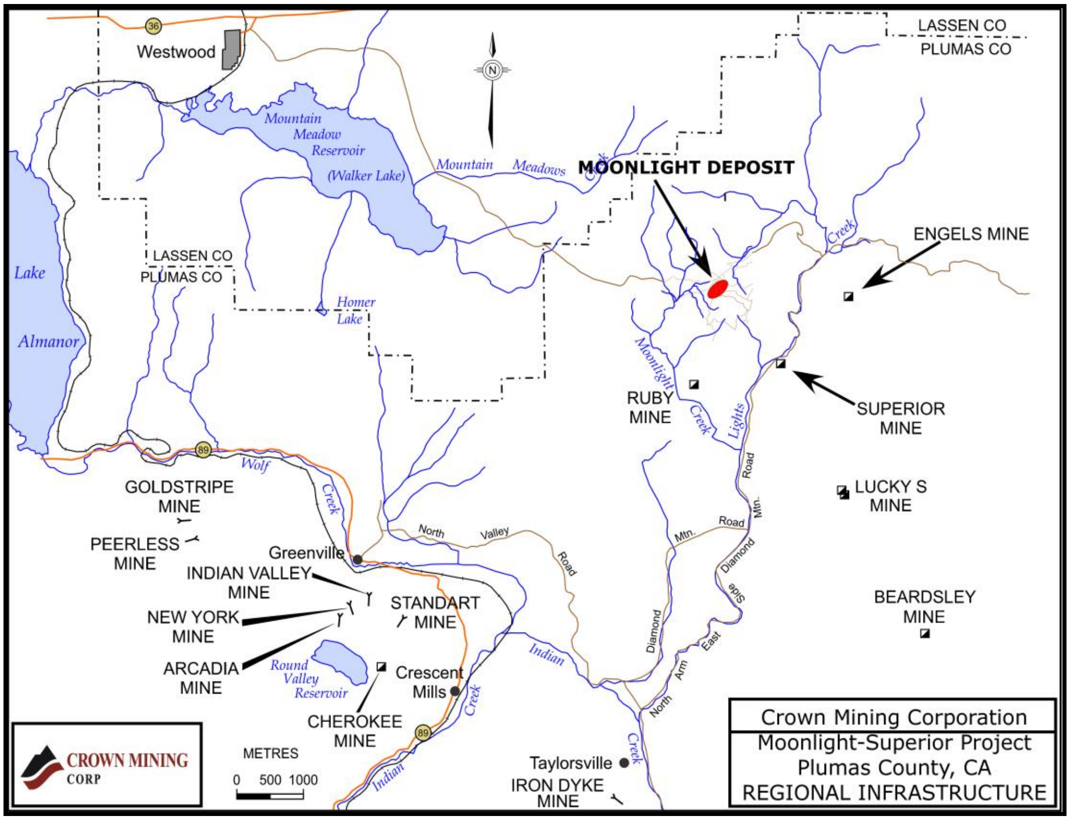
5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Property is accessed from the Reno Nevada International Airport by taking US Interstate 395 northwest for approximately 85 mi to the town of Susanville, California. At the town, turn south onto State Highway 36 towards the town of Westwood for approximately 18.6 mi to a secondary road heading south (approximately 2.2 mi east of Westwood). The western most edge of the Moonlight claim blocks is approximately 12.6 mi from the turnoff of Highway 36 via a series of gravel roads, many of which are actively used by logging companies operating east of Crown Mining's claim block. The access is good across the current Project ground utilizing active USFS roads and many old drill access roads completed by Placer-Amex in the 1960s and 1970s. No homes are located on the Property. One ranch and home is located approximately 7.4 mi west-northwest on the western Property border. The regional infrastructure is illustrated in Figure 5.1.

The Project is situated in the Sierra Nevada province of California, characterized by north-northwest trending mountain ranges separated by alluvial filled valleys. The claims vary in elevation from a low of approximately 3,760 ft (1,146 m) along Lights Creek at the southern edge of the Project area, to a high of approximately 6,621 ft (2,018 m) on a peak in the southeast corner of the claim block. In the Moonlight deposit area, elevations vary from a low of approximately 5,520 ft (1,682 m) in the Moonlight Valley, just west of the deposit. Outside the claims to the northwest of the claim block, elevation rises steeply to Moonlight Peak where elevations reach approximately 6,828 ft (2,081 m).

There are a few bedrock exposures on the Property and only a thin soil development on the upland portion of the blocks. The Moonlight Valley floor has virtually no bedrock exposure.

Figure 5.1 Regional Infrastructure Map



The climate is defined by hot summers to a maximum of 100°F and cold, windy winters with lows to -10°F. Precipitation is moderately light with an average rainfall of 30 in and an average snowfall of approximately 140 in. Spring and autumn months are moderate in temperature. The vegetation varies depending on elevation and moisture. Cedar, lodgepole pine, mountain mahogany, and juniper grow on the slopes of the Project ground. The Project area is fairly dry with numerous small dry drainages scattered throughout the claim block. Water will need to be trucked in during drilling phases. The Mountain Meadows Reservoir is located approximately six miles to the west-northwest of the Property and could supply water for all advanced exploration activities on the Property.

The area is serviced by Pacific Gas & Electric Company (PG&E) and significant high-tension power lines lie close to the Project ground and parallel Highway 36. The nearest rail line is the Western Pacific that runs through the town of Westwood, approximately 15 road miles to the west of the Property. International air services are located in Reno, approximately 85 mi southeast of Susanville. The closest deep-water port is Sacramento, which is located approximately 150 mi to the southwest.

A highly-trained mining-industrial workforce is available in the Carlin-Elko area of northern Nevada, which is located approximately 250 road miles from the Project area. All the needed equipment, supplies, and services for mining companies to conduct full exploration and mining development projects is available in Carlin, Elko, or Reno.

Exploration and mining could be conducted year-round, due to the established roads and the Project's proximity to nearby towns. The Property has sufficient surface rights for future exploration or mining operations, although there may be a requirement to lease nearby flat land available within a 6 mi radius for potential waste disposal areas, heap leach pads areas, and potential process plant sites.

6.0 HISTORY

Gold was discovered in Plumas County in 1850. Copper deposits were noted but were not exploited until the American Civil War (1861-1865) when a smelter was built in Genesee Valley. Copper was mined and shipped from the LCD during this period. Henry A. Engels and sons acquired the Superior Mine in 1880 and discovered the Engels Mine in 1883. The Engels Mine is located approximately 3 mi east of the Moonlight deposit. The Superior Mine is approximately 2.2 mi southeast of the Moonlight deposit. Both are now a part of the Crown Mining claim and lease holdings. Both mines shut down in 1930. Since that time there have been sporadic periods of exploration activity.

Copper production from Plumas County began in 1863 at the Cosmopolitan Mine (later patented as the Reward Mine) (N. Lamb, pers. comm. 2018) located approximately 10 mi south of what would be the Engels Mine. Operations at the Superior and Engels mines began in 1890. The main period of production began in 1915 and lasted until 1930 when mining activities were halted at both operations.

Historic production from the Superior and Engels mines was summarized as follows by Robert Wetzel (2009):

> *The Engels and Superior Mines have reported joint production of about 161.5 million pounds of copper, 23,000 ounces of gold and 1.9 million ounces of silver recovered from 4.7 million tons of ore between 1914 and 1930. (Lamb, 2006) Mill recovery averaged about 80% during this period of operation, indicating a feed grade of about 2.2% copper and 0.5opt* [ounces per ton] *Ag and 0.005 opt* [ounces per ton] *Au.*

Elsewhere in the Plumas Copper Belt, the Walker Mine, discovered in 1905 and located approximately 12 mi southeast of the Property, is reported to have produced approximately 168 million lb of copper, 180,000 oz of gold and 3.6 million oz of silver from 5.3 million st of ore from 1916-1941 (Wetzel 2009).

The Engels and Superior mines were jointly operated by California-Engels between 1915-1930. Approximately 60% of the production came from the Engels ore body. The ore was processed in the first all flotation mill for copper in the US, which operated from 1915-1930 at approximately 1,200 st/d (N. Lamb, pers. comm. 2018).

The following paragraphs summarize the exploration history of the Moonlight prospect. It is based on the Placer-Amex 1972 Summary Report by Leonid Bryner (Bryner 1972), Cavey and Giroux's 2007 Technical Report (Cavey and Giroux 2007), and Robert Wetzel's 2009 report (Wetzel 2009).

In 1953-54, Newmont Mining Co. completed a preliminary aerial geologic map of the Lights Creek area. Phelps-Dodge looked at the district in the early 1960s. In 1961, Placer-Amex, a subsidiary of Placer-Amex began an initial investigation of the LCD. In

October 1962, Placer-Amex completed preliminary geological investigations of the various known mineral occurrences in the area. That was followed by geochemical stream-sediment and soil sampling of the area that outlined six large anomalous zones within an area of approximately 6 sq mi with values commonly >1,000 ppm of copper. One of the copper soil anomalies coincides with the Moonlight deposit. This resulted in the first claims staked in the Moonlight Valley in late 1964. Preliminary drilling was completed on the Sulfide Ridge (now Lambs Ridge) geochemical anomaly. In mid-1966, IP surveys were conducted over many of the known mineral occurrences in the LCD. This survey produced anomalies in the area of the Moonlight deposit.

The first drillhole in the Moonlight deposit was completed in August 1966. This hole, ML-1, showed encouraging results including disseminated bornite in the top 220 ft of the hole, which returned a grade of 0.59% copper. Encouraged by the results, Placer-Amex acquired more claims and continued drilling through to December 1966. This work indicated that a large low-grade, disseminated copper ore body was present. During the summer of 1967 and through to December of that year, Placer-Amex completed additional drilling and initiated detailed petrological studies. Total footage to that point was 142,093 ft of diamond drilling in the area, including drilling completed on all mineral showings including Moonlight, Superior, Engels and other satellite showings. At this juncture in the program, Placer-Amex determined that there was a problem with all the previous analyses that had been completed at their own Golden Sunlight Mine assay facilities in Montana. Therefore, they decided that all drill samples must be re-assayed by an independent firm, Union Assay located in Salt Lake City, Utah. Further discussions of the analytical problems are found in Section 11.0 of this report.

A number of Mineral Resource estimates were generated by Placer-Amex. These Mineral Resource estimates shown in Table 6.1 do not follow the requirements for Mineral Reserves and Resources outlined in NI 43-101, as they were estimated prior to the inception of NI 43-101. The QP is not aware if these estimates were derived using the standards now outlined in NI 43-101. The historic Mineral Resources are presented here to show the progression of development of the Mineral Resources over the years on the Property. The Mineral Resource estimates are considered historic, and have now been replaced with current Mineral Resources that are discussed in Section 14.0 of this report.

Table 6.1 Historic Moonlight Mineral Resource Estimates

Year	Tons	Grade (Cu %)	Cut Off (Cu %)	Category (pre NI 43-101)	Estimation Method	Author
1972	174,000,000	0.406	0.25	Geological Reserve	Inverse distance to the 5th power as a block estimator	Rivera, Amex
1972	180,000,000	0.390	0.23	Mineable Reserve	Inverse distance to the 5th power as a block estimator, Strip Ratio 2.7:1	Rivera, Amex
1991	161,000,000	0.319	0.25	Ore Reserves	Inverse distance to the 5th power as a block estimator	Geasan, Placer-Amex
1991	80,190,000	0.366	0.30	Ore Reserves	Inverse distance to the 5th power as a block estimator	Geasan, Placer-Amex
1991	171,000,000	0.315	0.25	Ore Reserves	Ordinary Kriging	Hartzell, Placer-Amex
1991	91,965,000	0.357	0.30	Ore Reserves	Ordinary Kriging	Hartzell, Placer-Amex

Subsequent to the earlier 1972 Placer-Amex Mineral Resource estimates, Placer-Amex completed a study on the deposit concentrating on just the oxide component contained within the Moonlight body. The oxide material was noted by the various workers who generated the Mineral Resource estimates and was included in the Mineral Resource estimates. Sheffield obtained assays >0.25% copper from the near surface when drilling adjacent to holes where Placer reported 20 ft (6 m) of overburden. This suggests that the target size for an oxide Mineral Resource at the Moonlight deposit may be larger than the 12 million st estimated by Placer-Amex and in addition, it would have a low stripping ratio. A 1988 study (Gillette) reviewed just the oxide material. Gillette determined that there were four distinct oxide bodies contained within the Moonlight copper body.

Table 6.2 summarizes Gillette's Mineral Resource estimates for the oxide material. The estimates shown in Table 6.2 do not follow the requirements for Mineral Reserves and Resources outlined in NI 43-101 as they were estimated prior to the inception of NI 43-101. The terminology used in Table 6.2 is from Placer-Amex files which pre-dated NI 43-101, and the table should be considered only as a summary of historical estimates.

Table 6.2 Historic Moonlight Oxide Resource Estimates

Area	No. of Holes	Area (ft)	Material (not to NI 43-101)	Tons	Grade (Cu %)
North	17	2,300 x 500 x 33	Ore	3,200,000	0.55
		2,322 x 522 x 22	Waste	2,200,000	
North Central	10	1,800 x 600 x 54	Ore	4,900,000	0.60
		1,837 x 637 x 37	Waste	3,600,000	
South Central	10	2,000 x 400 x 25	Ore	1,700,000	0.54
		2,040 x 440 x 40	Waste	3,000,000	
South	11	1,150 x 800 x 31	Ore	2,400,000	0.42
		1,174 x 824 x 24	Waste	1,900,000	

The Project was put on hold from 1971 to1994, with respect to any new field exploration, due to declining copper prices in the early 1970s. In 1994, Placer dropped all interest in the Project, allowed the claims to lapse, and in September of that year Les Storey staked the eight Diane claims, which are now part of the Moonlight option.

Subsequently (2004-2012), a succession of Canadian junior companies (Sheffield, Nevoro, and Starfield) re-assembled the Property and completed some focused, but limited work, including drilling. Between 2004 and 2008 Sheffield staked an additional 410 unpatented lode claims in the district. In April 2006, Sheffield optioned the California-Engels land consisting of approximately 894 ac of deeded land covering the historic Engels and Superior mines. In 2005-2006, Sheffield drilled 14 HQ core holes (11,135 ft) on the Moonlight deposit, all but two of which were angle holes.

In April, 2007 Sheffield contracted Orequest to produce a NI 43-101 Mineral Resource estimate for the Moonlight deposit, as filed on SEDAR and shown in Table 6.3 (Cavey and Giroux 2007).

Table 6.3 Moonlight Deposit Historic NI 43-101 Mineral Resource Estimate – April 2007

Cut-off (Cu%)	Tons > Cut-off (tons)	Grade > Cut-off		
		Cu (%)	Au (oz/st)	Ag (oz/st)
Moonlight Indicated Resource Grade-Tonnage Table				
0.20	161,570,000	0.324	0.003	0.099
0.25	114,570,000	0.366	0.003	0.112
0.30	76,150,000	0.413	0.003	0.124
Moonlight Inferred Resource Grade-Tonnage Table				
0.20	88,350,000	0.282	0.003	0.089
0.25	48,820,000	0.329	0.003	0.107
0.30	23,720,000	0.390	0.003	0.118

The 2007 estimate should be considered only as a historic Mineral Resource, which has been superseded by the current Mineral Resource in Section 14.0 of this report.

Sheffield was acquired by Nevoro Copper in July 2008. In addition to drilling at Engels in the fall of 2008, Nevoro Copper completed seven vertical core holes totaling 2,603 ft (794 m) at the Moonlight deposit. Additional unpatented lode claims were staked in 2007 (33 total), 2008 (23 total), and 2011 (12 total).

In 2009, Starfield acquired Nevoro the parent company of Nevoro Copper, and conducted a limited drilling program at Engels in 2009 and 2010. Starfield also contracted a property-wide airborne geophysical survey conducted by Fugro. Starfield dropped the unpatented claims encompassing the Moonlight deposit in 2012.

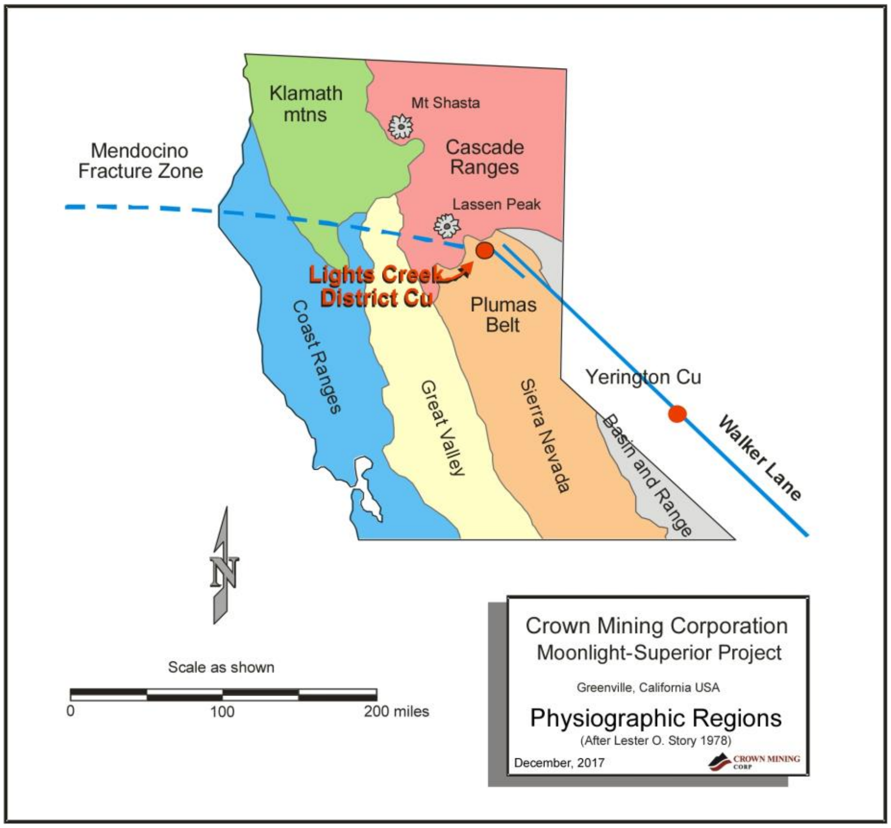
By 2013 the district was again split with the Moonlight deposit controlled by Canyon Copper by virtue of an assignment from Starfield, while the Superior-Engels lease was acquired out of bankruptcy court by Crown Gold Corp. The Superior-Engels acquisition included the complete database held by Nevoro, which comprehensively documented all known exploration activity on the Property from 1960 through 2013. In February 2016, Crown Mining Corp. (re-named from Crown Gold Corp. in 2014) optioned Canyon's position and the district again became a unified property.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Project area covers most of the historic LCD, located at the northern end of the Sierra Nevada physiographic province at the juncture with the late-Tertiary-to-Recent Cascade volcanic province to the north, and the Basin and Range province immediately to the east. The LCD lies at the northern end of the 25-mile-long, 5-mile-wide, N20W trending Plumas Copper Belt, interpreted to represent an extension of the north-northwest trending Walker Lane structural lineament, and at the eastern terminus of the Mendocino Fracture Zone (Figure 7.1).

Figure 7.1 Physiographic Regions Map



Source: after Tanaka (2014)

The Walker Mine is located at the south end of the Plumas Copper Belt, approximately 12 mi southeast of the Property. Numerous small mines and copper showings are located between the Walker Mine and the LCD.

7.2 LOCAL GEOLOGY

The LCD copper deposits are primarily hosted in the early Jurassic (178 Ma), multi-phased, quartz monzonite LCS, one of several Jurassic-age plutons which intrude slightly older meta-volcanic rocks, and which is itself intruded by the younger granodiorite of the Sierra Nevada Batholith. Older meta-volcanic rocks are locally mineralized where they sit as roof pendants on top of the Moonlight and Engels deposits.

The LCS is a roughly circular, fine- to medium-grained quartz monzonite to granodioritic tourmaline-rich intrusive, with an exposure of approximately 7 sq mi (Figure 7.1). Both Sheffield and Placer-Amex geologists noted that the quartz monzonite tends to be finer grained with a more porphyritic texture near the contact with meta-volcanic rocks and less potassium feldspar-rich and more equigranular with depth and towards the center of the quartz monzonite stock.

Early work by Anderson (1931) and later by Lester Storey (1978) suggest that five distinct batholithic differentiates exist in the LCD. Storey (1978) described five rock types as follows:

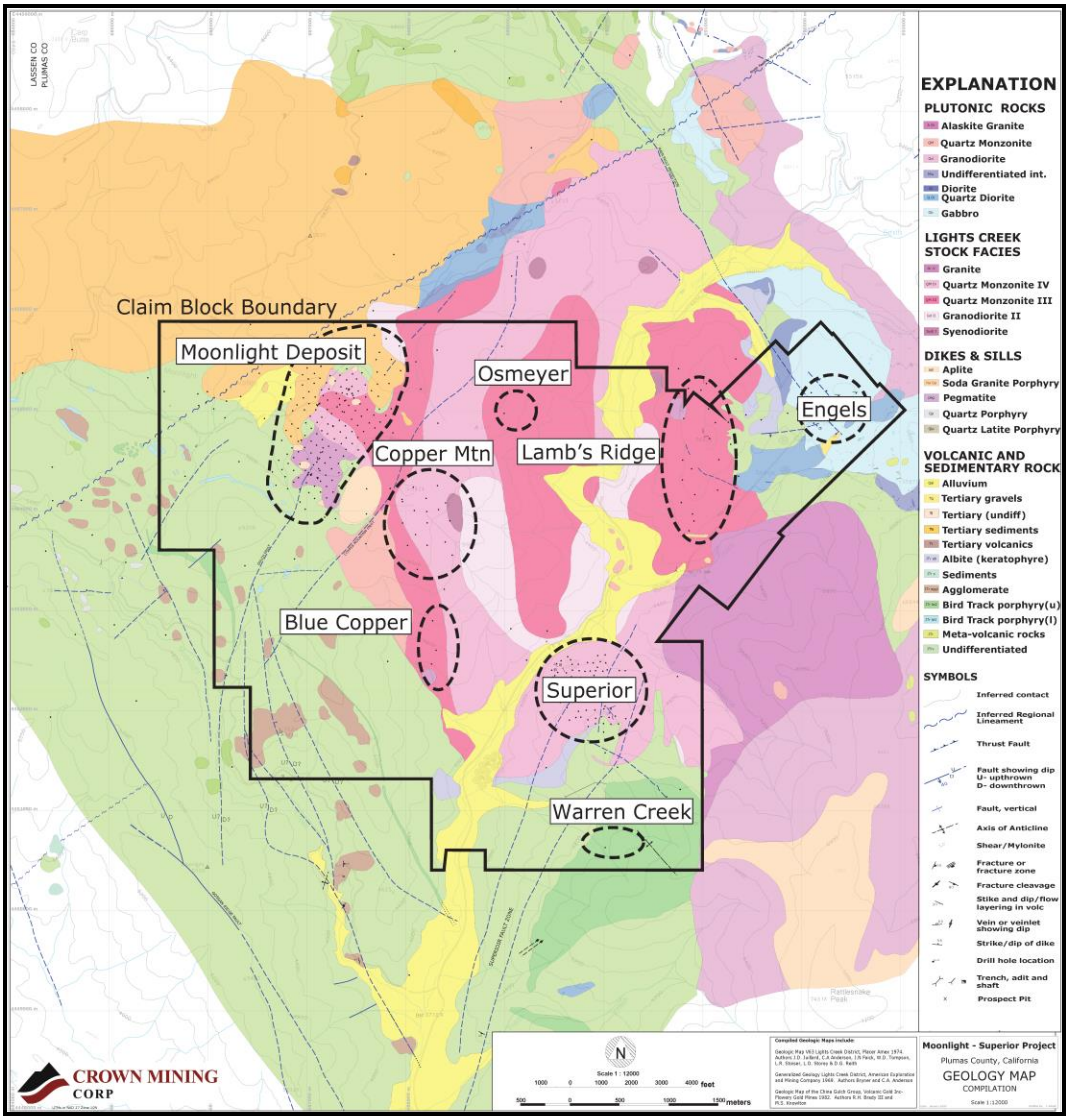
"These are from oldest to youngest:

- *Engels Mine gabbro (main host to high-temperature mine copper deposit)*
- *Quartz diorite (also host to Engels Mine ore).*
- *Granodiorite (main batholith, non-mineralized)*
- *Quartz monzonite (host to porphyry-type copper occurrence of intermediate temperature).*
- *Coarse-grained granite (non-copper bearing with rare molybdenum occurrences).*

The quartz monzonite is the most heterogeneous in overall make-up of any of the segregated intrusive bodies."

Several of these phases are shown in the simplified Placer-Amex map (Figure 7.2) as various shades of pink.

Figure 7.2 Simplified Local Geology



Note: Adapted from mapping by Pacer-Amex

The multi-phase nature of the igneous rocks hosting the Moonlight deposit were not confirmed by Sheffield geologists or the QPs for this report.

Several 1960s era holes drilled on the western side of the Moonlight deposit carry very low-grade copper mineralization in intercepts that were logged as Tertiary sediment. Examination of the original drill logs shows that with two exceptions, MN-15 and MN-247, mineralized intercepts in Tertiary sediment consistently average 0.09% copper or less. Many of the intercepts are described as carrying fragments of quartz monzonite and/or meta-volcanic rock. Three intercepts are described as volcanic agglomerate with fragments of quartz monzonite. The intercept in MN-015 is logged as conglomerate cemented with arkosic sand and hematite rich mud. From 60 to 110 ft it contains quartz veins with specular hematite as well as minor malachite and azurite and carries 0.23% copper. Coal seams and a reddish color reminiscent of the meta-volcanic rock that underlies it are mentioned in the MN-247 log. The age of this mineralization is not known and its relationship to the Moonlight deposit is not understood.

Apart from the Moonlight deposit, the LCD contains several other areas of copper mineralization. Two areas, Engels and Superior, have current NI 43-101 Technical Reports on Mineral Resource estimates filed on SEDAR (Tanaka 2014).

7.2.1 Engels Mine Area

The Engels deposit is hosted by hornblende gabbro immediately adjacent to the LCS. Quartz diorite, diorite and roof pendant meta-volcanic rocks are associated with the gabbro and may have played a role in the placement of the copper mineralization.

Mineralization in the Engels Mine area occurs in a 1200 ft by 600 ft pipe like zone. Mineralization is associated with brecciated zones that exhibit features of both an intrusion breccia and a hydrothermal breccia. Copper grades exceeding 15% copper have been encountered in several 2 m core intercepts.

Copper mineralization at Engels is strongly oxidized to depths of 230 ft. Primary copper oxide minerals are malachite with lesser chryscolla and azurite. Bornite and chalcopyrite are the principal sulfide minerals.

7.2.2 Superior Mine Area

Most of the historical production at Superior came from seven parallel, northeast striking, and east dipping (55 to 80°) vein zones. The veins principally consist of chalcopyrite and some bornite, along with associated magnetite and pyrite and are 8 to 20 ft thick. Magnetite is more prevalent at the Superior Mine than at the Moonlight deposit, while specularite, common at Moonlight, is non-existent at Superior. The Superior Mine exhibits some similarities to Moonlight in that they are both found within an intrusive body in close proximity to an older meta-volcanic sequence. In addition to the steeply dipping, thick, chalcopyrite rich veins which were the source of historical production, Placer-Amex drilled 96 holes at Superior from 1964 to 1968 and identified a large body of disseminated copper mineralization.

7.2.3 Structural Control

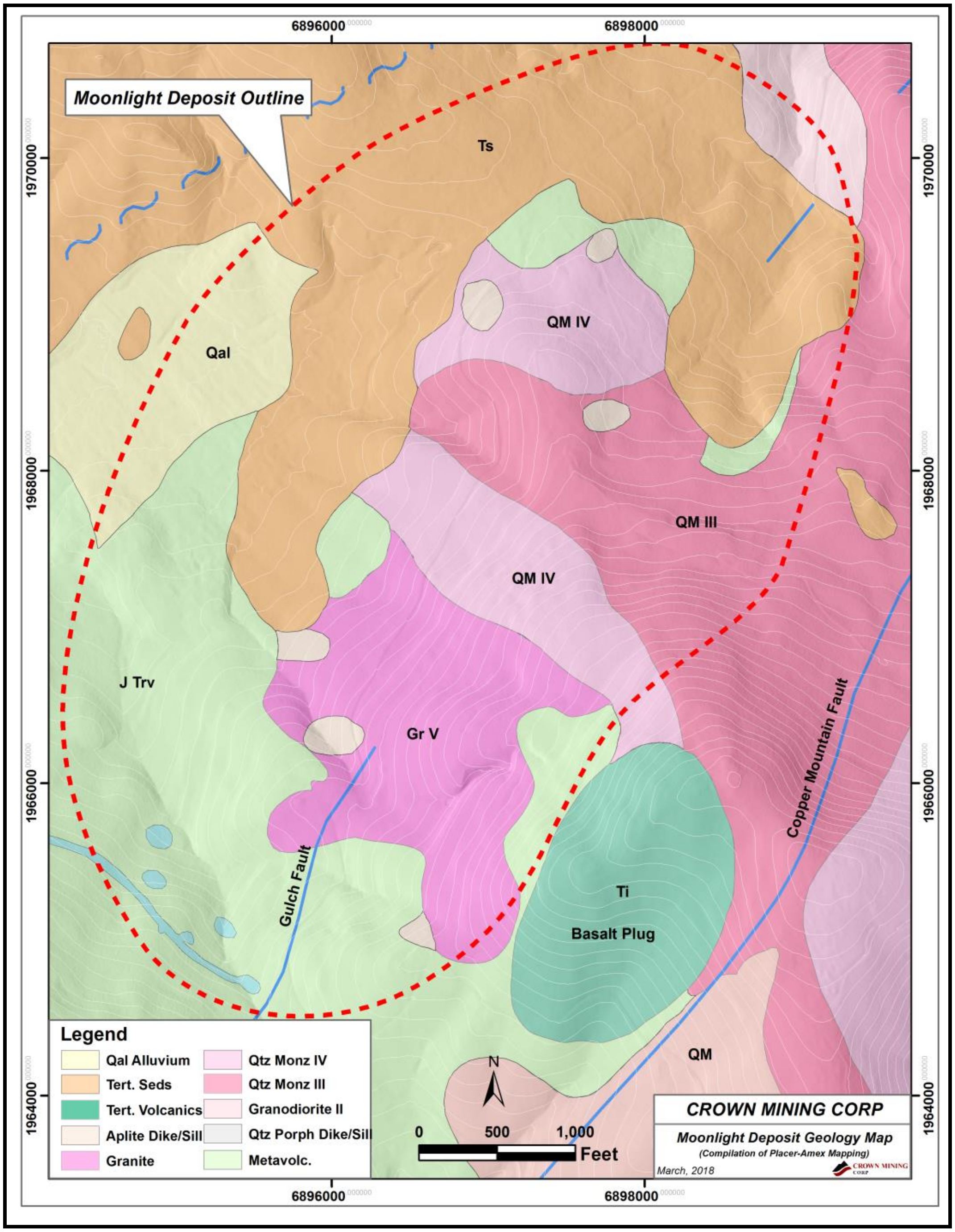
Structural preparation appears to be important in localizing mineralization throughout the LCD. Mineralized structures of varying attitudes are found throughout the district. At the Ruby Mine, south of the Moonlight deposit, mineralized structures strike N20W and dip steeply to the northeast paralleling the trend of the Plumas Copper Belt and the Walker Lane. Structures striking N10E and dipping steeply or moderately eastward are the primary host for copper mineralization at the Superior deposit. Similarly oriented mineralized fracture zones are evident at the Moonlight and Engels deposits. Northwest striking, gently southwest dipping fracture zones host significant copper mineralization throughout the district. At the deposit scale mineralization is preferentially located in stockwork zones with fractures of multiple orientations or at the intersection of structures and lithologic contacts.

The importance of structural controls notwithstanding, a significant part of the copper mineralization at the Moonlight and Superior deposits is disseminated and not associated with fractures or veinlets. At both deposits disseminated mineralization is typically associated with 2 to 10 mm blebs of tourmaline.

7.3 MOONLIGHT DEPOSIT GEOLOGY AND MINERALIZATION

Figure 7.3 illustrates the potential lithologic complexity of the Moonlight deposit. According to Placer-Amex surface maps, several intrusive phases host the Moonlight deposit. A large part of the deposit lies within two phases of the LSC quartz monzonite designated as QM III and QM IV. Granitic intrusive (Gr V) hosts the southern third of the deposit. Granodiorite carries copper mineralization at the northern tip of the deposit. Jurassic-Triassic roof pendant meta-volcanic rocks overlay the deposit and crop out on the southern, western and northern perimeters. On the western flank of the deposit meta-volcanic rocks are overlain by Tertiary sediments.

Figure 7.3 Moonlight Deposit Geology



Source: Adapted from mapping by Placer-Amex.

Multiple intrusions at Moonlight were not logged in the Sheffield drilling and were not confirmed by QPs preparing the report. Some of the variations in intrusion texture and composition may be related to hydrothermal alteration.

The bulk of the following discussion is after R. G. Wetzel from his January, 2009 report describing the Moonlight deposit. Placer-Amex, Sheffield, and Sheffield's successors recognized that there are at least two styles or stages of mineralization at the Moonlight deposit. The paragenetically earlier style is characterized by disseminated copper minerals located interstitial to quartz, feldspar, chlorite and especially disseminated rosettes of tourmaline. This mineralization usually consists of fine-grained chalcopyrite but zones of disseminated bornite are also common. High in the system disseminated hypogene chalcocite has also been occasionally observed. Bornite rims chalcopyrite grains in some places. This style of mineralization shows some association with potassium feldspar, a very strong association with tourmaline and sometimes chlorite. Unless overprinted by second stage fracture or breccia hosted mineralization, this earlier style of mineralization typically assays at 0.1% to 0.8% copper. The second stage of mineralization is characterized by veinlets, or stockwork breccias, which often have a gangue of tourmaline and lesser quartz with strong hematite. Strong copper mineralization is commonly observed on veinlets trending N20-35W and dipping 15-35SW southwest. The vein orientation suggests a good exploration target beneath the meta-volcanic rocks to the southwest. In addition to the mineralization in shallow dipping fractures, copper is contained on north-south, steep to moderately east dipping veinlets, N60-75E steeply north dipping veinlets, and N70-85W steeply south dipping veinlets. Although fracture hosted mineralization is widespread and often high grade at Moonlight, drilling to date has not revealed extensive vein-like structures similar to those mined at the Superior Mine.

The copper sulfides show a vertical zonation, with chalcocite or digenite predominating in the upper levels of the deposit. With increasing depth, bornite predominates and chalcopyrite appears. Bornite is often observed to rim or cut chalcopyrite. Bornite and chalcopyrite may also be cut by chalcocite veinlets. At the deeper levels chalcopyrite typically dominates in fracture hosted mineralization, but bornite is often still abundant. Magnetite can sometimes appear with hematite decreasing in abundance with depth. Rare pyrite may appear in veinlets at depth. Iron or magnesium-rich carbonates are also common in fracture hosted mineralization. Late-stage copper-poor calcite and quartz veinlets that cut both preceding types of mineralization are also common.

Veinlet-or-breccia hosted mineralization dominates the northern part of the Moonlight deposit, where chalcocite-rich mineralization commonly grades more than 1% copper. In holes 06MN-9, 10, 11, and 12 chalcocite-rich mineralization grades quickly into chalcopyrite with depth and bornite is not very abundant. In the southern and central parts of the deposit the chalcocite-bornite-chalcopyrite zonation is well-developed. Fracture-hosted mineralization may grade more than 1% copper in the central and southern portions of the deposit.

Sericitic, chloritic, and albitic alteration may form halos around veinlets and breccia zones. Epidote becomes more abundant in and around veinlets with depth. Potassium feldspar is abundant. In addition to the quartz, feldspar and 1 to 5% disseminated tourmaline that characterizes the Lights Creek quartz monzonite, it also contains 2 to 8% finely disseminated hematite and magnetite. The hematite is typically specular and thin section work indicates that it usually rims and replaces magnetite. Hematite replacement decreases with depth with the result that the LCS at Moonlight becomes increasingly magnetic with depth.

8.0 DEPOSIT TYPES

The Moonlight deposit was historically classified as a porphyry copper deposit with associated gold, silver, and molybdenum credits. However, Placer-Amex geologists recognized that the LCD deposits had many characteristics that were not typical of porphyry copper deposits and lacked many of the typical features. L.O. Storey (1978) noted, "Typical porphyry copper type alteration zonation as illustrated by Lowell and Guilbert is nonexistent." Recent work noting the lack of porphyry style veining, the ubiquitous presence of magnetite at Superior and specular hematite at Moonlight, and the relative scarcity of pyrite suggest an IOCG affinity (Stephens 2011) (Cole 2015).

Many copper deposits which had previously been classified as porphyry copper type have now been characterized as IOCG deposits. There is considerable evidence that the LCD deposits should be included in this group. Deposits with a fairly wide variety of characteristics have been classified as belonging to the IOCG group; however, the following characteristics are consistently used to classify these types of deposits:

- abundant magnetite and/or hematite, which is often specular; if both are present, hematite is more common higher in the system
- low-pyrite content, with increased pyrite often located beneath and adjacent to the ore zone
- typically tabular shaped orebody rather than cylindrical or deep sided cupola-shaped like porphyry copper deposits
- abundant bornite and/or hypogene chalcocite often as a late fracture filling phase of mineralization
- anomalous gold, silver, uranium, and rare earth elements.

The LCD deposits show the majority of these characteristics. A number of deposits have been classified by various authors as belonging to the IOCG group including Olympic Dam in Australia, Candelaria and Mantos Blancos in Chile, Luz del Cobre in Mexico, Marcona in Peru, and Minto in the Yukon. All of these deposits show significant tonnages of plus 2% copper mineralization.

Regarding IOCG deposits, Sillitoe (2003) noted, "The deposits…reveal evidence of an upward and outward zonation from magnetite-actinolite-apatite to specularite-chlorite-sericite and possess a Cu-Au-Co-Ni-As-Mo-LREE (light rare earth element) signature…". The high-grade mineralization at Superior is associated with magnetite-actinolite-tourmaline-apatite. At Moonlight, copper mineralization is associated with tourmaline-specularite-chlorite-sericite. During an April 2015 field visit to the district Sillitoe categorized Engels, Lambs Ridge, Superior and Moonlight as IOCG deposits (Cole 2015)

Mineralized diabase dikes have been observed at the Moonlight deposit and at the Superior Mine raising the question, how long after the crystallization of the quartz monzonite did some of the mineralization occur? More study is needed before a more complete genetic model can be developed for the LCD (Wetzel 2009).

9.0 EXPLORATION

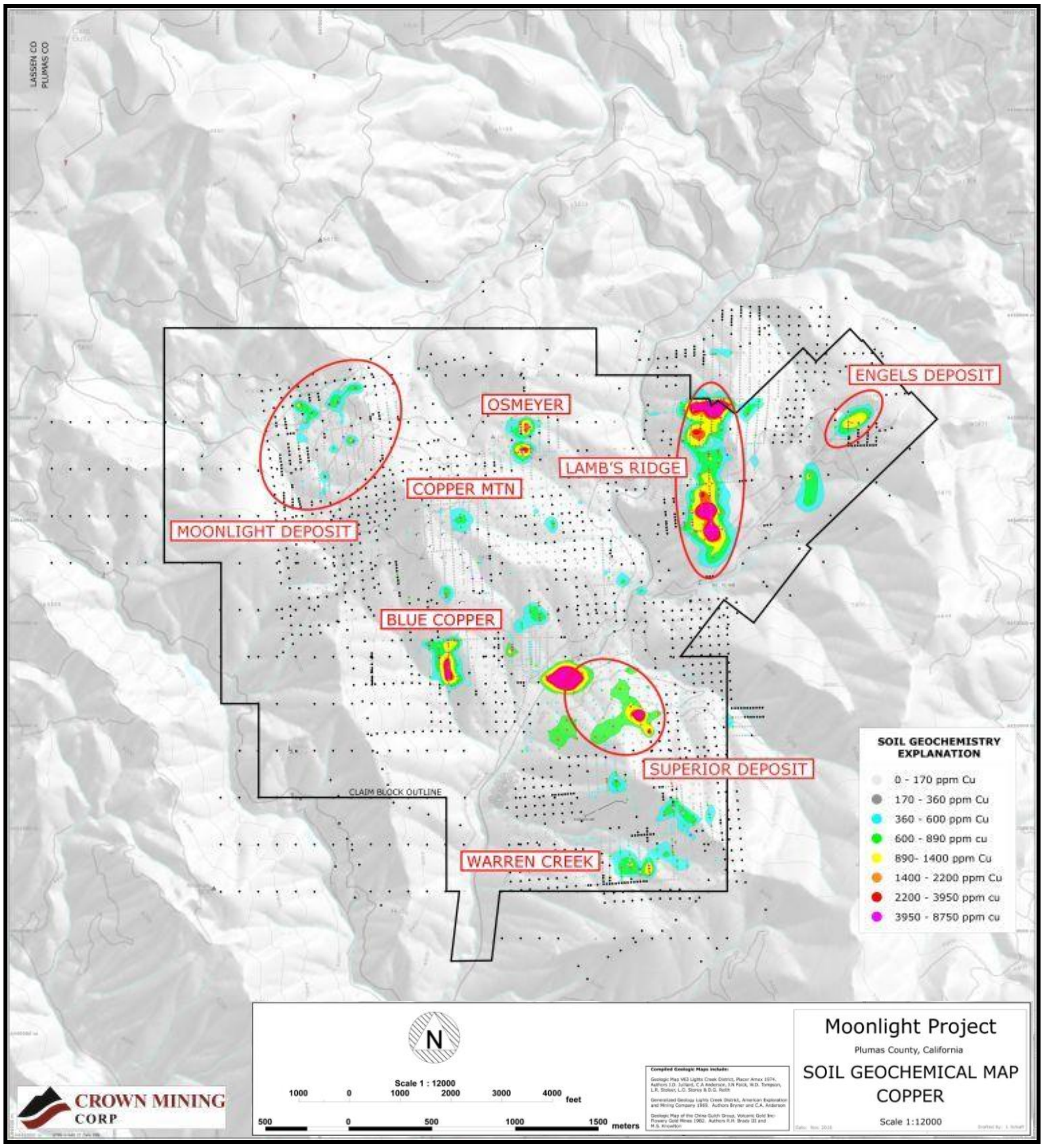
The following section is partly based on Tanaka (2014), Wetzel (2009), and Placer-Amex (1972).

9.1 LIGHTS CREEK DISTRICT HISTORICAL EXPLORATION

9.1.1 GEOCHEMISTRY

Beginning in 1963 and continuing into 1965, Placer-Amex conducted a series of stream sediment, soil and rock geochemical surveys for copper. Soil sampling was initially done on 300 ft centers over an area of roughly 10 sq mi. This program produced six large (>1,000 ppm) copper anomalies (Superior, Moonlight, Engels, Warren Creek, Blue Copper and Sulfide Ridge now re-named Lamb's Ridge) and several more of lesser magnitude. Follow-up sampling on 100 ft centers was carried out over most of the anomalous areas. This work identified a number of exploration targets in the district, including the Moonlight deposit. Figure 9.1 shows the results of the soil sampling campaign and identifies the named anomalies.

Figure 9.1 Soil Geochemistry Map – Copper



In 1966, in addition to the district-wide soil sampling program, Placer-Amex undertook extensive chip-channel sampling of the 1 Level workings at the Superior Mine. From 2005 through 2007, Sheffield completed its own program of underground sampling at Superior. A total of 151 chip-channel or select grab samples were collected in addition to 32 samples of splits from the old Placer-Amex underground drill core. The chip-channel sampling at Superior generally confirmed the results of Placer-Amex sampling which defined the broad-scale disseminated copper mineralization between and beyond the higher-grade breccia veins historically mined. Table 9.1 lists the results of Sheffield's underground sampling program.

Table 9.1 Results of Sheffield Underground Sampling Program – Superior Mine

No. of Samples	Mine Area	Average Width (m)	Average Cu (%)	Average Au (g/t)	Average Ag (g/t)
32	Underground Drill Core Re-samples	n/a	0.59	0.026	5.48
38	A Level Underground Samples	2.69	0.20	0.042	8.9
113	1 Level Underground Samples	2.88	2.43	0.028	39.8

9.1.2 GEOPHYSICS

In 1965, Placer-Amex initiated a ground-based IP survey over the Lamb's Ridge (formerly Sulfide Ridge) anomaly. The survey was conducted by HGC of Tucson, Arizona. In 1966, the same group ran a follow-up IP survey over Lamb's Ridge and expanded the IP work to Moonlight, Copper Mountain, Blue Copper, Osmeyer Ridge and Warren Creek. Their conclusions recommended follow-up drilling at several targets including at Moonlight, Copper Mountain, Blue Copper, Lamb's Ridge and Warren Creek.

In 1969, Placer-Amex initiated an airborne magnetic and gamma ray survey conducted by Geophoto Services Inc., a subsidiary of Texas Instruments Co., over the LCS. The results were regarded as inconclusive by Placer-Amex. In June of 1970, McPhar Geophysics began IP surveys on Gossan Ridge southwest of Moonlight. In 2009, Garrry Carlson of Gradient Geophysics reviewed the existing geophysical data and recommended an airborne magnetic and EM survey, a Deep IP-Resistivity survey, and a CSAMT survey.

In 2010, Starfield commissioned Fugro to conduct an airborne magnetic and EM geophysical survey of the district. The purpose of the survey was to collect magnetic and EM data to be used to enhance the understanding of the geology of the area and possibly to locate new mineral deposits. The survey provided a great deal of geophysical data that could be used to improve the geological mapping in the area. The magnetic data from the survey clearly shows major structures on the property and permits distinction of lithological/alteration differences within the Lights Creek intrusive complex. However, as far as the authors of this report are aware, to date, the results of the Fugro airborne survey have not been applied in a systematic way to exploration of the district.

9.1.3 DRILLING

From 1964 through November of 1970, Placer-Amex conducted an extensive drilling program totaling 198,916 ft in 409 drillholes that covered much of the LCD. This included 133 diamond drillholes at the Superior deposit between 1964 and 1970, 28 holes on Lamb's Ridge between 1966 and 1970, and 10 holes at the Engels Mine in 1967. Beginning in 1966, and continuing through 1970, Placer-Amex drilled 99,436 ft of BX core in 199 holes at the Moonlight deposit (Placer-Amex 1972). Sheffield completed an 11,135 ft, 14-hole diamond drill program on the Moonlight deposit in 2006. In 2007, Sheffield drilled 1,390 ft in 15 rotary holes at Moonlight. In 2008, Nevoro drilled 2,603 ft in 7 core holes at Moonlight. A more detailed discussion of drilling results can be found in Section 10.0.

9.2 Lights Creek District Exploration Targets

A large number of copper mineralized zones defined by soil sampling exist on the Property. Some show potential for containing additional economic Mineral Resources in the LCD. These include the immediate vicinity of the Engels and Superior Mines, Lamb's Ridge, Copper Mountain, the area surrounding the Moonlight deposit, and several others. All of the anomalous areas were tested by varying amounts of drilling. The following description of the soil sampling anomalies and discussion of their exploration potential has been adapted from Wetzel (2009) and from Tanaka (2014), but also includes information gleaned from geophysical reports detailing IP-Resistivity surveys conducted over the soil sampling anomalies.

9.2.1 Moonlight Deposit

Placer-Amex soil sampling produced a very irregular, <500 ppm copper anomaly that measures approximately 4,500 ft in a north-northeast direction and 3,500 ft in a west-northwest direction. There are numerous internal lows and local highs up to 5,000 ppm copper within the anomaly. Wetzel (2009) stated that the anomalously high zones do not usually coincide with the location of near surface >0.5% copper mineralization known from drilling. South of Moonlight widespread areas of specular hematite and some quartz veinlets with scattered copper oxides in the meta-volcanic rocks may indicate the presence of significant copper mineralization not intersected by Placer-Amex and Sheffield's drilling. Drilling indicates that mineralization at Moonlight is plunging to the south west underneath the roof pendant meta-volcanic rocks. The Ruby Mine, located approximately 1.5 mi south of the Moonlight deposit, is a collapsed adit with quartz vein material on an adjacent dump. Three grab samples collected by Sheffield returned an average grade of 5.28% copper, 1.87 g/t gold, and 211 g/t silver. This mineralization is in volcanic rocks above the projected Moonlight copper mineralization.

Limited surface sampling has shown high-grade copper in structures with a wide variety of orientations in the meta-volcanic rocks south of the Moonlight deposit. In addition to high-grade copper these samples have shown higher grades of gold and silver than have been found elsewhere in the district. ML-503, approximately 0.5 mi south of Moonlight hit 20 ft of 3.4% copper in meta-volcanic rocks. A zone of high-grade copper oxide with gold and silver credits is postulated but will need further drilling to define.

9.2.2 Engels Mine

Over the Engels Mine, a symmetrical, concentric, >500 ppm copper in soil anomaly extends about 1,800 ft in a northeast direction, is about 1,000 ft wide and contains a core that carries >5,000 ppm copper. Placer-Amex drilled a total of 10 holes at Engels in 1966 and 1967. Only three of these holes were within the 1,000 ppm contour. Core recoveries were typically poor but holes E-2 and E-7 carried >0.5% copper. Sheffield/Nevoro drilled 44 core holes in 2007 and 2008. In 2009, Wetzel observed that >0.3% copper mineralization encountered in Sheffield/Nevoro's drilling coincided with the 1,000 ppm copper contour. At Engels drilling is tightly confined to the immediate vicinity of the historically mined volume, and does not test the along-strike, or down dip extent of mineralization.

9.2.3 SUPERIOR

Placer-Amex soil sampling produced a concentric symmetrical 1,000 ft by 1,400 ft >2,000 ppm copper anomaly with a 600 ft by 800 ft core carrying >5,000 ppm copper. The long direction of the anomaly is oriented approximately east-west perpendicular to the dominant north-south structural fabric of the mineralization at Superior. Placer-Amex drilled 91 holes and defined a historical Mineral Resource which roughly coincides with the 2,000 ppm copper contour. Both Placer-Amex and Sheffield's underground sampling suggests the presence of a higher-grade core to this mineralization based on results, which include 260 ft of 1.83% copper and 29 g/t silver, and 220 ft of 1.63% copper and 20 g/t silver in underground workings. Superior drilling appears to better define the limits of known mineralization than the drilling at Engels. Any additional drilling should investigate the possible existence of other high-grade structurally-controlled segregations of high grade to the northeast and at depth.

9.2.4 LAMB'S RIDGE

At Lamb's Ridge, soil sampling produced a >1,000 ppm copper anomaly that extends 6,000 ft in a north-south direction, averages 1,000 ft in width and contains localized high-grade zones that carry >5,000 ppm copper. Placer-Amex drilled a total of 28 holes at Lamb's Ridge. Wetzel (2009) noted that outcrop and talus are very abundant along Sulfide Ridge and speculated that copper from fractures may have been over-represented in the sieved samples.

Lamb's Ridge drilling is very widely spaced with intervals of between 300 and 1,000 ft, relatively shallow for the lateral extent of mineralization observed and entirely vertical. The grades present in the 28 drillholes were not of interest to Placer-Amex at the time of drilling and, while generally lower than those present at both Engels and Superior, some drilling intercepted copper mineralization within the range of contemporary economic interest. The 1965 and 1966 IP-Resistivity survey identified three areas of interest (Ludwig 1966). Lamb's Ridge should be tested further with angled core holes in at least two orientations and extending to greater depth than previous drilling. The extent of copper mineralization at Lamb's Ridge is untested in any direction.

9.2.5 WARREN CREEK-UPPER SUPERIOR

Soil sampling produced an irregular 3,500 by 2,500 ft >1,000 ppm copper anomaly in the Warren Creek drainage that contains localized highs >3,000 ppm copper. This is the only large-scale soil anomaly presently known to be hosted in meta-volcanic rocks in the LCD. According to their 1972 report, Placer-Amex drilled one 1,515 ft hole, DDH-01A, at Warren Creek. Wetzel (2009) reports that Placer-Amex drilled at least four holes, including a 2,000 ft deep hole. The authors have not completely resolved this discrepancy, but assume that Wetzel was including the US series holes drilled 2,500 ft northeast of DDH-01A. It should be noted that in the records reviewed by the authors, there is no 2,000 ft deep drillhole in the Warren Creek area. Wetzel also states that >200 ft intercepts of 0.1 to 0.2% copper mineralization were encountered in the meta-volcanic rocks but that no high-grade copper was intersected either in the meta-volcanic rocks or in the underlying Lights Creek quartz monzonite.

9.2.6 Blue Copper

At Blue Copper soil sampling produced an 1,800 ft, north-south geochemical anomaly carrying >1,000 ppm copper. This area shows abundant outcrop and talus and poorly developed soil similar to Lamb's Ridge. Placer-Amex drilled four holes here with disappointing results. Conclusions from the 1966 IP-Resistivity survey recommended a drill test of two, perhaps three, anomalies (Ludwig 1966). It is not known at this time if the Blue Copper drilling tested those anomalies.

9.2.7 Copper Mountain

The Copper Mountain geochemical anomaly, an irregular 2,000 by 200 ft >500 ppm copper anomaly is located approximately 2500 ft southeast of the Moonlight deposit. A few >2,000 ppm copper highs are present within the 500 ppm contour. Placer-Amex drilled 14,226 ft in 25 holes over a 3,500 by 4,000 ft area at Copper Mountain. A number of encouraging intercepts were encountered. These include 210 ft of 0.39% copper in CM-11, 390 ft of 0.36% copper in CM-12, and an average of 0.224% copper from 490 to 1,965 ft in CM-29. All drilling at Copper Mountain is vertical. As at Lamb's Ridge, the drilholes are widely separated with spacing ranging from 300 ft to over 700 ft, with the exception of CM-29 the drilling depths average <500 ft. The 1966 IP survey identified several chargeability anomalies which were recommended for drilling (Ludwig 1966).

9.2.8 Osmeyer Prospect

At the Osmeyer Prospect, 4,000 ft east of the Moonlight deposit, soil sampling produced an irregular 1,500 ft by 600 ft >1,000 ppm copper anomaly. Only two holes, both vertical, were drilled in the vicinity of the Osmeyer Prospect. DDH-04A, located within the northern lobe of the anomaly, was drilled to a depth of 296 ft. From 10 to 110 ft the hole averaged 0.15% copper. The remainder of the hole carried consistent low-grade (<0.05% copper) mineralization. CM-22, located 300 ft west of the anomaly, was drilled to a depth of 200 ft and intercepted consistent low-grade (<0.1% copper) mineralization in quartz monzonite throughout the entire hole. Wetzel (2009) states that "Several other early holes were drilled nearby but records of this drilling are presently unavailable. This anomaly appears to be largely untested." The authors have found no record of other drilling in the immediate vicinity of the Osmeyer Prospect.

10.0 DRILLING

This section of the report is based primarily on the QPs review of various internal reports written by Robert G. Wetzel in his capacity as Project Geologist for Sheffield and Nevoro Copper, and in part on Cavey and Giroux (2007).

10.1 PLACER-AMEX DRILLING

In 1964, Placer-Amex contracted Boyles Brothers Drilling (Boyles Brothers) for the company's LCD drilling program. Beginning in 1966, and continuing through 1970, the contractor drilled 99,436 ft in 199 vertical holes at the Moonlight deposit (Placer-Amex 1972). A typical sequence for Boyles Brothers at Moonlight was to drill 5 to 40 ft from the collar with a rock bit and then set casing. NX core was then drilled to a depth of 100 to 200 ft, after which the hole was completed with BX core (Wetzel 2009). Mineralization occurs from surface to over 1,000 ft in depth. Many holes bottomed in mineralization. As part of an extensive drilling program covering much of the district. Placer-Amex completed 133 diamond drillholes at the Superior deposit between 1964 and 1970, 28 holes on Lamb's Ridge between 1966 and 1970, and 10 holes at the Engels Mine in 1967. Other drilled showings included Copper Mountain, Blue Copper, Warren Creek, Osmeyer Ridge, and Gossan Ridge.

Core was initially analyzed at the Placer-Amex laboratory at the Golden Sunlight operation in Montana. In mid-1967, Placer-Amex geologists realized that LCD assays from Golden Sunlight were unreliable and instituted a program to re-assay all pulps at Union Assay in Salt Lake City. According to Wetzel (2009), the Sheffield Project Geologist: “There is documentation that virtually all the core from Superior and Moonlight was re-assayed at Union with quality control.” Unfortunately, Mr. Wetzel did not provide any details for that documentation. However, in a recent (November 2017) telephone conversation with L.O. Storey, the District Geologist at the time, Mr. Storey confirmed the extensive re-assay program undertaken by Union Assay (L.O. Storey, pers. comm. 2017).

Union Assay ceased operations in the late 1990s. There appears to be no record of the ultimate disposition of assay certificates for the LCD drilling. In the same conversation referenced above, Mr. Storey indicated that he had no knowledge regarding the possible location of the assay certificates.

In 1981, Placer-Amex completed a study of the gold and silver values at the deposit. The early work on the project in the 1960s and 1970s composited 50 to 100 ft sample intervals which were analyzed for their precious metals content. In 1981, Placer-Amex was able to resample eight broadly spaced drillholes from within the Moonlight deposit. These holes, a total of 1,622 ft of core samples, were re-analyzed in close to 10 ft sample intervals within the copper zones. The original sampling of the 100 ft intervals allowed Placer-Amex to determine that the deposit contained an average grade of

0.092 oz/st silver and 0.0014 oz/st gold. The 1981 re-analyses of the 10 ft sections from the eight holes allowed Placer-Amex to estimate that the silver grade was 0.26 oz/st, almost a three-fold increase in the silver grade. Gold values were disappointingly low based on the results of the eight holes. However, there were several drillholes that returned better gold values from the original 100 ft sample intervals such as 0.015 oz/st over 100 ft (ML-13), 0.040 oz/st over 100 ft (ML-223) and 0.080 oz/st over 100 ft (ML-232). The 1981 program was encouraging and indicated that precious metal values must be considered and all future drill samples need to be analyzed for their gold and silver content.

10.2 SHEFFIELD/NEVORO DRILLING

In 2005 and 2006, Sheffield drilled 14 HQ core holes (11,135 ft) on the Moonlight deposit, all but two of which were angle holes. The two-hole 2005 drilling program was contracted to Kirkness Drilling headquartered in Carson City, Nevada. The remainder of the core drilling at Moonlight was contracted to Ruen Drilling of Clark Fork, Idaho. Sheffield's drilling had the following objectives:

- to confirm that the grade of mineralization reported from Placer-Amex's drilling at the Moonlight deposit was reliable or conservative
- to determine if metals other than copper could add value to the Moonlight mineralization
- to demonstrate that potentially ore grade mineralization extended laterally and to depth beyond the limits of Placer-Amex's historical drilling
- to confirm that mineralization was continuous between Placer-Amex's vertical holes drilled on approximate 300 ft centers
- to determine an accurate tonnage factor to be used in a resource estimate
- to determine the structural orientations and/or mineral zonation which control higher grade at Moonlight.

Cavey and Giroux, in their 2007 technical report on the Moonlight Deposit stated that "Inspections of cross sections indicates that Sheffield's drilling typically shows higher grades than Placer reported from nearby drill holes." This impression led to speculation that Placer-Amex vertical drilling had underestimated copper grade at the Moonlight deposit. Various reasons were put forth to explain this apparent discrepancy, but a rigorous analysis of the data was not conducted until March of 2017 when author Donald Cameron and Bob Suda, consultant for Crown Mining, undertook a comparison of the effect of angled drilling on the Moonlight Mineral Resource. They concluded that no apparent strong bias exists between the angle holes and the vertical holes. A full discussion of the issue in given in Section 11.0 of this report.

In 2007, Sheffield drilled 1,390 ft in 15 shallow reverse circulation holes designed primarily as a test of the copper oxide potential at the Moonlight deposit. Lang Drilling was contracted to complete the reverse circulation program. All reverse circulation holes except two returned significantly lower copper values than adjacent Placer-Amex core holes. Holes 07MRC-2 and 07MRC-3, drilled 4 ft apart and 6 ft from MN-505, returned nearly equal average copper values that were 15% lower than MN-505. Holes 07MRC-13 and 07MRC-14, the two reverse circulation holes that returned higher copper grades, were 50 ft from the nearest Placer-Amex core hole and may have simply been intercepting an area of higher-grade mineralization (Wetzel 2008). Based on these results, Wetzel concluded in his 2009 report that reverse circulation drilling is likely to underestimate copper grade at the Moonlight deposit.

Sheffield was acquired by Nevoro Copper in July 2008. In addition to drilling at Engels in the fall of 2008, Nevoro Copper completed seven vertical core holes totaling 2603.5 ft at the Moonlight deposit. The 2008 drillholes were twinned to selected Placer-Amex core holes. The Nevoro holes averaged 0.282% copper. Over the same footage intervals, the adjacent Placer-Amex holes averaged 0.274% copper. The Nevoro holes were not used to calculate a Mineral Resource estimate for this report. Table 10.1 is a summary of the Sheffield and Nevoro holes drilled between December 2005 and November 2008 at the Moonlight deposit.

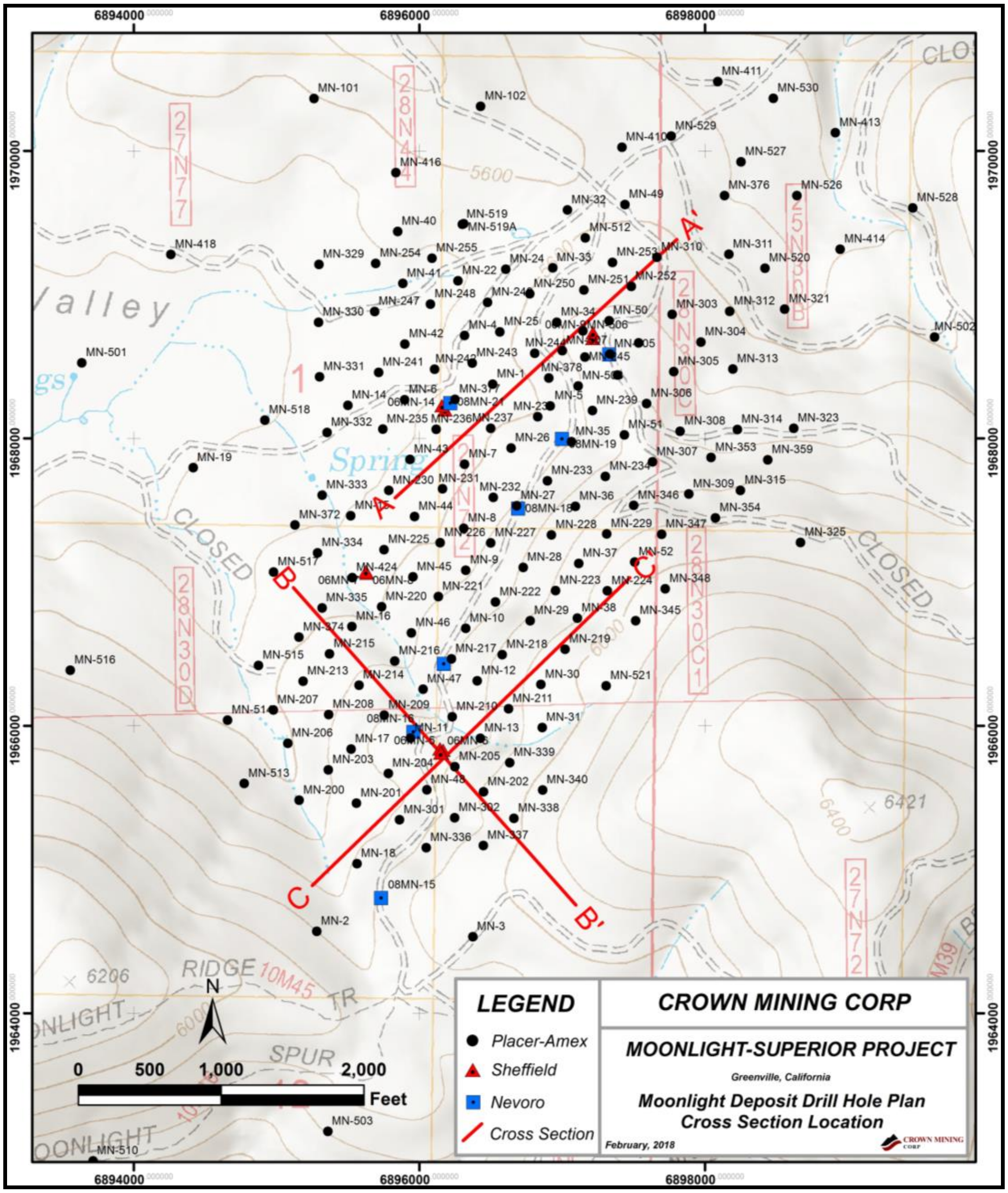
Table 10.1 Sheffield/Nevoro Moonlight Deposit Drilling Summary

Hole ID	Easting*	Northing*	Elevation (ft)	Length (ft)	Azimuth (°)	Dip (°)
05MN-1	6895634	1967063	5585.0	1,150.3	95	-45
05MN-2	6895634	1967063	5585.0	688.3	97	-60
06MN-3	6896145	1965841	5815.0	1,269.4	308	-45
06MN-4	6896168	1965830	5815.0	1,209.3	129	-45
06MN-5	6896158	1965811	5815.0	1,276.3	218.5	-55
06MN-6	6896151	1965811	5815.0	875.3	0	-90
06MN-7	6895627	1967073	5585.0	38.1	41	-45
06MN-8	6895627	1967073	5585.0	1,100.4	61.5	-45
06MN-9	6897217	1968696	5755.0	339.2	210	-45
06MN-10	6897217	1968696	5755.0	485.2	0	-90
06MN-11	6897217	1968696	5755.0	370.1	184	-45
06MN-12	6897217	1968735	5760.0	403.2	270.9	-45
06MN-13	6896181	1968199	5602.0	1,179.5	134.4	-47.5
06MN-14	6896159	1968229	5575.0	750.3	64.7	-46.5
08MN-15	6895733	1964802	5885.0	1,420.0	0	-90
08MN-16	6895959	1965957	5789.0	200.0	0	-90
08MN-17	6896171	1966431	5762.0	200.0	0	-90
08MN-18	6896692	1967511	5696.0	188.5	0	-90
08MN-19	6896999	1967996	5673.0	200.0	0	-90
08MN-20	6897330	1968583	5743.0	195.0	0	-90
08MN-21	6896218	1968245	5570.0	200.0	0	-90

Note: North American Datum (NAD)83 CA State Plane Zone 0401

Figure 10.1 illustrates the location of Placer-Amex. Sheffield, and Nevoro drillhole collars at the Moonlight deposit and shows the placement of three cross sections.

Figure 10.1 Moonlight Deposit Drillhole Plan



Cross sections A-A', B-B' and C-C' (Figure 10.2, Figure 10.3, and Figure 10.4, respectively) all indicate that mineralization extends between the vertical drillholes completed by Placer-Amex. Cross section B-B' shows low grade mineralization continuing to the east. Cross section C-C' clearly demonstrates mineralization in drill hole 06MN-05 extending to the south and west beneath a poorly mineralized Placer-Amex hole.

Further step out drilling, particularly to the south and west, is likely to intersect additional mineralization.

Figure 10.2 Cross Section A-A'

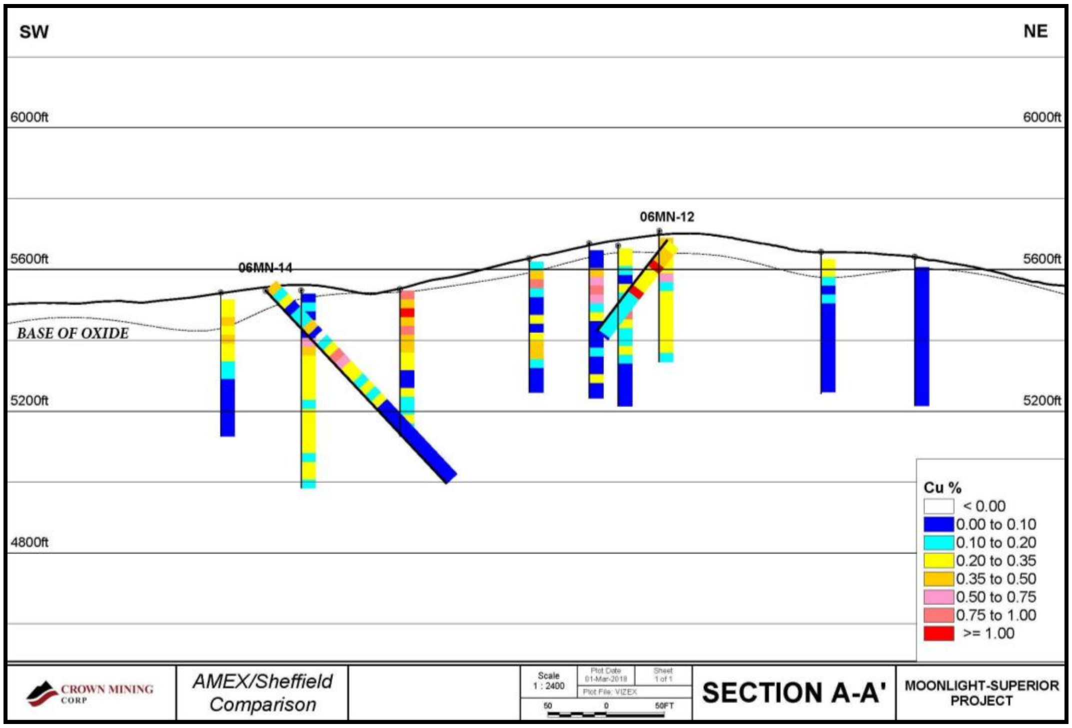


Figure 10.3 Cross Section B-B'

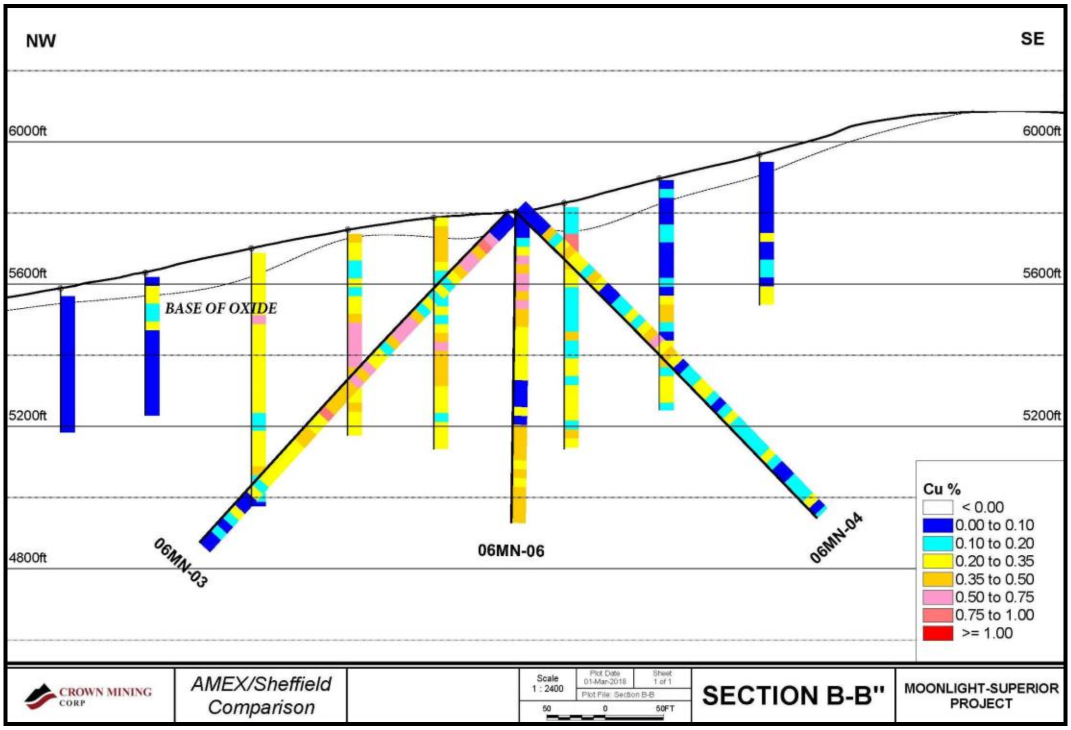
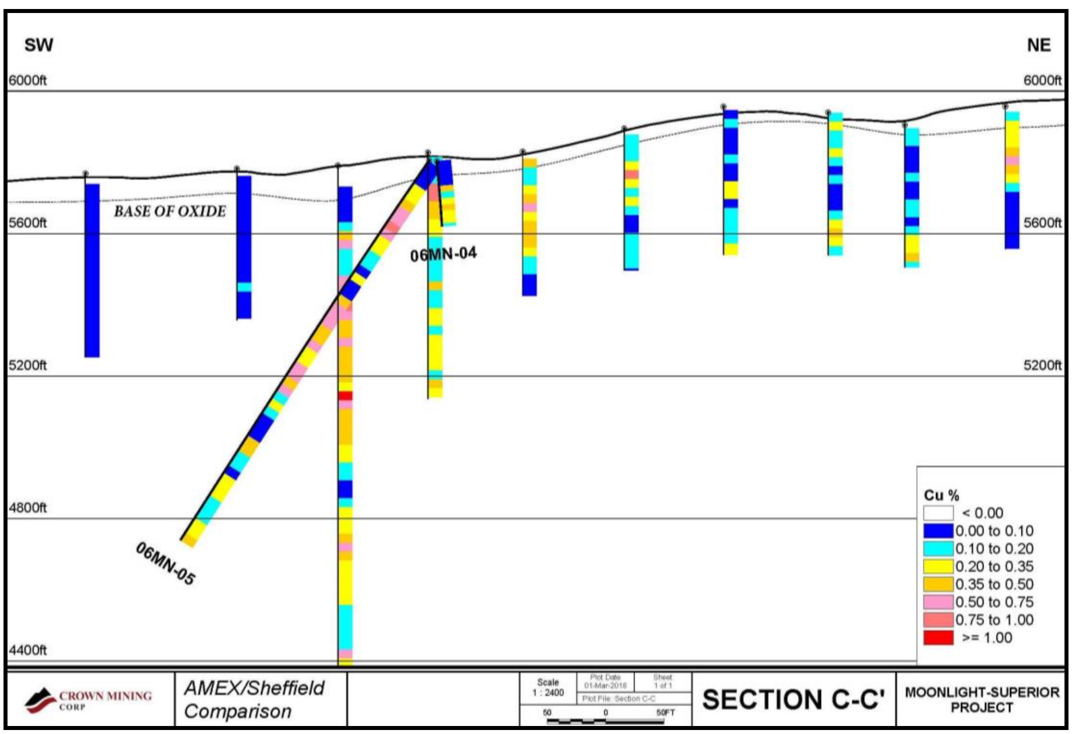


Figure 10.4 Cross Section C-C'



11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 PLACER-AMEX PROGRAM

11.1.1 DOCUMENTATION ISSUES

Core was initially analyzed at the Placer-Amex laboratory at the Golden Sunlight operation in Montana. In mid-1967, Placer-Amex geologists realized that LCD assays from Golden Sunlight were unreliable and instituted a program to re-assay all pulps at Union Assay in Salt Lake City. In a recent (November 2017) telephone conversation with L.O. Storey, the District Geologist at the time, Mr. Storey confirmed the extensive re-assay program undertaken by Union Assay (L.O. Storey, pers. comm. 2017).

Union Assay ceased operations in the late 1990s. There appears to be no record of the ultimate disposition of assay certificates for the LCD drilling. In the same conversation referenced above, Mr. Storey indicated that he had no knowledge regarding the possible location of the assay certificates.

Sheffield recovered a few intact samples of core from the Placer-Amex Superior Mine drilling program, and one sample ofMoonlight deposit drillhole MN-245, from storage at the Superior Mine. They then submitted samples of the core to ALS Chemex in Sparks, Nevada that included one 10 ft sample from MN-245. These analyses provide the only direct comparison with Union Assay results reported by Placer-Amex. The Union Assay result for the 30 to 40 ft interval of drillhole MN-245 was 6.7% copper. The ALS Chemex assay result for the same interval ran 7.42% copper (Wetzel 2009).

Neither the Placer-Amex (1972) summary report or Wetzel (2009) discuss the details of sample handling; sample preparation; QA/QC procedures, including addition of standards, blanks, and duplicates to the sample stream; or analytical methods for the Placer-Amex LCD drilling program. As these procedures are not available for review, the authors must assume that work done by employees of Placer-Amex, a well-known international mining company at the time, was done in accordance with best practices of the time.

11.2 SHEFFIELD/NEVORO PROGRAM

Between 2005 and 2008, Sheffield and its successor company, Nevoro, drilled 21 core holes and 15 reverse circulation holes at the Moonlight deposit in order to verify the Union Assay results reported by Placer-Amex. Moonlight deposit core drilling in 2005 and 2006 consisted of 12 angle holes and 2 vertical holes totaling 11,134.84 ft. Wetzel, in his LCD summary report (Wetzel 2006), stated that "Inspections of cross sections indicates that Sheffield's drilling typically shows higher grades than Placer reported from nearby drill holes." This was the first mention of the possibility that vertical drilling would underestimate grade at the Moonlight deposit. Insofar as the authors can determine, the Sheffield/Nevoro team never carried out a rigorous analysis of the drilling data to determine the effect of angle drilling on the grade of the Moonlight deposit resource. Without such an analysis, the reliability of the Placer-Amex vertical drilling compared to the Sheffield angled drilling remained an open question. In March 2017, author Donald Cameron and Bob Suda undertook a comparison of the effect of angled drilling on the Moonlight resource using three methods:

- exploratory graphic analysis comparing assays along fence diagrams and sections
- Micromine software's paired data analysis tool, comparing the nearest vertical and angled hole assays
- comparison of block models generated using all holes and only vertical holes.

The principal conclusion was that no apparent strong bias exists between the angle holes and the vertical holes. The angled drilling appears to validate the predominantly vertical Placer-era drilling that covers the deposit footprint. Nevertheless, Cameron recommended inclusion of at least a nominal percentage of angled drilling in any future programs as a precaution.

11.2.1 DRILLING PRACTICES

Ruen Drilling operated on a one-shift-per-day schedule. Downhole surveying was accomplished with a Tropari survey instrument. At the drill site, the drill contractor's staff placed core in wooden or cardboard boxes with appropriate footage blocks. In contrast to the NX and BX core favored by Placer-Amex in the 1960s, Sheffield consistently drilled HQ core with the exception of hole 08MN-15, which was reduced to NQ core below 800 ft. At the end of every shift either the drill contractor's staff or a Sheffield/Nevoro geologist transported core to a fenced and locked core logging facility in Crescent Mills, California. Drill collars were located with a Garmin global positioning system (GPS) instrument, presumably a handheld model. Drillholes were re-located with the GPS after reclamation of the drill site and marked with a permanent marker (Wetzel 2009). It should be noted that, depending on local conditions and satellite availability, handheld GPS measurements typically return an error of several feet in easting and northing measurements and an often-greater error in elevation.

11.2.2 SAMPLE PREPARATION

At the logging facility Sheffield/Nevoro staff photographed whole core in the box, measured recovery, documented fractures per meter, or per 5 ft interval, and marked assay intervals. Sheffield/Nevoro geologists completed a geological log of the core before sawing it in half with a diamond saw along a vertical axis. Sheffield/Nevoro staff then determined specific gravity from the sawn core. Samples from half of the core were placed in sample bags and tagged with a preprinted, uniquely numbered sample tag. At this point, standards and blanks were bagged and tagged so as to be indistinguishable from actual samples, and then added to the sample stream. All samples were sealed and stored in rice bags for transport to the ALS Chemex laboratory located in Sparks, Nevada. Samples were transported periodically to the ALS Chemex facility by ALS Chemex personnel or by Sheffield/Nevoro staff (Wetzel 2009).

11.2.3 ANALYSES

All samples submitted to the ALS Chemex laboratory in Sparks, Nevada were first logged into the ALS Chemex tracking system. Each sample then underwent the following procedures:

- Samples were weighed, dried and crushed to 70% minus 2 mm using ALS Chemex method WEI-21.
- A 250 g split was pulverized to 85% minus 75 µm.
- A 0.5 g split was analyzed for 34 elements including copper using inductively coupled plasma (ICP) (method MEICP41).
- In 2005 and 2006 ore grade copper was analyzed by atomic absorption (AA) using method CuAA62 after a four-acid (hydrofluoric acid [HF], nitric acid [HNO_3], perchloric acid [$HClO_4$], and hydrochloric acid [HCl]) digestion. After 2006 a 0.4 g split was dissolved using the same four-acid digestion and analyzed for ore grade copper by ICP using the OG62Cu method.
- Sulfuric acid soluble copper was assayed by ALS Chemex method CuAA05, wherein the sample is leached at room temp in 5% sulfuric acid, agitated for an hour and analyzed by AA.

After reviewing ALS Chemex assay certificates, the author is able to confirm that the analytical procedures listed above and in Wetzel (2009) are an accurate representation of the methodologies used.

11.2.4 QA/QC PROCEDURES

According to Wetzel (2009), Sheffield applied the following QA/QC procedures to their 2005-2008 LCD drilling program. An uncrushed blank sample, reportedly comprising unmineralized quartz monzonite (Cavey and Giroux 2007), was inserted as the first sample in each drillhole as a check for contamination from previous samples. Standard samples inserted into the sample stream included five commercial standards from CDN Laboratories in Surrey, British Colombia, Canada with values ranging from 0.13% to 5.07% copper and two standards prepared by CDN Laboratories from core saw cuttings from Moonlight core. The prepared standards had values of 0.5% and 1.038% copper. Wetzel (2009) describes these values as *"accepted"*, but the acceptance criteria have not been retrieved for these materials. According to Wetzel, one of these seven standards was inserted into the sample stream every 20th sample totaling 5% of any given sample sequence.

An examination of drill logs by the author shows that, with four exceptions, standards were inserted every 20th sample at a minimum. Standards added to these four drillholes made up a minimum of 4% of the sample sequence. Standards added to five of the 21 drillholes exceeded 5% of the total sample sequence for those holes. Four commercial standards (CDN-CGS-1, 4, 5, and 7) and the two prepared standards (SHLG and SHHG) were inserted into the Moonlight deposit drillhole sample stream. Four drillholes were submitted without blanks, ten drillholes contained one blank, and the remaining seven drillholes ranged from two to seven blanks. The logs for drillholes 05MN-01 and 05MN-02 note the insertion of one duplicate sample each. In the remaining drillhole logs no duplicates were noted.

Five control charts for copper and five for gold compose Figure 11.1 and Figure 11.2, respectively. Four of the charts in each figure compare ALS Chemex copper and gold assays for the four CDN Laboratories standards used at the Moonlight deposit (CDN-CGS-1, 4, 5, and 7) with their certified value. The fifth chart in each figure illustrates gold or copper values for blanks used in the 2005-2006 drilling at Moonlight.

Figure 11.1 Copper Control Charts

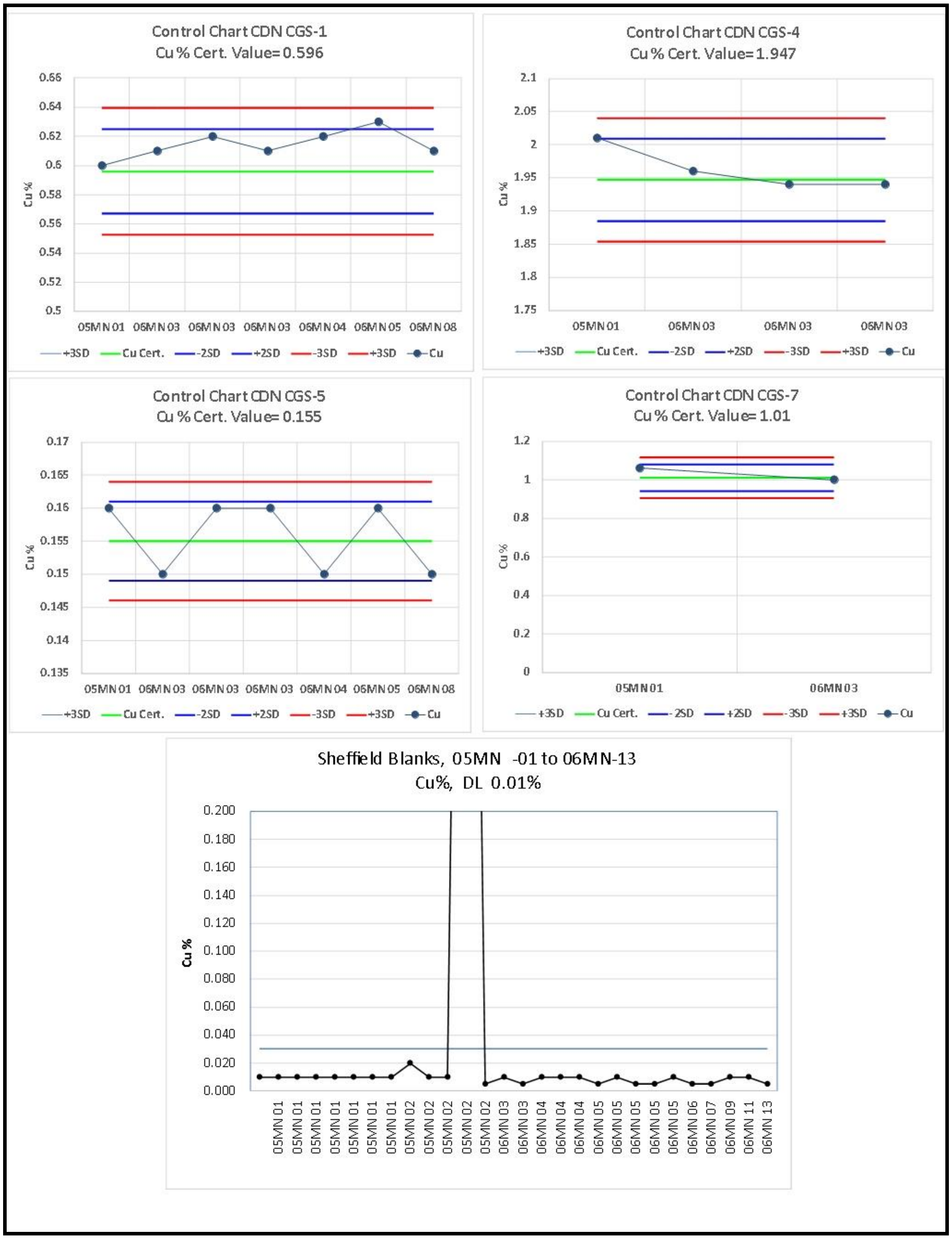
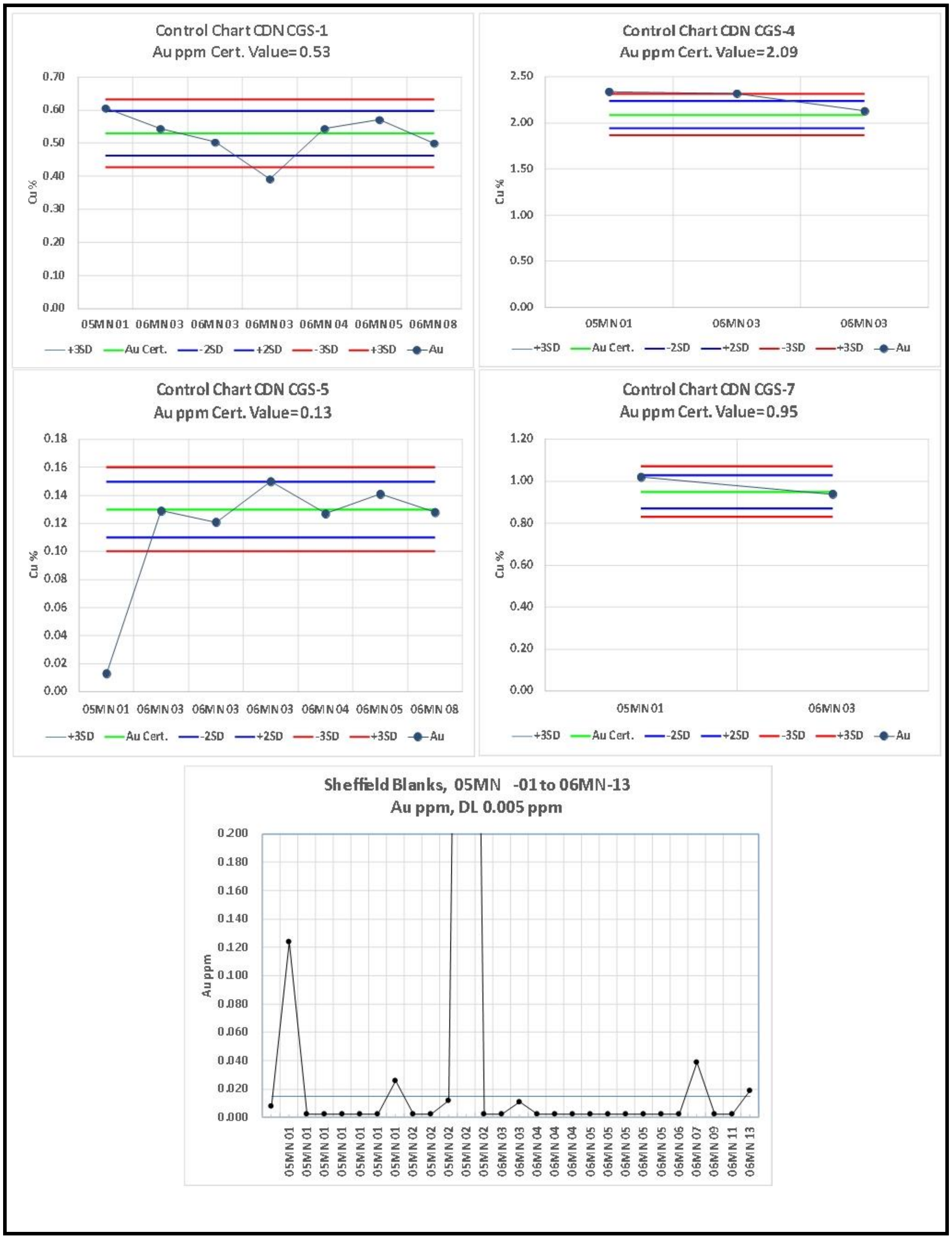


Figure 11.2 Gold Control Charts



In the copper control charts, all standards except one, CDN-CGS-1 in drillhole 06MN-05, assayed at less than two standard deviations from their certified values. One copper blank failed in drillhole 05MN-02. The same blank failed the gold assay suggesting that a mineralized sample may have been mistakenly included in the sample stream as a blank.

The gold control charts show that three standards had results outside of ±3 standard deviations, although only two, the low results in CGS-1 and 5, are fairly certain failures. More assay lots would be needed to ascertain appropriate failure criteria for all of the standards for which gold results seem consistent, if high-biased. The Sheffield records do not contain any information on silver assay control if it was performed.

11.2.5 Comments on Sample Preparation, Analyses and Security

Supporting information for drill results reported on Placer-Amex logs and electronic files, including survey, assay procedures and certificates, and QC measures have not yet been located. Drilling performed in 2005-2006 by Sheffield, with the exception of reliable collar surveying, appears to have been conducted in conformance with current industry best practices; QA/QC results for copper in this drilling campaign are acceptable. Results from Nevoro's 2008 drilling are, according to Wetzel, nearly identical with the twinned Placer-Amex holes. The ALS Chemex assay for the 30 to 40 ft interval of MN-245 returned 7.42% copper versus the original Union Assay result of 6.7% copper, a reasonably close result. Cameron, in his summary report (Cameron 2017), which found little difference between results from Sheffield's angle holes and results from the Placer-Amex vertical drilling stated, “This could be taken to validate the predominantly vertical drilling that covers the wider area.” All of the above suggests that Union Assay copper results reported by Placer-Amex for the Moonlight deposit are reasonably reliable, despite the lack of certificates and other supporting information for the 1960s-era campaign.

The 2005-2006 program QA/QC results for gold should have triggered a re-assay of selected samples. To date, this author has not seen specific evidence that a re-assay took place. In his 2009 report Wetzel states that “A number of high unexpected gold values ranging from 1 to 12 g/t were reported from several ALS Chemex jobs. Reanalysis of these samples showed >100 ppb Au for all of the suspect values.” He does not identify the deposit or deposits associated with the failed analyses, but it is clear that he is referencing the 2005-2008 time period. Based on the 2005-2006 QA/QC results for gold, and the failed analyses noted by Wetzel, gold results from Sheffield's 2005-2006 drilling at Moonlight must be considered to be less reliable than the copper results.

12.0 DATA VERIFICATION

Donald E. Cameron (CRC), an independent QP for this report, undertook steps to verify and validate Moonlight-Superior Project information in areas critical to the Mineral Resource base by way of a site visit and follow-up review and attention to key audit points in common industry use. The deposit has a long history with substantial information generated in the 1960s and 1970s by a major mining company, Placer-Amex, and its predecessors, including surface mapping, soil and rock chip sampling, geophysics and drilling. Several historic resource estimates were performed for the deposit up to the present. While some of the original information is lost for the present, notably assay certificates, most drillholes are cased at the surface and many can be located in the field. There is little doubt of a substantial effort made to the standards of the day which are pre-NI 43-101. Moreover, Sheffield drilling in 2005-2006 can be verified in nearly all key aspects and constitutes approximately 15% of the drillhole data. It should be considered a validation of the legacy Placer-Amex database, if not in its entirety, at least to the degree it is used in Mineral Resource estimation, as discussed in Section 14.0.

In addition to reviewing general practices, procedure documents, and in-house reports, it is standard to select specific areas for detailed checking in order to validate data and ensure the integrity of databases. The author followed this path in auditing the Moonlight deposit database and this was the focus of a site visit on September 26-27, 2017 and subsequent review of specific data at intervals between March 2017 and November 2017.

The drillhole database for the Project is a compilation of two spreadsheets separately constructed by previous workers for the two drilling campaigns, Placer-Amex and Sheffield. The Placer-Amex database has a long history--the oldest version encountered was a scan of a dBase software printout from 1990. Original assay certificates could not be found from Placer-Amex's Golden Sunlight mine lab and Union Assay Lab of Salt Lake City, Utah, a reputable lab that closed in the late 1990s, and reference to which is hand-written on some of the log

To check the Placer-Amex drill logs, CRC made a list of all drill holes with more than five 25 ft copper composites, comprising 109 drillholes. Six of these were selected at random, or 6% of the Placer-Amex drill hole list and 7.5% of the assays. Each assay interval was checked for its copper, gold and silver entries. A table of failures was compiled showing the results for each drillhole (Table 12.1).

Table 12.1 Table of Failures for Placer-Amex Campaign Portion of Drillhole Database

Hole	Total Assays	Cu	Au	Ag
ML-27	47	6	0	3
ML-50	40	0	0	0
ML-201	60	1	0	3
ML-221	71	5	0	4
ML-302	51	1	0	0
ML-411	41	0	0	1
Total	**310**	**13**	**0**	**11**
Failures	**-**	**4%**	**0%**	**4%**

Two types of copper assay errors were made: 1) Transcription; and 2) Out-of-sequence. The worst of the first type was an assay entered as 0.295% copper, but 1.295% was written in the log. All but one of the others was minor. The sequence errors affected two or more records. Several copper errors were cross-checked against the 2007 database where they were also present, and they may represent longstanding issues. Silver errors were almost all some sort of keyboard error that affected seemingly random records. These were entered as 0.0X oz/st silver where the correct entry was 0.00 oz/st silver.

In addition to the transcription errors, CRC noted inconsistencies with the handling of gold, and to a much less extent, silver values entered as ‘0’, ‘None’ and ‘Tr’ in the logs. In two drillholes, ‘Tr’ was set to 0.003 oz/st gold and ‘None’ as 0.001 oz/st gold. In some holes, ‘Tr’ was entered as ‘0’ and in others to 0.002 oz/st gold. In the 2007 study, ‘0’s were set to 0.001 oz/st gold and ‘Tr’ to 0.003 oz/st gold. In this study, the assays were maintained without corrections or changes of any kind, including the ones made in 2007. In all, 8,788 assays out of 9,745 Placer-Amex assays have a below detection (BDL) or detection level (DL) assay entered in the database. Of these, 5,840 are (appropriately) entered as '0', another 3,000 have BDL values entered as 0.002 oz/st gold, with a few exceptions (30). Unfortunately, with a detection level of 0.005 oz/st gold in the historic assaying and the preponderance of the deposit having gold grade below this, the handling of BDL and DL values biased, especially, the gold database. In summary, the Placer-Amex database checks suggest the need for a 100% check of the copper assay database and building a new table of gold and silver composites.

The Sheffield database was reconstructed independently by two persons including CRC by pasting the values entered in Microsoft® Excel drill logs into a compilation sheet. The numeric fields in the two sheets were sorted identically and compared to ensure no differences. Sheffield gold and silver assays stated in ppm were correctly converted to troy oz/st using a conversion factor of 34.3857, correcting a longstanding database error. This compilation was merged with the Placer-Amex data sheet to compose the estimate database. When the merged Placer-Amex-Sheffield assay file was loaded to Micromine, database validation utilities were run to check for assay and survey records greater than the drill hole depth, overlapping assay intervals and repeated assay intervals. Spot checks of ALS Chemex assay certificates were made by CRC to verify individual values, assay method and units.

While no obvious problems were noted during review and importation of the Sheffield data, a limited audit of two of the thirteen drill holes was undertaken to check the values of copper, gold and silver (Table 12.2).

Table 12.2 Table of Failures for Sheffield Campaign Portion of Drillhole Database

Hole	Total Assays	Cu	Au	Ag
06MN-03	236	3	4	4
06MN-08	151	4	4	5
Total	387	7	8	9
Failures		2%	2%	2%

In this case, assay data could be pasted into the Sheffield database and compared across columns. The errors comprised data offset a single row instead of transcription errors.

12.1 Drillhole Collars

Collar locations for Placer-Amex drilling are stored with decimal precision in a local coordinate system. These were converted by Sheffield to NAD27 State Plane coordinates in the 2005-2007 period based on limited ground surveying. A few Placer-Amex drill holes have rounded local coordinates that suggest that they were never surveyed. Some Sheffield collars may have been surveyed but some were shot in with a handheld GPS unit. No survey notes are available for either drill campaign. For the current study, all data were converted to NAD83 California State Plane 0401 and compared to the new topography at 6 ft resolution. The Placer-Amex drillholes required an average adjustment of <1 ft to plot on the new surface, but the Sheffield drillholes required an average vertical translation of 30 ft. Collars were adjusted by a drape onto the surface using Micromine software.

12.2 QA/QC

CRC reviewed the QA/QC discussion in the previous NI 43-101 Technical Report (Cavey and Giroux 2007) and compilations of QA/QC information from the historic campaigns prepared by Crown Mining consultants for the current study and presented in Section 11.0. In addition, CRC generated control charts for blanks and standards used in the Sheffield program, finding few significant failures, as discussed in the previous chapter. Sheffield twins of Placer-Amex drillholes were compared and found to generally validate the previous drilling. CRC believes that the quality of the program is sufficient to support its use in estimation of Mineral Resources.

No Placer-Amex QA/QC data is known or available for inspection. This is certainly regarded by the authors as an area of weakness that must be addressed by additional confirmation drilling.

CRC conducted a paired data analysis whereby nearest Sheffield-Placer-Amex pairs were extracted in a file using a Micromine software utility. The pairs were plotted on scatterplots and Q-Q plots, and block models were constructed using each data set. The campaigns were also visually compared on plans and sections. The principal conclusion from all of the tools employed was that there is no apparent strong bias between the angled Sheffield and Placer-Amex vertical drilling.

12.3 SITE VISIT

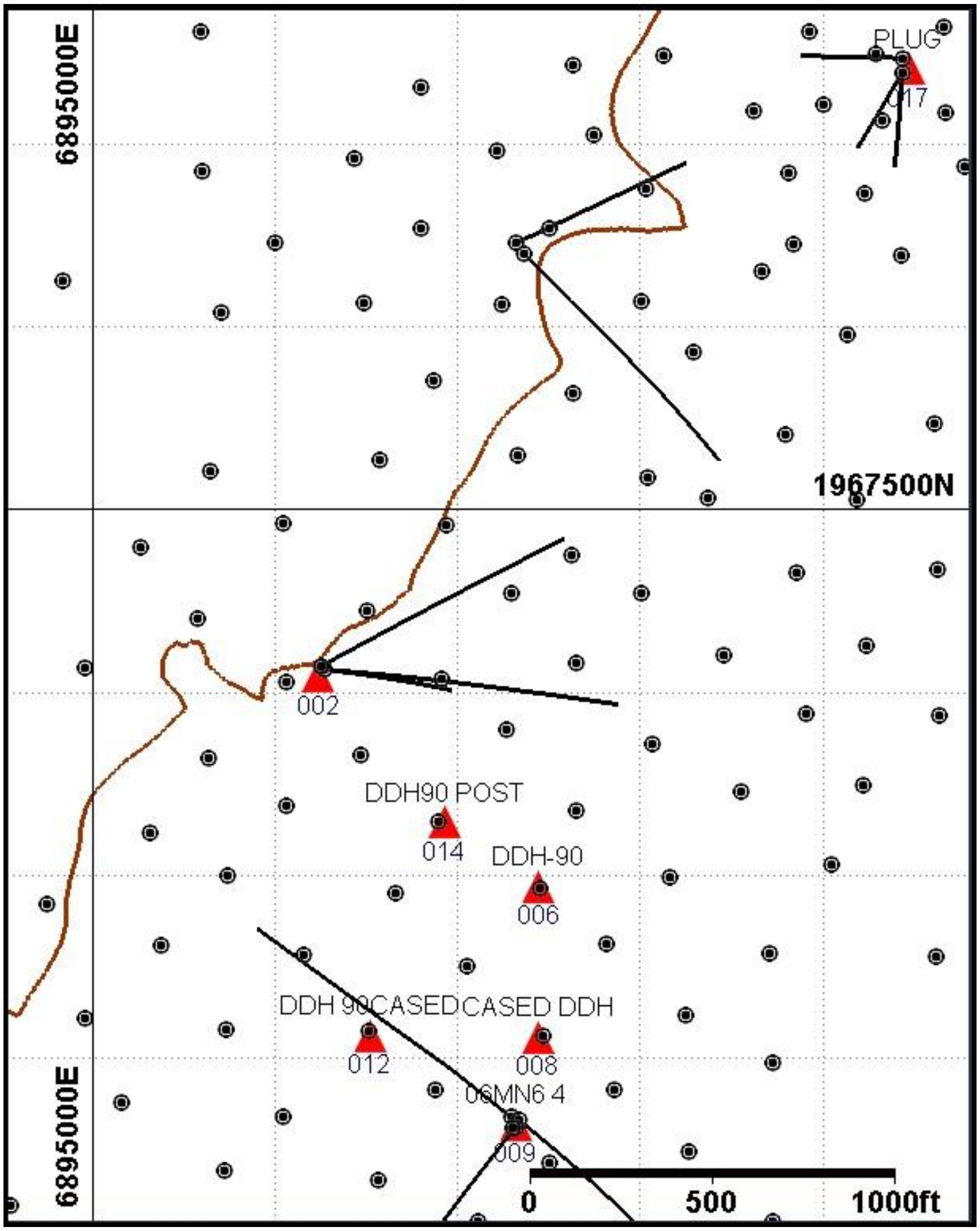
CRC visited the Moonlight-Superior Project for two full days on September 26-27, 2017, accompanied by a consultant for Crown Mining. September 26, 2017 was spent at the Moonlight deposit with short visits made to Engels and Superior prospects. At Moonlight, the goal was to visit three of four Sheffield drill sites to verify their coordinates in the database and find any evidence of hole markers. A search was also made for Placer-Amex-era drillholes or drill platforms. The three Sheffield sites were located by GPS where expected and hole plugs found in two cases. The GPS coordinates were compared to the drillhole database coordinates in Universal Transverse Mercator (UTM) World Goedetic System (WGS)84 as recorded on Sheffield logs (Table 12.3).

Table 12.3 Comparison of Sheffield Database Coordinates with Handheld GPS (UTM, metric)

Hole	Easting	Northing	Elevation	GPS-DB X	GPS-DB Y	GPS-DB Z
05-MN-01	686937	4454708	5,518	-1.0	-6.0	43.6
06-MN-06	687105	4454326	5,804	1.0	5.0	2.2
06-MN-11	687420	4455209	5,755	4	6	-36

One of the elevations is grossly in error, possibly due to tree cover. Five Placer-Amex cased holes with illegible drillhole markers were also located. The waypoints, converted to NAD83 property coordinates are shown relative to the database drillhole collars in Figure 12.1

Figure 12.1 GPS Checks of Drillhole Database Collars



With the noted exception of elevation measurements in the Sheffield drillholes, the GPS checks give some confidence in the previous surveying, particularly when roads are plotted and compared with CRC's location notes.

Several other items were reviewed and verified while on site:

- accessibility and local infrastructure
- inspection of outcropping mineralization and prospect pits
- review of core and core storage

- collection of samples for independent verification of mineralization.

Access to the site is good with roads maintained by state, county and private timber companies. We reviewed mineralization at Moonlight in several prospect pits, noting any obvious mineralogical and structural controls, as well as variations in alteration and rock type. One of the key differences between the Placer-Amex geology interpretation and drillhole coding with Sheffield's is the variety of igneous phases identified in the former, and the simple scheme of the latter. This could not be resolved in the short field and core shack time, but the Sheffield interpretation seemed tenable if the variations in the igneous host are simply due to alteration intensity.

September 27, 2017 was spent inspecting available from intervals in three drillholes selected by CRC. The intervals reviewed were:

- 05MN-01 10 to 100 ft/150 to 550 ft/650 to 825 ft
- 06MN-04 20 to 45 ft/45 to 145 ft/275 to 340 ft
- 06MN-11 0 to 65 ft/65 – end of hole (EOH).

Each was selected to review a particular feature of the deposit: oxidation, high grade, low grade, tourmalinization, possible intrusive phases and grade breaks. Among mineralization features, several occurrences of native copper, copper oxide minerals and copper sulfides were noted during the review. Variations in the quartz monzonite (QM) appeared to be a function of alteration; multiple intrusion phases ranging from granodiorite to granite as mapped on the surface may exist, but the grouping of the intrusions into a single QM unit fit the observations made in the core review. Evidence of dominant high-angle structural controls to copper mineralization, a possible problem for characterization of mineralization with vertical drilling, was not observed in outcrop or core.

The core had been sawed by Sheffield previously and stored in wooden boxes with screwed lids inside locked storage buildings. The core run blocks, sampling marks and tags were intact and there was evidence of orderly and standard procedures employed by Sheffield in the original sampling. The location of core from the Placer-Amex campaigns, if any still exists, is unknown and could not be inspected.

12.4 CHECK SAMPLES

CRC took its own samples for independent verification of copper mineralization in outcrop and for comparison to assayed drill intervals. The outcrop samples were selected and taken by CRC, placing them in sample bags sealed with his own cable tie. The core samples were taken from quarter-sawed sections of core halves remaining from the Sheffield program. CRC place each section of core himself into the sample bags with an identifying tag and sealed them with his own cable tie. All samples were in CRC custody or under lock and key at all times and were personally delivered by CRC to the Bureau Veritas's Sparks, Nevada laboratory for analysis of copper, gold and silver. Results are listed in Table 12.4.

Table 12.4 Independent Sampling Results for Moonlight Copper Deposit

Sample	Type	Interval (m)	Cu (ppm) (BV)	Au (ppm) (BV)	Ag (ppm) (BV)	Cu (DB)	Au (DB)	Ag (DB)
68901	Chip, prospect pit	N/A	1,230	0.021	3.2	N/A	N/A	N/A
68902	Chip, outcrop w/ CuOx	N/A	1,401	0.009	1.2	N/A	N/A	N/A
68903	Chip, pit near 06MN-06	N/A	33,010	0.138	>100.0	N/A	N/A	N/A
68904	Chip, outcrop by 06MN-04	N/A	3,717	0.009	2.3	N/A	N/A	N/A
68905	Chip, drill site 07MRC-05	N/A	9,405	0.012	1.4	N/A	N/A	N/A
68906	Drill core, ¼, 05MN-01	102-104	3,854	0.016	4.9	4,200*	0.026	4.0
68907	Drill core, ¼, 06MN-11	8-10	2,532	0.010	1.9	2,500	<0.005	2.7

Note: *Sample interval 102.03 – 108.6 m

CRC's outcrop sampling verified the presence of significant copper in several locations on the property and in proximity to collars of the historic drillholes that are located inside the deposit outline. For the core quarters, the corresponding Sheffield assay is listed for comparison. Results show excellent correspondence between the database values and the check samples.

SG determinations were made by Bureau Veritas on two specimens from each of the two core intervals were submitted for assays. The SG of fresh material in drill hole 05-MN-01 was measured as 2.72, whereas the near-surface oxidized specimen from 06MN-11 had an SG of 2.49. The result for the fresh sample is close to the 2.67 average value calculated for all historic samples. The oxide check sample result suggests the need for more extensive SG determinations.

12.5 COMMENTS ON DATA VERIFICATION

Data verification included examination of assay certificates and cross-checks against the assay values entered in the database, comparison and correction of collar coordinates with the surface topography, inspection of outcrops, drill hole collar locations and drill core, independent check samples and a review of QA/QC.

In the opinion of CRC, Sheffield drilling programs substantially complied with current Exploration Best Practices recommended by CIM and the drilling information is suitable for estimation of Mineral Resources under Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines (CIM 2003). A large portion of the Sheffield core is preserved and can be examined or tested. Original assay certificates are complete.

CRC notes that Placer-Amex-era drill holes are not surveyed and original assay certificates have not been located. While these are significant deficiencies, there can be little doubt based on the logs and the extensive contemporaneous correspondence and reporting related to the Placer-Amex campaigns in the historical records available that the drilling and assaying occurred and was conducted according to the standards of care at the time. Furthermore, Sheffield drilling generally confirms the Placer-Amex copper results, as discussed in Section 11.0 of this report. Based on these findings, copper and silver assays from Placer-Amex drill campaigns are also suitable for use in Mineral Resource estimation.

Recommendations for advancing the Moonlight deposit based on the data verification activities include:

- additional surveying to more accurately tie marked drill hole collars to the topographic grid
- re-construct the Placer-Amex portion of the assay table from the logs and Sheffield portion from certificates
- collect additional specific gravity samples
- perform additional infill drilling to reduce proportion of Placer-Amex information in the database and to replace Placer-Amex precious metal assays deposit-wide.

These activities will ensure that the deposit grade and tons are accurately estimated from data that have been fully validated by information collected in conformance with current industry Best Practice.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 REVIEW OF HISTORICAL METALLURGICAL TESTING

In June 1956, Haley provided a report to Indian Valley Chemical Corporation. This was based upon his experience working on the Property in 1915. He and a colleague, John Scott, proposed a heap leach coupled with a sulfide roasting operation.

In 1967, Canadian Exploration Limited carried out a series of metallurgical investigations in a series of six reports issued to American Exploration and Mining Co. These were entitled *Venture 63 – Moonlight Metallurgical Investigations*. Only the transmittal communications concerning these studies were available, except for parts of Report No. 1, which focused on comminution and flotation testing. The studies indicated that a grind of 80% passing 100 mesh was sufficient for a copper recovery of 90% using Z-200 as a collector. Further, by use of cyanamid xanthate S-3501 on Moonlight material, a coarser grind of 50 to 60% passing 100 mesh achieved the same recovery. As well, the Bond Work Index for the Superior material was 18.8 kWh/st, while the Moonlight material was 20.0 kWh/st. Further testing was also reported on the oxide portion of the materials.

Placer-Amex completed several metallurgical studies during the early phases of drilling in the LCD to quantify the recovery of copper and silver. Most studies were focused on the copper leach extraction of copper oxide mineralization at the Moonlight deposit, oxide mineralization at the Engels Mine, and sulfide mineralization at the Superior Mine. In 1989, toward the end of their tenure on the Property, Placer-Amex completed metallurgical testing on five bulk composite samples of cores from the Moonlight deposit. Placer-Amex collected composite core samples and sent to the Kappes, Cassidy & Associates (KCA) laboratory in Sparks, Nevada. Three of the five samples contained oxide material, one each from the North, Central and South oxide zones. The remaining two samples were sulfide material.

KCA completed sulfuric acid leaching tests utilizing 500 g head splits from each of the five composites. Copper recoveries from the oxide samples after 72 hours of tests were quite different for the various samples. Results from the South oxide composite returned a 97.9% recovery. Recoveries for the North oxide and Central oxide samples were considerably lower at 52.8% and 55.8%, respectively. The results from the acid leaching tests on the sulfide composites were predictably low and consistent at 24.8% and 24.6% for the two samples. Sulfuric acid consumption ranged from 37 to 108 lb/st. Ten kilograms of finer than one inch crushed composite was also leached with similar results. The South oxide composite returned a 92% recovery. The Central oxide composite and the North oxide composite returned 65% and 57% recoveries, respectively. The sulfide composites returned a 27% copper recovery.

In July 1989, Placer-Amex completed a metallurgical study with KCA performing ferric sulfate leaching tests. This was done to evaluate heap leaching of the deposit. Sulfide and oxide materials were tested. This was a continuation of previous work completed earlier in 1989 by KCA. Both small scale beaker tests using 50 g of material were carried out along with larger tests utilizing 500 g of material. The results of this test program showed that the use of ferric sulfate in the leach solutions will increase copper recovery. However, to determine if ferric sulfate addition to the leach solution would be economical, additional test work to optimize both acid and ferric sulfate additions would be required.

In August 1989, Placer-Amex completed preliminary acid bottle roll leach tests at Metcon on three samples from this deposit. The main objective of the study was to determine the amount of copper that may be recovered from a finer than 100 mesh sample leached for 24 hours in a 10% sulfuric acid solution. Copper recoveries above 60% were observed using sulfuric acid on the samples with higher non sulfide content. Moreover, some of the sample treated consumed up to 180 lb of sulfuric acid per ton.

Further work is warranted to determine the reason for the poor copper extraction in oxides in the North and Central oxide zones. Mineralogical studies would probably indicate whether or not complete oxidation occurs or if some oxide mineralogy is present that is not amenable to the leaching. Flotation tests on sulfide material to determine grind requirements, expected recoveries, and concentrate grades will also be needed at some time.

In 2007, Sheffield drilled 15 reverse circulation holes in the Moonlight deposit to test and confirm the copper oxide Mineral Resource defined by Placer-Amex. Drilling was completed on all areas of the deposit, with most holes twinning Placer-Amex holes, which had defined the copper oxide Mineral Resource for Placer-Amex. Preliminary leachability tests were completed on all reverse circulation samples. Geochemical analysis of the drill samples included soluble copper assays on all samples using a sulfuric acid leach analysis (method code Cu-AA05). The oxide leach recoveries ranged between 49 and 78% copper.

Sheffield also completed bottle roll tests on seven reverse circulation drill samples using KCA. Two samples were from Moonlight reverse circulation holes 07MRC-03 from 20 to 25 ft and 07MRC-06 from 10 to 15 ft. Samples were coarse-crushed. Distilled water was added to make a slurry to which sulfuric acid was then added. The samples were bottle-rolled for 144 hours. The 07MRC-3 sample returned a 65% copper recovery. The 07MRC-06 sample returned an 81% copper recovery. Silver recovery was negligible in both samples.

Additional bottle roll and column leach tests were conducted on Moonlight deposit oxide mineralization samples in 2013. A company interested in acquiring the Property, Sandfield Resources, through a company known as Exploration Alliance, S. A., collected core samples from available Moonlight Sheffield core. These samples were composited into six samples which were submitted to SGS Labs in Tucson, Arizona for bottle roll and column leach testing. The specific drillholes from which the composite samples were collected are not known. The composited samples of crushed cores were sized and column leach tests were conducted on the size fractions of 1 in, ¾ in, and ½ in Closed column tests were cured for five days then leached for 30 days. The main results of the study concluded that the ½ in size fraction had the best copper leach recoveries of 89%. A sulfuric acid cure dosage of 15.5 lb/st (7.75 kg/t) gave the optimum cure dosage to obtain the highest copper extraction (SGS 2013).

13.2 CURRENT METALLURGICAL TEST WORK

In 2017, Crown Mining requested Allihies to complete a metallurgical test program, in collaboration with Continental, to confirm previous work and carry out testing associated with the Moonlight-Superior deposit. This deposit consists of the Moonlight, Superior, and Engels deposits. The material provided by Crown Mining was identified as follows (as identified by the shipping documents and sample bags): Moonlight Sulfide, Moonlight Oxide, and Superior Sulfide. The intention of the test program was to confirm effective flotation reagent conditions and demonstrate the recoveries and concentrate quality that can be achieved with the tested material. The baseline conditions were developed based on previous work so the results would be comparable. The scope of the program included sample preparation, sample characterization, grinding tests, and batch flotation test work that included both rougher and cleaner testing.

Table 13.1 to

Table 13.3 show the results of the automated mineralogical analysis of each material.

Table 13.1 Moonlight Oxide Automated Mineralogical Analysis

Mineral	Chemistry	Percentage
Quartz	SiO_2	32.94
Orthoclase	$KAlSi_3O_8$	24.70
Albite	$NaAlSi_3O_8$	18.56
Andalusite	Al_2SiO_5	15.19
Hematite	Fe_2O_3	5.81
Chlorite	$(Fe,Mg,Al)_6Si_4O_{10}(OH)_8$	2.07
Calcite	$CaCO_3$	0.38
Dolomiate	$CaMg(CO_3)_2$	0.09
Barite	$BaSO_4$	0.07
Apatite	$Ca_5(PO_4)_3OH$	0.07
Ilmenite	$FeTiO_3$	0.05
Malachite	$Cu_2CO_3(OH)_2$	0.04
Anorthite	$CaAl_2Si_2O_8$	0.02
Chromite	$FeCr_2O_4$	0.01
Tetrahedrite	$(Cu,Fe)_{12}Sb_4S_{13}$	<0.01
Bornite	Cu_5FeS_4	<0.01
Galena	PbS	<0.01
Titanite	$CaTiSiO_5$	<0.01
Chalcopyrite	$CuFeS_2$	<0.01

Table 13.2 Superior Sulfide Automated Mineralogical Analysis

Mineral	Chemistry	Percentage
Orthoclase	$KalSi_3O_8$	30.24
Quartz	SiO_2	25.99
Albite	$NaAlSi_3O_8$	22.79
Chlorite	$(Fe,Mg,Al)_6Si_4O_{10}(OH)_8$	9.87
Hematite	Fe_2O_3	3.52
Andalusite	Al_2SiO_5	2.04
Chromite	$FeCr_2O_4$	1.92
Chalcopyrite	$CuFeS_2$	1.52
Anorthite	$CaAl_2Si_2O_8$	0.75
Calcite	$CaCO_3$	0.44
Ilmenite	$FeTiO_3$	0.40
Titanite	$CaTiSiO_5$	0.28
Bornite	Cu_5FeS_4	0.10
Apatite	$Ca_5(PO_4)_3OH$	0.14
Galena	PbS	<0.01
Malachite	$Cu_2CO_3(OH)_2$	<0.01
Dolomite	$CaMg(CO_3)_2$	<0.01
Tetrahedrite	$(Cu,Fe)_{12}Sb_4S_{13}$	<0.01
Barite	$BaSO_4$	<0.01

Table 13.3 Moonlight Sulphide Automated Mineralogical Analysis

Mineral	Chemistry	Percentage
Quartz	SiO_2	33.13
Orthoclase	$KAlSi_3O_8$	24.83
Albite	$NaAlSi_3O_8$	17.01
Andalusite	Al_2SiO_5	13.35
Hematite	Fe_2O_3	6.92
Chlorite	$(Fe,Mg,Al)_6Si_4O_{10}(OH)_8$	1.91
Calcite	$CaCO_3$	0.65
Chromite	$FeCr_2O_4$	0.62
Bornite	Cu_5FeS_4	0.51
Dolomite	$CaMg(CO_3)_2$	0.30
Apatite	$Ca_5(PO_4)_3OH$	0.25
Galena	PbS	0.24
Barite	$BaSO_4$	0.08
Anorthite	$CaAl_2Si_2O_8$	0.07
Ilmenite	$FeTiO_3$	0.07
Chalcopyrite	$CuFeS_2$	0.03
Tetrahedrite	$(Cu,Fe)_{12}Sb_4S_{13}$	0.03

Titanite	$CaTiSiO_5$	<0.01
Malachite	$Cu_2CO_3(OH)_2$	<0.01

A key aspect is the absence of pyrite in the Superior Sulfide and Moonlight Sulfide materials, which is advantageous in the flotation of the copper mineralization noted.

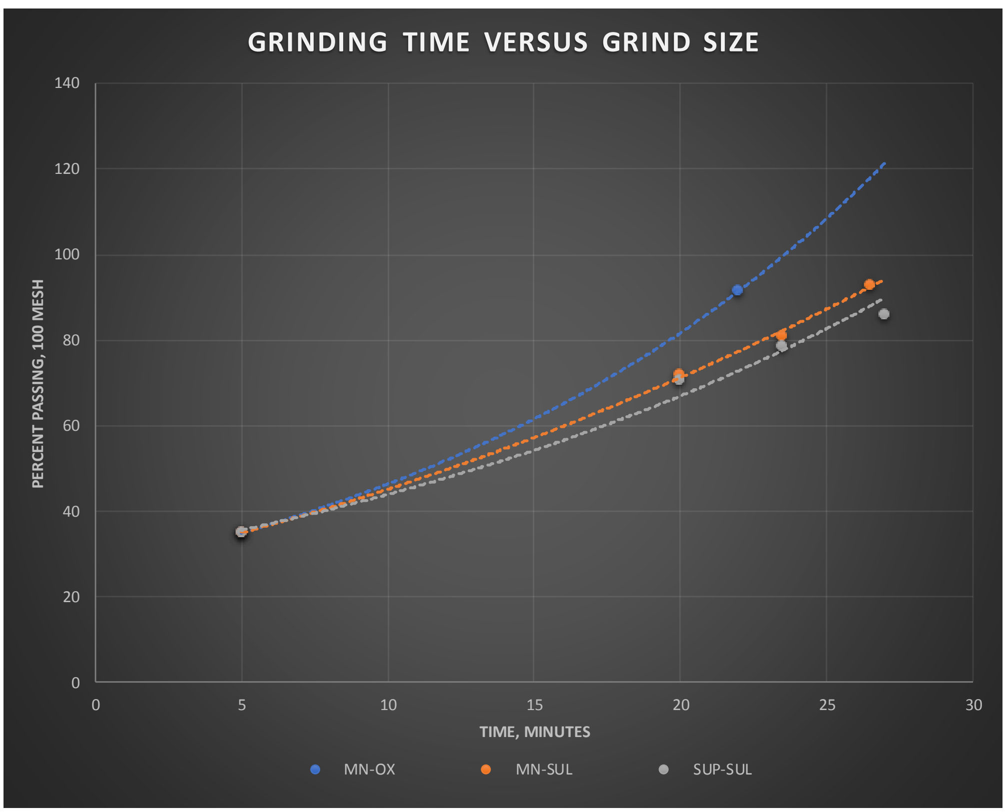
The Bond Work Index testing results for the three composites were as follows.

- Moonlight Oxide 18.1 kWh/st
- Superior Sulfide 21.3 kWh/st
- Moonlight Sulfide 19.7 kWh/st

Based on these Bond Work Index values, these materials would be classified as very hard.

Grinding testing using a rod mill was performed on all three composites to identify the laboratory requirements for grinding to 80% passing 100 mesh (149 µm). Figure 13.1 summarizes these results.

Figure 13.1 Grinding Time Versus Grind Size



For the Superior Sulfide and Moonlight Sulfide materials, both rougher and cleaner flotation testing were then undertaken. The testing was based on optimizing the responses to copper grade and copper recovery.

The following variables were set for testing:

- grinding size – P_{80} 100 mesh, P_{70} 100 mesh, P_{90} 100 mesh
- conditioning time – 5 minutes
- float time – 6 minutes
- reagent type – Aerofloat 3477 and potassium amyl xanthate (PAX) – fixed addition rates of 0.06 and 0.03 kg/st, respectively
- frother – methyl isobutyl carbinol (MIBC) – as required
- pH – 10.0
- pulp bulk density – 40% solids by weight.

Table 13.4 identifies the rougher flotation test results, shown in decreasing order based on grind size.

Table 13.4 Rougher Flotation Test Results

Material	Grind Size, Passing 100 mesh (%)	Cu Recovery (%)	Au Recovery (%)	Ag Recovery (%)	Back Calculated Cu Head Grade (%)	Back Calculated Au Head Grade (oz/st)	Back Calculated Ag Head Grade (oz/st)	Cu Rougher Conc.. Grade (%)	Au Rougher Conc.. Grade (oz/st)	Ag Rougher Conc.. Grade (oz/st)	Cu Tailings Grade (%)	Au Tailings Grade (oz/st	Ag Tailings Grade (oz/st)
Moonlight Sulfide #3	72	81.5	60.1	34.0	0.57	0.002	0.41	11.3	0.035	3.35	0.11	0.0010	0.28
Moonlight Sulfide #1	81	79.6	52.2	39.1	0.52	0.001	0.32	13.4	0.017	4.00	0.11	0.0005	0.20
Moonlight Sulfide #2	93	88.0	100.0	72.3	0.56	0.001	0.21	11.5	0.023	3.50	0.07	0.0000	0.06
Superior Sulfide #1	71	82.3	38.4	41.7	0.44	0.001	0.30	21.5	0.018	7.44	0.08	0.0005	0.18
Superior Sulfide #3	79	85.6	79.3	41.8	0.39	0.002	0.36	5.6	0.030	2.48	0.06	0.0005	0.22
Superior Sulfide #2	86	86.8	100.0	60.4	0.43	0.000	0.24	7.9	0.009	3.06	0.06	0.0000	0.10

Note: conc. = concentrate

The rougher concentrates were then cleaned using the following parameters:

- grinding size – P_{90} 100 mesh
- float time – 3 minutes
- conditioning time – 3 minutes
- reagent type – Aerofloat 3466 and PAX – addition rates of 0.02 and 0.01 kg/st, respectively
- frother – MIBC – as required
- pH – 10
- pulp bulk density – 25% solids by weight.

Table 13.5 identifies the results of the cleaner flotation tests.

Table 13.5 Cleaner Concentrate Results

Sample ID	Au (oz/st)	Ag (oz/st)	Cu (%)
Moonlight Sulfide Concentrate	**0.016**	**7.00**	**32.1**
Moonlight Sulfide Concentrate Tailings	0.001	1.60	6.0
Moonlight Sulfide Rougher Grade*	0.006	2.06	9.1
Moonlight Sulfide Cleaner Recovery	38.7%	58.0%	62.8%
Superior Sulfide Concentrate	**0.017**	**11.30**	**22.10**
Superior Sulfide Concentrate Tailings	0.134	2.25	8.36
Superior Sulfide Rougher Grade*	0.046	4.17	9.40
Superior Sulfide Cleaner Recovery	2.7%	52.3%	36.6%

Note: *back calculated grade

A review of the concentrate results identifies that a good grade copper concentrate can be expected. These results are consistent with the potential need of a regrind mill. Chalcopyrite tends to be harder and floats at a coarser size. The regrind will lower the size and remove any entrained particles. As a next step, locked cycle flotation testing should be performed.

The targeted metal grades in the Moonlight sample tested by the 2017 test program are higher than the average contents, compared to the resource estimate data. This suggests that the samples tested may not be well presentative to the mineralization. Further tests on better representative samples should be conducted. The recommended test work is discussed in Section 26.4.

14.0 MINERAL RESOURCE ESTIMATES

An updated Mineral Resource estimate of copper, silver and gold for the Moonlight copper deposit has been prepared for Crown Mining by CRC, with an effective date of December 15, 2017. The Mineral Resource estimate incorporates geologic interpretations and a drillhole database modified since the previous Mineral Resource estimate and NI 43-101 Technical Report (Cavey and Giroux 2007). The resource database comprises 202 drillholes with 11,005 copper assays, 10,555 gold assays, and 10,675 silver assays from 189 vertical N- and B-sized diamond drillholes drilled by Placer-Amex from 1966 – 1970 and 13 angled HQ diamond drillholes completed between 2005 - 2006 by Sheffield. Holes drilled by Placer-Amex and Sheffield, either outside the deposit area or with insufficient information, are excluded from the resource database.

14.1 TERMS OF REFERENCE

The Mineral Resource estimates presented in Section 14.0 are prepared and classified according to the guidelines of the Canadian Securities Administrators' NI 43-101, CIM Estimation of Mineral Resource and Mineral Reserves Best Practices, and CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM 2014) , excerpted below:

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.

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.

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.

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.

Mineral Resources are exclusive of Mineral Reserves and do not include dilution or other modifying factors applied that are needed to convert Mineral Resources into Mineral Reserves. Classification of resources under CIM definitions includes a test of potential economic feasibility. Readers are cautioned that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.2 Previous Estimates

Information on previous estimates of Mineral Resources is given in Section 6.0 of this report. The authors have undertaken their own review of the Mineral Resource inventory with respect to the available data and the conceptual study parameters of this PEA, as discussed in the following sections. They have not relied on previous estimates and the previous estimates should not be considered current.

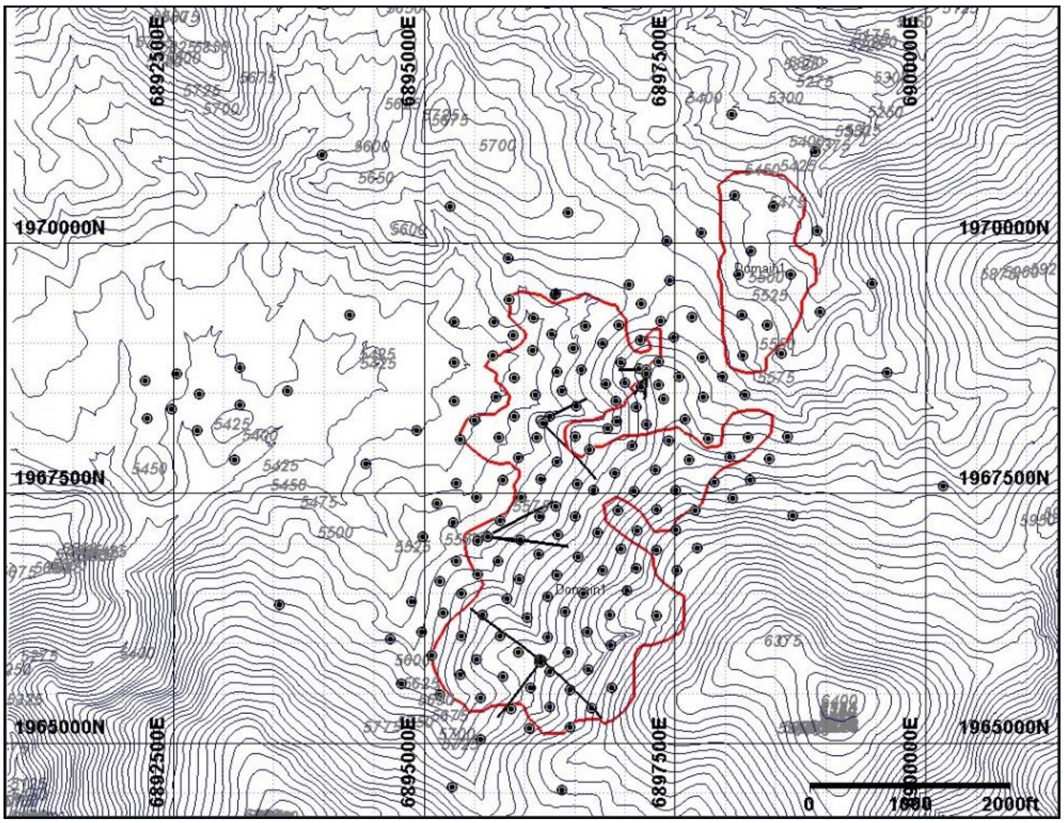
14.3 Software Used

The project drillhole database was compiled in Microsoft® Excel from different sources and imported to Micromine data files. Topography files consisting of 6 ft by 6 ft-spaced points were received from Pacific Geomatics and imported to Micromine for gridding and digital terrain model (DTM) generation. It was then simplified by eliminating those points that could be removed without significant displacement of the DTM. Wireframe geological model items were constructed in Micromine. GSLIB; Microsoft® Excel; and plug-ins Micromine, Sage, and CRC proprietary software were used for statistical analysis. Micromine version 2016.1 was used for block model creation, grade estimation, and reporting. Whittle™ software was used for pit optimization.

14.4 DATA ARRAY

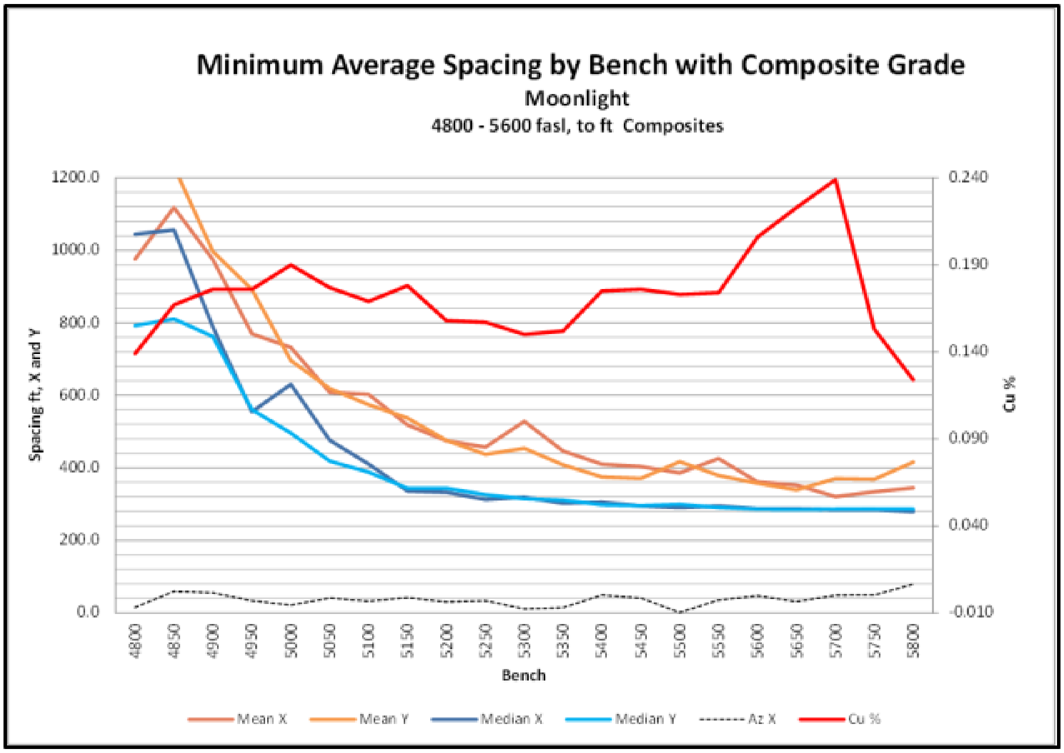
Drillhole collars are shown in plan over topography in Figure 14.1.

Figure 14.1 Moonlight Drillhole Plan Showing Topography and Projection of 0.1% Copper Shell at 5,350 Elevation



Placer-Amex drilling is laid out on a fairly regular 300 ft grid throughout the core of the deposit. Drilling is more irregular at the deposit fringes and at depth. Figure 14.2 quantifies the drill spacing by bench throughout the deposit at 50 ft intervals.

Figure 14.2 Drillhole Spacing and Mean Copper Grade by Bench



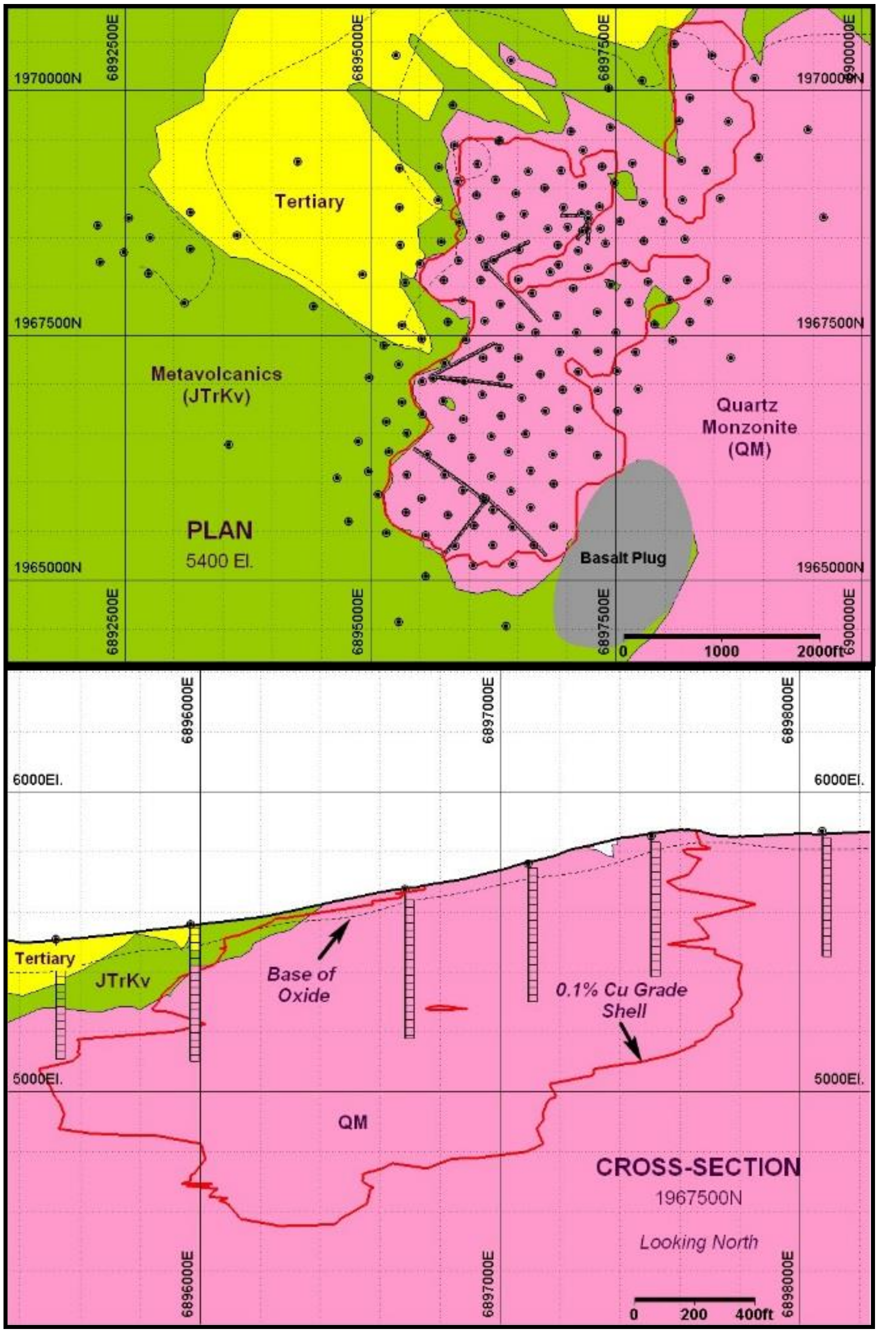
The median drill spacing curves for each direction are considered most representative for characterizing the majority of the grid. For much of the deposit, median drill spacing is approximately 300 ft. The mean lines reflect that some drilling is wider-spaced around the deposit fringes. The fact that the spacing curves for each orthogonal direction of minimum spacing plot nearly on top of one another for both mean and median confirms the regular drilling grid. The 300 ft grid is maintained to approximately the 5,150 ft elevation, or a vertical extent of 650 ft through the deposit. A copper grade trend is not very evident; of the other metals, silver shows a gradual decay in grade downward by bench. The regular drilling grid minimizes clustering of the data.

14.5 GEOLOGIC MODEL

In order to provide a geologic context for the grade modeling, Sheffield drillhole coding was translated to equivalent Placer-Amex lithology and alteration codes. Using the drillhole coding and a registered image of the Property geology map as reference, a 3D solid for the undifferentiated tertiary sediments (T) was digitized from sectional string interpretations. Based on CRC's field and drill core inspections, the intrusive rocks were treated as a single unit. A QM solid was digitized based on a plan interpretation reconciled and snapped to the drillholes on cross section. The outline of a tertiary basalt plug was projected from the surface map vertically downward to the model extents. This unit was not cut by any drillholes but was presumed to be a limiting, post-mineral intrusion. All other material in the model was considered to belong to the Triassic-Cretaceous (JTrKv) metavolcanics and metasediments.

The QM body, the principal material host, is shown in section and plan (Figure 14.3). It is exposed in the central portion of the deposit, plunging gently westward under a thin cover of JTrKv and T.

Figure 14.3 Geology Model Plan and Section, 5,400 ft elevation and 1967500N, with 0.1% Copper Shell and Surface Representing the Bottom of Oxide



A few small inliers of JTrKv are included in the QM solid. Tertiary rocks cap only a portion of the northern half of the deposit. The section also shows a surface representing the bottom of oxide, constructed by attempting to identify the point representing the transition to fresh rock and digitizing it. This was relatively easy for the Sheffield logs because comments were included in a description field. The Placer-Amex holes were hard to interpret. If no log data were available, the contact was either digitized at the surface, or where the soluble copper/total copper ratio of less than 0.5 could be calculated from the assays, or at the deepest occurrence of limonite or native copper. A surface was estimated for the contact using the array of points and a thin plate spline method that made projections sufficient to cover the mineralized area. The type of soluble copper assay used is not clear from the logs. Since chalcocite was logged in some holes, some of the copper not dissolved by an acid leach might be dissolved by a cyanide leach. In any case, material coded as oxide in this study forms a relatively thin skin over the deposit and is effectively a combination of oxide and transition zones based on mineralogy and limited oxide/total copper data.

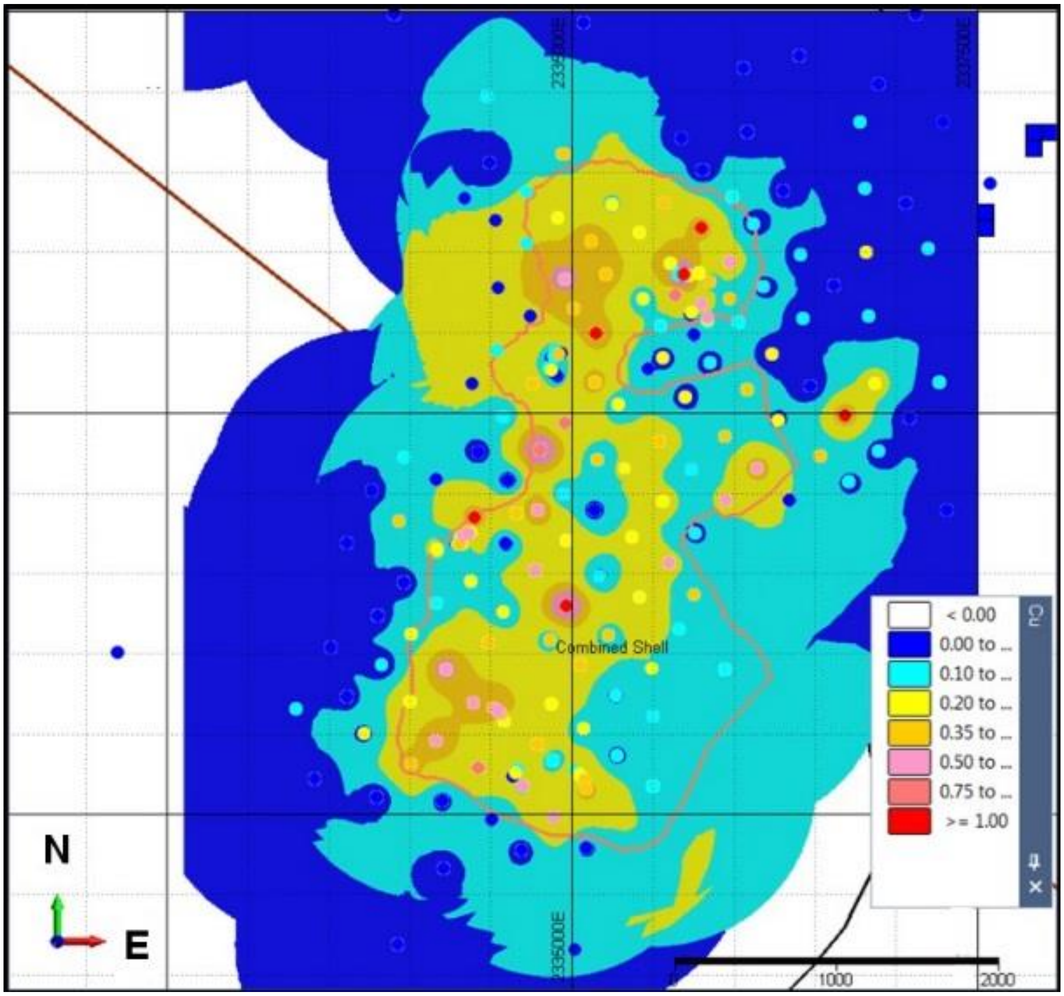
Tourmaline has a positive association with copper but it was not clear enough to model a distinct zone based on it. All of the assays were tagged to the rock solids and the redox surface to enable exploratory data analysis (EDA).

14.6 Exploratory Data Analysis

14.6.1 Metal Grids

Data gridding is a useful tool to identify areas of stronger mineralization, mineral trends, and association with other displayed features such as lithology and faults. For Moonlight, the contouring is performed by an isotropic search using an inverse distance interpolator of a maximum of 30 bench composites to 5 ft cells. The bench composite inputs and a copper grade shell based on an indicator estimate (discussed in Section 14.8) are superimposed on the plan in Figure 14.4.

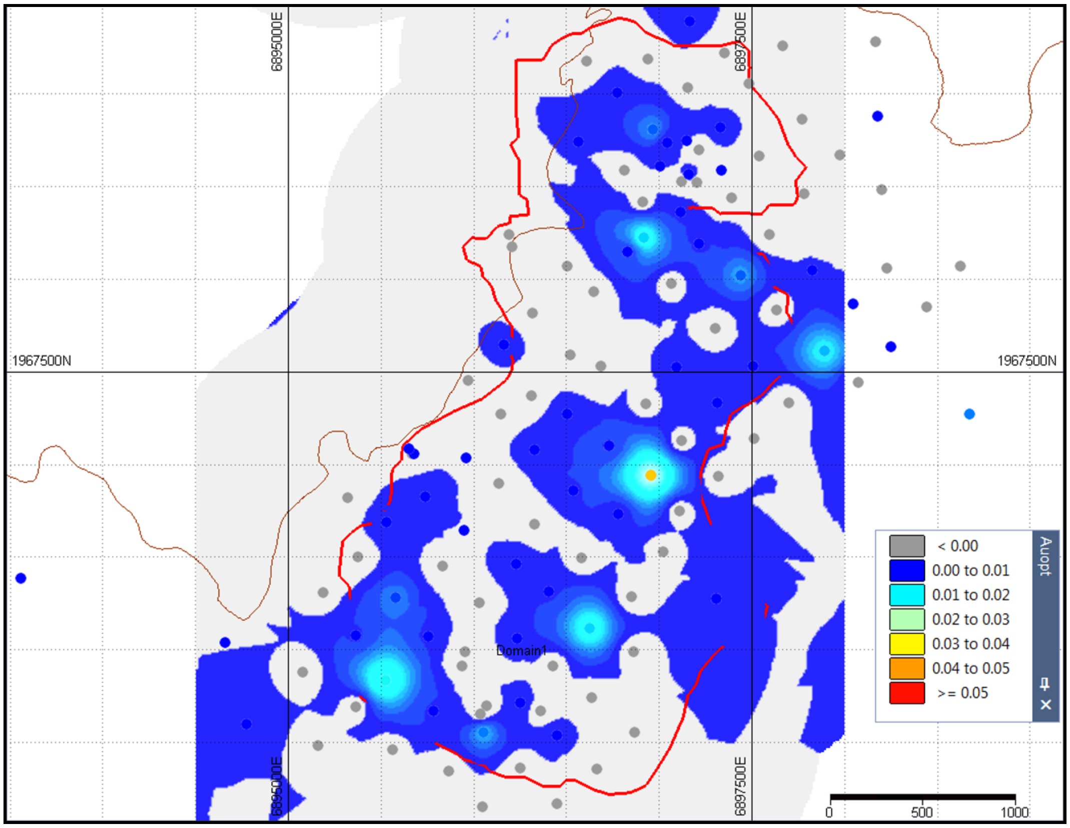
Figure 14.4 Copper Grid for 5,500 Bench With 0.1% Copper Shell



The plan shows a north-northeasterly mineralization trend with northwesterly internal lobes. The brown line was tested as a possible break between the south and north halves of the deposit, and may represent a structural zone. In section, the deposit appears to have significant continuity vertically. The broad zone shown in Figure 14.4 appears to break up into separate near-vertical bands further to the north, possibly representing the limbs of a partially eroded carapace of mineralization.

Gold and silver grids are less insightful because metal values are very low. The gold grid contours for the same bench as shown for copper in Figure 14.4 suggest a slightly more northeasterly trend (Figure 14.5).

Figure 14.5 Gold Grid for 5,550 Bench Showing 0.1% Copper Shell

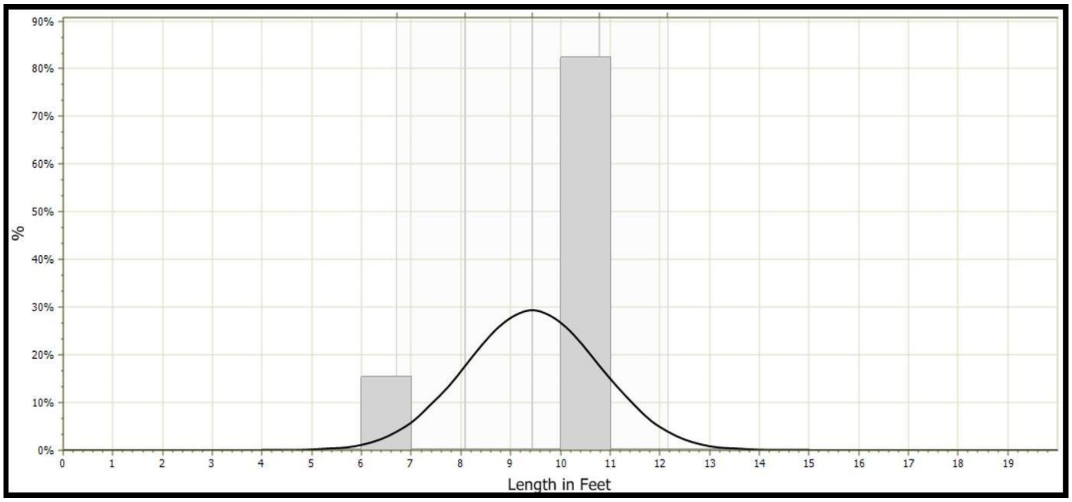


The internal northwest trend seen in the copper grid is evident in the gold grid, too. The surface geology map shows faults with northeast trends in the deposit vicinity. The outcrop pattern of the intrusive rocks is a combination of northeast and northwest trends. Thus, the simple metal grids show patterns that parallel some of the deposit geology trends.

14.6.2 Assay Statistics

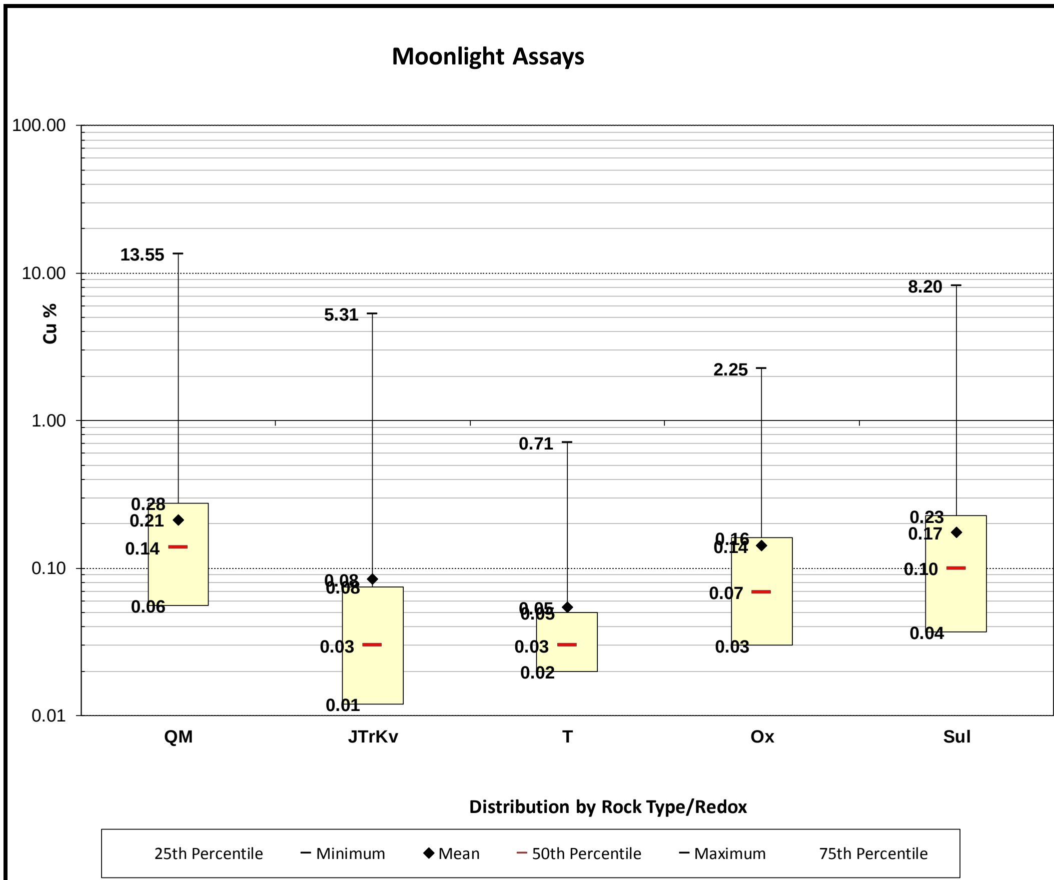
Placer-Amex assayed copper on 10 ft intervals and Sheffield assayed 6.5 ft intervals (2 m), resulting in two clusters on the histogram of assay length (Figure 14.6).

Figure 14.6 Histogram of Copper Assay Lengths



The mean assay length is 9.5 ft. The boxplot in Figure 14.7 shows the distribution of total copper in each geology model unit. Most of the mineralization occurs in the QM unit, but the other units are not completely barren. The co-efficient of variation (CV) is moderate in the QM (1.5) and high in the JTrKv. Copper grade in the oxide zone is somewhat lower compared to fresh rock.

Figure 14.7 Boxplot of Copper Assays by Geology Model Item



	QM	JTrKv	T	Ox	Sul
25th Percentile	0.06	0.01	0.02	0.03	0.04
Minimum	0.00	0.00	0.00	0.00	0.00
Mean	**0.21**	0.08	**0.05**	**0.14**	**0.17**
50th Percentile	0.14	0.03	0.03	0.07	0.10
Maximum	13.55	5.31	0.71	2.25	8.20
75th Percentile	0.28	0.08	0.05	0.16	0.23
Sample Count	**8,757**	1,932	**316**	860	**8,757**
SD	0.321	0.221	0.080	0.206	0.268
CV	1.5	2.6	1.5	1.5	1.5

Gold and silver were composited by Placer-Amex on 50 to 100 ft intervals before assay, but the composites were decomposed to 10 ft intervals in the preparation of the electronic database by Placer-Amex and others. Sheffield assaying was on 6.5 ft intervals. The data were imported to Micromine with three-decimal precision, conforming to the precision of the bulk of the data which were originated by Placer-Amex. A statistical comparison of data from the two campaigns is shown in Table 14.1.

Table 14.1 Comparison of Placer-Amex and Sheffield Decomposed Gold and Silver Assays

Placer-Amex Assays			Sheffield Assays		
Parameter	Au (oz/st)	Ag (oz/st)	Parameter	Au (oz/st)	Ag (oz/st)
Count	9,390	9,398	Count	1,536	1,648
Mean	0.0014	0.057	Mean	0.0001	0.096
Standard Deviation	0.0058	0.1058	Standard Deviation	0.0010	0.1769
Range	0.1000	3.800	Range	0.0370	3.450
Minimum	0.0000	0.000	Minimum	0.0000	0.000
Lower Quartile	0.0000	0.000	Lower Quartile	0.0000	0.020
Median	0.0000	0.020	Median	0.0000	0.060
Upper Quartile	0.0020	0.080	Upper Quartile	0.0000	0.120
Maximum	0.1	3.800	Maximum	0.0370	3.450
CV	407%	184%	CV	732%	184%
10% Percentile	0.0000	0.000	10% Percentile	0.0000	0.010
90% Percentile	0.0020	0.160	90% Percentile	0.0000	0.200
95% Percentile	0.0050	0.210	95% Percentile	0.0010	0.290
97% Percentile	0.0050	0.300	97% Percentile	0.0010	0.370
98% Percentile	0.0100	0.330	98% Percentile	0.0010	0.470
99% Percentile	0.0150	0.400	99% Percentile	0.0020	0.626

Placer-Amex gold assays are high biased in all ranges compared to Sheffield, whereas silver assays are more comparable, but low biased in the Placer-Amex set. Most of the Sheffield gold values are less than the below-detection defaults set for the Placer-Amex data when it was migrated from the written logs; differences in the campaigns are due to assay method, not spatial variation. Although gold assays are shown to four decimal places, the reader should note that the real precision of the majority of the database (Placer-Amex) is less.

14.7 COMPOSITING

The copper assays are composited to equal-length 25 ft intervals down-the-hole with no breaks on geology or other features and a 6 ft minimum length. The interval chosen is half of the block height and provides some resolution around contacts that would otherwise be lost with a longer interval. Copper composite statistics by rock type show a reduction in CV from 1.5 to 1.2 (Table 14.2).

Table 14.2 Copper Composites Statistics by Rock Type

Rock Type	Count	Minimum	Maximum	Standard Deviation	Mean Cu %
Rock					
JTrKv	789	0.00	2.50	0.172	0.08
QM	3,299	0.00	4.28	0.227	0.21
T	124	0.00	0.49	0.064	0.05
Total	**4,212**	**0.00**	**4.28**	**0.221**	**0.18**
Redox					
1 (Ox)	369	0.00	1.20	0.177	0.15
2 (Fresh)	3,843	0.00	4.28	0.225	0.18
Grand Total	**4,212**	**0.00**	**4.28**	**0.221**	**0.18**

The oxide zone has 20% lower copper mean and maximum values than the fresh rock, probably due to leaching by surface waters. The Sheffield data cause a degree of local clustering because they are drilled in angled fans crossing the previous pattern in better-mineralized areas of the deposit.

Gold and silver are composited down-the-hole with no geology breaks on 100 ft intervals, both to put the data on the same support and because of the data quality. This compromise creates approximately the same number of full-length 100 ft composites from both drill campaigns, 85%, with less than 5% of the composites less than 50 ft in length. Pivot statistics of the gold composites are shown in Table 14.3.

Table 14.3 Gold Composite Statistics by Rock Type

Rock Type	Count	Minimum	Maximum	Standard Deviation	Mean Cu %
JTrKv	195	0.0000	0.0260	0.0023	0.0013
QM	819	0.0000	0.0900	0.0054	0.0013
T	30	0.0000	0.0020	0.0008	0.0004
Total	**1,044**	**0.0000**	**0.0900**	**0.0049**	**0.0013**

The simple mean of the Sheffield composites in the QM unit is 0.0002 oz/st gold, nearly an order of magnitude less than the Placer-Amex drillholes largely due to assay method sensitivity, as discussed above.

A pivot table shows that mean silver is somewhat higher in the QM than in the JTrKv unit (Table 14.4).

Table 14.4 Silver Composite Statistics by Rock Type

Rock Type	Count	Minimum	Maximum	Standard Deviation	Mean Cu %
JTrKv	195	0.000	1.380	0.113	0.044
QM	827	0.000	0.738	0.076	0.066
T	30	0.000	0.156	0.035	0.021
Total	**1052**	**0.000**	**1.380**	**0.084**	**0.061**

Silver is also slightly elevated in the oxide zone, but the difference is only 20%. Descriptive statistics for copper, gold, and silver are listed side-by-side in Table 14.5.

Table 14.5 Descriptive Statistics for Copper, Gold, and Silver Composites

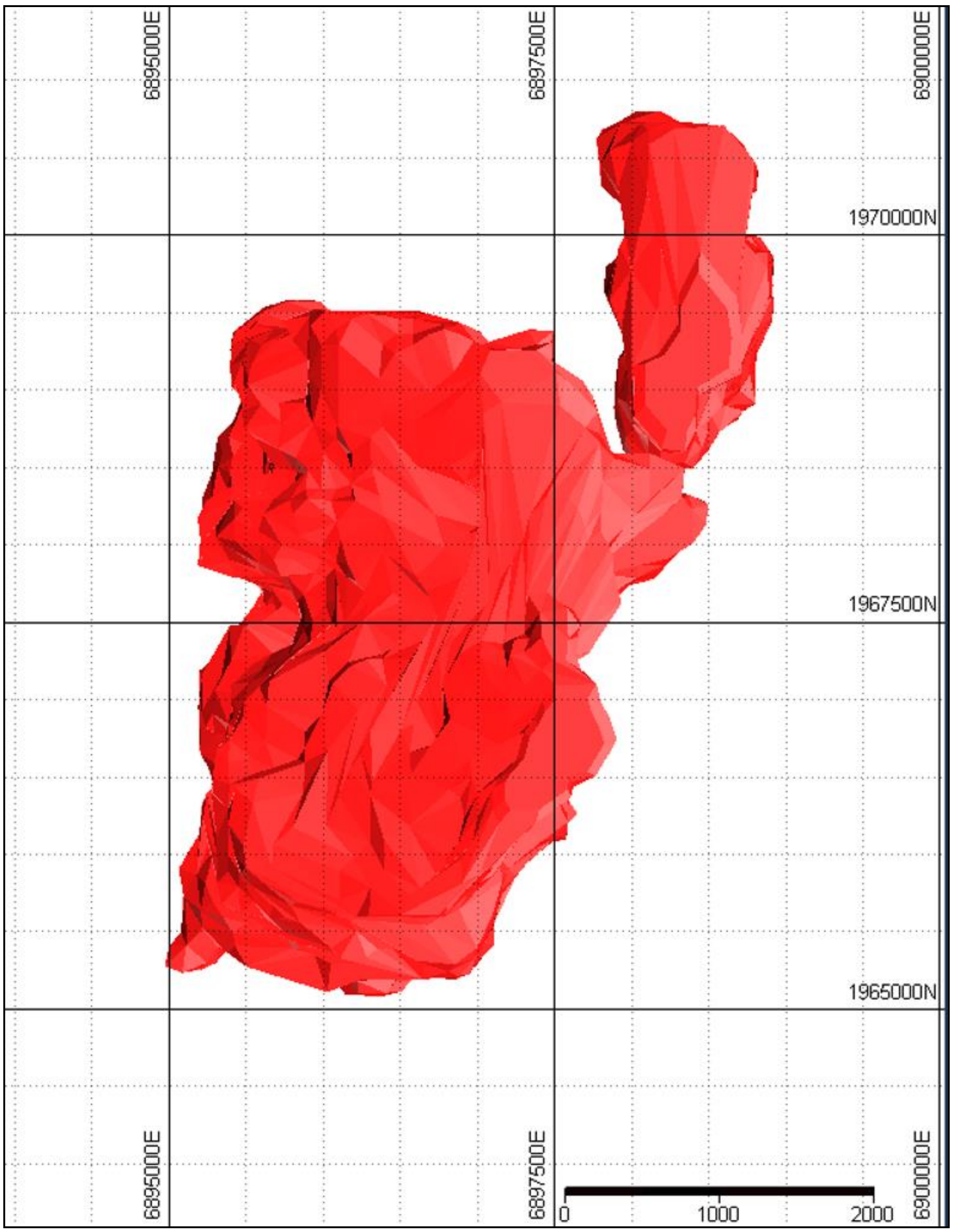
Parameter	CuT	Au (oz/st)	Ag (oz/st)
Number of Numeric Values	4,212	1,044	1,052
Mean	0.179	0.001	0.061
Standard Deviation	0.221	0.005	0.084
Range	4.280	0.090	1.380
Minimum	0.000	0.000	0.000
Lower Quartile	0.048	0.000	0.006
Median	0.116	0.000	0.039
Upper Quartile	0.246	0.002	0.081
Maximum	4.280	0.090	1.380
CV	123%	378%	138%
10% Percentile	0.018	0.000	0.000
90% Percentile	0.400	0.002	0.150
95% Percentile	0.526	0.004	0.200
97% Percentile	0.635	0.005	0.243
98% Percentile	0.752	0.009	0.262
99% Percentile	0.904	0.012	0.333

The pivots and descriptive statistics by rock type and redox show support for definition of copper domains in the differences in means between rock and redox types. Support for gold and silver domains from the statistics is less obvious, possibly, in the case of gold, due to the sensitivity of the assay methods employed by Placer-Amex.

14.8 GRADE SHELL

Assay sections, bench plans and the copper grids demonstrate distinct areas of high and low grade material within the QM host. A grade shell within the QM was digitized to control smoothing across grade boundaries based on a 0.1% copper indicator threshold assigned to drillhole composites. This value was chosen because it is close to the median copper value and slightly below likely mining cut-off grades. The estimated model blocks were contoured at a +0.5 indicator result and on 50 ft spaced levels from 4,475 to 6,050 ft elevation. A filter was imposed on the model blocks to exclude those greater than 300 ft from a data point. Each contour was smoothed using a minimum spacing between points of 100 ft and by "light-tabling" reconciliation between adjacent levels. The resulting contours excluded non-continuous and poorly supported indicator projections, and locally included blocks with lower indicator values in order to produce a logical shell shape (Figure 14.8).

Figure 14.8 Quartz Monzonite 0.1% Copper Grade Shell (Scale in feet)



The grade shell has maximum continuity north-south, with roughly subequal continuity east-to-west and vertically. It is reasonably effective in separating high and low copper grade, as shown in Table 14.6.

Table 14.6 Basic Statistics of Composites after Tagging to Copper Grade Shell

0.1% Cu Shell	Count	Minimum	Maximum	Standard Deviation	Mean Cu%
In	2,357	0.00	4.28	0.246	0.26
Out	1,855	0.00	2.50	0.129	0.08
Total	**4,212**	**0.00**	**4.28**	**0.221**	**0.18**

Material inside the shell has grades 25% higher than the mean grade of the QM unit and the material outside the shell has a mean grade lower than the 0.1% copper shell threshold.

14.9 Estimation Domains

Copper estimation domains are specified to confine estimates to similar materials and ones that have similar geologic controls. Criteria for domains is that they must be geologically and statistically consistent, they must form discrete and continuous shapes, and they should result in lowering the variance of samples in the best mineralized zones. Copper domains selected for separate estimation are listed in Table 14.7.

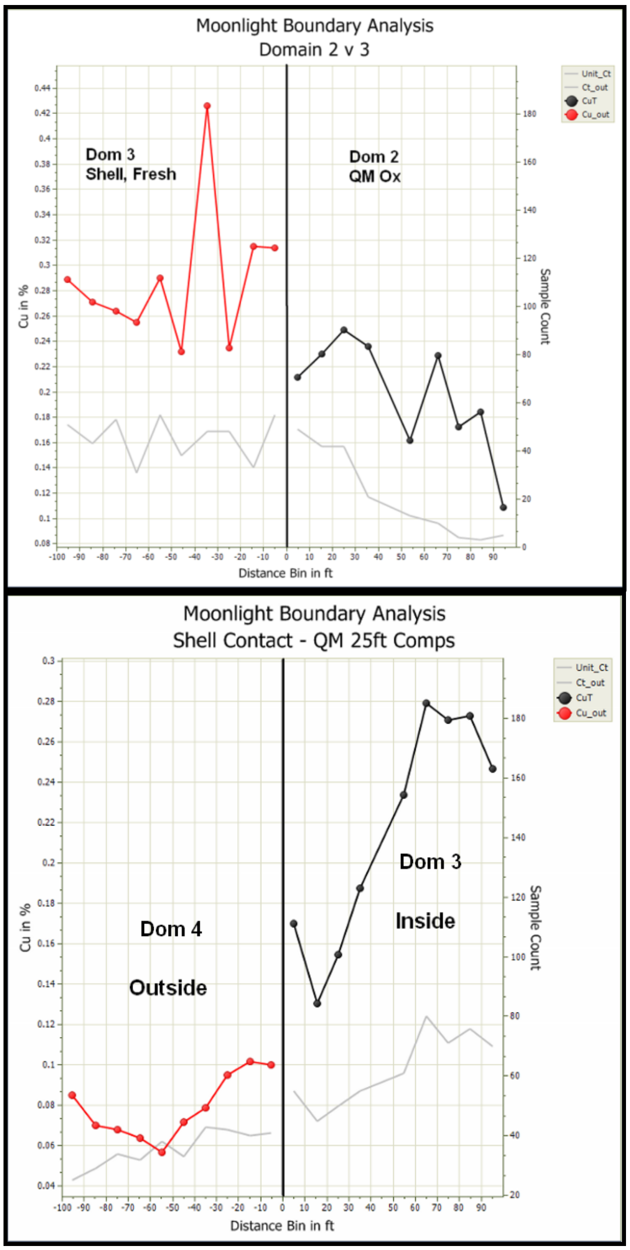
Table 14.7 Copper Estimation Domains and Boundary Characteristics

Domain	Code	No. of Comps	Description	1	2	3	4	5
JTrKv+Tertiary	1	913	-	X	25	5	25	Hard
QM_Ox	2	212	QM, Redox=1	25	X	25	25	Hard
QM_Shell	3	2,180	QM, Redox=-2, Shell	5	25	X	5	Hard
QM_Outside	4	907	QM, Redox=2, Outside	25	25	5	X	Hard
Plug and Unmodeled	5	0	Basalt plug	Hard	Hard	Hard	Hard	X

The principal mineralized domain is the QM inside the 0.1% copper grade shell. Approximately 10% of the potentially economic mineralization is carried by the other domains. The domain code matrix to the right shows which domain boundaries are "hard" and which are "soft" and by how many feet, determined by geological considerations and boundary analysis.

Boundary, or contact analysis is a tool used to understand the behavior of grade on either side of proposed boundaries for statistical analysis and grade estimation. These boundaries may be lithologic or alteration contacts, or grade shells. From an input file containing grades, coordinates, and codes to identify the boundaries, a program determines the average grade of assays or composites on either side of the contact in bins representing increasing distance from the contact. A line graph displays the average grades per bin relative to the contact. An abrupt change in grade across a contact is deemed a “hard” boundary. A smoother gradient across a boundary is termed a “soft” boundary, suggesting that the estimate of a block on one side of a contact should have some influence from composites on the other side. Lack of any definite break or slope suggests that the proposed boundary is ineffective for separating grade populations. The boundary analysis was done for the composites with Micromine, calculating the true distances to contacts represented by the geological model solids. Two plots supporting the QM domain soft and hard characteristics are shown in Figure 14.9.

Figure 14.9 Copper Domain Boundary Analysis Plots Supporting Hard (top) and Soft (bottom) Boundaries



The QM boundary between the shell and material outside is clearly hard. The QM shell and oxidized QM boundary have an apparent break at the contact that does not persist away from it, moreover the construction of the bottom-of-oxide surface is not robust. Thus, the redox boundary is treated as a soft boundary for estimation. The soft boundary distances selected, 25 ft, make it unlikely that more than one composite tagged to another domain will be selected to estimate a block.

Only two silver domains are specified: 1) QM; and 2) Other, comprising the JTrKv and T units. The gold estimate is not performed using separate domains.

14.10 TREATMENT OF OUTLIERS

The objective of grade capping is to remove metal-at-risk arising from statistical outliers included in the data set. CRC selected grade caps for each metal based on decile analysis and probability plots, according to the values listed in Table 14.8.

Table 14.8 Capping Levels by Domain for Each Metal

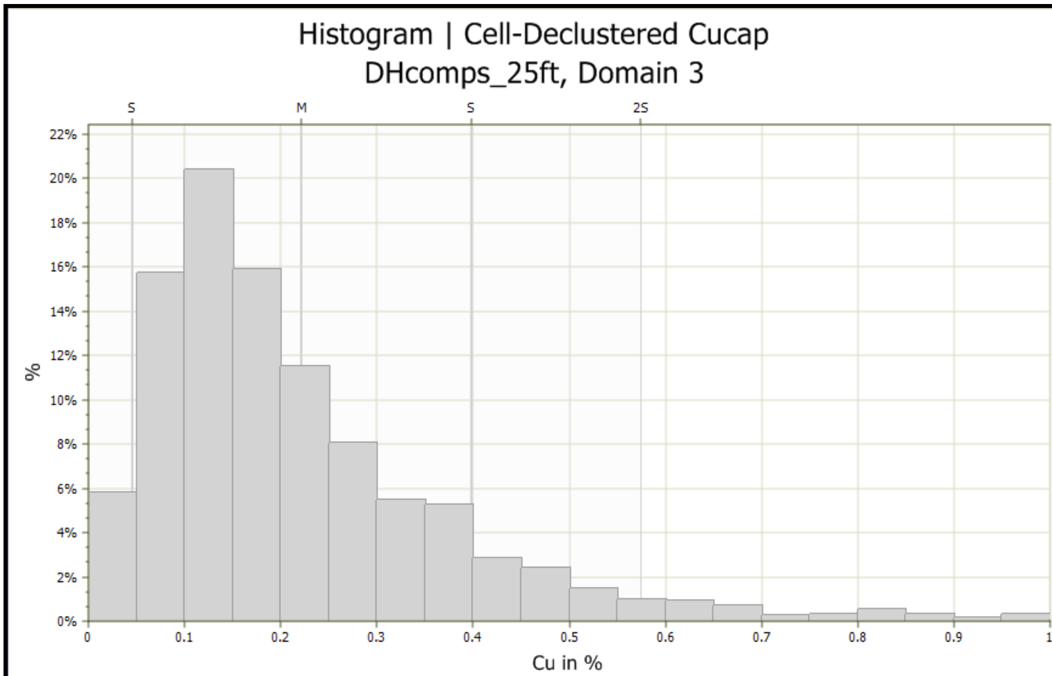
Domain	Cu (%)	Ag (oz/st)	Au (oz/st)
1	0.7	0.5	0.4
2	0.4	n/a	0.4
3	1.2	n/a	0.4
4	0.4	n/a	0.4
5	n/a	n/a	n/a

Copper capping affects 14 composites in domain 3, and eight others in the rest of the domains combined. A comparison of estimates made with capped and raw composites shows copper metal removal of less than 2% overall and less than 1% in domain 3, the principal mineralized domain. Capping affects three gold composites and one silver composite, removing 9% of gold and 1.5% of silver.

14.10.1 DOMAIN STATISTICS

The focus of copper mineralization is domain 3, for which cell de-clustering was performed to provide a comparison with the grade estimation done by kriging methods. A histogram for the domain is shown in Figure 14.10.

Figure 14.10 De-clustered Histogram of Copper Domain 3



Minimum Value	0.000	Mean	0.251	Standard Deviation	0.196	Geometric Std Dev	
Maximum Value	1.200	Variance	0.038	Coeff. of Variation	0.780	Sichel's Estimator	
2nd Highest	1.200	Sum of Weights	2179.999	Median	0.200	Sichel's V	
3rd Highest	1.200	Weighted Mean	0.222	Ln Mean	-1.670	Sichel's Gamma	
4th Highest	1.200	Weighted Variance	0.031	Ln Std Deviation	0.816	Chi-Square Test	
N	2180	Weighted Std. Dev.	0.176	Geometric Mean	0.188	Degrees of Freedom	

The de-clustered mean is 12% lower than the raw composite mean. Non-de-clustered composite statistics for all of the domains, capped and uncapped, are shown in Table 14.9.

Table 14.9 Non-de-clustered Copper Composite Statistics by Domain

Parameter	Cu% Raw	Cu% Cap	1	2	3	4
Number of Numeric Values	4,212	4,212	913	212	2,180	907
Mean	0.174	0.179	0.075	0.212	0.251	0.081
Standard Deviation	0.182	0.221	0.118	0.187	0.196	0.078
Range	1.200	4.280	0.700	1.196	1.200	0.400
Minimum	0.000	0.000	0.000	0.004	0.000	0.000
Lower Quartile	0.048	0.048	0.014	0.082	0.119	0.032
Median	0.116	0.116	0.036	0.158	0.200	0.057
Upper Quartile	0.246	0.246	0.074	0.277	0.331	0.096
Maximum	1.200	4.280	0.700	1.200	1.200	0.400
CV	104%	123%	157%	88%	78%	96%
10% Percentile	0.018	0.018	0.003	0.046	0.065	0.012
90% Percentile	0.400	0.400	0.185	0.441	0.486	0.181
95% Percentile	0.521	0.526	0.324	0.560	0.623	0.259
97% Percentile	0.634	0.635	0.426	0.632	0.726	0.300
98% Percentile	0.700	0.752	0.488	0.711	0.844	0.336
99% Percentile	0.859	0.904	0.700	0.892	1.038	0.400

The raw (uncapped) copper values are shown in the first column for comparison.

14.11 VARIOGRAPHY

Correlograms for copper and silver were generated for the estimation domains using Sage software. Table 14.10 summarizes the models generated.

Table 14.10 Correlogram Parameters used for Copper and Silver

Item	Cu	Cu	Ag
Type	Correlogram	Correlogram	Correlogram
Domain(s)	1	2,3,4	n/a
C^0	0.135	0.184	0.350
C^1	0.865/Exp	0.540/Exp	0.385/Exp
C^2	n/a	0.276/Exp	0.265/Exp
Azimuth 1	283	265	0
Azimuth 2	28	33	90
Azimuth 3	118	123	0
Plunge 1	89	87	0
Plunge 2	0	2	0
Plunge 3	1	2	90
Range 1	415	243/724	485/522
Range 2	126	84/1,105	485/522
Range 3	252	96/1,442	485/522

The copper correlogram for domain 1 shows more vertical continuity and a northwest alignment in plan. The domain 3 correlogram shows more vertical than horizontal continuity up to 175 ft, but the opposite at longer ranges. It is only moderately anisotropic. An omnidirectional correlogram was the best result for modeling spatial correlation of silver. No variograms or correlograms were obtained for gold.

14.12 Estimation Plan

14.12.1 Model Definition

The block model comprises unrotated 100 ft by 100 ft by 50 ft blocks constructed to cover the full extent of the drilled deposit. The scheme is summarized in Figure 14.11.

Figure 14.11 Moonlight Block Model Definition

	Min Centre	Block Size	Max Centre	# Blocks	Min Size
East	2330050	100	2339450	95	100
North	323050	100	331450	85	100
Z	4225	50	6025	37	50

Block height is compatible with the mining plan for 50 ft high benches. The X and Y block dimensions are reasonable based on the current data spacing (300 ft by 300 ft). Each model block is assigned a default SG of 2.67, equivalent to a tonnage factor of 12 cu ft/st, and a default rock type code JTrKv. The rock type is subsequently overwritten as necessary by wireframe tagging of the modeled rock types (e.g., QM). Blocks discretization is 2 x 2 x 2 for copper and 1 x 1 x 2 for gold and silver.

14.12.2 Methods

All estimates are performed in Micromine software. Each step in the estimation process is scripted in a Micromine macro with replaceable parameters in order to reduce the number of filter and process profiles Besides the interpolated grade estimates, each process includes parallel uncapped and nearest-neighbor estimates for validation purposes.

The estimation strategy is to separately estimate data that demonstrate different lithologic and structural controls by identification of estimation domains, discussed in Section 14.9. The main copper domain is defined by, and constrained to a 0.1% copper grade shell based on smoothed contouring of an indicator kriging estimate, discussed above. Estimates for the copper and silver domains are by ordinary kriging using correlogram model weighting, except for the silver hosted by JTrKv which uses inverse distance weighting (ID3). Estimates for gold are by ID3. Copper estimation is performed in two passes; for kriged estimates the first pass is limited to a search based on the correlogram ranges and anisotropy at 90% of the sill. The second pass search is expanded to a multiple of the first pass range with less anisotropy. The estimation plan is summarized in Table 14.11.

Table 14.11 Estimation Plan Summary for Copper, Silver and Gold

Description					Correlogram	Search Orientation				Srch Dist/Ratios			Composites						
Domain	Input File	Variable	Pass	Method		Az1/Plg	Az2/Plg	Az3/Plg	Max Dist	1st	2nd	3rd	Min Comps	Max/Sector	Sectors	Min Holes	Max/Hole	Units	Cap
1	DHComps_25ft	Cucap	1	OK	JTrKv	283/89	13/0	283/-1	750	1.00	0.30	0.61	3	4	4	2	4	%	0.7
1	DHComps_25ft	Cucap	2	OK	JTrKv	283/89	13/0	283/-1	750	1.00	0.30	0.61	3	10	1	1	5	%	0.7
2	DHComps_25ft	Cucap	1	OK	QMVario	265/87	37/2	307/-2	300	1.00	0.90	0.95	3	4	4	2	4	%	0.4
2	DHComps_25ft	Cucap	2	OK	QMVario	265/87	37/2	307/-2	600	0.70	0.90	1.00	3	10	1	1	5	%	0.4
3	DHComps_25ft	Cucap	1	OK	QMVario	265/87	37/2	307/-2	300	1.00	0.90	0.95	3	4	4	2	4	%	0.4
3	DHComps_25ft	Cucap	2	OK	QMVario	265/87	37/2	307/-2	600	0.70	0.90	1.00	3	10	1	1	5	%	0.4
4	DHComps_25ft	Cucap	1	OK	QMVario	265/87	37/2	307/-2	300	1.00	0.90	0.95	3	4	4	2	4	%	0.4
4	DHComps_25ft	Cucap	2	OK	QMVario	265/87	37/2	307/-2	600	0.70	0.90	1.00	3	10	1	1	5	%	0.4
Shef/Nevoro	AuComps_100ft	Aucap	1	ID3	N/A	135/0	210/90	225/0	450	1.00	1.00	0.67	1	8	1	N/A	N/A	oz/st	0.04
All	AuComps_100ft	Aucap	2	ID3	N/A	135/0	210/90	225/0	600	1.00	1.00	0.80	1	8	1	N/A	N/A	oz/st	0.04
1	AgComps_100ft	Agcap	1	OK	AgOmni	135/0	210/90	225/0	450	1.00	1.00	0.67	1	8	1	N/A	N/A	oz/st	N/A
1	AgComps_100ft	Agcap	2	OK	AgOmni	135/0	210/90	225/0	600	1.00	1.00	0.80	1	8	1	N/A	N/A	oz/st	N/A
2	AgComps_100ft	Agcap	3	ID3	N/A	135/0	210/90	225/0	600	1.00	1.00	0.80	1	8	1	N/A	N/A	oz/st	0.5

Soft and hard boundaries are incorporated in the composite neighborhood searches as described in Section 14.9.

Silver is estimated in three steps: blocks coded as QM are estimated by QM composites in two passes, using less restrictive parameters to fill the model in the second pass, then, the JTrKv blocks are estimated by ID3 in a single pass.

Gold estimation comprised two passes, the first using only Sheffield data which is the most reliable. Blocks not estimated in the first pass were estimated using all data and using a larger search neighborhood in order to fill the model. Blocks estimated for gold in the second pass were re-set to a default value subsequent to validation procedures, discussed below.

14.13 VALIDATION

The estimates are validated using the following methods:

- global bias checks comparing interpolated and nearest-neighbour and/or de-clustered composite statistics estimates
- check of metal removal by capping
- drift analysis for check of local bias
- graphic validation of the following elements:
  - domain tagging of composites and model
  - grade
  - average distance from composites
  - estimation pass
- check variable value ranges for consistency with caps and other parameters
- change-of-support (COS) from composites to blocks.

The most extensive checks are for copper, comprising all of the steps listed above. Silver checks included all steps except for COS. Grade capping removes 2% of copper metal overall and 1% of metal in domain 3, the principal mineralized domain. A COS analysis performed using the HERCO method for copper domains 3 and 4 shows that the estimates of block grades and tons are reasonable at cut-off grades below 0.4% copper. All estimates are checked for global bias and are checked graphically.

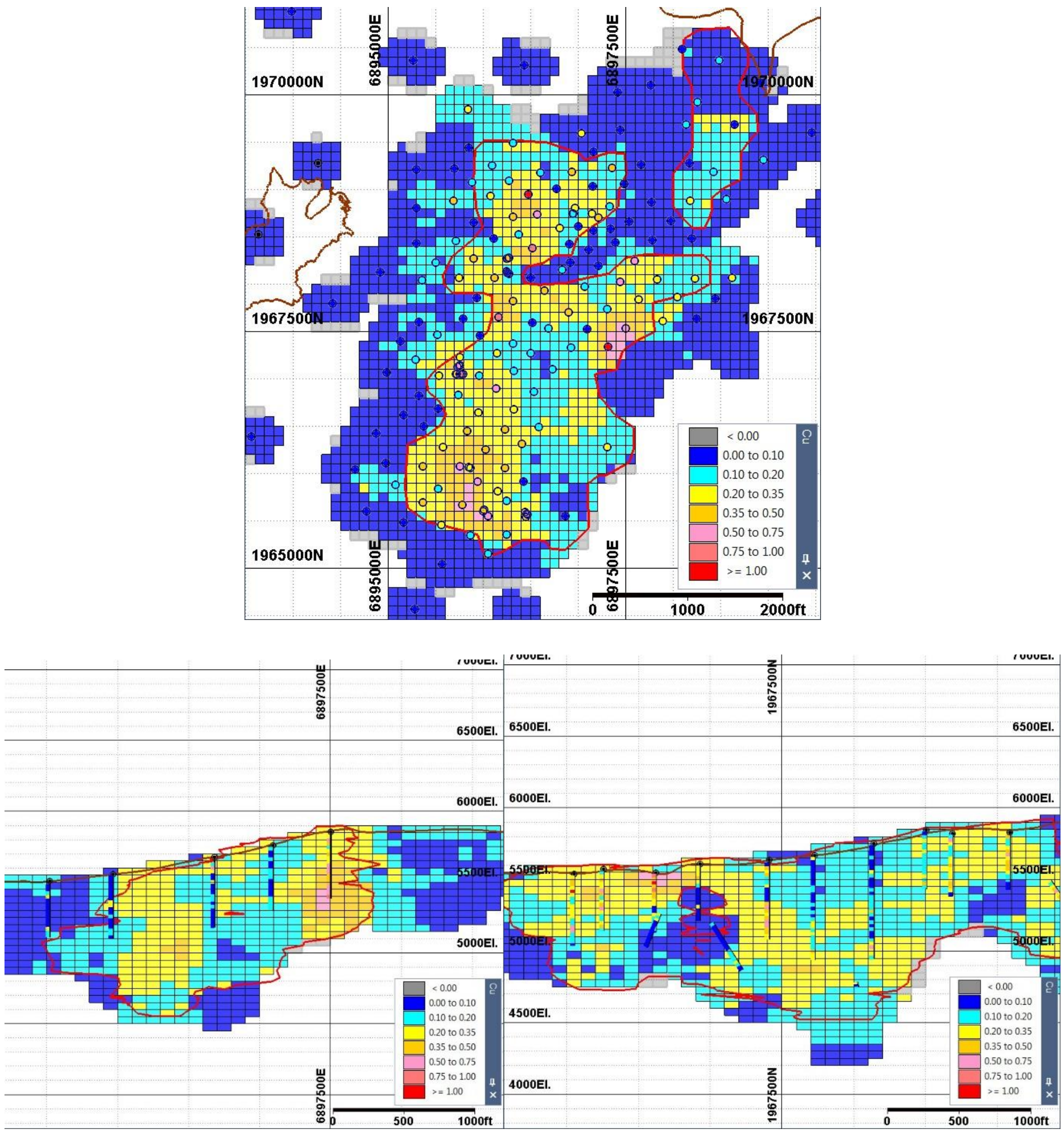
Graphic and statistical comparison of estimates from the two gold estimation passes highlighted the impact of the Placer-Amex gold assay detection level issue, where the second estimation pass using all data created much higher estimates than the first pass estimate. Based on the validation, gold values in all blocks estimated by the second pass were re-set to a lower default value of 0.00007 oz/st gold, the mean of the first pass estimate.

14.13.1 Graphic Checks

Figure 14.12 shows a representative plan and sections of the estimated block model copper grades relative to the 0.1% copper shell, topography, and the supporting drillhole composites.

The plots show that the block grades honor the composite grades and the domain 3 boundary with other domains. Copper grade shows a significant degree of continuity between drillholes. Angled Sheffield holes, shown on the north-south section, have similar grades as the Placer-Amex neighboring drillholes.

Figure 14.12 Copper Model Views Showing 5,425 ft elevation (top), East-West Section 1967500N (bottom left) and North-South Section 6896500E (bottom right) with 0.1% Copper Grade Shell and Intersection of Topographic Surface



14.13.2 Global Bias

Global bias of the grade model introduced by the estimation plan was checked by comparing the interpolated estimate grades versus the de-clustered composites and/or nearest-neighbor estimate. The latter was performed for copper using 50 ft composites, equal to the bench height. For the precious metals, the nearest-neighbor input was the closest 100 ft composite. Results for each metal are shown in Table 14.12.

Table 14.12 Global Bias Check for Copper, Gold and Silver

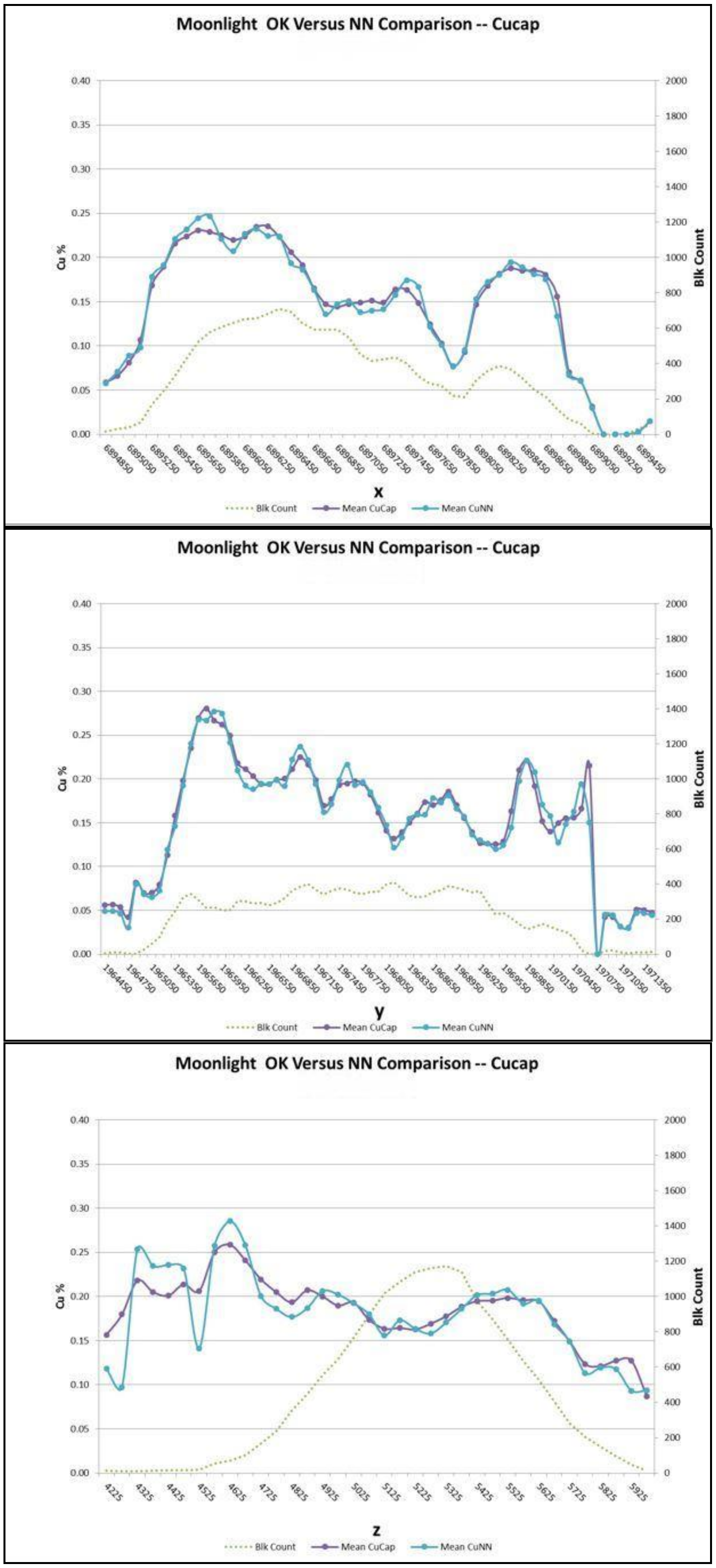
Domain	Tons ('000 st)	CuCap (%)	CuCap (st)	CuNN (%)	CuNN (st)
Copper					
1	505,259	0.054	544,497	0.048	489,729
2	53,550	0.144	154,505	0.153	164,111
3	566,382	0.225	2,546,606	0.224	2,537,200
4	1,102,646	0.092	2,030,245	0.088	1,933,862
Total	**2,227,837**	**0.118**	**5,275,853**	**0.115**	**5,124,903**
Gold					
1	505,259	0.001	505,549	0.001	431,466
2	53,550	0.001	53,946	0.001	51,976
3	566,382	0.001	485,329	0.001	479,698
4	1,102,646	0.001	887,650	0.001	820,888
Total	**2,227,837**	**0.001**	**1,932,474**	**0.001**	**1,794,119**
Silver					
1	505,259	0.036	18,236,011	0.033	16,862,943
2	53,550	0.059	3,182,239	0.060	3,194,660
3	566,382	0.070	39,514,958	0.070	39,664,679
4	1,102,646	0.045	49,644,643	0.045	49,632,576
Total	**2,227,837**	**0.050**	**110,577,851**	**0.049**	**109,354,858**

Mean kriged copper grade in domain 3 compares well to 0.223 %Cu for the cell-declustered composites and to the nearest-neighbor estimate (CuNN). Overall, the estimates for all metals show no indication of significant bias compared to the underlying composite support.

14.13.3 Local Bias

The presence of any local bias created by the estimation plan was checked with drift (swath) plots, line plots comparing the average grade of interpolated grades and nearest-neighbor grades along the coordinate axes. Results for copper are shown in Figure 14.13.

Figure 14.13 Swath Plot Local Bias Check for Copper Across Model X, Y, and Z



The plots also show a dotted line representing the number of blocks used in the averages for each coordinate bin, tied to the second Y axis of the graphs on the right edge. The only significant problems with bias are on the Z plot below 4,900 ft elevation where block counts are small. Silver model drift analysis shows similar results to copper with no problem areas except at the lowest, thinly populated levels.

The following observations are made from the various checks of the grade model:

- Estimation domains coincide with the current structural and lithological understanding of the deposit; they also help control over-projection of higher grade into areas with lower grade drillhole results.
- Block grade estimates reasonably represent composite grades.
- The copper grade for domain 3, QM inside the grade shell, is within 1% of the cell de-clustered composite and nearest-neighbour block grades tagged to the domain.
- The copper, gold and silver models are not globally biased and the copper and silver models shows no evidence of local bias in drift (swath) analysis.

Continuity of copper and silver grades are less beyond the regular-spaced drilling grid and at lower elevations due to variable drillhole lengths.

14.14 SPECIFIC GRAVITY

All of the information about SG and bulk density are from samples collected and tested by Sheffield from the MN-series core holes. These are reported as SG, converted for this study into Imperial system-equivalent tonnage factors by the formula:

$$TF = (2000\ lb/st)/(SG * 62.44\ lb/cu\ ft)$$

All 68 samples lie within the QM solid. The simple mean of the SG's is 2.67 and all samples lie within two standard deviations of the mean. Fifteen samples above the oxide surface have the same mean as those below. Thus, a single value of 2.67, equivalent to a tonnage factor 12.0 cu ft/st, is assigned to all blocks in the model.

14.15 CLASSIFICATION

14.15.1 PRE-CLASSIFICATION

Geological continuity has been established for the most part through diamond drilling of the QM zone. Pre-classification of potential Mineral Resources is according to criteria that address confidence in the copper grade estimate. These include search and composite selection and average distance of composites to a block compared to the drilling grid. Blocks are coded from 2 to 3 in descending order of confidence (e.g., blocks coded "2" are eligible for classification as Indicated and "3" for Inferred Resources). A block with a pre-classification code of "2" may be classified as Indicated Resource if it meets the test for potential economics. No mineralization in the Moonlight deposit is considered for classification as Measured.

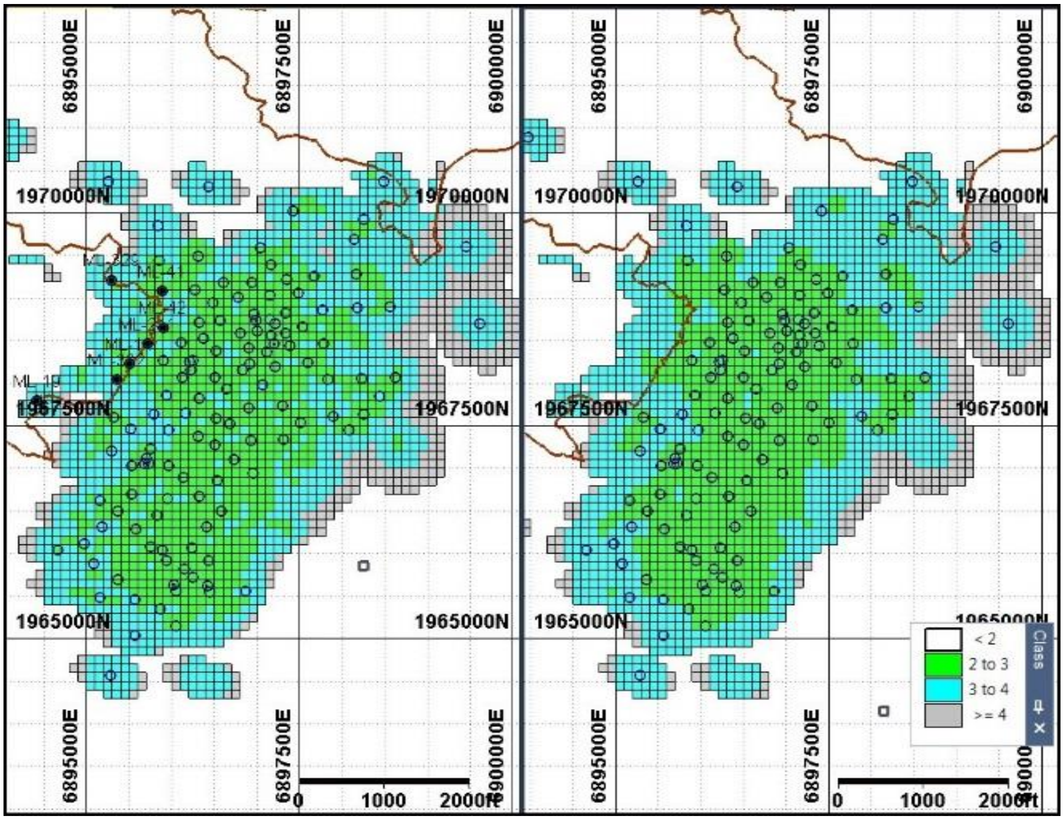
Specific criteria for pre-classification of blocks with code equal to 2 are estimation on the first pass with the average distance to composites less than 300 ft, the distance corresponding to 90% of the correlogram sill and the nominal deposit drillhole spacing, or if estimated on the second pass, the nearest anisotropic distance is 150 ft, equal to one-half the drillhole spacing. The mean block average distance from composites (after smoothing, discussed below) is 175 ft. Blocks with a pre-class code equal to 3 must be within an anisotropic distance of 300 ft from the nearest composite in the search neighborhood.

14.15.2 SMOOTHING

The next step in producing the model class coding involves a limited degree of smoothing to eliminate isolated blocks with different pre-class codes than their neighbors. These can occur due to edge effects, irregular boundaries, and step-wise cutoffs for sample and search criteria. Smoothing for the Moonlight Mineral Resource model comprises a computer-assisted method. Separate block estimates to a dummy variable are made for each class code category, in the Moonlight case, “2”, “3”, and “4”, where blocks coded “4” are all those not in the other recognized categories. The estimates use a restricted search radius to include only the adjacent block centroids and their assigned pre-class codes as input. The number of input centroids is recorded for each estimate which gives a count of the number of “2”s, “3”s, and “4”s in each block. A script compares the counts and assigns the dominant pre-class code to a new block variable “class”. Thus, the initial pre-class code is objectively smoothed in 3D by the pre-class codes of the neighboring blocks, resulting in a model with reasonably contiguous groups of blocks having the same class coding (Figure 14.14).

The smoothing eliminates many of the pre-class inliers equal to 3 and islands of pre-class equal to 2 on the fringes of the main body of mineralization. The class 2 perimeter conforms more closely to the perimeter of the 300 ft-spaced drilling grid. In section, the smoothing reduces the projection of class 2 material below the bottoms of the drillholes by one or two benches.

Figure 14.14 Comparison of Block Classification before and after Smoothing Procedure, 5,475 Elevation, Showing Bench Composites for Reference



14.15.3 REASONABLE PROSPECTS FOR EXTRACTION

Classification of Mineral Resources under CIM definitions includes a test of potential economic viability. The mining method for the Moonlight deposit will be open pit, for which pit optimizations discussed in Section 16.0 form the basis for demonstration of "reasonable prospects for economic extraction" of the Mineral Resources discussed here. Pit optimization parameters determined by Tetra Tech for this PEA are summarized in Table 14.13.

Table 14.13 Pit Optimization Parameters for Resource Estimation Constraint

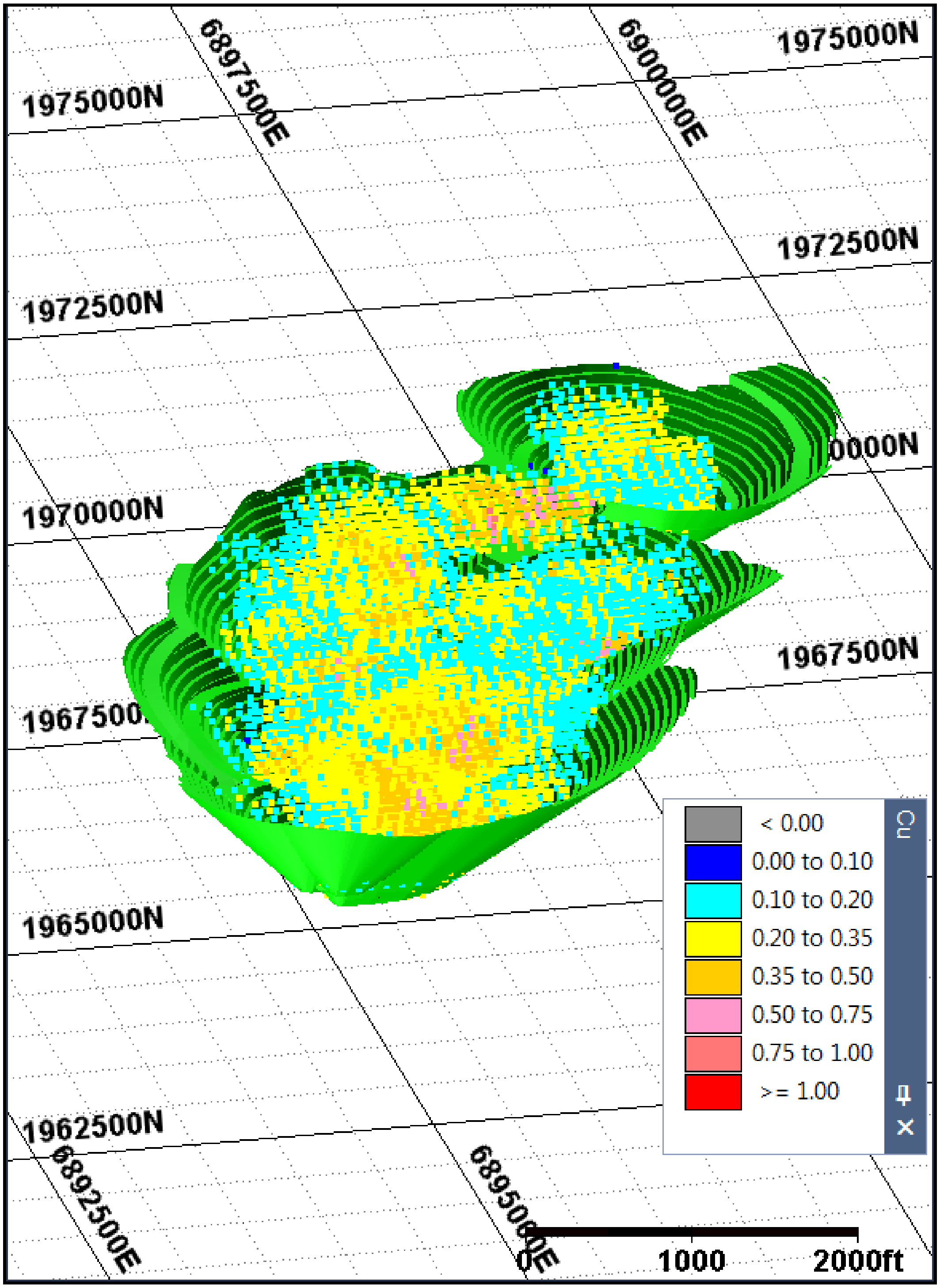
Item	Unit	Assumption
Revenue, Smelting and Refining		
Copper Price	US$/lb	3.00
Gold Price*	US$/oz	1275
Silver Price	US$/oz	17.50
Copper Recovery	%	86
Gold Recovery	%	70
Silver Recovery	%	70
Copper Concentrate Deductions	%	1.00
Gold Concentrate Deductions	oz/st	0.03
Silver Concentrate Deductions	oz/st	1.00
Gold Payable	%	98
Silver Payable	%	90
Refining Charges Copper	US$/lb	0.08
Refining Charges Silver	US$/oz	0.30
Transport and Concentrate Loadout	US$/st concentrate	40
Ocean Freight	US$/st concentrate	25
Other Off-site Costs (Losses, Ins, Sale Rep. Assay)	US$/st concentrate	15
Treatment Charges Copper	US$/st concentrate	80
Operating Cost Estimates		
Open Pit Mining Cost	$/st mined	1.25
Processing Cost	$/st milled	6.00
G&A	$/st milled	0.25
Process and Mining Recovery		
Mining Recovery	%	99.5
Dilution	%	2.0
Geotechnical		
Slope Angles (Overall)	degrees	45
Production Limits		
Process Throughput	st/a	60,000

Note: *The PEA was completed using annually average grades. The annually averaged gold grades are too low to result in a payable credit in the concentrates. It however possible that on a concentrate consignment basis, some consignments will have sufficient grade to warrant payability of the gold.

Only Indicated Mineral Resources were used for the pit optimization. A NSR value was calculated for each block lying in the pit using the inputs in Table 14.13 and the formula: NSR=(Cu%*44.08)+(0.348*3.10348*Ag(troy oz/st)). Blocks with an NSR greater than the sum of processing cost and G&A ($6.00 + $0.25 = $6.25) were considered to have economic potential justifying consideration as Mineral Resources. Mineral Resources lie within the pit shell (Figure 14.15) which covers most of the principal mineralized zone to as deep as 4,600 ft elevation.

The main pit is connected to two smaller sub-pits at the northeastern and eastern edges of the deposit.

Figure 14.15 Perspective View (Looking Down to Northeast) of Mineral Resources Constrained by Optimized Pit Shell



14.16 MINERAL RESOURCE STATEMENT

Mineral Resources for the Moonlight copper deposit are listed in Table 14.14 for Indicated and Inferred Mineral Resources, respectively. There are no Measured Resources or Mineral Reserves for the Moonlight copper deposit.

Table 14.14 Moonlight Mineral Resources as of December 15, 2017[1,2,3,4,5]

Class	Tons ('000 st)	Cu (%)	Au (oz/st)	Ag (oz/st)	Cu ('000 st)	Au ('000 oz)	Ag ('000 oz)
Indicated	252,000	0.25	0.0001	0.07	636	18	18,400
Inferred	109,000	0.24	0.0001	0.08	267	9	9,000

Notes: [1]The QP for the Mineral Resource estimate is Donald E. Cameron, Registered Geologist, SME.
[2]Rounding as required by reporting guidelines may result in apparent differences between tons, grade, and contained metal content.
[3]Mineral Resources are reported above a US$6.25 NSR cut-off (NSR=44.08*Cu + .348*31.10348*Ag) and within a conceptual pit shell using copper, gold, and silver prices of US$3.00/lb, US$1,275/oz, and US$17.50/oz, respectively, and preliminary operating costs as of the effective date of this Mineral Resource.
[4]Effective date of the Mineral Resource estimate is December 15, 2017.
[5]There is no assurance that Mineral Resources will be converted into Mineral Reserves. Mineral Resources are subject to Modifying Factors and inclusion in a mine plan that demonstrates economics and feasibility of extraction in order to be considered Mineral Reserves.

All Mineral Resources are fresh material; oxidized material is treated as waste in the pit optimization and has been excluded from Mineral Resources.

Recommendations with respect to the Mineral Resource estimate include additional drilling to confirm Placer-Amex assay results for copper, silver and gold. Additional density measurements representative of the entire resource volume should be taken on core samples collected from infill drilling.

14.17 RISK FACTORS

There is no assurance that Mineral Resources will be converted into Mineral Reserves. Mineral Resources are subject to modifying factors and inclusion in a mine plan that demonstrates economics and feasibility of extraction in order to be considered Mineral Reserves. Estimates for some Mineral Resources rely on historical data which cannot be verified without re-sampling. No production records with which to validate the estimates are available since the Moonlight deposit has not been previously mined.

Certain weaknesses and deficiencies have been identified in the drillhole database that should be addressed with the work program discussed in Section 26.0. While none of these, separately or in aggregate, will invalidate the Mineral Resource, they could impact it at the margin, resulting in changes. Further metallurgical and marketing studies will be necessary to verify that gold, in particular, is payable in the concentrate.

14.18 SENSITIVITY OF MINERAL RESOURCE TO CUT-OFF GRADE

Moonlight Mineral Resources are moderately sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the block model tonnage and grade estimates are presented at various cutoffs in Table 14.15 and Figure 14.16 for blocks within the Moonlight deposit optimized pit shell. The reader is cautioned that the figures presented in the tables and graphs should not be confused with a Mineral Resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of NSR cut-off. The Mineral Resource estimates within the conceptual pit shell and above an NSR cut-off of U$6.25 reported in Table 14.14 are highlighted for reference in Table 14.15.

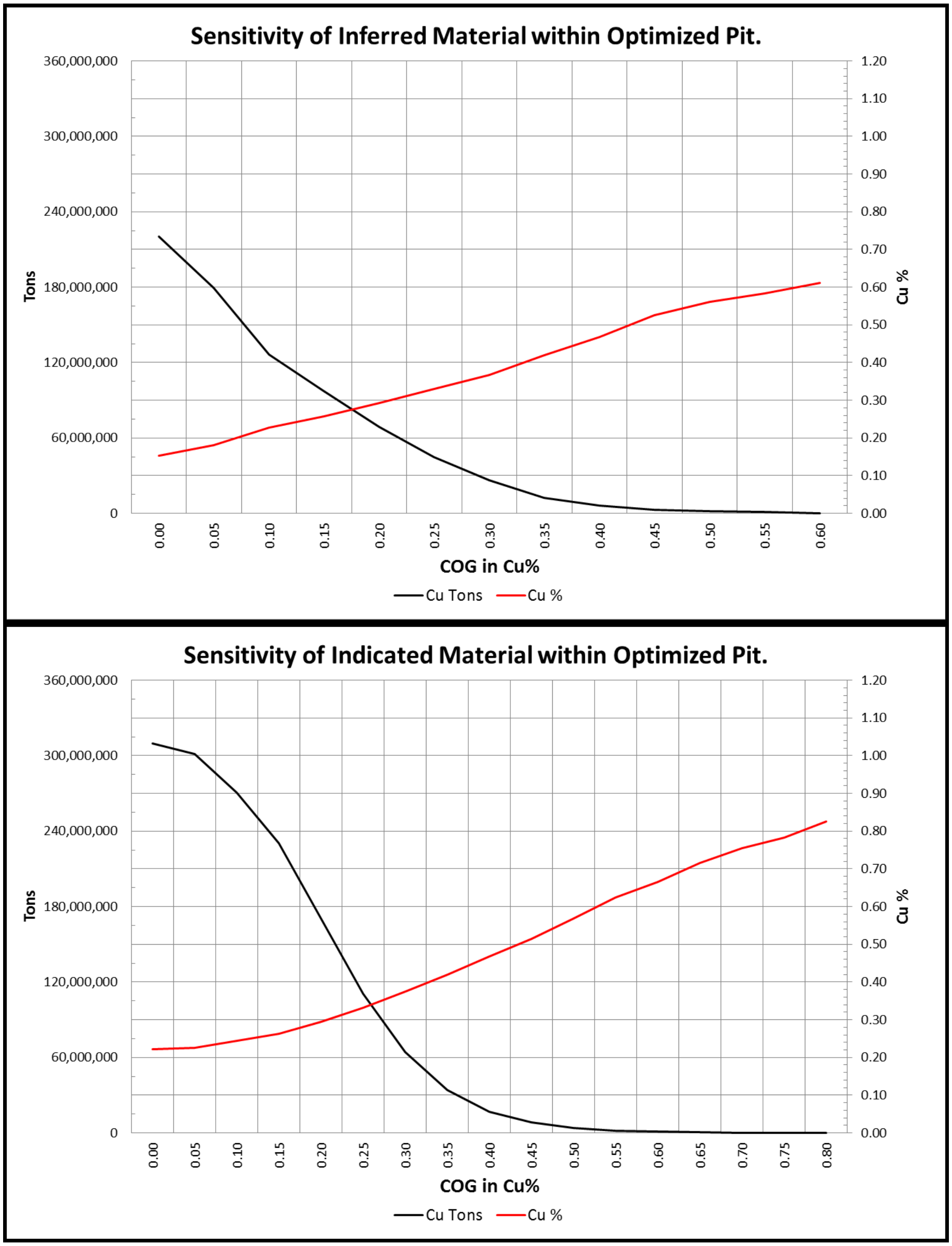
Table 14.15 Mineral Resource Sensitivity to Block NSR Cut-off

NSR Cut-off (US$)	Tons ('000 st)	Cu (%)	Cu ('000 st)	Au (oz/st)	Au ('000 oz)	Ag (oz/st)	Ag ('000 oz)
Indicated							
2	307,000	0.223	684,000	0.0001	20	0.07	20,900
3	296,000	0.229	678,000	0.0001	20	0.07	20,500
4	283,000	0.237	669,000	0.0001	19	0.07	19,900
5	270,000	0.244	658,000	0.0001	19	0.07	19,300
6.25	**252,000**	**0.253**	**636,000**	**0.0001**	**18**	**0.07**	**18,400**
7	237,000	0.260	616,000	0.0001	17	0.07	17,700
8	211,000	0.272	576,000	0.0001	15	0.08	16,300
9	186,000	0.285	531,000	0.0001	14	0.08	14,800
10	158,000	0.301	475,000	0.0001	12	0.08	13,000
Inferred							
2	204,000	0.164	334,000	0.0001	16	0.07	13,400
3	167,000	0.189	317,000	0.0001	14	0.07	11,900
4	141,000	0.212	299,000	0.0001	12	0.08	10,700
5	126,000	0.227	286,000	0.0001	11	0.08	9,900
6.25	**109,000**	**0.245**	**267,000**	**0.0001**	**9**	**0.08**	**9,000**
7	100,000	0.254	255,000	0.0001	9	0.09	8,600
8	87,000	0.269	234,000	0.0001	8	0.09	8,000
9	77,000	0.280	217,000	0.0001	7	0.09	7,300
10	64,000	0.298	191,000	0.0001	6	0.10	6,300

Reporting from the same conceptual pit, additional cases shown in the table illustrate the sensitivity of the estimate to changes in NSR cut-off. The estimate of contained copper is moderately sensitive to NSR values between US$6.25 and US$7.00. Copper is the principal contributor to NSR.

The grade-tonnage curves for blocks within the optimized pit shell at various copper cut-off grades are shown in Figure 14.16 demonstrate a sensitivity of potential resources to copper grade.

Figure 14.16 Grade and Tonnage Curves for Copper Based on Cu% Cut-off



The copper cut-off that gives a similar tonnage and copper grade to the NSR Mineral Resource cut-off is 0.12% copper.

14.19 COMPARISON TO PREVIOUS RESOURCE MODELS

The most recent and comparable Mineral Resource estimate by Orequest and Giroux Consultants Ltd. (Cavey and Giroux 2007) used ordinary kriging for copper, gold and silver within an interpreted QM body solid. Separate low-grade and high-grade population estimates were made for each grade element and combined in proportion to a high-grade indicator factor determined from a separate indicator estimate. The global high-grade mean was used for the high-grade proportion in each block. Block sizes were identical to the current study and drillhole databases were to a large degree comparable except with respect to gold and silver.

Differences in the estimates of current Mineral Resources from the historical estimate presented in the 2007 NI 43-101 Technical Report (Cavey and Giroux 2007) are summarized below:

- additions and modifications to the drillhole database
- estimates constrained by oxidation model
- more detailed lithologic model
- grade shell constraint for copper estimate with no separation of high- and low-grade populations
- capping of outlier grades
- changes in estimation strategy
- classification based on drill spacing and pit shell constraint.

The current estimate estimated copper in four domains specified by lithology and a 0.1% copper grade shell, the latter based on contouring an indicator estimate. Domain boundaries were identified as “hard” or “soft” based on boundary analysis. Directional variograms and searches were oriented less vertically than the previous estimate. The influence of Placer-Amex gold assays was eliminated in the current estimate. The model was checked for global and local biases compared to the underlying composite support, and for change-of-support. The historical estimate based classification exclusively on the variogram ranges, whereas changes in NI 43-101 requirements and CIM definitions since 2014 require tests for reasonable expectation of economic extraction and conversion of a majority of Inferred Resource to Indicated Resource. Table 14.16 shows a comparison of the current and historical estimates at various copper cut-offs; the figures shown as “current” should not be confused with Mineral Resources reported in Table 14.14 which are reported with different cut-off criteria.

Table 14.16 Comparison of Current and Historical Estimates at Various Cu% Cut-off Grades

Cut-off Grade	Current			Historical			Variances (%)		
	Tons	Cu (%)	Cu (st)	Tons	Cu (%)	Cu (st)	Tons	Cu%	Cu Metal
Indicated									
0.1	270,400,000	0.243	658,000	305,270,000	0.240	732,648	89	101	90
0.15	230,100,000	0.263	606,000	227,140,000	0.281	638,263	101	94	95
0.2	170,400,000	0.294	501,000	161,570,000	0.324	523,487	105	91	96
0.25	110,700,000	0.332	367,000	114,570,000	0.366	419,326	97	91	88
0.3	64,500,000	0.374	241,000	76,150,000	0.413	314,500	85	90	77
Inferred									
0.1	126,300,000	0.227	287,000	272,940,000	0.187	510,398	46	121	56
0.15	97,200,000	0.258	250,000	158,250,000	0.234	370,305	61	11	68
0.2	68,600,000	0.292	201,000	88,350,000	0.282	249,147	78	104	81
0.25	44,600,000	0.329	147,000	48,820,000	0.329	160,618	91	100	92
0.3	26,300,000	0.368	97,000	23,720,000	0.390	92,508	111	94	105

The current estimate of Indicated category material is moderately smoothed with respect to copper compared to the historical estimate. The tonnage variance is positive at 0.15 and 0.2% copper cut-offs, but is somewhat reduced by the pit constraint to demonstrate economic potential of Mineral Resources in current NI 43-101 guidance. Less Inferred category tons are estimated, but the grade is higher than the historical estimate. Most of the Inferred variance is actually due to the application of the pit constraint, without which, the current estimate would appear more smoothed relative to the historical one.

14.20 COMMENTS ON CHAPTER 14

The QP is of the opinion that estimation of Mineral Resources for the Moonlight copper deposit has been performed to Estimation of Mineral Resources and Mineral Reserves Best Practices (CIM 2003), and conforms to the requirements of CIM Definition Standards (CIM 2014). Recommendations are made for further work to improve the quality of the database used for Mineral Resource estimation. In particular, assay certificates and core for the Placer-Amex portion of the database have been misplaced or destroyed and can only be verified by the trail of historical reports and assay values written on logs. Furthermore, gold, and to a lesser extent silver assaying by Placer-Amex was carried out by methods that lacked sufficient sensitivity for the low levels of these metals in the deposit, and are composites of long intervals. Preliminary metallurgical tests and the limited Sheffield drilling program show that gold is only a minor potential credit in concentrates. Additional infill drilling will further mitigate the Placer-Amex data issue and will allow formulation of a definitive grade control plan and budget.

Discussion of environmental, permitting, legal, title, taxation, socio-economic, marketing, political and other relevant factors that could materially affect the Mineral Resource estimates are included in Chapters 18 to 20, and 22.

15.0 MINERAL RESERVE ESTIMATES

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource and has not been estimated for the Project as part of this PEA.

16.0 MINING METHODS

16.1 INTRODUCTION

All mining for the Moonlight deposit will be conducted utilizing conventional open pit mining methods with drill and blast, followed by load and haul with large diesel truck and shovel equipment.

Limited information on rock quality is available for mine planning, and as such Tetra Tech has assumed the bulk of the mining will be hard rock excavation, which requires drill and blast.

The preliminary mine plan includes inferred Mineral Resources. Preliminary technical and economic considerations have been applied in this study and all material classified as Inferred was considered material.

Industry standard methodologies were adopted for cut-off grade estimation, pit optimization, detailed design, and mine scheduling/sequencing. The following steps were taken in part of the planning process:

- NSR values were assigned to the Mineral Resource block model by determining commodity prices, transport/freight costs, as well as treatment and refining charges inclusive of any metal deductions.
- Pit optimization parameters, such as mining; processing; and G&A costs, as well as defined pit wall angles and metallurgical recoveries were determined.
- Economic ultimate pit limits on the deposit were selected using GEOVIA Whittle™ software. A series of nested pit shells were generated by decreasing/increasing the revenue factor, and the most economical pit was selected.
- Pit design parameters and design of ultimate pit using GEOVIA GEMS™ mining software were selected.
- The ultimate pit schedule was created using GEOVIA Whittle™ software and realistic operational pushbacks throughout the LOM were introduced.
- Equipment fleet requirements for the LOM production schedule were estimated.
- Waste rock storage facility (WRSF) was designed and the volume calculated.
- TMF was designed and the volume calculated.

16.2 OPEN PIT OPTIMIZATION

An open pit optimization was completed prior to developing an open pit mine design. The optimization used the Lerchs-Grossman algorithm in the GEOVIA Whittle™ software package.

Prior to importing the block model into the optimization software, Tetra Tech estimated an NSR value which accounts for metal content, metal recoveries, all off-site charges, and metal deductions. The NSR was then used as the selling value for each block, as opposed to assigning metal prices directly in the optimization software. Using this method, all off-site costs applicable to the product in the form of copper concentrate are accounted for in the NSR values, and only mine site costs are used in the GEOVIA Whittle™ optimization.

It must be noted that the GEOVIA Whittle™ optimization was conducted in US imperial units, using tons as the primary mass unit.

The optimization was run by inputting mining costs, processing costs, G&A costs, pit slope, and mining throughput limit, which in turn determines the ultimate pit shell that effectively maximizes value.

The software generated a series of nested pit shells using a range of commodity selling prices, and in this case the NSR value. Tetra Tech selected revenue factors between 0.3 and 2.0, increasing every 0.02, which yielded 86 nested pits increasing in size. This process was used to evaluate the effect of the pit size and stripping ratios on the Project NPV. In theory, the software pushes out the less profitable material in to the larger pit shells when the NSR value is increased.

The optimization parameters used are shown in Table 16.1 and are based on preliminary estimates. These preliminary estimates differ slightly from the final estimates used in the economic model; however, overall differences between the preliminary and final numbers are considered insignificant. The pit optimizations were run using Indicated and Inferred Mineral Resources.

Table 16.1 Open Pit Optimization Parameters

Item	Unit	Assumption
Revenue, Smelting and Refining		
Copper Price	US$/lb	3.00
Gold Price*	US$/oz	1275
Silver Price	US$/oz	17.50
Copper Recovery	%	86
Gold Recovery	%	70
Silver Recovery	%	70
Copper Concentrate Deductions	%	1.00
Gold Concentrate Deductions	oz/st	0.03
Silver Concentrate Deductions	oz/st	1.00
Gold Payable	%	98
Silver Payable	%	90
Refining Charges Copper	US$/lb	0.08
Refining Charges Silver	US$/oz	0.30
Transport and Concentrate Loadout	US$/st concentrate	40
Ocean Freight	US$/st concentrate	25
Other Off-site Costs (Losses, Ins, Sale Rep. Assay)	US$/st concentrate	15
Treatment Charges Copper	US$/st concentrate	80
Operating Cost Estimates		
Open Pit Mining Cost	$/st mined	1.25
Processing Cost	$/st milled	6.00
G&A	$/st milled	0.25
Process and Mining Recovery		
Mining Recovery	%	99.5
Dilution	%	2.0
Geotechnical		
Slope Angles (Overall)	degrees	45
Production Limits		
Process Throughput	ton/a	60,000

Note: *The PEA was completed using annually average grades. The annually averaged gold grades are too low to result in a payable credit in the concentrates. It however possible that on a concentrate consignment basis, some consignments will have sufficient grade to warrant payability of the gold.

A NSR value was calculated for each block lying in the pit using the inputs in Table 16.1 and the formula: NSR=(Cu%*44.08) + (0.348*3.10348*Ag(troy oz/st)). Blocks with an NSR greater than the sum of processing cost and G&A ($6.25) were considered to have economic potential justifying consideration as Mineral Resources. Mineral Resources lie within the pit shell (Figure 14.15 and 16.5) which covers most of the principal mineralized zone to as deep as 4,600 ft elevation.

16.3 OPEN PIT OPTIMIZATION RESULTS

The results of the pit optimization exercise are presented in Figure 16.1. Pit shell No. 28 was selected as the basis for the engineered pit design and mine planning moving forward. The specific results of pit No. 28 are shown in Table 16.2.

Figure 16.1 Moonlight Project Pit by Pit Graph

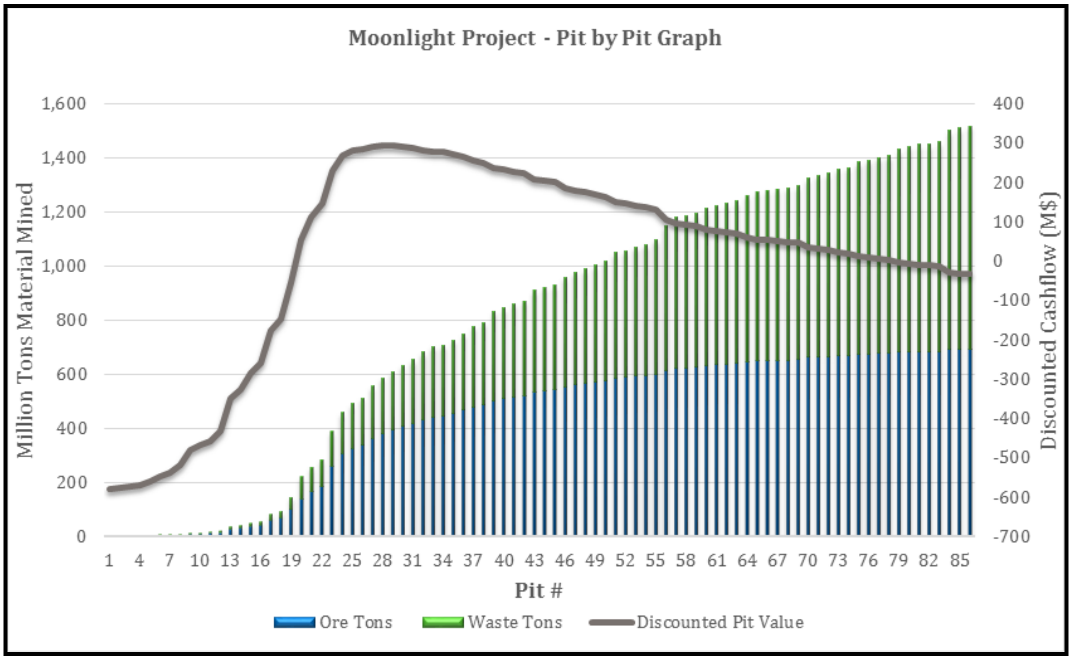


Table 16.2 Pit Optimization Results – Pit 28

Item	Units	Results
Mineralized Material	st	381,977,460
Diluted Copper Grade	%	0.244
Contained Copper	st	932,025
Diluted Gold Grade	oz/st	0.00007
Contained Gold	oz	26,738
Diluted Silver Grade	oz/st	0.075
Contained Silver	oz	28,648,310
Waste	st	204,410,854
Total Material	st	586,388,314
Strip Ratio	st:st	0.54

Crown Mining originally provided the block model to Tetra Tech with imperial units; therefore, the optimization was carried out using short tons.

Looking at Figure 16.1, there are visual indications of larger spikes in excavated volume between pits. As such, these were selected and assigned as pushbacks, and used in the optimization scenario to influence the pit discounted cash flow. The pit shells selected for pushbacks included: pit shell No. 13, pit shell No. 19, pit shell No. 20, and pit shell No. 23.

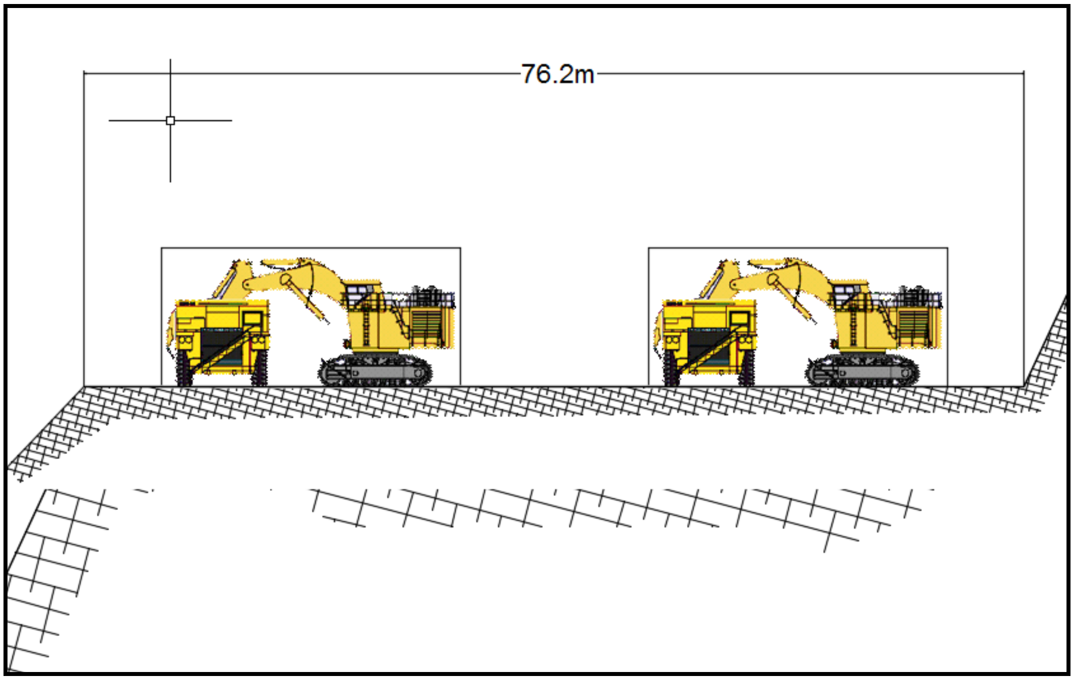
16.4 MINE DESIGN

Tetra Tech modelled the optimized pit shell (pit No. 28) using GEOVIA GEMS™ software. The pit shell generated by GEOVIA Whittle™ software was imported into the mine design software and used as a guideline. Tetra Tech defined parameters incorporated into the final pit design (Table 16.3 and Figure 16.2). The resulting pit design includes practical geometry that is required in an operational scenario, such as the haul road to access all mining faces, pit slopes with berms, and standard benching configuration.

Table 16.3 Pit Design Parameters

Item	Imperial		Metric	
	Unit	Value	Unit	Value
Pit Walls				
Bench Height	ft	50	m	15.2
Berm Width	ft	27	m	8.2
Bench Face Angle	degrees	65	degrees	65
Overall Pit Slope Angle	degrees	44.8	degrees	44.8
Haul Roads				
Double Lane Road	ft	100	m	30.5
Single Lane Road	ft	72	m	22
Max ramp Grade	%	8	%	8
Mining				
Minimum Pushback Operating Width	ft	250	m	76.2
Minimum Pit Bottom Mining Width	ft	250	m	76.2

Figure 16.2 Minimum Mining Widths



16.4.1 Ramp Design

The pit was designed to have varying haul road widths. A majority of the pit has a double-lane road, which is 100 ft (30.5 m) wide. This is based on the industry standard for the running width of a haul road to be a minimum of 3.5 times the width of the largest equipment, which in this case is a Komatsu 830 E haul truck. This does not include additional allocation for a drainage ditch and safety berm.

The depths of the pit were transitioned to a single-lane road to achieve a steeper pit slope and minimize the excavation and removal of uneconomical material. The single-lane road dimensions are 72 ft (22 m) wide.

The overall ramp gradient is 8% for both double lane and single lane roads.

Figure 16.3 Double Lane Haul Road

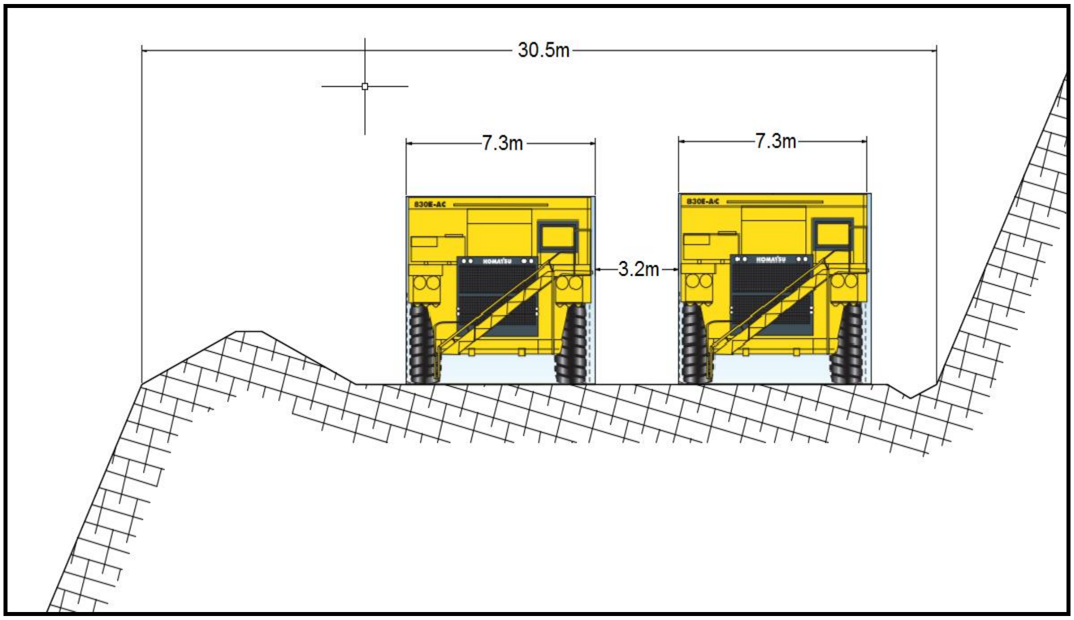
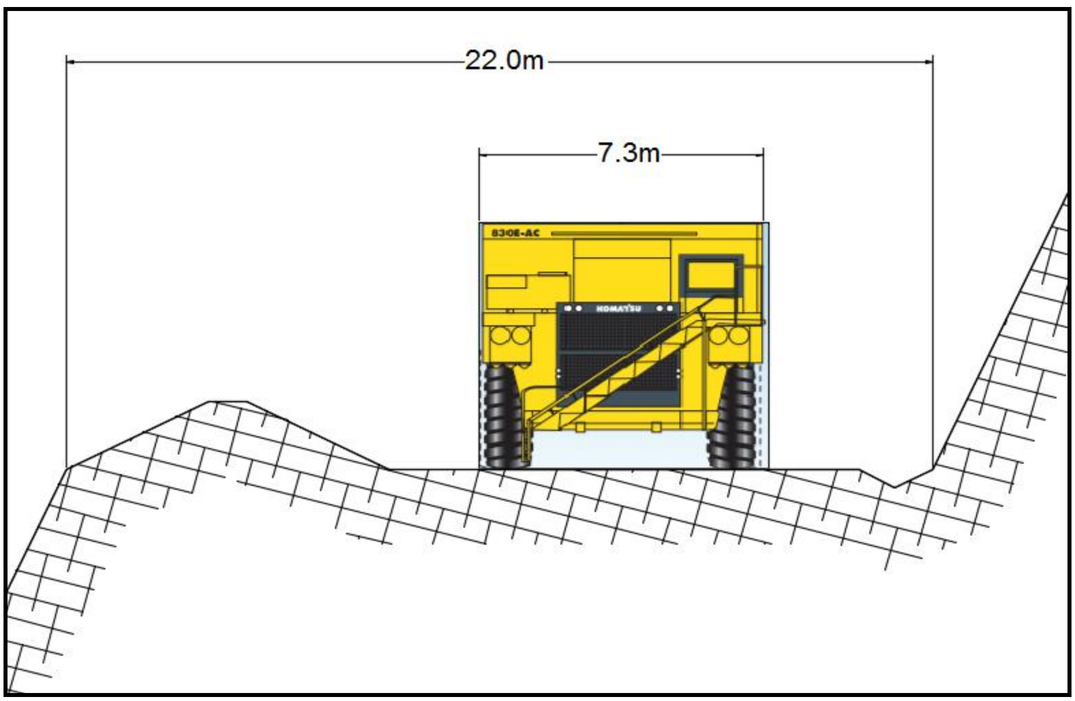


Figure 16.4 Six Lane Haul Road, Used for Pit Bottom Access Only



16.4.2 Moonlight Pit Design Results

The detailed mine design final pit result is shown in Figure 16.5.

Figure 16.5 Pit Design Result

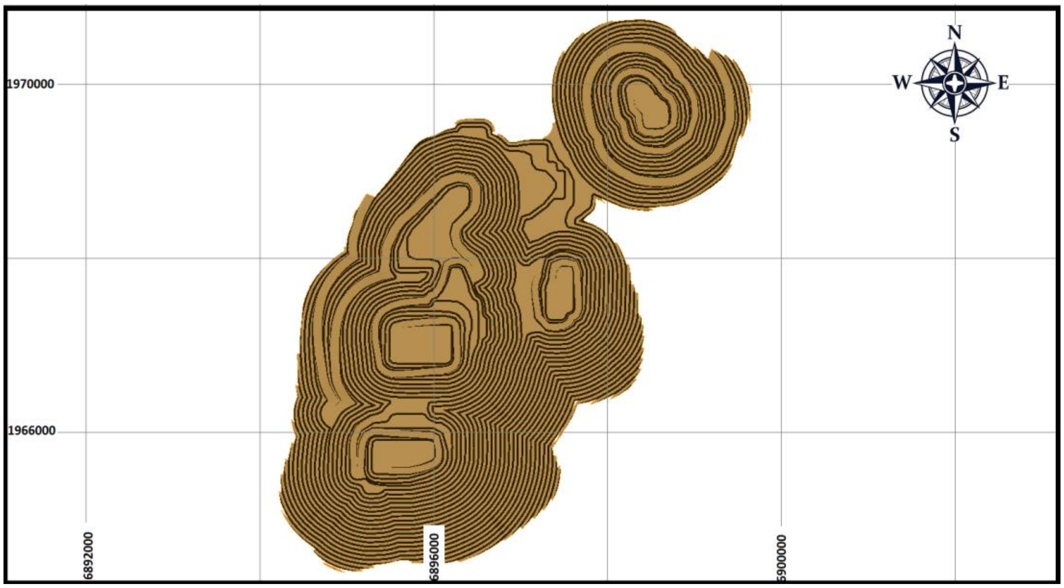


Table 16.4 highlights the tons and grades of the material extracted from the engineered designed pit.

Table 16.4 Designed Pit Results

Item	Units	Results
Mineralized Material	million st	365
Diluted Copper Grade	%	0.25
Contained Copper	'000 st	912
Diluted Silver Grade	oz/st	0.075
Contained Silver	million oz	27
Waste	million st	286
Total Material	million st	651
Strip Ratio	st:stn	0.78

16.4.3 Interim Pit Phases (Pushback) Designs

Tetra Tech designed the first three pushbacks for the Moonlight deposit, recognizing that the project economics are sensitive to scheduling. The pushbacks discussed in Section 16.3 were used as guides to design pits that had practical geometry similar to the optimized pit shape. Figure 16.6 to Figure 16.8 highlight these first few phases of mining.

Figure 16.6 Phase 1 Design

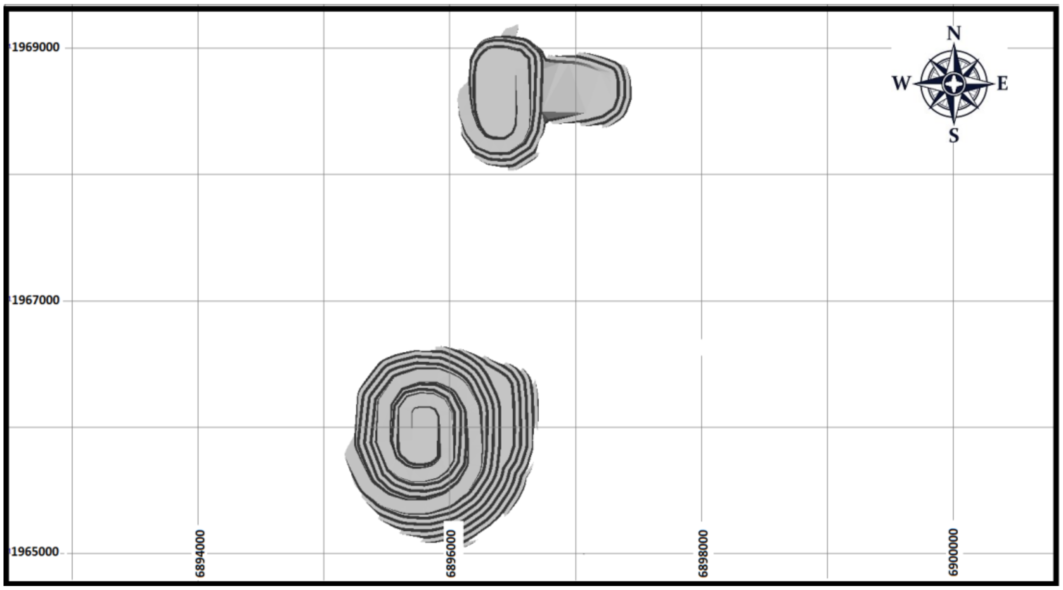


Figure 16.7 Phase 2 Design

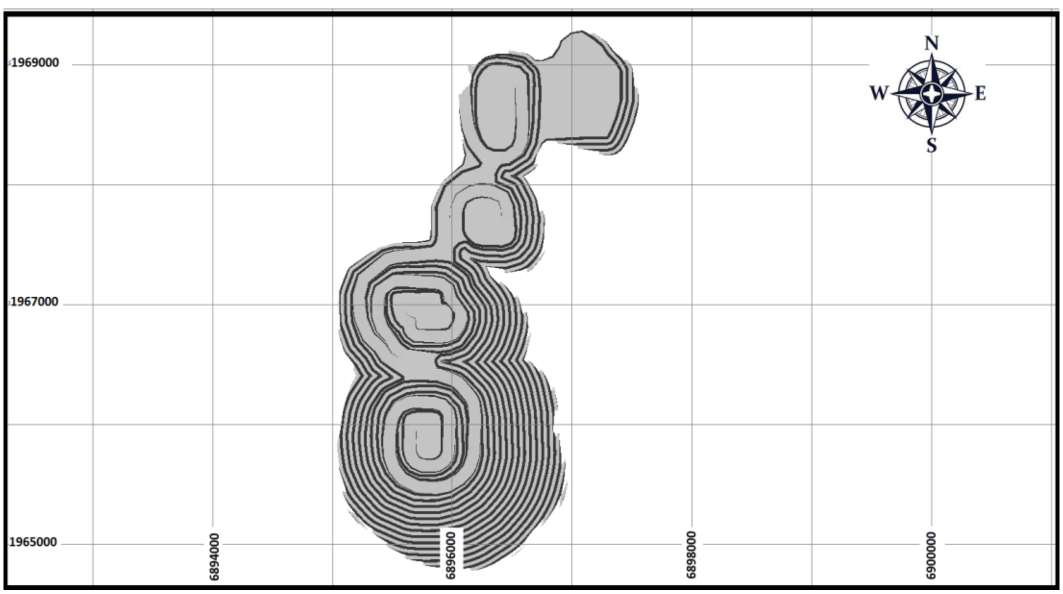
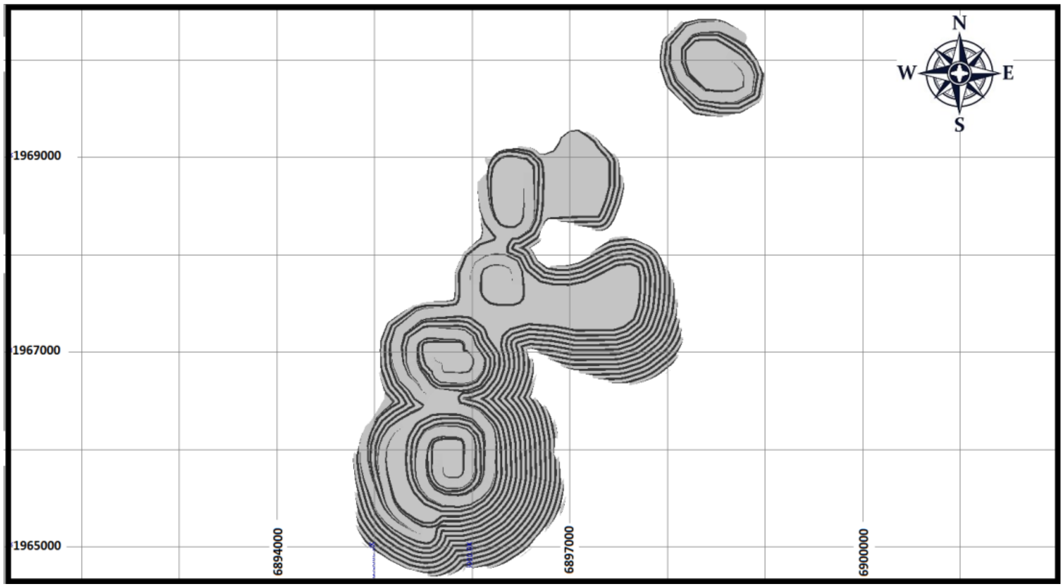


Figure 16.8 Phase 3 Design



16.5 Mining Operations

Mining for the Moonlight deposit will be conducted utilizing conventional open pit mining methods with drill and blast followed by load and haul with large diesel truck and shovel equipment.

Material will be drilled and blasted and then loaded with a hydraulic shovel into a fleet of haul trucks. This material will then be hauled from the open pit location to the primary crusher, where it will be end dumped into the crushing pocket. Waste will also be drilled and blasted, but will be loaded via one of two wheeled loaders. This waste will then be hauled to the WRMF and end dumped. From here a tracked dozer will push the dumped piles and shape the management facility as required.

Support labour and equipment has been selected for the mine operation, which will aid in the overall mining activities.

16.5.1 Drilling and Blasting

Based on the selected bench height and the production schedule, a 10-inch diameter production drill was selected with a smaller drill rig to be used for pre-splitting. The blast design concept anticipates use of ammonium nitrate fuel oil (ANFO) as the main explosive for the blast holes (90%) with 10% by emulsion. For the PEA, explosives handling, transport and delivery to the mine site is assumed to be conducted by explosives contractor as a “down the hole” service.

Table 16.5 shows details of the drilling and blasting parameters.

Table 16.5 Ore/Waste Drill and Blast Parameters

Blasting Specifications	Unit	Ore	Waste
Powder Factor (Medium Rock)*	lb/ton	0.41	0.41
Explosive Cost Used (Down the Hole Service)**	US$/lb	0.65	0.65
Hole Diameter Used	in	10	10
Burden	ft	25	25
Spacing	ft	30	30
Bench Height	ft	50	50
Tons Blasted per Hole	ton	3,383	3,383
Explosive per Hole Required	lb	1,387	1,387

Notes: *Industry typical
**Dyno Nobel/Alpha Explosives budgetary quote

16.5.2 LOADING AND HAULING

Tetra Tech selected primary loading equipment based on its ability to mine selectively, while also matching the bucket size to the capacity of the trucks selected for the operation.

Primary loading is planned to be performed by a diesel hydraulic front shovel (Komatsu PC4000) with support from two-wheeled loaders (Komatsu WA1200) that will typically be allocated to loading waste. The shovel has a 29 cu yd capacity while the wheeled loaders have a 26 cu yd capacity.

Tetra Tech has also selected a smaller excavator and wheeled loader for support around the mine as needed. The smaller excavator will also be utilized for pushback areas that don't meet the minimum mining width, allowing for further flexibility and selectivity. This smaller excavator can be coupled with the smaller Komatsu trucks Tetra Tech has included in the full equipment list (Table 16.6).

Komatsu 830E trucks were selected for the mining operation, which have a 240 st (218 t) capacity. Tetra Tech developed haul profiles for the final pit as well as the initial starter pit to both the WRSF as well as the crusher area for dumping. This initial and end-of-mine-life haul profiles were then used to estimate distances for years in between. These profiles were imported to Runge TALPAC® truck and loader productivity and cost analysis software to determine cycle times and ultimately come up with the number of trucks required for the operation.

16.5.3 SUPPORT EQUIPMENT

Tetra Tech selected auxiliary and support equipment based on the size and type of primary loading/hauling equipment, which in turn decides geometries of the open pit. Table 16.6 lists the number and type of each of the support equipment that has been selected for the Moonlight operation.

The major tasks that for each type of support equipment include:

- Track Dozer – used for shovel support and cleanup, maintenance at the WRSF, road construction, as well as other projects as needed.
- Wheel Dozer – used to support WRSF, and shovel floor maintenance.
- Grader – predominantly used for road grading/maintenance, road construction services, WRSF and pit floor maintenance.
- Compactor – used for road construction, foundation preparation for civil works, and other projects as needed.
- Fuel and Lube Truck – supply and deliver diesel fuel and lube to all major and minor equipment around site as required.
- Mechanical Truck – equipped with tools, welding machine, common replacement parts to provide preventative and corrective maintenance to equipment around site as required.
- Water Truck – predominantly used for dust suppression along service and open pit haul roads.
- Low-boy Flatbed and Truck – used to transport smaller equipment around site as well as other major components.
- Pumps – used for pit dewatering.
- Lighting Towers – required for illuminating pits, at WRSF, and other construction areas.
- Mobile Crane – used for field maintenance and hoisting for construction projects.

16.5.4 Mine Equipment Requirements

Table 16.6 highlights all major and minor equipment envisaged for the Moonlight deposit. The table shows equipment required for the initial four years of mine life, and then shows the numbers required for each equipment type to be added to the fleet to meet production requirements.

Table 16.6 Mining Equipment List

Task	Selected Equipment	No. of Units Years 1 to 4	No. of Units Years 4 to 17
Primary Equipment			
Blast hole Drilling	Atlas Copco PV-235	2	1
Pre-split Drilling	Atlas Copco D65	1	1
Loading	Komatsu PC4000	1	0
Loading	Komatsu WA1200	2	0
Loading/Spill Cleanup	Komatsu WA900	1	1
Excavator	Komatsu PC800LC	1	1
Haul Trucks	Komatsu 830E	11	6

table continues...

Task	Selected Equipment	No. of Units Years 1 to 4	No. of Units Years 4 to 17
Haul Trucks	Komatsu HD325	2	0
Dozers	Komatsu D375A	4	2
Dozers	Komatsu D155AX	2	0
Support/Ancillary Equipment			
Grader	-	1	1
Compactor	-	1	0
Water Truck	-	1	0
Wheel Dozer	-	1	1
Excavator with Rock Hammer	-	1	0
Lowboy/Flatbed	-	1	0
Prime Mover for Lowboy	-	1	0
Crane – 30 t	-	1	0
Maintenance with Jib Crane Truck	-	1	0
Welding Truck	-	1	0
Fuel and Lube Truck	-	1	1
Boom/Bucket Truck (Cherry Picker)	-	1	0
Light Vehicles	-	5	5
Lighting Towers	-	8	0
Pumps	-	1	5

16.6 Mine Scheduling

A preliminary mine schedule was generated using GEOVIA Whittle™ software and Microsoft® Excel. Mine scheduling was based on use of the GEOVIA Whittle™ Milawa algorithm to optimize project value. Scheduling was completed using designed pushbacks and use of pit lists. The pushbacks are then used to create mine phases. No specific cut-off grade was used other than a break-even cut-off grade. No stockpiles are used in the schedule.

A summary of the mine schedule is shown in Table 16.7. Mill feed of 365 million st will be mined, along with 286 million st of waste rock

.

Table 16.7 Mine Schedule

Year	Unit	Years																	
		-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Mill Feed	st		19	22	22	22	22	22	22	22	22	22	20	20	22	22	22	22	22
Waste Rock	st	6	12	19	15	23	23	2	19	23	23	23	24	25	15	13	10	6	4
Strip Ratio	st:st		0.7	1	0.7	1.1	1	0.3	0.7	1.1	1.1	1.1	1.4	1.2	0.7	0.6	0.4	0.3	0.2
Au Grade	oz/'000 st		0.12	0.08	0.10	0.08	0.07	0.10	0.07	0.07	0.10	0.07	0.05	0.06	0.06	0.05	0.05	0.05	0.04
Cu Grade	%		0.32	0.3	0.27	0.23	0.23	0.31	0.2	0.21	0.23	0.24	0.25	0.24	0.23	0.22	0.21	0.23	0.27
Ag Grade	oz/st		0.09	0.07	0.08	0.06	0.06	0.07	0.05	0.06	0.06	0.08	0.10	0.07	0.07	0.07	0.08	0.09	0.11

Note: *The PEA was completed using annually average grades. The annually averaged gold grades are too low to result in a payable credit in the concentrates. It however possible that on a concentrate consignment basis, some consignments will have sufficient grade to warrant payability of the gold.

16.7 WASTE ROCK MANAGEMENT

The WRSF will be located on the northeast side of the pit. Tetra Tech placed the waste rock as close to the pit perimeter as possible, due to the requirement of having to backfill the open pit with the waste rock during closure. This will minimize transportation costs not only during mining of the open pit but during backfill operations as well for closure. Placing waste close to the pit also provides the option of dozing the waste into the pit, eliminating the requirement for back haul all together. This will ultimately depend on the operational practicality but has been envisaged as the method of backfilling the pit for this study.

The pit slope geotechnical implications of having waste rock immediately above has not been investigated, and it is recommended that in subsequent studies further analysis be completed.

Waste rock has also been assumed as the main borrow material to be used for the construction of mill feed stockpile pad, TMF, and infrastructure facilities, as necessary during the site construction phase.

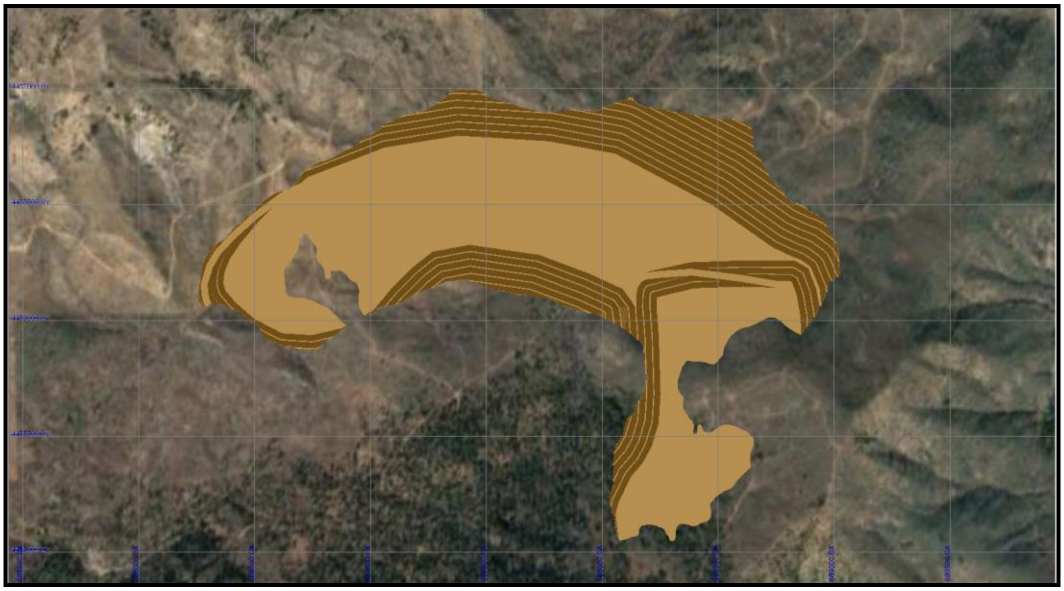
An access ramp along the west side of the WRSF is designed for two-way traffic to allow for access as placement of material advances.

The WRSF has been designed to a capacity of 134 million m^3 based on a density of 0.083 st/cu ft (2.66 t/m^3) and a 30% swell factor, which accommodates the LOM waste rock volume as indicated by the mining schedule. The WRSF was designed according to the geotechnical specifications detailed in Table 16.8. The WRSF design is shown in Figure 16.9

Table 16.8 WRMF Design Parameters

Item	Imperial		Metric	
	Unit	Value	Unit	Value
WRSF Slope				
Dump Bench Height	ft	49.2	m	15.0
Dumpy Bench Width	ft	16.4	m	5.0
Dump Face Angle	degrees	30.0	degrees	30.0
Overall Pit Slope Angle	degrees	25.8	degrees	25.8
Haul Roads				
Double Lane Road	ft	100	m	30.5
Ramp Grade	%	8	%	8
Mining				
Minimum Pushback Operating Width	ft	250	m	76.2

Figure 16.9 WRMF Design



16.8 Mine Personnel Requirements

The mine will operate 24 h/d, 7 d/wk, and assumed it will operate 365 d/a with currently no scheduled non-production days. Operations and mining personnel would work two, 12 h shifts per day. All staff required to keep the mine up and running around the clock would primarily work rotational shifts with two-weeks on and two-weeks off. All hourly labour and supervisory personnel would rotate between day and night shifts. Management and technical staff would work the day shift only.

Equipment operator labour requirements are based on the estimated equipment hours that were derived using equipment productivities, quantity of the various material streams being moved, mechanical availability, as well as utilization rates. Maintenance and support labor requirements are estimated based on Tetra Tech's experience as well as by benchmarking with other similar sized projects in similar environments.

16.8.1 Organizational Structure

The mine will follow a structure similar to a majority of mines, and will be overseen by the mine manager, who will report to the general manager. The mine operations is planned to have three major departments: mine operations, mine maintenance, and technical services consisting of engineering and geology.

The mine operations department will be responsible for all operator training and day-to-day operations of the open pit and the WRSF. This includes drilling, blasting, loading, and hauling of material; dump/open pit haul road construction; as well as pit dewatering activities. Each crew will be led and supervised by a mine shift foreman who will report to the mine superintendent.

The mine maintenance department will fall under the leadership of the maintenance superintendent. Working the same shifts as the mine operations department, the department will be responsible for all preventative and corrective maintenance on the mining fleet, which includes all primary and auxiliary equipment.

A chief mine engineer will head the engineering department, whose responsibility would be to develop short-, medium-, and long-range mine plans. The chief geologist and his team will be responsible for updating the Mineral Resource and Reserve models and providing grade control.

Staff and labor requirements are summarized in Table 16.9 for the LOM.

Table 16.9 LOM Staff and Labour Requirements

Personnel (maximums)	No. of Personnel Years1 to 4	No. of Personnel Years 4 to 17
Management and Technical	18	18
Maintenance Staff	37	37
Operators	89	122
Total	**144**	**177**

16.8.2 Grade Control

Grade control activities considered in the PEA include:

- assaying of blast hole drilling chips
- reverse circulation drilling ahead of active mining benches
- employment of grade control technicians and geologists to work with mining crews
- use of the mill facility assay lab to conduct daily assaying of blast holes and reverse circulation drilling samples.

17.0 RECOVERY METHODS

17.1 SUMMARY

The concentrator has been designed to process a nominal 60,000 st/d (54,400 t/d) of copper-silver mineralized material and is expected to produce a marketable copper concentrate of 28% copper.

The unit processes selected were based on the results of metallurgical testing performed by Continental and Allihies, along with resources set out by Crown Mining. The metallurgical processing procedures have been designed to produce a saleable high-grade copper-silver concentrate. The recovery process for gold has been excluded from the process design.

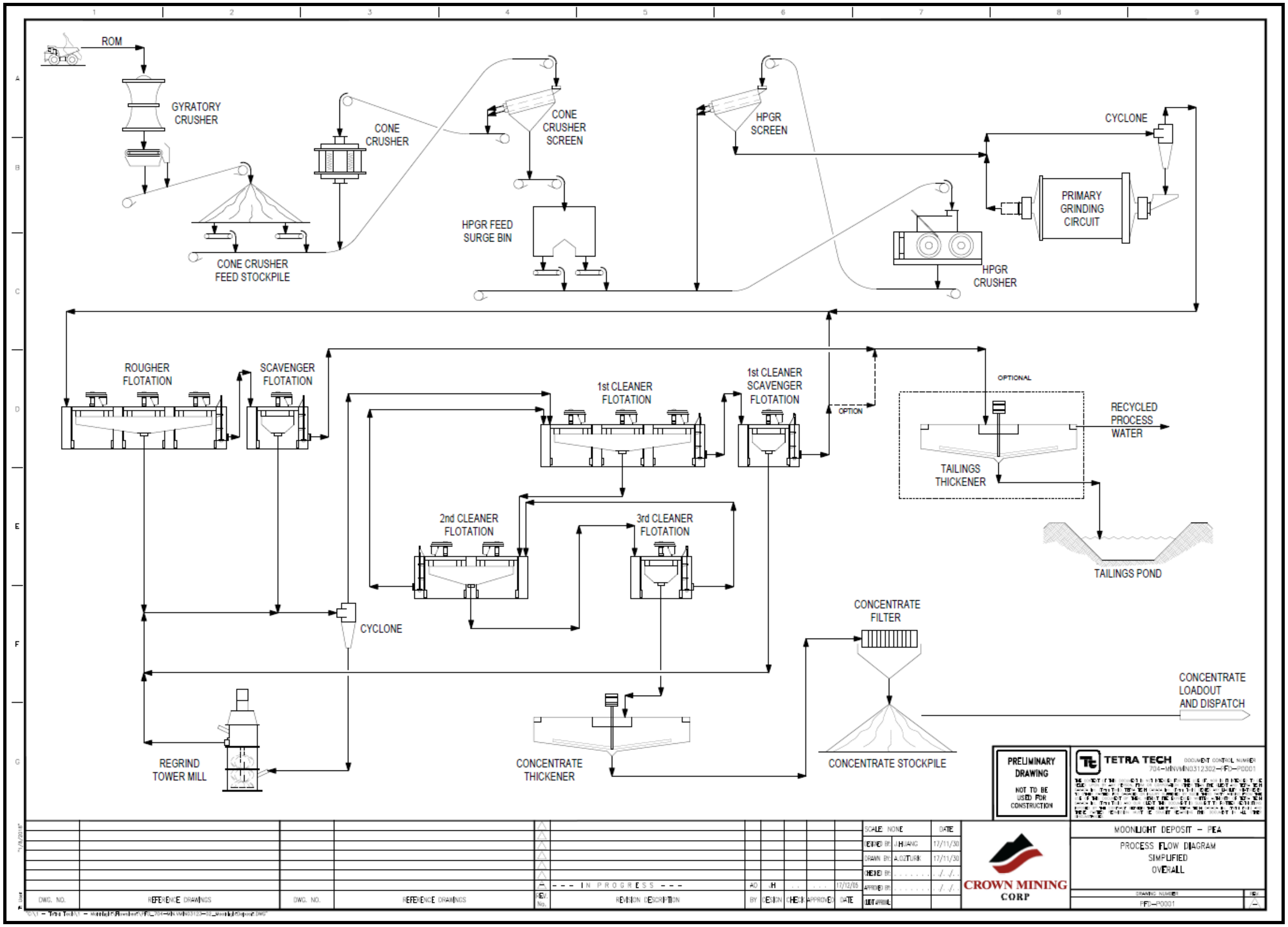
As shown in the simplified flowsheet (Figure 17.1), the treatment plant will consist of crushing and grinding circuits, followed by a flotation process to recover and upgrade copper from the feed material. The flotation concentrate produced will be thickened and filtered and sent to the concentrate stockpile for subsequent shipping to smelters.

The final flotation tailings will be sent and stored in a TMF. Process water will be recycled from the tailings pond. Fresh water will only be used for gland service and reagent preparation.

The process plant will consist of the following unit operations and facilities:

- run-of-mine (ROM) mineralized material receiving
- primary (gyratory) crushing
- crushed mineralized material stockpile and reclaim
- secondary crushing (cone crushers)
- tertiary crushing (HPGR crushers)
- ball mill grinding circuit incorporating with hydro-cyclones for classification
- copper rougher and scavenger flotation
- copper concentrate regrinding
- copper cleaner flotation
- copper concentrate thickening, filtration, and dispatch
- tailings disposal to a tailings pond.

Figure 17.1 Simplified Process Flowsheet



17.2 MAJOR DESIGN CRITERIA

The concentrator has been designed to process 60,000 st/d (54,400 t/d), equivalent to 21,896,000 st/a (19,863,000 t/a). The major criteria used in the design are outlined in Table 17.1.

Table 17.1 Major Process Design Criteria

Criteria	Unit	Value
Operating Year	d	365
Primary Crushing Availability	%	70
Second and Third Crushing Availability	%	92
Grinding and Flotation Availability	%	92
Primary Crushing Rate	st/h	3,571
	t/h	3,239
Milling and Flotation Process Rate	st/h	2,717
	t/h	2,465
HPGR Crusher Discharge Size, 80% Passing	mm	3.0 to 3.5
Ball Mill Grind Size, 80% Passing	µm	110
Ball Mill Circulating Load	%	250
Bond Ball Mill Work Index	kWh/t	21.7
Bond Abrasion Index	g	TBD
Concentrate Regrind Size, 80% Passing	µm	50

The design parameters are based on test work results obtained by Continental and Allihies, as well as Tetra Tech's experience.

The grinding mills were sized based on the Bond Work Index data and related mill feed particle size and product particle size.

The flotation cells were sized based on the flotation retention times as determined during the laboratory test work. Typical scale-up factors have been applied.

17.3 OPERATING SCHEDULE AND AVAILABILITY

The crushing and downstream process plant are designed to operate on the basis of two 12-hour shifts per day, for 365 d/a.

The primary crushing overall availability will be 70% and the cone crushing, HPGR crushing, and grinding and flotation circuit availability will be 92%. This will allow sufficient downtime for the scheduled and unscheduled maintenance of the crushing and process plant equipment.

17.4 PROCESS PLANT DESCRIPTION

17.4.1 PRIMARY CRUSHING AND CRUSHED MINERALIZED MATERIAL STOCKPILE AND RECLAIM

A conventional gyratory crusher facility has been designed to crush ROM mineralized material, to reduce the size of the materials to approximately 80% passing 150 mm at an average rate of 3,571 st/h (3,239 t/h).

The major equipment and facilities in this area include:

- dump pocket
- stationary grizzly
- hydraulic rock breaker
- gyratory crusher – 1,600 mm x 2,286 mm (60 in x 90 in) or equivalent
- crusher discharge apron feeder
- crushed mineralized material stockpile, 55,100 st (50,000 t) live capacity
- reclaim apron feeders
- conveyor belts, metal detectors, self-cleaning magnets, and belt tear detectors
- belt scale
- dust collection system.

The ROM material will be trucked from the open pit to the primary crusher by240 st haul trucks. The mineralized material will be reduced to 80% passing 150 mm using a gyratory crusher. A rock breaker will be installed to break any oversize rocks retained by the static grizzly.

The crusher product will be discharged into an approximately 400 st dump pocket and then onto an apron feeder. The apron feeder discharge will be conveyed to the crushed mineralized material stockpile.

The stockpile will have a live capacity of 55,100 st. The mineralized material will then be reclaimed from this stockpile by apron feeders at a nominal rate of 2,717 st/h. The apron feeders will feed a 72 in (1,830 mm) wide conveyor, which in turn will feed the cone crusher screen. The conveyor belt will be equipped with a belt scale.

17.4.2 SECONDARY CRUSHING

The secondary circuit will consist of three cone crushers (each with an installed power of 1,000 hp (750 kW), two in operation and one on standby) and will have a crushing circuit capacity of 2,717 st/h. The cone crushers will operate in closed-circuit with sizing screens. Reclaimed material from the coarse mineralized material stockpile will be conveyed to the cone crusher screen feed chute, which will dry feed two vibrating double-deck screens. Screen oversize materials will be directed to the cone crusher feed surge bins. The cone crusher product will return to the screen feed conveyor where it will combine with freshly reclaimed material prior to feeding the vibratory screens. The fine product screened will be delivered to the HPGR feed surge bin by a conveyor.

The facilities for the crushed mineralized material will be equipped with dust collection systems to control fugitive dust that may be generated during crushing, conveyor loading, and the transportation of the mineralized material.

17.4.3 HPGR Feed and Surge Bin

The HPGR feed surge bin will have a live capacity of 1,320 st. The feed material will be reclaimed from this bin by two belt feeders at a nominal rate of 2,717 st/h (each at 1,358 st/h). Each of the belt feeders will feed each of the HPGR feed chutes and then to the HPGR units.

17.4.4 Tertiary Crushing

Tertiary crushing will be completed using two HPGR units to further crush the material to a product size of 80% passing approximately 3.0 to 3.5 mm (average) prior to entering the grinding circuit.

The major equipment and facilities in this area include:

- belt feeders
- two HPGR crushers: each 94 in x 67 in (2,400 x 1,700 mm) with two 3,755 hp (2,800 kW) motors
- HPGR feed surge bin
- two double-deck vibratory screens: 12 ft wide x 28 ft long (3.7 m wide x 8.5 m long), 3/5 in and 1/4 in (15 and 6 mm) apertures (wet screening).

There will be two HPGR crushers, each fed independently via a belt feeder from the HPGR feed surge bin. The HPGR circuit will operate in closed circuit with a vibrating double deck screen system. The HPGR product will be conveyed to the HPGR screens. The screen oversize will be returned to the HPGR feed surge bin by conveying. The screen undersize will leave the crushing circuit as independent streams via a pipeline launder and report to the grinding circuit at a process flowrate of 2,717 st/h, or 1,358 st/h per one HPGR line.

The tertiary crushing facility will be equipped with a dust collection system to control fugitive dust that will be generated during crushing, conveyor loading and the transportation of the crushed materials.

17.4.5 Grinding and Classification

The primary grinding circuit will consist of two ball mills in closed circuit with classifying hydro-cyclones. The grinding will be conducted as a wet process at a nominal rate of 2,717 st/h of material. The grinding circuit will include:

- two ball mills – 8.2 m diameter x 13.2 m long each (26.9 ft x 43.3 ft); each with an installed power of 24,800 hp (18,500 kW)

- two ball mill discharge pump boxes
- two sets of hydro-cyclone feed slurry pumps
- two hydro-cyclone clusters
- one particle size analyzer
- samplers.

Each ball mill will operate independently in closed-circuit, with a hydro-cyclone cluster. The product from each ball mill will be discharged into its separate hydro-cyclone feed pump box, receiving with a portion of the HPGR screen discharge to become the hydro-cyclone feed. The slurry in each mill discharge pump box will be pumped to a hydro-cyclone cluster for classification. The cut size for the hydro-cyclones will be 80% passing 110 µm, and the circulating load to the individual ball mill circuits will be 250%, with the hydro-cyclone underflow returning to the ball mill as feed material.

The new feed to each ball mill circuit will be 1,358 st/h and the combined total of the two mills (2,717 st/h) will constitute the feed rate to the copper flotation circuit. The ball mills will operate at a speed of approximately 78% of the critical speed. Dilution water will be added to the grinding circuit as required.

Using a ball charging kibble, steel balls will be periodically added to the mills as grinding media in order to maintain the grinding efficiency.

17.4.6 Flotation Circuit

The ground material will be subjected to flotation to recover the targeted minerals into a high-grade copper concentrate containing silver.

Copper Flotation Circuit

The copper flotation circuit will include the following equipment:

- flotation reagent addition facilities
- rougher/scavenger flotation tank cells – 6 x 10,600 cu ft (300 m^3)
- regrind tower mill one classification hydro-cyclone cluster
- first cleaner flotation tank cells – 5 x 1,800 cu ft (50 m^3)
- first cleaner scavenger flotation tank cell – 1 x 1,800 cu ft (50 m^3)
- second cleaner flotation tank cells – 2 x 1,800 cu ft (50 m^3)
- third cleaner flotation tank cell – 1 x 1,800 cu ft (50 m^3)
- pump boxes
- slurry pumps
- one on-stream analyzer
- sampling system.

The hydro-cyclone overflows from the primary grinding circuit will be combined to feed the flotation circuit by gravity flow from the hydro-cyclone clusters. The rougher and scavenger flotation circuit will have a design feed rate of 2,717 st/h. The first cleaner scavenger tailings will report to the flotation rougher cells for reprocessing. The flotation reagents added will be the collectors, PAX and A3477, and the frother, MIBC. Lime will be used as a pH modifier throughout the process as required. Provision will be made for the staged addition of the reagents in the cleaner stage of the flotation circuit.

Rougher and scavenger concentrates will be sent to the regrind mill circuit hydro-cyclone feed pump box from where it will be pumped to the regrind classification hydro-cyclone. The scavenger tailings will be sampled automatically prior to being discharged into the final tailings pump box. This stream will constitute the final tailings reporting the TMF.

To liberate the fine-sized grains of the copper minerals from the gangue constituents, and to enhance upgrading of the copper concentrate, regrinding and cleaning will be incorporated in the cleaner flotation circuit.

The rougher regrind circuit hydro-cyclone will separate the finely ground flotation concentrate into a hydro-cyclone overflow product according to the design particle size of 80% passing 50 µm. The hydro-cyclone underflow will be the feed for the regrind mill. The mill will discharge into the hydro-cyclone feed pump box where will receive the rougher and scavenger flotation concentrates and the first cleaner scavenger concentrate.

The hydro-cyclone overflow from the regrind circuit will combine with the second cleaner tailings as feed to the first cleaner stage. The first cleaner concentrate will report to the second cleaner flotation stage, while the second cleaner concentrate will report to the third cleaner flotation stage. The concentrate from the third cleaner flotation stage will be the final copper concentrate which will feed directly to a copper concentrate thickener. The tailings from the third cleaner stage will be returned to the feed of the second cleaner stage. Tailings from the second cleaner flotation stage will be recycled back to the first cleaner flotation stage. Concentrate from the first cleaner scavenger flotation stage will report to the regrinding circuit while the tailings will report to the rougher/scavenger flotation for retreatment. Operationally, there will be the option of directing the first cleaner scavenger tailings to the final tailings.

17.4.7 Concentrate Handling

The final cleaner flotation concentrate will be thickened, filtered, and stored prior to shipment to the smelter. The concentrate handling circuit will have the following equipment:

- concentrate thickener
- concentrate stock tank
- concentrate filter press
- related slurry pumps and concentrate thickener overflow standpipe
- concentrate storage and dispatch facility.

The concentrate produced will be pumped from the final cleaner flotation stage to the concentrate thickener. Flocculant will be added to the thickener feed to aid the settling process. The thickened concentrate will be pumped to the concentrate stock tank. The underflow density will be approximately 60% solids. The concentrate filter will be a filter press unit. The filter press will reduce the concentrate water content to approximately 8%. The filtrate will be returned to the concentrate thickener. The thickener overflow will be re-used in the process plant as make-up water.

The cakes from the filter will be discharged to the concentrate stockpile. The dewatered concentrate will be stored in a designated storage facility prior to being periodically dispatched off the property to the smelter via trucks.

17.4.8 TAILINGS HANDLING

The flotation tailings from the flotation circuit will be the final plant tailings. The final tailings will be sent to the tailings pond, located south of the processing plant, for final deposition.

There is an opportunity to install a tailings thickening circuit to reduce potential water losses at TMF due to evaporation and the cost of pumping reclaim water from TMF to the processing plant. It is suggested that a detail overall site water balance and trade-off study to be conducted during the next phase of the study to determine the economic viability of tailings thickening.

The tailings thickening circuit is presented as provisional in the simplified process flowsheet (Figure 17.1).

17.4.9 REAGENT HANDLING AND STORAGE

Various chemical reagents will be added to the process slurry streams to facilitate the copper flotation process. The preparation of the various reagents will require:

- bulk reagent handling systems
- mixing and holding tanks
- metering pumps
- a flocculant preparation facility
- a lime slaking and distribution facility
- eye-wash and safety showers and other applicable safety equipment.

The chemical reagents, including collectors, frother, and pH regulator, will be added to the grinding and flotation circuits to modify the mineral particle surfaces and enhance the floatability of the valuable mineral particles into the copper-silver concentrate product. Fresh water will be used for making up the reagents that will be supplied in powder/solids form. These prepared reagent solutions will be added to the addition points in the grinding and flotation circuits using metering pumps. The PAX collector reagent will be made up to a solution of 15% strength in a mixing tank, and then transferred to the holding tank, from where the solution will be pumped to the addition points. Collector, A3744, and frother, MIBC, will not be diluted and will be pumped directly from the bulk containers to the points of addition using metering pumps.

Flocculant will be prepared in the standard manner as a dilute solution with 0.20% solution strength. This will be further diluted prior to being added to the thickener feed well.

Lime, as quick-lime, will be delivered in bulk and will be off-loaded pneumatically into a silo. The lime will then be prepared in a lime slaking system as 20% concentration slurry. This lime slurry will be pumped to the points of addition using a closed loop system. The valves will be controlled by pH monitors that will control the amount of lime added.

To ensure spill containment, the reagent preparation and storage facility will be located within a containment area designed to accommodate 110% of the content of the largest tank. In addition, each reagent will be prepared in its own bunded area in order to limit spillage and facilitate its return to its respective mixing tank. The storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, and Material Safety Data Sheet (MSDS) stations will be provided at the facility.

17.4.10 Assay and Metallurgical Laboratory

The assay laboratory will be equipped with the necessary analytical instruments to provide all routine assays for the mine, the processing plant, and the environment departments. The most important instruments to be installed in the assay laboratory includes:

- atomic absorption spectrophotometer (AAS)
- inductively coupled plasma atomic emission spectrometry (ICP-AES)
- x-ray fluorescence spectrometer (XRF)
- LECO furnace.

The metallurgical laboratory will undertake all necessary test work to monitor metallurgical performance and, more importantly, to improve process operations and efficiencies. The laboratory will be equipped with laboratory crushers, ball and stirred mills, particle size analysis sieves, flotation cells, filtering devices, balances, and pH meters.

17.4.11 Water Supply

Two separate water supply systems for fresh water and process water will be provided to support the operation.

Fresh Water Supply System

Fresh and potable water will be supplied to a fresh/fire water storage tank from the pits and from wells. Fresh water will be used primarily for the following:

- fire water for emergency use
- cooling water for mill motors and mill lubrication systems
- gland service for the slurry pumps
- reagent make-up
- potable water supply.

The fresh/fire water tank will be equipped with a standpipe which will ensure that the tank is always holding sufficient fresh water, equivalent to a 2 hour supply of fire water.

The potable water from a fresh water well will be treated and stored in the potable water storage tank prior to delivery to various service points.

PROCESS WATER SUPPLY SYSTEM

Some process water generated from the concentrate thickener will be re-used in the grinding circuit. Reclaimed water will be pumped from the tailings pond to the process water tank for distribution to the points of usage.

17.4.12 AIR SUPPLY

Separate air service systems will supply air to the following areas:

- Low-pressure air for flotation cells will be provided by air blowers.
- High-pressure air for the filter press, including drying of the concentrate, will be provided by dedicated air compressors.
- Instrument air will come from the plant air compressors and will be dried and stored in a dedicated air receiver.

17.4.13 ON-LINE SAMPLE ANALYSIS

The plant will rely on the on-stream analyzer for process control. An on-line analyzer will analyze and monitor the key streams in the grinding and flotation circuits. Specific samples that will also be taken for metallurgical accounting purposes will be the flotation feed to the circuit, the final tailings, and the final concentrate sample; these samples will be assayed in the assay laboratory. An on-stream particle size analyser will determine the particle sizes of the primary grinding circuit and regrind circuit products.

17.5 PLANT PROCESS CONTROL

17.5.1 PLANT CONTROL

The type of plant control system used will be a distributed control system (DCS) that will provide equipment interlocking, process monitoring and control functions, supervisory control and expert control systems. The DCS will generate production reports and provide for data and malfunction analysis as well as a log of all process upsets. All process alarms and events will be also logged by the DCS.

Operator interface to the DCS will be via programmable computer- (PC-) based operator workstations located in the following control rooms of the following areas:

- primary crusher
- process plant including cone crushers, HPGR units, ball mills, flotation circuit and concentrate handling
- tailings area.

The plant control rooms will be staffed by trained personnel 24 h/d.

Programmable logic controllers (PLCs) or other third-party control systems supplied as part of mechanical packages will interface with the plant control system via ethernet network interfaces.

Control strategies within the plant control system will be applied to control product particle size and to improve plant feed tonnage through the crushing, grinding, and flotation circuits. Slurry solid concentration control, dilution water flow rate control and level control will be carried out to reach the control targets.

Operator workstations will be capable of monitoring the entire plant site process operations, and will be capable of viewing alarms and controlling equipment within the plant. Supervisory workstations will be provided in the key management offices.

Field instruments will be microprocessor-based “smart” type devices. Instruments will be grouped by process area, and wired to each respective area local field instrument junction boxes. Signal trunk cables will connect the field instrument junction boxes to DCS inlet/outlet (I/O) cabinets.

Intelligent-type motor control centres (MCCs) will be located in electrical rooms throughout the plant. A serial interface to the DCS will facilitate the MCC’s remote operation and monitoring.

An automatic sampling system will collect samples from various product streams for on-line analysis and daily metallurgical balance.

An on-line dispersive XRF analyzer will be used to monitor the performance of the flotation process in various streams in order to optimize concentrate grade and metal recoveries.

A particle size-based computer control system will be used to maintain the optimum grind sizes for the primary grinding and concentrate regrinding circuits. Particle-size analyzers will provide the main inputs to the control system.

The control objective of the tailings facility will be to dispose of tailings and will include water storage, reclamation, and distribution back to the process plant.

Closed-circuit television (CCTV) cameras will be installed at various locations throughout the plant, including the stockpile conveyor discharge points, the stockpile reclaim area, the cone crusher and HPGR crushing areas, the ball mill grinding area, the flotation area, the concentrate handling building and the tailings pond area. The cameras will be monitored from the plant control rooms.

17.5.2 COMMUNICATIONS

Site wide communications will incorporate proven, reliable and state-of-the-art systems to ensure that personnel at the mine site have adequate voice, data and other communication channels available.

The communication systems will include a Voice-over Internet Protocol (VoIP) telephone system which utilizes the plant wide fibre optic network.

Hand-held mobile and base radios will be used by operating and maintenance personnel.

Internet connection and phone service external to the plant will be sourced via a local telephone company or internet service provider.

18.0 PROJECT INFRASTRUCTURE

18.1 INTRODUCTION

The Property is located in Plumas County, California, US, approximately 12 mi southeast of Westwood, California, and approximately 70 mi northwest of Reno, Nevada. The Property is approximately 1.5 mi from Diamond Mountain Road, a two-lane paved all-weather highway. The Property is accessible through a network of existing forestry service roads, designed for accommodating heavy equipment and vehicles used for logging activities.

18.2 ACCESS ROADS

The Project will be accessed by an existing network of logging roads, mostly from the nearest paved highway (Diamond Mountain Road) (Figure 18.1). A connecting network of roads that are required to access the various ancillary facilities including the laydown area, the open pits, the process plant, auxiliary buildings, the primary crusher, the TMF, and the mining operations staging points will be constructed.

Re-alignment of the existing road within the proposed TMF area will be required (Figure 18.2).

Figure 18.1 Diamond Mountain Road

Figure 18.2 Existing Access Road at Site

18.3 MAJOR BUILDINGS AND FACILITIES

The major buildings at the plant site will include the mill building, administration building, truck shop complex, assay laboratory, primary crushing (gyratory crusher) building, secondary crushing (cone crusher) and tertiary crushing (high-pressure grinding roll [HPGR]) building, concentrate storage and concentrate loadout facility, substation, warehouse, and cold storage.

Figure 18.3 to Figure 18.5 illustrate the overall Project site layout, the general arrangement of the plant site, and the plant and ancillary facility layout, respectively.

Figure 18.3 Overall Project Site Layout

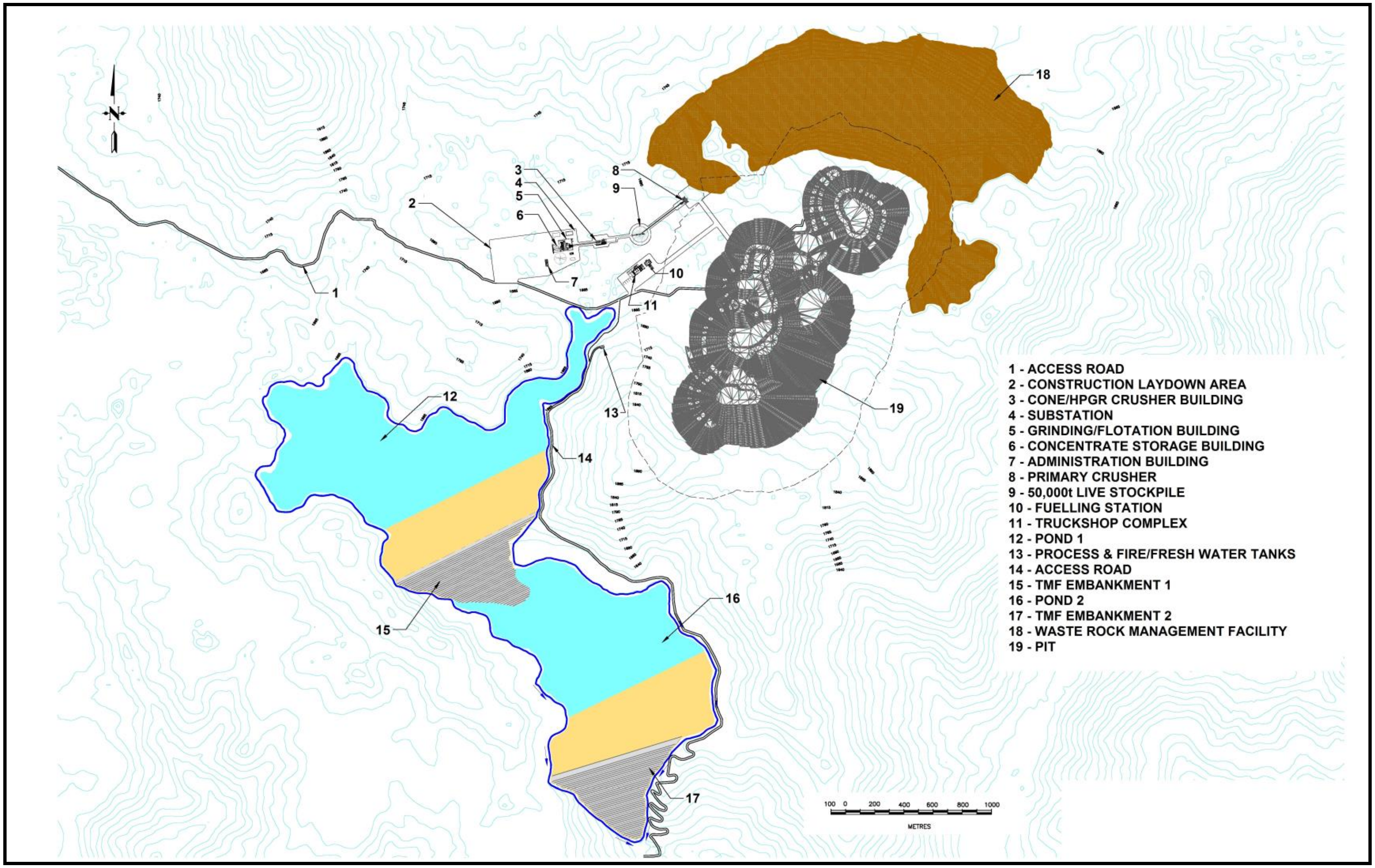


Figure 18.4 Plant Site General Arrangement

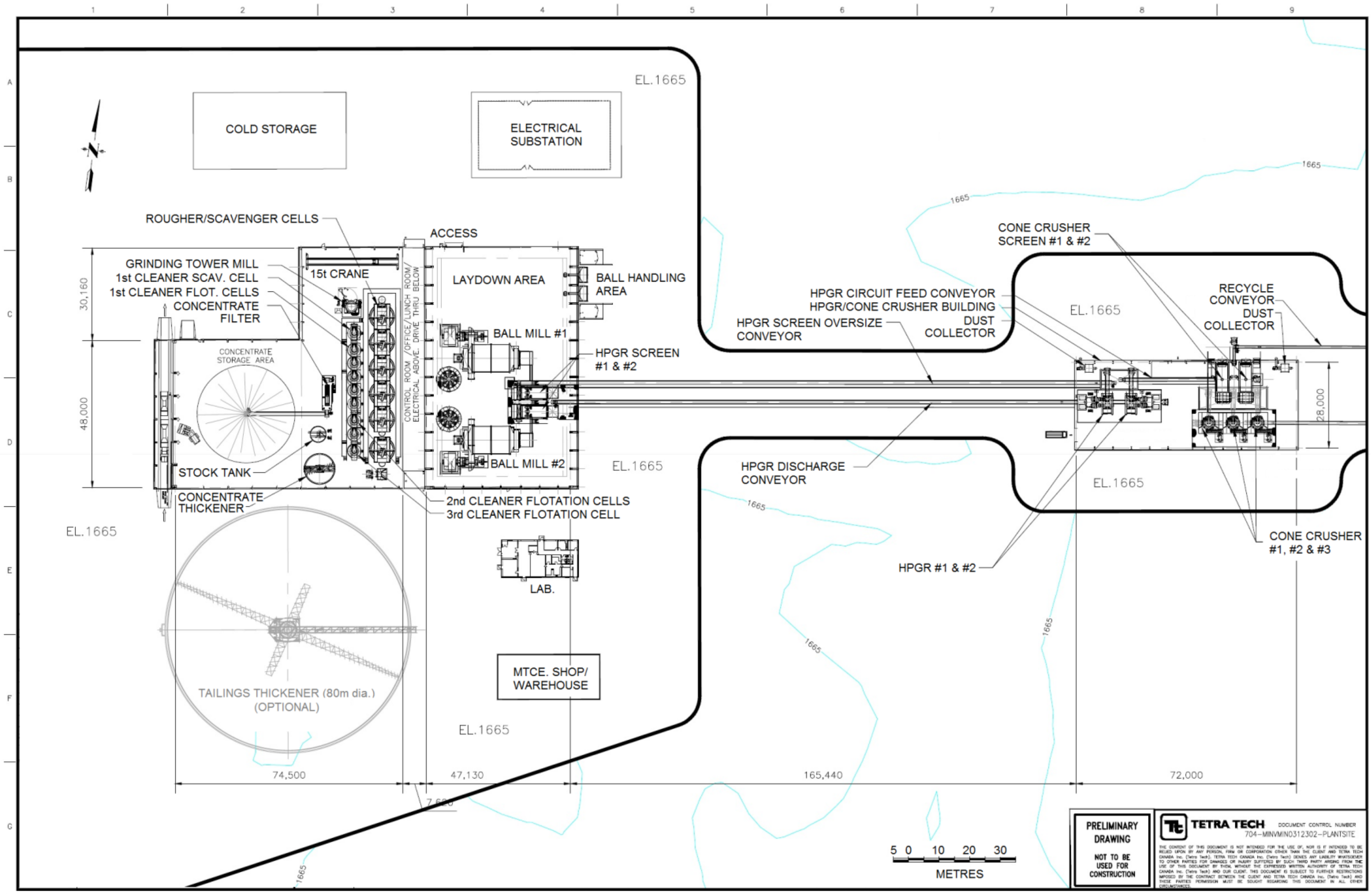
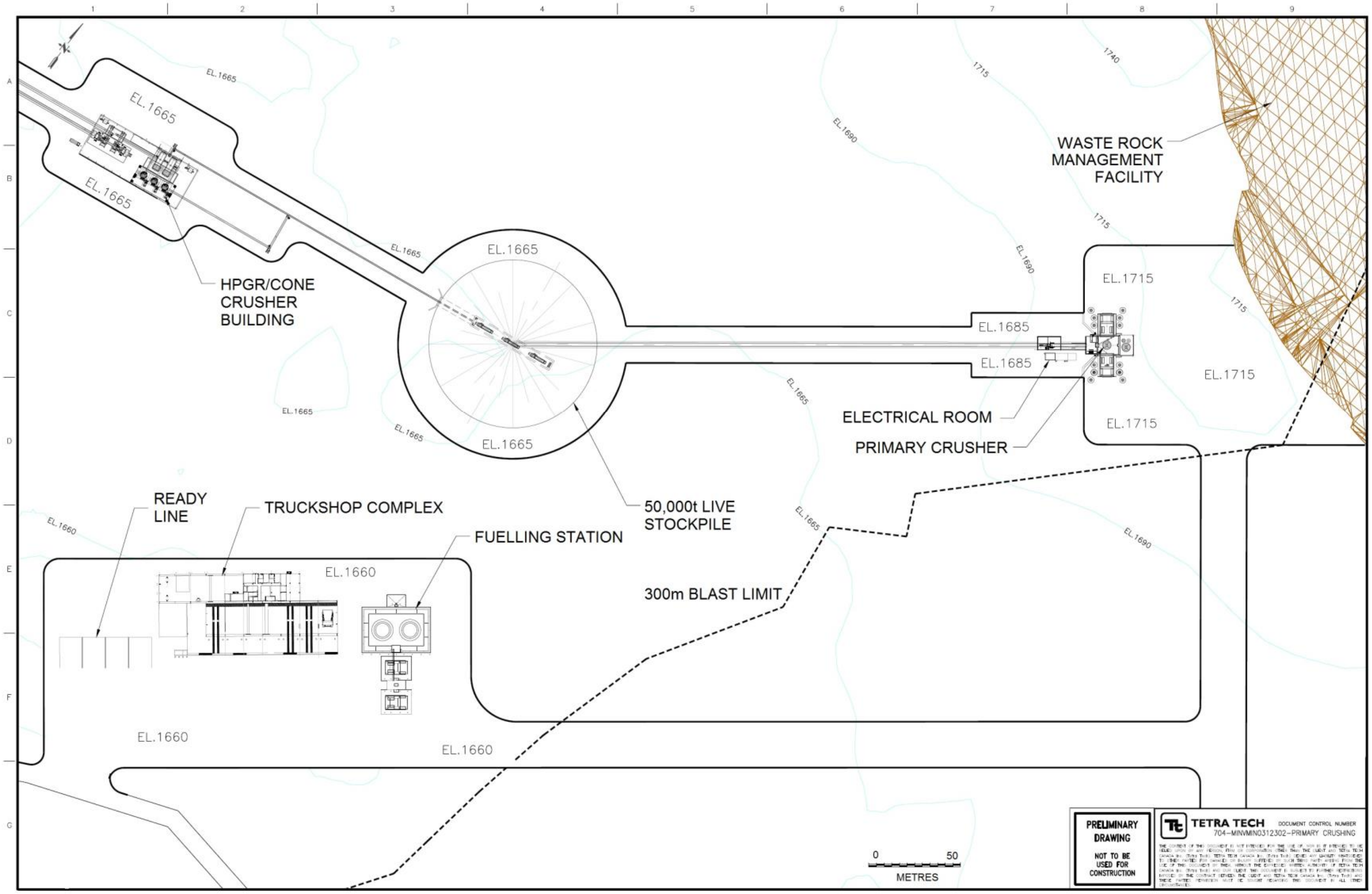


Figure 18.5 Plant and Ancillary Facility Layout



18.3.1 Process Plant

The mill building will be a pre-engineered steel structure with insulated steel roof and walls. The building will have elevated steel platforms throughout for ongoing operations and maintenance and will house an overhead crane coverage clear-span. The building foundation will consist of concrete spread footings, grade walls along the building perimeters and a slab-on-grade floor. The floor surfaces will have localized areas that are sloped toward sumps for clean-up operations. Operations and maintenance activities will be staged in the designated laydown area.

The building will house the grinding, rougher flotation and cleaner flotation, regrind area, reagents area, concentrate surge tank, concentrate filter press and laydown areas. There will be a mezzanine level above for the control room, offices, and electrical room.

The mill building will also include the concentrate stockpile and loadout facility with a full clear-span interior. The area will be partitioned with an “almost zero” air leakage envelope to contain or limit dispersing of concentrate dust. Modular, steel interior retaining walls will be provided for fleet load out vehicles, to operate and manage the concentrate and loadout facility.

An optical fibre backbone is included throughout the plant in order to provide a path for the data requirements for voice, data, and control system communications. A fibre backbone for a site ethernet-type system is included, which will provide data and voice bandwidth.

18.3.2 Primary Crushing (Gyratory Crusher) Building

The primary crushing building will be of stick-built concrete and steel construction, with multiple levels housing the ROM mineralized material feed hopper, gyratory crusher, apron feeders, conveyor support structures, ancillary equipment and utilities. The structure will be supported on concrete spread footings and concrete grade walls along its perimeters. Interior steel platforms will be provided to support equipment for ongoing operations and maintenance. There will be a control room and a rock breaker adjacent to the dump pocket. The facility will be equipped with a dust collection system to control fugitive dust that will be generated during crushing, feeding and conveyor loading of the mineralized material.

18.3.3 Cone Crusher Feed Stockpile

The cone crusher feed stockpile will be a production surge facility, which will allow for a controlled feed of crushed mineralized material to the cone-crusher feed circuit. The stockpile will have a live capacity of 55,000 st. The crushed materials from the gyratory crusher will be conveyed to the stockpile and then be reclaimed by apron feeders onto a cone crusher feed belt conveyor which transport the crushed materials to the cone crusher screen feed chute. The area will be equipped with feed weight scale and dust collection systems.

18.3.4 Secondary Crushing (Cone Crusher) and Tertiary Crushing (HPGR) Building

The secondary and tertiary crushing buildings will be pre-engineered structures with insulated steel roof deck and insulated wall cladding panels. A 120 st and 25 st overhead crane will be included and supported off the main building columns. Interior steel platforms on multiple levels will be provided for ongoing operation and maintenance needs. Several means of egress and staircases are also provided.

Equipment will be supported on independent steel platforms, complete with steel grating and handrails. The cone crushers and HPGR will be supported on heavy concrete mat foundation with reinforced concrete piers.

The secondary and tertiary crushing buildings will be supported on isolated spread footings complete with perimeter grade beams. Building slab-on-grade will be included. Interior footings to support various structures and equipment will also be included.

18.3.5 Conveying

Conveyors will be vendor supplied, including all structural support frames, trusses, bents, and take-up structures. Elevated conveyors will be supported with vendor supplied steel trusses and bents on concrete foundations.

18.3.6 Administration Building

The administration building will be a single-storey steel structure with insulated steel roof and walls located in close proximity to the process area. The building will be supported on concrete spread footings with concrete grade walls along its perimeter. This facility will house mine dry, lockers, shower facilities, first aid, with emergency vehicle parking and office areas for the administrative, engineering, and geology staff.

18.3.7 Truck Shop Complex

The facility will be a pre-engineered steel structure with insulated roof and walls, and limited interior support steel structures. The building will be supported on concrete spread footings and concrete grade walls along its perimeters. Sumps and trenches will be constructed to collect wastewater in the maintenance bays. Floor hardener will be applied to concrete surfaces in high-traffic areas.

The facility will house a wash bay complete with repair bays, parts storage area, welding area, machine shop, electrical room, mechanical room, compressor room, and lube storage room. It will also house the cold/warm storage warehouse and areas to support warehouse and maintenance personnel, with offices and mine dry.

The facility is designed to service and maintain both the mining haul fleet and the process plant fleet.

18.3.8 Fuel Storage

Diesel fuel requirements for the mining equipment, and process and ancillary facilities will be supplied from above-ground diesel fuel storage tanks located near the truck shop. The diesel fuel storage tank will have a capacity sufficient for approximately five days of operation. Diesel storage will consist of above-ground tanks and a containment pad, complete with loading and dispensing equipment conforming to regulations. A fuel dedicated service truck will transport diesel to the mining equipment and the process plant fleet.

18.3.9 Assay Laboratory

The assay laboratory will be a single-storey modular building. The building foundation will consist of concrete spread footings. The facility will house the assay and metallurgical laboratory equipment required for necessary grade assay and control and metallurgical testing, and will be equipped with all appropriate HVAC and chemical disposal equipment as needed. The facility floor will be reinforced as needed to accommodate specialized equipment.

18.3.10 HVAC and Fire Protection

All process areas will be heated to a minimum temperature of 5°C on a design winter day. This will be achieved by providing electric heating units along the perimeter walls and above all doorways. All process areas will be ventilated year-round to prevent a build-up of contaminants and humidity.

All occupied areas, such as offices, first aid, washrooms and change rooms, will be heated to a minimum temperature of 20°C on a design winter day. This will be achieved by supplying filtered and tempered outdoor air mixed with return air. The air will be distributed through ductwork into the individual rooms.

Air conditioning will be limited to control rooms, laboratories, and those electrical rooms where heat gains from electrical equipment are excessive. Electrical rooms where heat gains are not significant will be cooled using filtered outdoor air.

Small rooms, electrical rooms and remote buildings will be heated using electric heat.

Washrooms, change rooms and janitorial rooms will be mechanically exhausted to atmosphere. Make-up air will either be transferred from adjacent areas or supplied as filtered, tempered outdoor air.

18.3.11 Plumbing

All plumbing fixtures will be hard-piped by gravity to a sanitary drainage system.

All sinks and showers will be hard-piped with both potable hot and potable cold water.

Water will be heated in hot water storage tanks near the end users. Electrical heating will be used.

All fixtures connected to the sanitary system will be vented.

All cold-water piping will be insulated to prevent condensation, and all hot-water piping will be insulated for heat conservation.

Oil separators will be provided in truck shops and truck washes.

18.3.12 Fire Protection

A fire water tank will be built capable of sustaining two hours of firefighting at the design water flow rate. Firewater will be distributed around the site in valved loops, enabling water to flow in either direction.

Branches from the firewater distribution into each building will be provided with isolating valves.

The fire water system will be pressurized by a firewater pump package that consisting of a jockey pump, a main electric pump and a standby diesel-fired pump.

Yard hydrants will be positioned around the site such that all the buildings outside walls and all fuel tanks can be reached by a standard firehose and hose stream.

Sprinkler systems will be provided in lube rooms, air compressor rooms, blower rooms, truck shops, warehouse, laboratories, elevated mill offices, the mining equipment storage building and the administration building. Sprinklers will also be used to protect conveyors located in enclosed areas.

18.3.13 Dust Control

Dust control systems will be provided at the primary crushing, stockpile, the secondary and tertiary crushing and concentrate loadout areas.

The dust collection equipment will consist of dry baghouse and the collected fines will be returned to the process stream.

The dust will be pneumatically conveyed from the exhaust hood to the dust collector through steel ducting.

The dust ducting will include test ports, dampers and clean-outs.

18.4 Power Supply and Distribution

The Project is estimated to have a running load between 60 to 70 MW. Power is expected to be drawn from the existing network of transmission lines located in

Westwood, CA, approximately 10 mi northwest of the Project site. A new high-voltage power line will be constructed for bringing the power from Westwood, CA to site. The line is expected to be routed alongside the site access road for ease of construction and maintenance.

The on-site electrical substation will be located as close as possible to the grinding/mill loads as these are the largest loads. Utility voltage will be stepped down to 4,160 V at mill and mine for site-wide power distribution.

A single 2,000 hp, 600 V standby rated diesel generator set will be provided at the mill building to provide standby power to mill building and HPGR building critical power loads.

18.5 WATER MANAGEMENT

The key facilities for the water management plan are:

- open pit
- mill (including fresh and process water tanks)
- TMF
- diversion and water management structures
- fresh water supply
- sediment and erosion control measures for the facilities.

The water management strategy utilizes water within the Project area to the maximum practical extent. The plan involves collecting and managing site runoff from disturbed areas and maximizing the recycle of process water. Site run-off water will be stored on site within the TMF. The water supply sources for the Project are as follows:

- precipitation runoff from the mine site facilities
- water recycle from the tailings supernatant ponds
- groundwater wells for fresh water supply and potable water
- treated black and grey water, in small quantities, from the buildings.

A detailed site water balance assessment will be carried out to determine the water management strategy and process makeup water requirements during the next phase of the Project.

18.5.1 RECLAIM WATER SYSTEM

Reclaim water for use in the mill processes will be pumped from a floating barge to a reclaim head tank at the crest of the hill located south of the plant site. This head tank will store a 24-hour supply of mill process water, which will be gravity fed to the plant site. The water will be pumped to the head tank using a HDPE pipe. The barge will be positioned at the south end of the pond to minimize the pumping distance to the head tank.

18.5.2 Additional Water Management Facilities

Additional facilities have been identified for water management. The conceptual level design of these facilities has not yet been completed at this stage of development. However, an allowance for these items (including an allowance for cost) are included as they will need to be evaluated and incorporated into subsequent design studies.

18.6 Tailings Management Facility

The TMF are designed to accommodate over 370 million st of tailings, to be generated over the 17-year LOM. The design mill throughput rate is nominally 21.9 million st/a. The tailings are expected to be non-acid generating. Metal leachability of the tailings will be investigated during next phase of the project.

18.6.1 TMF Design Requirements And Concept

A cross-valley type TMF concept, with embankments constructed of cyclone sand, has been adopted based on the mine plan, the limited available construction materials, and an assessment of the site topography. Over the LOM, two dams (TMF1 and TMF2) will be constructed in the same valley to store tailings and process water. The proposed TMF dams will be located to the south and at a lower elevation than the proposed process plant location.

The design will permit storage of approximately 315 million cu yd of tailings at an average tailings dry density of 87 lb/cu ft.

A summary of TMF design requirements and characteristics is provided in Table 18.1.

Table 18.1 TMF Requirements and Characteristics

TMF Feature/Requirement	Unit	Value
Required Tailings Storage Capacity	million st	370
Required Tailings Storage Capacity	million st/a	21.9
TMF1 Embankment and Basin	million cu yd	175
TMF2 Embankment and Basin	million cu yd	165

18.6.2 TMF Design And Construction

The TMF design is shown in plan and section in Figure 18.6 and Figure 18.7, respectively. The nominal 65.6 ft (20 m) high zoned earthfill starter embankments will be constructed using compacted mine waste and select borrow. The embankment foundation will be prepared and a compacted key trench incorporated to mitigate seepage. The embankments will be raised in stages by centreline methods using cyclone sand tailings and select borrow material as required. The zoned embankment will include a low-permeability compacted clayey zone keyed into competent and low permeability foundation.

The adopted embankment design geometry is 2.5H:1V downstream to suit typical stability and closure requirements.

A nominal 3 ft thick granular blanket drain shall be installed below the downstream portion of the embankment to improve downstream drainage and maintain a low phreatic surface in the embankment. Seepage through the dams will be collected at the base of the dams and pumped back to the tailings pond.

Surface water diversion ditches will be constructed to divert surface water from the TMF.

The TMF footprint will be grubbed and topsoil stripped and stockpiled for future reclamation.

Figure 18.6 Tailings Management Facility - Plan View

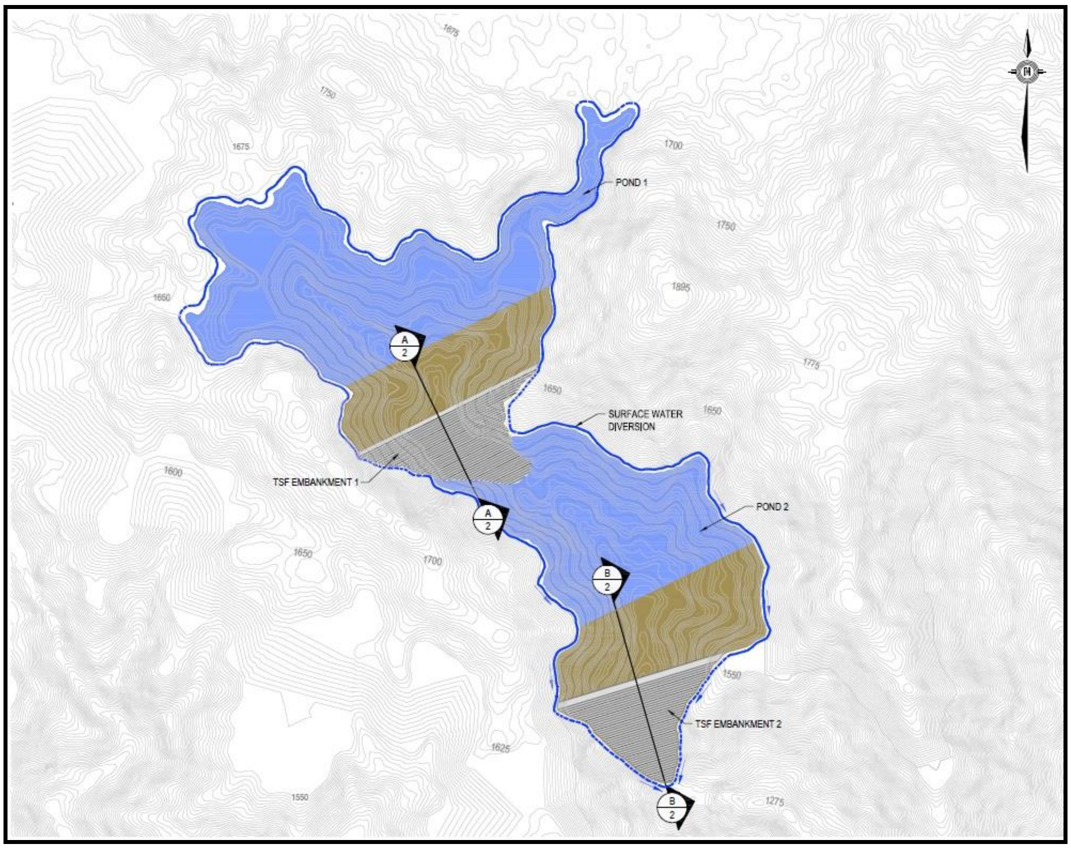
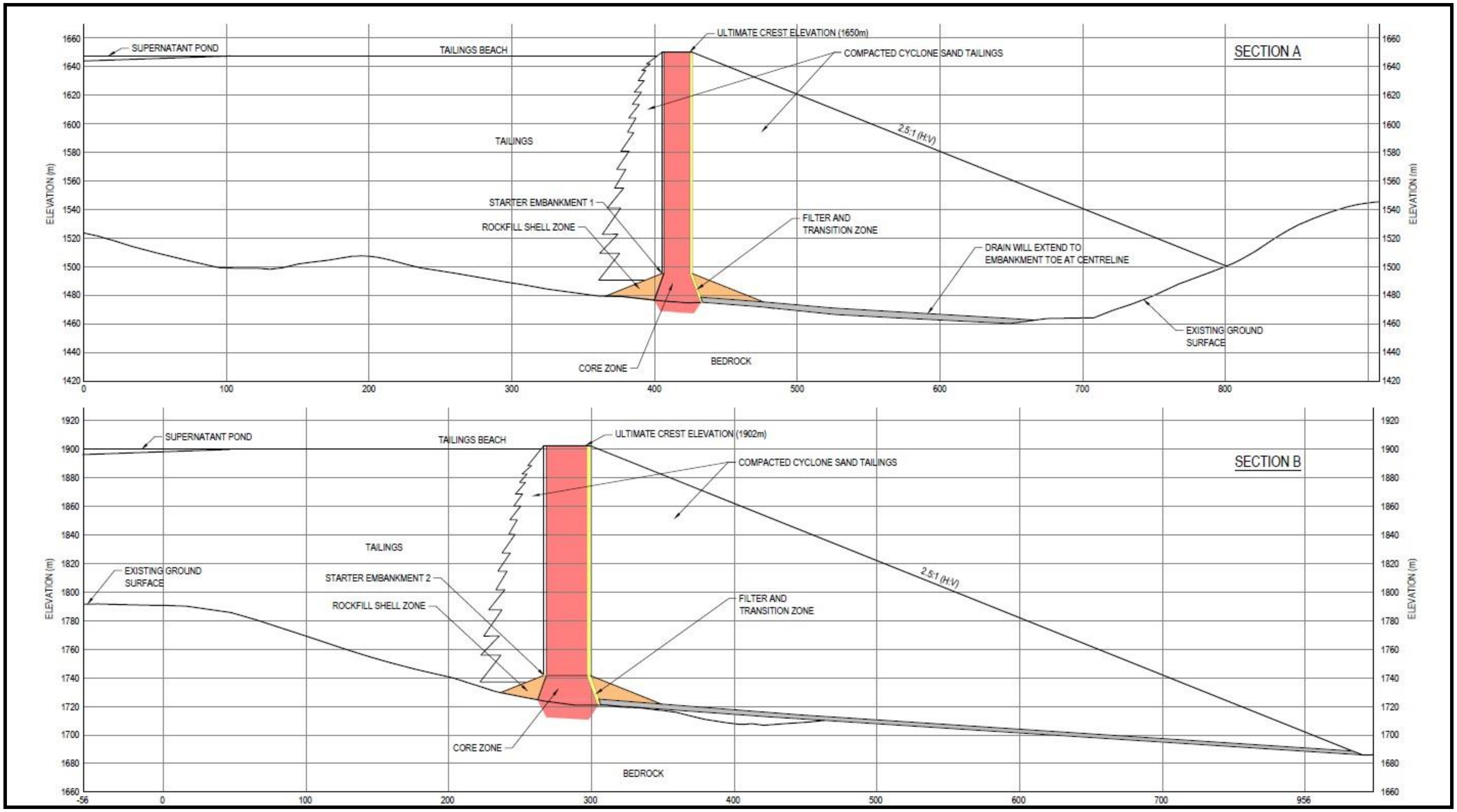


Figure 18.7 Tailings Management Facility – Sections A & B



18.6.3 TMF Operation

Coarse tailings from the cyclone underflow will be deposited in cells and compacted as part of embankment raising construction. Fine tailings from the cyclone overflow will be deposited into the basin upstream of the embankment. This approach will optimise tailings storage capacity while reducing the risks associated with embankment stability and seepage. Tailings deposition will be undertaken to maintain the decant water return pond adjacent to the water return intake. Decant water will be returned to the process plant for re-use.

The following practices are important in the operation of the TMF:

- The water pond size shall be kept to a minimum by optimising water return.
- Deposition should be cycled in such a manner as to concentrate and maintain the water pond around the water recovery point located in the valley area of the storage.
- The supernatant water should not be allowed to pond against the embankment.

18.6.4 TMF Monitoring

The TMF monitoring program will include the embankment stability, tailings storage management, and groundwater quality.

Embankment stability will be monitored by routine visual inspections and periodic measurements of slope inclinometers, survey stakes, and standpipe and/or vibrating wire piezometers.

Tailings management will be monitored by routine visual inspection by operations and management staff as well as annual audits by geotechnical specialists.

Piezometers will be installed to permit monitoring of groundwater flow and quality.

18.6.5 TMF Closure

The conceptual closure plan involves covering the surface and embankment slopes of the TMF with overburden and revegetating. The revegetation technique that is adopted will be based on site specific trials and experience.

A spillway will be required to facilitate controlled release of surface runoff from design storm events.

19.0 MARKET STUDIES AND CONTRACTS

There were no market studies conducted or contracts negotiated for this PEA.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 LOCATION, ENVIRONMENTAL AND PHYSICAL SETTING

The Moonlight-Superior Project is located in Plumas County Northern California approximately 100 mi northwest of Reno, Nevada. The Project is approximately 3,205 ac in size consisting of a mix of patented mining claims, unpatented mining claims and fee lands in the Plumas National Forest (Figure 20.1).

Figure 20.1 Project Location

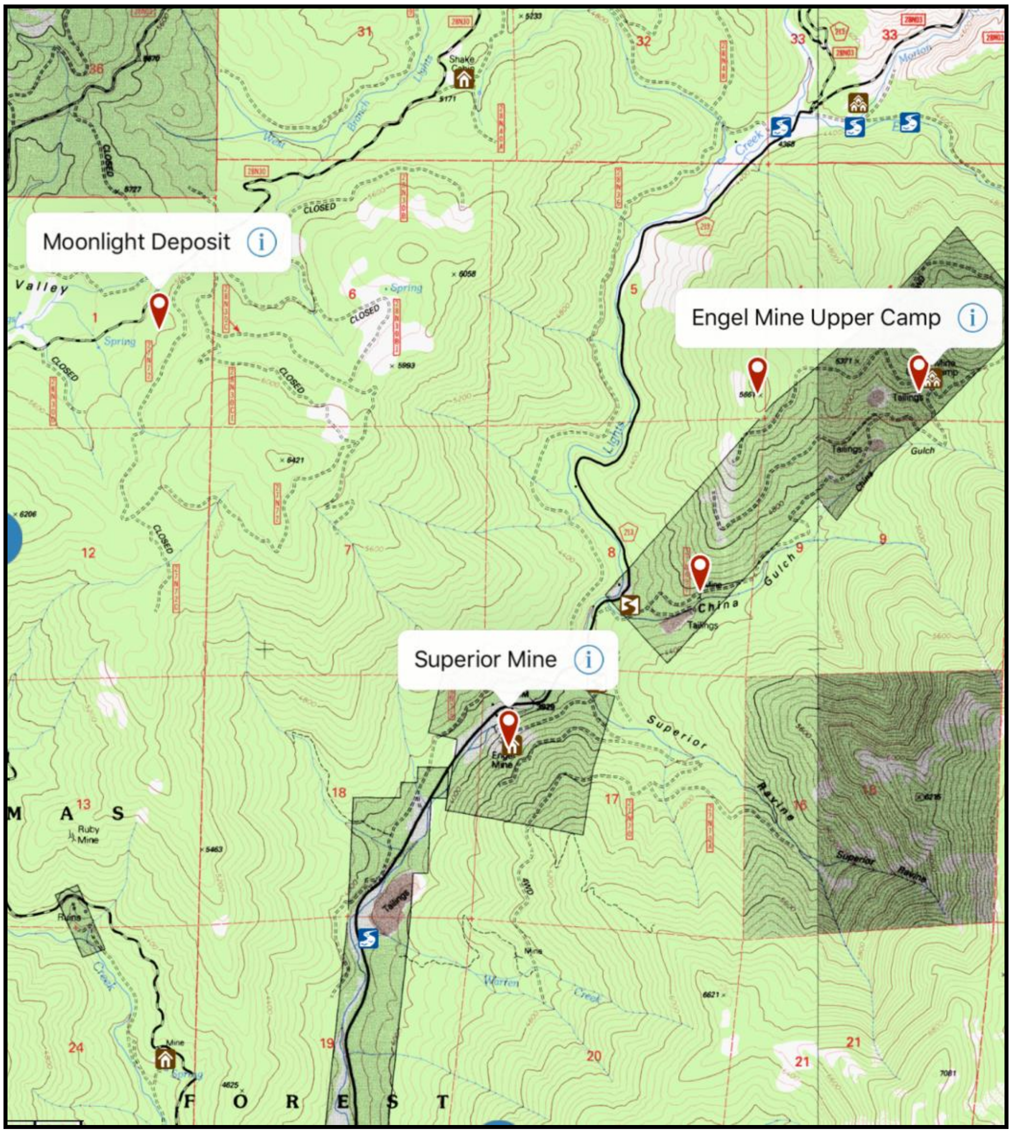
The site is in mountainous terrain with variable forest cover which has been subject to a recent forest fire. Elevations range from 3,600 ft above mean sea leve in the lower Lights Creek valley to over 5,600 ft above mean sea level at the Engels Mine with peaks in the 7,500 ft above mean sea level range. Lights Creek is a relatively small continuously flowing stream with variable riparian habitat which generally increases downstream towards the Indian Valley. Lights Creek flows southeast through the Indian Creek Valley before joining the Feather River. There are no gauging stations on Lights Creek so stream flow information is unavailable.

Access to the site is on a county road which passes through ranch lands and scattered residences along the lower portion of the Lights Creek valley. The towns of Greenville and Taylorsville are a few miles from the site and Quincy, the County Seat of Plumas and the nearest major community, is 20 mi to the south.

The Greenville Rancheria is located in Indian Valley just east of Greenville. The Rancheria is a Northern Maidu Indian Reservation headquartered in Red Bluff, California. Identification of cultural uses or claims to any area within the Moonlight-Superior Project was not performed for this assessment.

Three areas of potential mining interest are included in the Project: Superior, Engels, and Moonlight (Figure 20.2). Both Superior and Engels have been mined in the past and are located on patents and fee lands. Both contain existing ground disturbances including wastes and mine openings. Moonlight is largely un-impacted and completely located on unpatented claims in the National Forest.

Figure 20.2 Moonlight-Superior Project



The Superior Mine, the largest of the existing mines, is located on a steep valley wall above Lights Creek and has limited ground for development outside the valley bottom. Any mine planning will have to consider diverting Lights Creek around mine facilities and finding suitable locations for waste storage facilities. Existing impacts include mine water discharge, mine openings and structures, waste rock piles and a tailings impoundment of approximately 20 ac in size. A second tailings impoundment is located at the Lights

Creek valley mouth and is approximately 100 ac in size. The Project does not include ownership of this area.

The Engles Mine is located on a ridge above a tributary to Lights Creek and possesses more options for mine development and waste management. Existing impacts include mine water discharge, waste rock and tailings impoundments of approximately 1 and 5 ac in size.

The Moonlight deposit is located adjacent to a topographic depression (the Moonlight Valley) and has ample room for facilities and waste management. Existing impacts are minimal consisting of a caved adit, a small flooded shaft and numerous prospect pits.

20.2 MINE PERMITTING

20.2.1 STATE OVERVIEW

Project permitting in California is led by the County planning authority in which the Project is located. This local primacy is unlike most other states in which state-level authorities perform the role of the lead agency. This becomes more complex when permits other than land use certifications are required so it is likely that state agencies which grant water and waste permits and federal agencies for other reasons described below would be involved in any permitting at Moonlight.

In Plumas County decisions related to land use development, including permits to mine, are managed by the County Planning Commission, composed of five members one each from five County districts. With regard to mining specifically, the County is granted lead agency status by the California Department of Conservation, State Mining & Geology Board under the Surface Mining and Reclamation Act of 1975 (SMARA). SMARA is the state law that regulates surface mining activities in the state and contains numerous provisions which include pit backfilling during mine closure and reclamation. An application for a permit to mine must include an application, a reclamation plan and a financial surety to cover the cost of reclamation. Larger projects, and open pit mining projects subject to SMARA require environmental review under the California Environmental Quality Act (CEQA).

CEQA requires that larger projects produce environmental reviews called Environmental Impact Reports (EIRs). The EIR is meant to provide descriptions and impacts of the various project components on the environment in order to determine if the project can proceed. While similar to environmental review under federal law, the EIR does have stipulated timeframes that the lead agency is obligated to meet. The county acts as the lead in the production and review of the EIR.

California is fully delegated by the Environmental Protection Agency (EPA) to administer federal environmental regulations and grant permits. As a result, most permits for waste and water discharge or air emissions will be issued by the appropriate state agencies.

20.2.2 Federal Overview

Besides California's CEQA, it is likely that Federal review under the National Environmental Protection Act (NEPA) will be required for this project since federally managed resources will likely be impacted. This review, an Environmental Impact Statement (EIS) is completed by the lead Federal agency. Given that the unpatented mining claims are located in the Plumas National Forest the Federal lead agency would be the USFS. Federal EISs do not have time limits as anticipated under California law and tend to be larger and more complex in scope. The EPA, which itself issues no permits, acts as final arbiter to the adequacy of environmental review and can have significant impact on the outcome of the project.

As with County land use laws the Federal land manager will require submission and approval of plans that detail the operation and closure of the proposed project. In addition bonding of surface impacts will also be required. These requirements are summarized below.

Since these rules only apply to lands under management by a Federal Agency, some recent projects in the US have sought to mitigate the requirements by engaging in land exchanges with the USFS. This process while cumbersome and difficult, often involving a separate EIS to deal with the issues relate to the exchange process, can successfully remove a project from ongoing agency management issues during operation and closure. In theory, privately held lands within the Forest boundary can be exchanged for lands anticipated to be needed for the project. Often a premium is placed on the amount of land offered for exchange based on commercial and ecological values.

The only permits per se issued by Federal agencies would involve any wetlands or waterway impacts under section 404 of the Clean Water Act which regulates filling of Waters of the US. The US Army Corps of Engineers (USACE) issues 404 permits.

20.2.3 Permits Required

The major permits, approvals, and environmental reviews for a proposed project at Moonlight are presented in Table 20.1.

Table 20.1 Major Permits and Approvals

Permit, Approval or Review	Agency Responsible
Plan of Operations	USFS Plumas National Forest
Environmental Impact Statement	USFS Plumas National Forest
Clean Water Act Section 404 Permit	US Corps of Engineers
Reclamation Plan and Bonding	Plumas County
Environmental Impact Report	Plumas County
Conditional Use Permit	Plumas County
Air Quality Permit	Northern Sierra Air Quality Management District
CWA Section 401 Certification	Central Valley Regional Water Quality Control Board
Storm Water Pollution Control Permit	Central Valley Regional Water Quality Control Board
Waste Discharge Order	Central Valley Regional Water Quality Control Board

Plan of Operations

Under 36 CFR 228, the US Code that regulates mining on National Forest Lands, proposed mining operations must submit a document describing the activities that exceed 5 ac in size envisioned to take place on federally managed lands. This is formally known as Plan of Operations. The plan must include descriptions of construction and operation and must include plans for closure and reclamation. Bonding must also be estimated to cover the anticipated costs of reclamation. Any activity is prohibited until approval of the Plan and approval is subject to completion of a successful environmental review.

Environmental Impact Statement

Before a Federal agency can issue permits or approvals for projects impacting nationally managed resources NEPA requires that a thorough assessment of both the impacts and mitigations be conducted. Issues assessed include ecological, human, and cultural resources that are under held in trust by the US Government. A series of alternatives to the project including a no-action alternative must be assessed with the tacit assumption that development provides economic benefits and, therefore, has value. The alternative selection seeks to minimize environmental and social impacts while providing a maximum benefit to the community. In order for project approval to be granted, one of the alternatives other than the no-action alternative, must be selected.

The Lead Federal Agency is responsible for producing the EIS; they select the contractor and lead the determinations. A project proponent is required to fund the study and can only provide information to the EIS contractor for consideration. The adequacy of the EIS in thoroughly assessing impacts is a primary challenge point for project opponents.

In the case where both State and Federal regulators require environmental assessments a combined EIS can be produced under a Memorandum of Agreement between the appropriate agencies. This will likely be the case at Moonlight. The combined EIR/EIS would not be required to meet the statutory requirements of just the EIR.

Clean Water Act Section 404 Permit

The USACE is delegated with managing navigable waters within the US. Discharge of material such as tailings or waste rock into Waters of the US must be permitted under Section 404 of the Clean Water Act. In practice, any flowing water in the country is determined to be Water of the US, so tailings impoundments or other waste disposal activities that impact streams or wetlands at the headwaters of streams require a 404 Permit. Permits can be issued under a system of established Nationwide Permits or under a separate site permit. If impacts to these water are deemed significant, the USACE may decide to become co-lead with the land manager on the EIS.

Reclamation Plan and Bonding

California promulgated its SMARA to regulate surface mining activity reclamation in the State. It is unique in that for new mine open pits it requires backfilling of with all available waste rock material to original contour. Bonding is required on an on-going basis. To provide for this requirement, a Reclamation Plan is required before mining can be approved. The reclamation plan needs to include all aspects of the mine including remaining waste rock piles, tailings impoundments, processing facilities, roads and utilities. Where there are other reclamation requirements, such as under a USFS Plan Of Operation, the State only requires that its bond cover the reclamation costs not included in the Federal plan.

Environmental Impact Report

In California, permitting is coordinated with the CEQA process and no permits can be granted until a successful review under CEQA. Under State law, project review must comply with a set of established time lines under the California Streamline Permitting Act (Gov't Code Sec. 65920-65963.1). Once the lead agency is established, all other permitting agencies become responsible or trustee agencies, whether State or Federal. Responsible agencies do not usually prepare their own documents but rely on the lead agency. Each of these responsible agencies will comment on the adequacy of the EIR and propose mitigations. The lead agency will be responsible for consultation with Native Americans as well. In those cases where both an EIR and an EIS under NEPA are required, the lead agency may choose to utilize the EIS in lieu of completing a separate EIR or may join in the EIS process under a memorandum of understanding (MOU).

CEQA anticipates three phases: 1. Pre-application, 2. Application and 3. Review. The Pre-application phase begins with the project proponent supplying a detailed explanation of the project. This project scope, should be detailed enough to allow the lead and responsible agencies to determine the scale and potential impacts of the project. The applicant is supplied with detailed lists of the required permits, timelines and necessary documentation to complete a review. Multiple meetings are usually the case to establish the project parameters.

The Application phase begins with the proponent completing the applications for the required permits and request for land use determination and submitting them to the responsible agencies. Within 30-days the lead agency is required to determine if the applications are complete after consultation with the responsible agencies. The completeness review gives the agencies a chance to internally determine their satisfaction with the amount of documentation provide by the proponent. A determination of incompleteness stops all clocks. If after a second determination of incompleteness (after 30-days) an appeal process is initiated giving an answer to the proponent within 60-days.

The Review phase starts upon the lead agency's determination of completeness and starts a 30-day clock for it to prepare the Initial Study which establishes whether the project will have a significant impact on the environment. If so determined a Notice of Preparation is prepared and submitted to the responsible agencies and the public for review. This begins the preparation of the EIR. Which has a one-year timeline. The Draft EIR, upon its completion is subjected to public review and comment. Note that in the case of Federal agency involvement, the timeline may be waived. Permits can be issued at the successful close of this process.

Conditional Use Permit

Under its land use obligation, Plumas County must issue a permit to establish that land under its jurisdiction is in compliance with its zoning requirements. For mining projects, it issues such a land use permit with conditions on operating that comply with other State regulations such as SMARA. The plan has similar requirements as the USFS Plan of Operations and may provide the basis for the permit application. Transportation requirements with mitigations and revegetation are to be included in the application. The County Planning Board issues this permit subject to approval by the Board Supervisors. As with all other permits, the Conditional Use Permit cannot be issued without a successful environmental review under CEQA.

Air Quality Permit

Under the Clean Air Act and its California implementation, a permit to emit pollutants into the atmosphere is required before installation of any air emitting equipment can be completed. Regulated pollutants include the six criteria pollutants (carbon monoxide, lead, nitrogen dioxide, ozone, particles, and sulfur dioxide) and air toxics. Modelling of the emissions is required to demonstrate to the State that they will not degrade the airsheds with respect to the National Ambient Air Quality Standards (NAAQS).

Additionally, National Parks and Wilderness are statutorily protected from impacts to their air quality and visibility by the Clean Air Act as being designated Class I Airsheds. Mt. Lassen National Park lies approximately 30 mi northwest of the property. The National Park is bounded on the east by the Caribou Wilderness Area and has the Thousand Lakes Wilderness to its north approximately 56 mi from the project. Regional haze and visibility issues will be the concern.

Class 1 Airshed designation requires additional air quality monitoring and modelling to demonstrate that impacts will not affect air or view qualities of the Park or wilderness areas. This review is known as Prevention of Significant Deterioration (PSD). If deemed to have the potential to impact the airshed, additional mitigation may allow the project to move forward. Project proponents are required to contact the Federal Land Managers of

Class I Airsheds within 60 mi (100 km) of the proposed emission source. This will have the effect of bringing both the National Park Service and the Lassen National Forest into the discussion. Air modelling would be required for Environmental Review and would require at least one year of weather and air quality monitoring.

Section 401 Certification

Discharge of process water will require a permit under California's Water Code. Before it can be issued, the proposed discharge must be certified to meet the requirements of the US Clean Water Act at the point of compliance. Process water means any water that is used in processing or comes in contact with process materials. It is likely that some level of water treatment will be required before water is discharged and description of this processing and modelling of its results will be an important part of the EIS and supporting documentation to the project.

This certification is conducted by the local Regional Water Quality Control Board (RWQCB). The EPA in its oversight role will review and comment on adequacy of the certification process.

Storm Water General Permit

Non-process water discharge resulting from run-off during precipitation events from industrial operation to Waters of the US require permitting under the Storm Water provision of the Clean Water Act. Operators must demonstrate that these discharges will not adversely impact waterways. Storm Water Pollution Prevention Plans provide the State with the ability to include a facility under its Storm Water General Permit rather than requiring a separate site permit.

Waste Discharge Order

Process water discharges from industrial facilities requires permitting under the California Water Code. This will include discharges from tailings basins as well as any other direct discharges from the facility such as drainage from waste rock piles or mill effluents. Discharges will be required to meet applicable water quality standards either naturally or through treatment. The Waste Discharge Order will establish the water quality objectives and a point of compliance for these to be met. As discussed above, certification of the proposed discharge will be required before the order is issued. The Central Valley RWQCB will conduct the certification and issue the Order.

20.3 Water Quality Summary

The existing water quality of the Moonlight Project has been assessed in two rounds of sampling, one conducted by earlier owners from mid 2006 to late 2008 and the second sampling conducted as part of a Master of Science thesis from 2008 to 2009. The results of these studies show that despite existing mining impacts, the in-stream water quality of Lights Creek and Moonlight Creek are good. Discharges from mine adits are elevated with regard to copper, antimony, and arsenic, but these concentrations are lower than might be expected and are mitigated within relatively short distances downstream. Conventional analyses collected from 2000 to 2004 indicate pH in the circum-neutral to alkaline range and even at mine portals acid discharge has not been recorded. For the six sampling sites in this program, total dissolved solids (TDS) averages 54 ppm with highs from 100-150 recorded only at the site downstream from the Superior Mine Tailings. Temperature varies from slightly above freezing in winter to 79° F (26° C) in some summer sampling events indicating a small water flow stream that is directly affected by surface temperatures.

20.3.1 Water Quality Studies

The two water quality studies at Moonlight sampled at somewhat different sites in the project area. Common to both are sampling sites at the Engels #10 Level portal and the #2 Superior portal. The earlier study, which will be referred to as the Owners Study, utilized 9 sites, of which not all were sampled during each sampling event.

Table 20.2 Owners Study Sampling Sites

Site Sample ID	Alt Sample Site ID	Location
1	201	#10 Level Engels
2	202	Lights Creek Below Bridge
3	203	#2 Level Superior
4	204	Downstream of Superior
5	205	Lights Creek Upstream
6	206	Lights Creek Upstream
7	207	Moonlight Creek
8	208	Downstream of Superior Tails on Lights Creek
9	209	Confluence of Moonlight and Lights Creek

Sample sites 4 and 5 (204 & 205) can be considered background sites not directly affected by existing mining operation. Sample site 7 (207) is a measure of existing site conditions from the Moonlight Deposit.

Analyses were conducted on these samples by Sierra Foothill Laboratory of Jackson California using EPA approved methods for water quality employing ICP or graphite furnace atomic adsorption (GFAA) instrumentation. Reporting limits were generally acceptable with regard to regulatory standards.

Seventeen metals and metalloids were analysed in these reports. They are: barium, beryllium, cadmium, chromium, cobalt, copper, molybdenum, nickel, vanadium, zinc, antimony, arsenic, lead, mercury, selenium, silver and thallium. Of these, only three, copper, arsenic and antimony, proved to be of interest.

Analytical results were compared against a set of regulatory standards applicable to the jurisdiction. They are: the EPA Gold book aquatic and human health standards, the Safe Drinking Water Act MCLs, California Freshwater Aquatic Standards, and the California

Agricultural Water Quality Goals. In every case except copper and arsenic, the most stringent regulatory standard was employed. In the case of Copper, the aquatic Gold Book standard is calculated based on the Biotic Ligand Model (BLM). The additional analyses required for the BLM were not available to this report so the Alaska Freshwater Aquatic Standard based on a hardness of 100 was employed as a surrogate. For the purposes of this report this standard should be adequately protective. In the case of arsenic, the Human Health Goldbook standard is generally recognized as being of such a vanishingly low concentration (18 parts per trillion) that it is unworkable in real life conditions. Therefore, the Safe Drinking Water Act MCL was employed as the applicable standard for Arsenic.

Of the nine sites samples, only the two portal sampling sites (201 & 203) recorded analytes in excess of applicable regulatory standards. At 201 the Engles #10 portal site, copper and arsenic were constantly elevated above standards with copper averaging 153 µg/L and arsenic 20 µg/L. Of the remaining analytes, only zinc was consistently detected. At 203 the Superior #2 portal site copper, arsenic and antimony were consistently above standards, averaging 270 µg/L copper, 14.5 µg/L arsenic, and 20 µg/L antimony. Zinc was also consistently detected.

On the main stem of Lights Creek below the Engles but above the Superior, site 202, Copper was detected at 6 µg/L during one of two sampling events. No other analyte was detected.

At the project background sites 205 & 206, copper, vanadium and zinc were sporadically detected at low concentration. No other analytes were above the laboratory Reporting Limit.

A Master of Science Thesis by Kara E. Scheitlin and William M. Murphy published in 2009 at California State University at Chico and conducted under an agreement with Nevoro Inc. was summarized in a document titled: *Final monitoring Report, Moonlight Copper-Gold-Silver Project, Plumas County, California*, looked more closely at the geochemistry of surface waters in the Moonlight Project area. Eleven stations were sampled from October 2008 to May 2009 and analysed for a suite of 67 metals and metalloids using inductively coupled plasma-mass spectrometry (ICP-MS) to very low reporting limits. In addition, conventional parameters of pH, temperature, dissolved oxygen, and electroconductivity were collected periodically as well as major ions chlorine, alkalinity, sulphate, carbonate and bicarbonate. Some of the stations either directly matched earlier sampling or were in close proximity, others are new to geochemical sampling.

Table 20.3 Sheitlin Sample Locations

Site ID	Location Body of Water	UTM NAD 27 Coordinates (m)			
		Zone	Northing (m)	Easting (m)	
MN-WAT-01	Superior Mine #2 level adit entrance	10 T	4452549	689441	
MN-WAT-02	Blue Copper Mine adit entrance	10 T	4452453	688338	
MN-WAT-03	Lower Lights Creek downstream of tailings piles (upstream of bridge)	10 T	4451378	688607	
MN-WAT-04	Lights Creek at Moonlight Valley Road fish ladder Lights Creek	10 T	4449188	688182	
MN-WAT-05	Moonlight Creek Junction at Moonlight Valley Road Moonlight Creek	10 T	4453557	685175	
MN-WAT-06	Superior Ravine at Diamond Mountain Road Superior Ravine	10 T	4452765	689785	
MN-WAT-07	China Gulch at Diamond Mountain Road China Gulch	10 T	4453373	690131	
MN-WAT-08	Trout Bridge on Lights Creek 2.5 miles upstream of Superior Mine Lights Creek	10 T	4456485	691241	
MN-WAT-09	Engels Mine # 10 Level adit entrance	10 T	4453383	690703	
MN-WAT-10	Engels Mine drill pond Upper China Gulch	10 T	4454406	692255	
MN-WAT-11	50ft downstream of Engels Mine #10 level downstream from adit	10 T	4434041	671217	*

Analytical results were compared against the same set of standards as earlier results. No significant differences were observed in the data. Elevated concentrations of copper, arsenic and antimony were observed from the mine discharges. Baseline conditions in the streams showed similar results although with the lower detection limits used by the analytical procedures, most metals were detected in low concentrations.

Significant in the study was the identification that all the waters were calcium bicarbonate dominated except for the Superior mine drainage which was mixed calcium bicarbonate/calcium sulphate. These results support the other observations which suggest that acid generation from the dissolution of sulphide is not the major driver of metal leaching in this geological system.

20.4 ACID BASE ACCOUNTING

A limited number of rock and tailings samples have been subjected to ABA over the course of the project history. The results of seven samples collected and analysed by Sheffield Resources Ltd. In 2007 (Orequest Consultants Ltd, 2007) are presented below:

Table 20.4 Sheffield Resources ABA

Sample	Source	Cu%	MPA	NNP	NP	NP:MPA
			tCaCO3/t ore	tCaCO3/t ore	tCaCO3/t ore	
MNRW-35	SHM120 to 136	0.565	5.3	55	60	11.29
MNRW-36	SHM 137 to 143 SHM 58 to 60	1.02	12.5	36	48	3.84
MNRW-38	Saw Cuttings	0.56	6.6	45	52	7.9
MNRW-39	Saw Cuttings	0.4	7.5	50	57	7.6
MNRW-42	Engels Dump	0.39	6.3	28	34	5.44
MNRW-43	Engels Tailings	0.53	1.9	15	17	9.07
MNRW-44	Superior Tailings	0.22	1.6	25	27	17.28

Samples MNRW-35 and 36 are from drill core in the ore zone intercepted in drill hole 06MN-01 and represent a typical intercept in the Moonlight deposit. The saw cuttings samples MNRW-38 and 39 are from diamond saw residue of 300 ft of sawn drill intercepts in in the Moonlight deposit. Samples MNRW 42- through 43 are from weathered surface materials from the Engles mine and MNRW-44 are from tailings in the Superior tailings impoundment. All samples show neutralization potential (NP) above acid generation potential (MPA). The ratio of NP to MPA in the positive range suggests that acid generation is being consumed by the rock's natural ability to neutralize such acid. Significant is that even samples taken from material on the surface for 75 years still have excess neutralization capacity.

Two samples of tailings generated by Crown during flotation testing, Moonlight Sulfide and Superior Sulfide, were subjected to ABA under the supervision of Enviromin Inc. of Bozeman Montana (Enviromin 2017). The results are summarized below.

Table 20.5 Acid based accounting and Net Acid Generation Tests

	Analyte	Unit	Mn SUL	S SUL
Acid Base Accounting	FIZZ RATING	--	2	1
	AP	tCaCO3/1Kt	2.5	1.3
	NP	tCaCO3/1Kt	17	14
	pH	--	8.7	9.2
	NP:AP Ratio	--	6.8	11.2
	NNP (NPMAP)	tCaCO3/1Kt	15	13
	Total S	%	0.08	0.04
	Sulfate S (NaCO3 leach)	%	0.01	<0.01
	Sulfate S (HCl leach)	%	0.01	0.02
	Sulfide S	%	0.07	0.04
	Total C	%	0.51	0.1
	Inorganic C (CO2)	%	1.9	0.4
NAG test	NAG at pH 4.5	kg H2SO4/t	<0.01	<0.01
	NAG at pH 7.0 kg	H2SO4/t	<0.01	<0.01
	NAG pH	--	10.2	10.6

According to Enviromin both samples are classified as non-acid generating, with both exceeding the commonly employed threshold NP:AP of 3. Low sulfur and the presence of neutralizing carbon are responsible for these results. Enviromin does caution that the relatively low levels of carbon indicate low neutralizing capacity, however, tests of existing tailings as discussed earlier indicates that acid generation is not a problem in the tens of years range.

20.5 Social or Community Impact

According to their website (http://www.countyofplumas.com/index.aspx?NID=190), Plumas County had a population of 18,627 in 2016, the last year for which there are data but the county population has been in decline from a high of 20,800 in 2000. The median annual income in 2000 was $35,154, below the California median of $39,595.

Fifty-four percent of the population have received high school diplomas versus 42% for California in general indicating the presence of a strong local employment base for the project.

Historically mining, timber extraction, and ranching provided the economic drivers for the county as it grew. Today according the California State Economic Development Department (http://www.labormarketinfo.edd.ca.gov/majorer/countymajorer.asp?CountyCode=000063), Sierra Pacific Industries and Collins Pine Company, both in the timber industry, are the largest single non-governmental employers in the county. Some small-scale gold mining operations and quarrying remain active. However, recreation is growing in importance for the county's economic base especially in the Lake Almanor/Chester area as natural resource extraction industries dwindle.

The county seat Quincy, which is the largest city in the county, is the source for most mercantile activity in the county and hosts the County, State and Federal government offices. Portola, which lies east of Quincy is a major Union Pacific Railroad crew-change facility due to its proximity to Beckworth Pass, the lowest crossing of the Sierra Nevada Mountains. Rail access through the county on the Feather River Line provides a vital transportation corridor for the Union Pacific Railroad to the ports of Sacramento, California.

Two Indian Communities lie near to the project. In California, the name given to the tribal trust lands is Rancheria, a name derived from the Spanish for small village or habitation. As mentioned above the Greenville Rancheria is the closest with an enclave in the Indian Valley approximately 10 miles to the southwest. No population is given for the enclave, but based on a visual assessment, fewer than 100 households are present on it. It is interesting that the Headquarters for the Rancheria is in nearby Red Bluff, California. The Northern Maidu Tribe who belong to the Rancheria claim to be the native inhabitants of the region with historic range from the Feather to the Sacramento Rivers moving between the two as the weather changed throughout the year (http://www.greenvillerancheria.com/).

The Susanville Rancheria, while more distant in Susanville CA, appears to be better organized. Their people are from Washoe, Mountain Maidu, Achomawi, Northern Paiute, and Atsugewi Tribes (http://www.sir-nsn.gov/). The Rancheria has established a separately chartered economic development corporation named SIRCO for the purposes of developing sustainable economies for the members of the Rancheria.

There are no extant claims on the project from either Tribe and past activity at the Engles and Superior is reported to have provided jobs to tribal members. Consultation with the Rancherias will be important to the successful permitting of the project although no discussions have been had with either by Crown.

Given the location of the project, transportation will provide a significant hurdle to the the mine's acceptance in the community. The main road to Lights Creek, a narrow winding two-lane paved surface, passes through the Indian Valley which hosts a number of large ranches and private homes. These homes persist up the canyon within a few miles of the

Superior Mine. It could be expected that the owners of these parcels will not approve of the amount of traffic that a major mining operation will require and may oppose any mine plans that rely on this access corridor. Alternative roads to the mine are available and should be further assessed as the primary mine access.

County interest in further sustainable economic development will be an important aspect in the acceptance of the project. Initial discussions with County officials suggest that there will be support of the project. However, opposition in the area could be expected from those who see little direct economic benefit and may be subject to its negative implications. As might be expected in an economically challenged area, locals seeking employment or businesses anticipating direct or indirect benefit from the mine will be supportive of its development.

21.0 CAPITAL AND OPERATING COST ESTIMATES

21.1 CAPITAL COST ESTIMATES

Tetra Tech developed and prepared the capital cost estimate for the Project with input from Crown Mining.

Tetra Tech established the capital cost estimate using a hierarchical work breakdown structure (WBS). The accuracy range of the estimate is ±35%. The base currency of the estimate is US dollars. Tetra Tech used a foreign currency exchange rate of US$1.00 to Cdn$1.25, where applicable.

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is US$512.9 million. A summary breakdown of the initial capital cost is provided in Table 21.1.

Table 21.1 Capital Cost Summary

Area		Cost (US$ million)
Direct Costs		
10	Overall Site	40.4
30	Mining (excluding leased equipment)	15.6
40	Process	234.0
50	TMF	12.2
70	On-site Infrastructures	22.6
Direct Cost Subtotal		**324.8**
Indirect Costs		
X	Project Indirects	105.0
Y	Owner's Costs	12.4
Z	Contingencies	70.7
Indirect Cost Subtotal		**188.1**
Total		**512.9**

21.1.1 EXCLUSIONS

The following items are excluded from the capital cost estimate:

- pre-production mine pre-stripping (included in the financial model)

- working or deferred capital
- financing costs
- refundable taxes and duties
- land acquisition
- currency fluctuations
- lost time due to severe weather conditions
- lost time due to force majeure
- additional costs for accelerated or decelerated deliveries of equipment, materials, or services resultant from a change in project schedule
- warehouse inventories, other than those supplied in initial fills
- any project sunk costs (studies, exploration programs, etc.)
- mine reclamation costs (included in financial model)
- mine closure costs (included in financial model)
- escalation costs
- community relations.

21.1.2 Mining Capital Cost Estimate

The capital cost for mining is based on purchase of mining equipment and on pre-stripping.

A total of $84 million was estimated for mining equipment. An additional $7.5 million was allocated to the financial model for pre-stripping of waste rock.

The financial model for the Project is based on leasing mining equipment. Major mining equipment will be leased, with smaller support equipment purchased at the start of the mine life.

The LOM equipment capital cost includes:

- US$4.5 million purchased initially
- US$84 million in leased equipment
- US$50 million in sustaining capital (purchased)
- US$148 million in lease payments over the LOM

21.1.3 Processing and Overall Site Infrastructure Capital Cost Estimate

Major mechanical costs were prepared based on quotations from qualified vendors. All equipment and material costs are included as free carrier (FCA) or free board marine (FOB) manufacturer plants and are exclusive of spare parts, taxes, duties, freight, and packaging. These costs, if appropriate, are covered in the indirect cost section of the estimate.

Where appropriate, material quantities were developed from general arrangement drawings, process design criteria, process flow diagrams, and equipment lists. Electrical, platework, instrumentation, piping, and HVAC are based on historical information from similar projects.

A blended labour rate of US$73.00/h was used throughout the estimate. The labour rate was developed based on recent Northern California union rates. A productivity factor of 1.20 was applied to the labour portion of the estimate to allow for the inefficiency of long work hours, climatic conditions, and due to the three-week-in, one-week-out rotation.

Costs for the maintenance shop, truck shop, administration building, and cold warehouse are based on pre-engineered steel framed structure complete with roofing, cladding, doors and architectural finishes. The assay laboratory cost is based on a modular building.

Project indirect costs, including construction indirects, spare parts, and freight and logistics, are calculated on a percentage basis based on Tetra Tech work experience. Allowances for initial fills are provided for grinding media, reagents, lubricants and fuel. Engineering, procurement and construction management (EPCM) allowance is calculated on a percentage basis based on Tetra Tech in-house experience. Commissioning and start-up, and vendor assistance allowances are calculated based on the number of engineers required on site, estimated duration, and the average man-hour rates. The average commissioning and start-up hourly rate is US$120/h. The average vendor assistance hourly rate is US$150/h. An allowance of US$12,375,000 has been included for Owner's costs. The estimated contingencies are allowances for undefined items of work which is incurred within the defined scope of work covered by the estimate. Each discipline was allocated different contingency factors due to the varied risk level. The average contingency for the Project is 21.8% of the total direct costs.

21.1.4 Tailings Management Facility Capital Cost Estimate

The initial TMF capital costs include installation labour rates based on $73.00/h. Installation manhours were based on Tetra Tech in-house information. The cost of earthworks, footings, and foundations were determined using material take-offs from preliminary design drawings.

Pipeline costs were estimated based on supply and installation costs from suppliers and from previous projects.

The TMF capital cost summary is provided in Table 21.2.

Table 21.2 Capital Cost Summary of the TMF

Item Description	Material/ Equipment Total (US$000)	Labour Total (US$000)	Total (US$000)
Civil and Dam Construction	872	535	1,407
Reclaim and Pipeline	8,842	1,985	10,827
Total Direct	-	-	**12,234**

21.2 Operating Cost Estimates

21.2.1 Summary

On average, the LOM on-site operating costs for the Project were estimated as US$7.77/st of material processed. The operating costs are defined as the direct operating costs including mining, processing, site servicing, and G&A costs, including related freight costs. Table 21.3 shows the cost breakdown for various areas.

Table 21.3 LOM Average Operating Cost Summary

Area	Operating Cost (US$/st milled)*	Operating Cost (US$/t milled)*
Mining	2.35	2.59
Process and TMF	4.77	5.25
G&A and Site Services	0.65	0.70
Total Operating Cost	**7.77**	**8.53**

Note: *Rounded to the nearest hundredth.

The cost estimates in this section are based on the consumable prices and labour salaries/wages from Q3/Q4 2017, or information from Tetra Tech's in-house database. The expected accuracy range of the operating cost estimate is +35%/-25%. All the costs have been estimated in US dollars, unless specified.

It is assumed that operation personnel will reside in towns or villages nearby. There will be no accommodation or catering services provided at site. Personnel would commute to the site at their own expense.

The operating costs exclude shipping and refining charges for copper concentrate, which are included in financial analysis.

21.2.2 Mining

Tetra Tech estimated mining costs for each period of the mine life. Table 21.4 summarizes the mining costs over the LOM. An average cost of US$1.32/st moved was estimated. Costs vary for each year based on the scheduled throughput for each year and on haul distances from the pit to the mill and to the waste dumps. Mining costs vary from US$1.15 to US$1.58/st over the LOM. Average mining cost is US$2.35/st milled.

The key assumptions used in the estimate of the mining costs include:

- fuel cost of US$2.60/gal
- explosive cost of $0.65/lb (down-the-hole service by contractor)
- mining labour rates varying from US$33 to US$47/h excluding burden costs, or $43 to $62/h including burden costs
- four persons per role (where applicable).

Table 21.4 Mining Costs

Summary of Operating Costs	LOM Total (US$ million)	Unit Cost (US$/st mill feed)	Unit Cost (US$/st moved)
Explosives	167	0.46	0.25
Equipment Costs (Fuel and Maintenance)	45	1.24	0.69
Drill Bits	7	0.02	0.01
Labour	221	0.61	0.34
G&A (Mining Only)	9.5	0.03	0.02
Total Mining Operating Cost	**856**	**2.35**	**1.32**

21.2.3 Processing

Process Operating Costs

The average LOM unit process operating cost was estimated as US$4.58/st processed, at a nominal processing rate of 60,000 st/d, or 21,896,000 st/a, including the power cost for the processing plant. The estimate is based on 12-hour shifts, 24 h/d, and 365 d/a.

The breakdown for the estimated process operating cost is summarized in Table 21.5.

Table 21.5 Process Operating Cost Summary

Description	Unit Cost (US$/st milled)*	Unit Cost (US$/t milled)*
Manpower (109 persons)	0.48	0.53
Metal Consumables	1.67	1.84
Reagent Consumables	0.32	0.35
Maintenance Supplies	0.44	0.48
Operating Supplies	0.03	0.03
Power Supply	1.65	1.82
Total Process Operating Cost	**4.58**	**5.05**

Note: *Rounded to the nearest hundredth.

The process operating cost estimate includes:

- personnel requirements including supervision, operation and maintenance; and salary/wage levels, including burdens, based on the estimated 2017 Q3/Q4 labour rates in north-eastern California or Nevada.
- ball mill liner and grinding media consumption, estimated from the Bond ball mill work index and abrasion index equations and Tetra Tech's experience; steel ball, ball mill liner and crusher liner prices based on the estimated 2017 Q3/Q4 market prices.
- maintenance supplies, based on approximately 8% of major equipment capital costs or estimated based on the information from the Tetra Tech's database/experience
- reagent consumptions, based on test results and reagent prices from Tetra Tech's database
- other operation consumables, including laboratory and service vehicles consumables
- power consumption for the processing plant based on the preliminary plant equipment load estimates and a power unit cost of US$0.07/kWh.

All operating cost estimates exclude taxes unless otherwise specified.

Personnel

The estimated average personnel cost, at a nominal processing rate of 60,000 st/d, is US$0.48/st milled. The projected process personnel requirement is 109 persons, including:

- 20 staff for management and technical supports including personnel at laboratories for quality control, process optimization, and assaying.
- 44 operators servicing for overall operations from crushing to concentrate loadout

- 45 personnel for equipment maintenance, including maintenance management team.

The salaries and wages, including burdens, are based on the estimated 2017 Q3/Q4 labour rates in California/Nevada. The benefit burdens for the workers includes retirement savings plans, various life and accident insurances, medical benefits, unemployment insurance, tool allowance, and other benefits.

The labours required for the tailings and reclaimed water management are excluded in this estimate, but included in the tailings and reclaimed water management cost estimate.

Consumables and Maintenance/Operation Supplies

The operating costs for major consumables and maintenance/operation supplies were estimated at US$2.45/st milled, excluding the costs associated with concentrate off-site shipment and refining. The costs for major consumables, which include metal and reagent consumables, were estimated to be US$1.99/st milled. The consumable unit prices were based on the market prices in Q3/Q4 2017.

The cost for maintenance/operation supplies was estimated at $0.46/st milled. Maintenance supplies were estimated based on approximately 8% of major equipment capital costs and/or based on the information from the Tetra Tech's database/experience.

Power

The total process power cost was estimated to be US$1.65/st milled. Electricity is planned to be transmitted from Westwood, California, approximately 16 km northwest of the Project site. The power unit cost used in the estimate was approximately US$0.07/kWh. The power unit cost is estimated based on the information provided by Pacific Gas and Electric (the largest power supplier in California) and the published information of US Energy Information Administration.

The power consumption was estimated from the preliminary power loads estimated from major process equipment load list. The average annual power consumption was estimated to be approximately 516 GWh.

21.2.4 General and Administrative and Site Services

G&A and site service costs include the expenditures that do not relate directly to the mining or process operating costs. These costs were estimated as US$0.42/st milled for G&A and US$0.22/st milled for site services, based on a nominal mill feed processing rate of 60,000 st/d. The G&A and site service costs include:

- Personnel – General manager and staffing in accounting, purchasing, environmental, security, site maintenances and other G&A departments. The estimated total employee numbers are 22 for G&A and 28 for site services. Only personnel working at the Project site are included. Personnel at Crown Mining's corporate headquarters are not included.

The salaries and wages are based on the 2017 Q3/Q4 labour rates in California/Nevada, including base salary or wage and related burdens, including retirement savings plans, various life and accident insurances, extended medical benefits, unemployment insurance, tool allowance, and other benefits.

- General Expenses – General administration, contractor services, insurance, security, medical services, legal services, human resources, travel, communication services/supports, external assay/testing, overall site maintenance, electricity and fuel supplies, engineering consulting, and sustainability, including an environment and community liaison.

A summary of the G&A and site service cost estimates is shown in Table 21.6. The costs for management and service personnel were estimated to be US$0.10/st milled for G&A and US$0.12/st milled for the site services. The estimated other costs for G&A and site services are US$0.32/st milled and US$0.10/st milled, respectively.

Table 21.6 G&A and Site Service Cost Estimates

Description	Manpower	Annual Cost (US$/a)	Unit Cost (US$/st milled)*	Unit Cost (US$/t milled)*
G&A				
Labour	22	2,226,900	0.10	0.11
Other Costs	-	6,875,000	0.32	0.35
Subtotal	**22**	**9,101,900**	**0.42**	**0.46**
Site Services				
Labour	28	2,580,500	0.12	0.13
Other Costs	-	2,250,000	0.10	0.11
Subtotal	**28**	**4,830,500**	**0.22**	**0.24**

Note: *Rounded to the nearest hundredth.

21.2.5 Tailings Management Facility

Overall tailings and reclaimed water management cost was estimated as approximately US$0.18/st milled. The costs associated with the tailings dam construction and the closure are excluded from the estimates and estimated separately as sustaining capital costs.

22.0 ECONOMIC ANALYSIS

A PEA should not be considered a Prefeasibility or Feasibility study, as the economics and technical viability of the Project have not been demonstrated at this time. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results reported in the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Tetra Tech completed a pre-tax preliminary economic analysis based on estimated costs and revenues for mining and processing the Moonlight deposit. The economic analysis concluded the following financial results:

- pre-tax NPV of US$237 million at an 8% discount rate
- pre-tax IRR of 16.4%
- pre-tax payback period of 4.87 years

PwC prepared a post-tax estimate for the LOM.

- post-tax NPV of US$179 million at an 8% discount rate
- post tax IRR of 14.6%.

For the purpose of preparing the preliminary economic analysis, Tetra Tech made a number of assumptions, including:

- base case copper price of US$3.15/lb
- production rate of 60,000 st/d
- copper recovery of 86% and a silver recovery of 70%
- revenue from gold was excluded from the financial analysis
- pre-production capital costs of US$513 million, including a contingency of US$71 million
- leasing of key mining equipment for a LOM cost of US$148 million
- reclamation costs of US$60 million.

22.1 BASIS OF FINANCIAL EVALUATIONS

The production schedule has been incorporated into the 100% equity pre-tax financial model to develop annual recovered metal production from the relationships of tons processed, head grades, and recoveries.

Copper and silver payable values were calculated based on base case metal prices. Net invoice value was calculated each year by subtracting the applicable refining charges from the payable metal value. At-mine revenues are then estimated by subtracting transportation and insurance costs. Operating costs for mining, processing, and G&A were deducted from the at-mine revenues to derive annual operating cash flow.

Initial and sustaining capital costs as well as working capital have been incorporated on a year-by-year basis over the mine life. Salvage value and mine reclamation costs are applied to the capital expenditure in the last production year. Capital expenditures are then deducted from the operating cash flow to determine the net cash flow before taxes.

Initial capital expenditures include costs accumulated prior to first production of concentrate. Sustaining capital includes any capital expenditures required during the production period.

Working capital is assumed to be three months of the annual operating cost, reducing to one month from Year 3 onwards and fluctuates from year to year based on the annual cost. The working capital is recovered at the end of the mine life.

Mine closure and reclamation is assumed to be roughly US$0.09/st mined and incurred at the end of mine life.

Pre-production period is assumed to be two years. NPV and IRR reported in this section are estimated at the start of the two-year pre-production period.

22.2 SUMMARY OF KEY FINANCIAL RESULTS

A summary of key financial results is shown in Table 22.1. Tetra Tech evaluated the project financial performance as currently planned, at various copper prices ranging from US$3.00/lb to US$4.00/lb. The base case was evaluated at a price of $3.15, which is based on long-term forecasts for copper price.

Table 22.1 Key Financial Results

Item	Unit	Value				
Copper Price	US$/lb	3.15 (Base Case)	3.00	3.25	3.50	4.00
LOM	years	17	17	17	17	17
Annual Tonnage Processed	'000 st	21,469	19,476	19,476	19,476	19,476
Tons Mined including Waste Rock	'000 st	650,847	650,847	650,847	650,847	650,847
Tons Processed	'000 st	364,967	364,967	364,967	364,967	364,967
Tons of Concentrate Produced (dry mass)	'000 st	2,763	2,763	2,763	2,763	2,763
Copper Recovered to Concentrate	'000 st	774	774	774	774	774
Silver Recovered to Concentrate	'000 oz	19,141	19,141	19,141	19,141	19,141
Off-site costs	US$ millions	516	516	517	517	519
Net Revenue from Sales	US$ millions	4,468	4,245	4,617	4,990	5,734
Average NSR	US$/st	12.24	11.63	12.65	13.67	15.71
Operating Cost (LOM)						
Mining	US$ millions	856	856	856	856	856
Processing	US$ millions	1,740	1,740	1,740	1,740	1,740
G&A	US$ millions	237	237	237	237	237
Total LOM Operating Cost	**US$ millions**	**2,832**	**2,832**	**2,832**	**2,832**	**2,832**
Operating Cost (per st processed)						
Mining	US$/st mined	1.32	1.32	1.32	1.32	1.32
Mining	US$/st processed	2.35	2.35	2.35	2.35	2.35
Processing	US$/st processed	4.77	4.77	4.77	4.77	4.77
G&A	US$/st processed	0.65	0.65	0.65	0.65	0.65
Total LOM Operating Costs	**US$/st processed**	**7.76**	**7.76**	**7.76**	**7.76**	**7.76**
Capital Costs						
Pre-production Capital Costs	US$ millions	502	502	502	502	502
Mining Equipment Leasing costs (LOM)	US$ millions	148	148	148	148	148

table continues...

Item	Unit	Value				
Copper Price	US$/lb	3.15 (Base Case)	3.00	3.25	3.50	4.00
LOM Sustaining Costs	US$ millions	97	97	97	97	97
Reclamation Costs	US$ millions	60	60	60	60	60
Cash Flow						
Pre-tax Operating Cash Flow	US$ millions	851	628	1,000	1,373	2,117
Pre-tax NPV 8%	US$ millions	237	132	307	482	832
Pre-tax IRR	US$ millions	16.4	12.9	18.6	23.9	33.5
Cash Flow						
Post-tax Operating Cash Flow	US$ millions	708	529	827	1,119	1,706
Post tax NPV at 8%	US$ millions	179	91	236	376	653
Post tax IRR	US$ millions	14.6	11.5	16.6	21.1	29.4

22.3 POST TAX ANALYSIS

Crown Mining commissioned PwC to conduct a tax assessment for the Project. The taxes are based on the pre-tax cash flows provided by Tetra Tech.

PwC made the following assumptions for tax calculation purposes.

22.3.1 CORPORATE STRUCTURE

The Project will be operated by Crown Mining, a company listed on the Toronto Stock Exchange (TSX). For the purposes of the tax model, Crown Mining will be considered a single entity C- corporation with operations in the US.

The tax model does not take into account the impact of any affiliated US entities.

22.3.2 FINANCING

It is assumed the Project will be 100% funded by equity. No interest or principal repayments with respect to any debt will be included in the analysis because of this assumption.

22.3.3 TAXATION AUTHORITIES

The applicable tax jurisdictions of the Project for purposes of the tax model will be US federal and the state of California. The tax model was prepared based on the US federal Internal Revenue Code (IRC) and the Revenue and Taxation Code for the state of California respectively, and that were enacted on the date of the tax model.

22.3.4 TAX RATES

For the US federal income tax calculations, the statutory income tax rate of 21% for all the years of the Project was used. The 21% rate will be used under the assumption the currently enacted income tax rates will remain in effect for the life of the mine.

The California regular income tax rate of 8.84% was used for the California tax calculations and under the assumption no change will occur during the life of the mine. In addition, the California Alternative Minimum Tax (AMT) was also calculated for the life of the mine and a California AMT rate of 6.65% was used.

22.3.5 NET OPERATING LOSSES

All net operating losses (NOL) incurred will be utilized in future years in which there is taxable income. The usage and calculation of remaining NOLs will take into account the 80% taxable income limitation brought into law under the Tax Cuts and Jobs Act of 2017 (TCJA).

It is assumed no additional restrictions on NOL usage will be in place during the life of the mine.

22.3.6 Percentage Depletion and Cost Depletion

For the percentage depletion calculation, the Tax Model reflects only the mining costs that are allowed to be deducted under the IRC, and the "adjusted gross income" of the percentage depletion calculation was adjusted for all non-mining costs per discussions with Crown Mining management.

All off-site costs have been confirmed with the Tetra Tech to be excluded from mining costs, and the transportation costs are assumed to be in excess of fifty miles of the mine.

All general and administrative expenses included in mining costs will be treated as such for purposes of calculating percentage depletion. Additionally, the general and administrative costs not included in mining costs will be excluded in the depletion calculation.

It is assumed any processing costs incurred and included under the category "Mining Costs" meet the definition for Mining Costs per IRC section 613(c)(4).

It was also discussed with Crown Mining management, and thereby assumed in the tax model, that the tax basis of the mineral property is nil. Therefore, no cost depletion was calculated.

22.3.7 Reclamation

The tax model assumes the reclamation deduction for tax basis will be based on the accrual method under IRC section 468, and any imputed interest will not be significant and was not calculated.

22.3.8 Other Timing Differences and Assumptions

Assumed the Project will be at development stage in Year-2 and Year-1, and assumed Year-2 is year 2020 and Year-1 is year 2021 respectively. The remainder of the model will be considered in production in Year 1, which is year 2022 and onwards.

The capital equipment expenditures in Year-2, and Year-1 (the years 2020 and 2021 of the model respectively) are assumed to all be qualified assets that can take 100% deductions, and capitalized as tax development costs (except for building costs, see below). Thus, such capital expenditures took 70% deduction in the year the costs were incurred with the remaining 30% of the costs being amortized over 5 years.

The buildings costs in Year-2, and Year-1 (the years 2020 and 2021 of the model respectively) have not been included in development costs for those years, and have been depreciated per the MACRS three-year useful life.

We have assumed, after discussion with Crown Mining management, all capital expenditures not related to the building will be assumed to be MACRS 7 years and 15 years assets and placed into service when costs incurred.

Crown Mining will continue to own the building and equipment at the end of mining life.

The maximum bonus depreciation deduction will be taken in the years it is available per the TCJA of 2017.

We have assumed that California will not conform to the recently passed federal tax law changes (TCJA of 2017) and this assumption was applied in the construction of the Tax Model. As such the California regular, AMT and other California adjustments have been made to arrive at California taxable income for the life of the mine.

For regular California tax depreciation purposes the straight line method was used on capital expenditures. The 150% Declining Balance method was used for California's AMT tax depreciation on capital expenditures.

California Environmental fees and taxes, as well as any other indirect taxes, have not been factored in the tax model.

For federal taxable income calculation purposes, deducted California state tax using the lag method.

US Uniform Capitalization (UNICAP) adjustments governed by IRC section 263A were not incorporated. The US UNICAP adjustment normally results in a one-year timing difference in relation to the tax deduction for cost of goods sold.

Any withholding taxes on dividend payments as part of any repatriation strategy, or any other distributions, have not been included in the tax model and we have assumed the after-tax profits will remain in the US subsidiary.

22.3.9 Tax Results

Tetra Tech has determined the following tax results for the Project based on the tax assessment by PwC.

Table 22.2 Tax Results ($US millions)

Item	Amount
Project Revenue	4,985
Operating Costs	2,825
Off-site Costs	516
Project Income	1,643
Capital Costs	502
Sustaining Capital	97
Other Operating Costs	193
Income Before Taxes	851
Federal Taxes	78
California Taxes	65
Total Taxes	143
Income After Taxes	708

22.4 CASH FLOWS

Tetra Tech calculated cash flows based on revenue from sale of copper concentrate. Tetra Tech deducted off-site costs and smelter fees from revenue prior to deduction of operating costs to estimate operating income.

Initial, sustaining capital costs, equipment leasing costs, working capital and reclamation costs were deducted from the operating income to derive an estimate of pre-tax cash flow.

A summary of pre-tax and post-tax cashflows are show in Appendix A.

22.5 SENSITIVITY ANALYSIS

To evaluate Project economic risks, Tetra Tech conducted sensitivity analysis on the pre-tax financial performance. Figure 22.1 and Figure 22.2 show the Project economic sensitivities.

Tetra Tech evaluated the sensitivity of the project economics to copper price, capital costs, operating costs and combined capital change in capital and operating costs.

The Project, as planned, is most sensitive to copper price and least sensitive to capital costs. The Project is relatively sensitive to a combined increase in both operating and capital costs.

Figure 22.1 Sensitivity of NPV to Changes in Costs and Prices

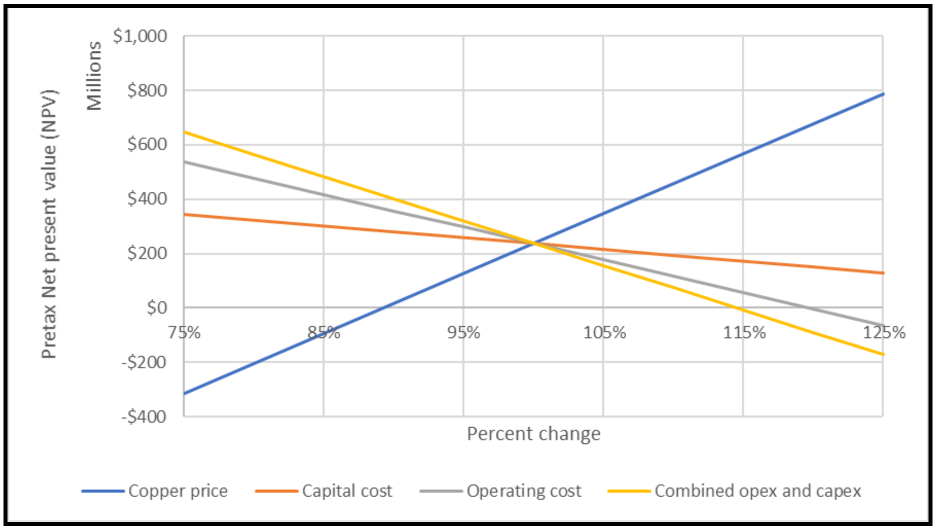
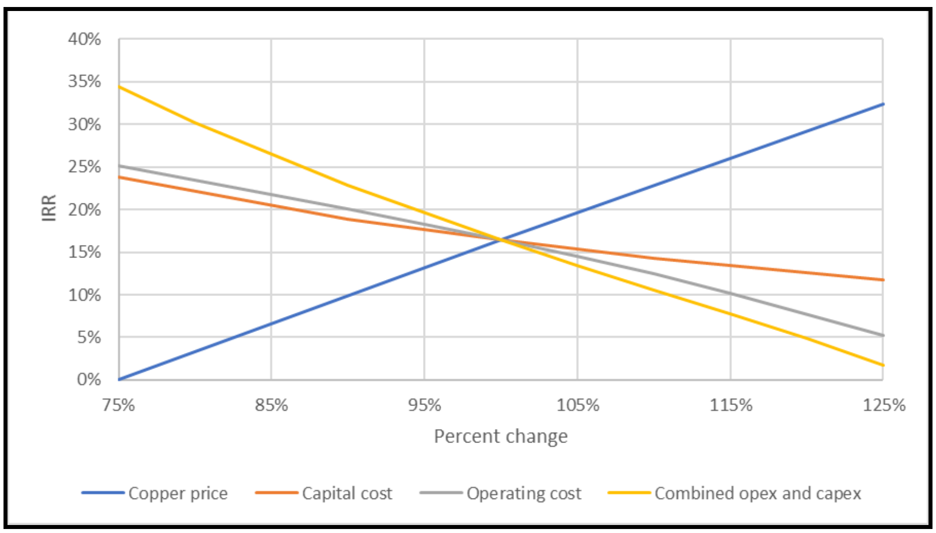


Figure 22.2 Sensitivity of IRR to Changes in Costs and Prices



The Project has a negative NPV at a copper price of less than US$2.85/lb.

22.6 ROYALTIES

Crown advised that there are no private royalties applicable to this Project. Therefore, no royalties are considered in this economic analysis.

23.0 ADJACENT PROPERTIES

In addition to the Moonlight deposit, Crown Mining's Moonlight-Superior Project controls the entire historic LCD which saw sustained activity from the late 1800s to 1930. Between 1914 and 1930 the Engels and Superior Mines, the two principal producers, reported joint production of approximately 161.5 million lb of copper, 23,000 oz of gold and 1.9 million oz of silver recovered from 4.7 million st of ore (Lamb 2006 pers. comm.). Both properties have been idle since the suspension of mining activities in 1930. Both deposits are contained within Crown Mining's current property position.

The Engels Mine lies three miles east of the Moonlight deposit. The Superior Mine is located two miles southeast of Moonlight and approximately 2.2 mi southwest of Engels. In the 1960s and 1970s Placer-Amex produced Mineral Resource estimates for both deposits, none of which meet NI 43-101 standards because they were estimated prior to the advent of NI 43-101.

At the Engels Mine, in the 1970s, Placer-Amex recognized the possibility that approximately two million st grading 0.65% copper (not to NI 43-101 standards) might remain in the pillars and immediate areas and be amenable to open pit mining. They reported an Indicated and Inferred Mineral Resource of 19 million st averaging 0.63% copperu (not to NI 43-101 standards) underground that was not considered amenable to open pit mining. Placer-Amex also reported 68,000 st of 2% copper (not to NI 43-101 standards) remaining in the shaft level sill pillar. The underground mineralized areas are no longer accessible by the previous production shafts and adits.

At Superior, from 1964-1968, Placer-Amex drilled 47,964 ft in 96 diamond drillholes and completed 3,550 ft of reverse circulation drilling outlining a substantial body of disseminated copper mineralization. Preliminary “ore reserves” (not to NI 43-101 standards) were estimated to be 43 million st grading 0.559% copper with a 0.3% copper cut-off. In 1971-1972 Placer-Amex completed another Mineral Resource estimate using a 0.25% cut-off and reported 39 million st grading 0.41% copper (not to NI 43-101 standards).

In the 2014 technical report covering the Superior and Engels deposits Tanaka produced the following NI 43-101 produced Mineral Resource estimates, a report on which is filed on SEDAR:

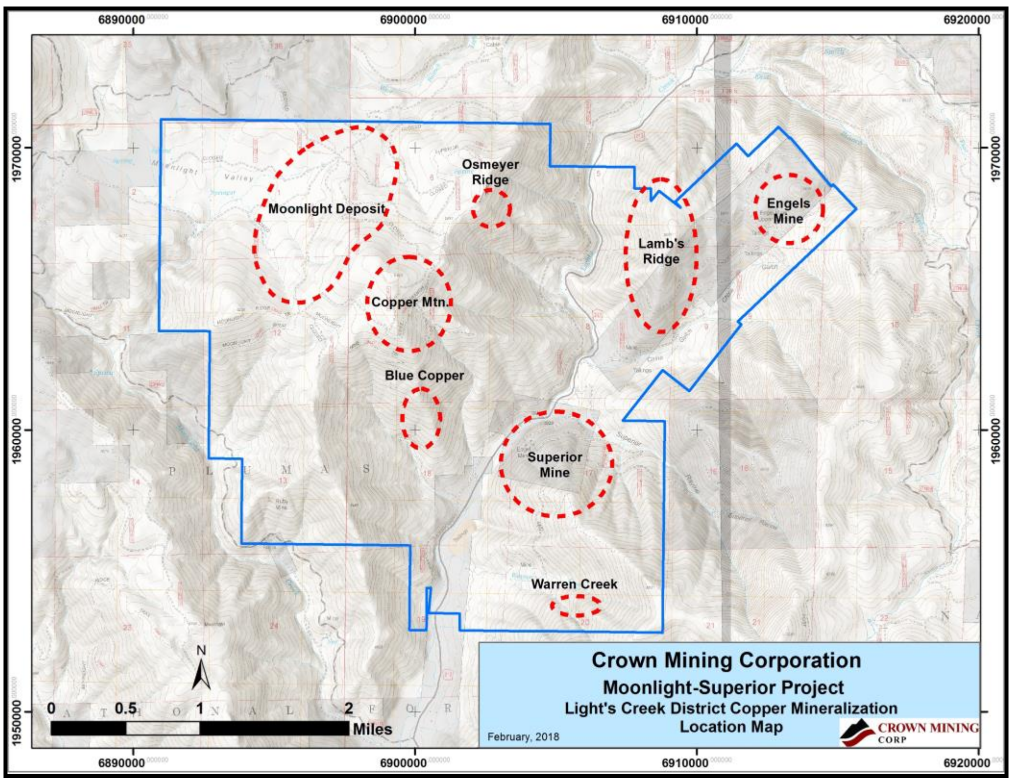
- Engels: Inferred 2.5 Mt @1.05% total copper
- Superior: Inferred 54 Mt @0.41% total copper
- Total: Inferred 57 Mt @0.43% total copper

As discussed in Section 9.0, a number of other copper mineralized areas exist on the Crown Mining Property. As shown in Figure 23.1, these include Lamb's Ridge, Copper Mountain, Blue Copper, Osmeyer Ridge, and Warren Creek.

Lamb's Ridge, 2.3 mi east of the Moonlight deposit, is the site of one of the strongest and most extensive copper in soil anomalies on the Crown Mining Property. Placer-Amex tested the anomaly with 28 widely spaced diamond drillholes. Crown Mining reports (W.D. Baker 1967) mention the possibility of a large tonnage of low-grade copper mineralization. The mineralization style and grade has been compared to that of the Moonlight deposit. To date, no follow-up drilling has been undertaken.

Additional exploration including utilization of the Fugro airborne data, a modern interpretation of the Placer-Amex era ground geophysical data (IP-Resistivity and ground magnetics), possibly a new IP survey and more drilling will be necessary to fully understand and evaluate these outlying mineralization centers.

Figure 23.1 Copper Mineralization Location Map



The Walker Mine is located at the southern end of the Plumas Copper Belt approximately 12 mi southeast of the Moonlight deposit. Numerous small mines and copper showings exist between the Walker Mine and the Crown Mining land package. The Walker Mine is reported to have produced about 168 million lb of copper, 180,000 oz of gold and 3.6 million oz of silver from 5.3 million st of ore between 1916 and 1941. Assuming 80% recovery, the feed grade would have been 1.98% copper, 0.85 oz/st silver, and 0.041 oz/st gold. The copper mineralization at the Walker Mine is contained in N20W, steeply northeast dipping zones of quartz, chlorite, magnetite and pyrite. Chalcopyrite is the predominant copper mineral but bornite is also abundant (Tanaka 2014).

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT DEVELOPMENT PLAN

To achieve the project schedule, the long-lead process equipment will need to be identified at the beginning of the Feasibility Study stage. The critical path for the Project will be the supply and delivery of this equipment.

The early-start date is driven by the civil construction work. To achieve this schedule, several construction packages will need to be issued as unit rate packages. The unit rate packages will include rough grading, concrete and structural steel buildings, and interior steel platforms.

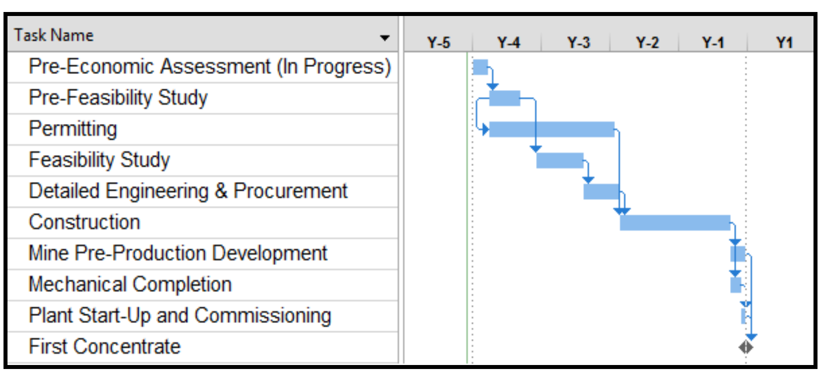
Upon construction commencement, the temporary construction facilities will be mobilized, including the batch plant and aggregate plant. Site preparation, grading, and road construction will commence immediately upon receipt of permits and approvals. Modular construction will be utilized wherever practical to reduce field construction.

Upon completion of foundation preparation, the concrete for the main process building, truck shop, and powerhouse building foundations will be poured to allow the buildings to be erected. Once the buildings are erected, the concrete inside the buildings (including equipment supports) can be poured in a controlled environment through the winter.

Electrical and mechanical installation contracts will be bid lump sum to qualified contractors. A start-up and commissioning period has been allowed at the completion of construction in order to complete mechanical check out and acceptance and commissioning of the facilities.

A conceptual summary schedule for the Project is shown in Figure 24.1.

Figure 24.1 Conceptual Project Summary Schedule



25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 MINERAL RESOURCE

A historic database comprising 202 core holes and 11,005 copper assays supports the Mineral Resource estimates presented in this report. The database is a combination of 189 N- and B-sized drillholes drilled by Placer-Amex and its predecessor companies in the period 1966-1970 and 13 HQ wireline drillholes drilled by Sheffield in 2005-2006. The Placer-Amex information for copper and silver is of sufficient confidence for grade estimation, but Placer-Amex gold assays were not used to estimate Mineral Resources. The Placer-Amex portion of the database has four significant deficiencies: 1) No core or samples remain for inspection; 2) Missing assay certificates; 3) Missing collar survey information; and 4) Database transcription errors. The Sheffield data, comprising 10.5% of the copper composites used for grade estimation, appears to have been collected according to current best practices. Sheffield core is available for inspection and evidence of drilling is still present at the drill platforms. Assay methods and detection limits were appropriate to the type of material and the metal concentrations of the deposit. The only significant deficiency is the quality of original collar survey which was corrected to the extent possible in the current work using a new topography DTM. The Sheffield data largely confirm the geologic logging and the copper and silver assaying performed by Placer-Amex in the earlier campaign within a significant volume of the deposit. Additional confirmation with core drilling will extend this level of confidence to the entire deposit.

Copper mineralization is largely confined to a quartz monzonitic stock with minor mineralization extending into metavolcanics and undifferentiated tertiary sediments that cap it. A strong correlation with hairline quartz and quartz-magnetite veining is noted, without a strong alignment in a preferred direction. A weak positive correlation is seen between copper and the occurrence of tourmaline and strong potassic alteration. Copper, and to a lesser extent silver mineralization trends are aligned north-northeast with a cross-trend striking northwest. These have been incorporated into the estimation search ellipse. Copper mineralization is strongest in large pods along these trends and gradually decays laterally outward. A 0.1% copper shell based on contouring an indicator estimate delimits most of the better copper mineralization in the deposit. Most of the copper mineralization occurs in fresh rock. Surface oxidation extends to variable, but generally shallow depths; material estimated to be oxidized is treated as waste in this study. Further sampling will be necessary to accurately interpret the surface or zone of transition from oxidized to fresh rock.

Geostatistical analysis of the drillhole data demonstrates significant continuity of copper mineralization in all directions, but with preferred trends. Trends are less pronounced for the silver and gold mineralization, perhaps because of the low concentrations of these metals. Historical Placer-Amex gold assaying is unsuitable for the low levels of gold in the deposit; the gold estimates only incorporate Sheffield data. The estimations of copper and silver by ordinary kriging are validated by comparison with the underlying composited drillhole support, for bias and degree of smoothing. The results are appropriate as an estimate of the Mineral Resources within the conceptual pit at the NSR cut-off stated; however, the silver estimate and the less important gold estimate will be significantly improved in quality by the recommended infill drilling.

A decrease in block size subsequent to recommended infill drilling could favorably impact selectivity and potential economics of the Project and allow planning of a definitive grade control plan. This could be investigated upon completion of the recommended work program (Section 26.0).

25.2 Mineral Processing and Metallurgical Testing

Several metallurgical test programs were conducted before 2017. The early work was mainly focused on heap leach processing although some test work had been conducted using flotation to concentrate the copper minerals. In 2017, Crown Mining undertook a metallurgical test work program for the Project to confirm previously completed test work, to confirm effective flotation reagent conditions and demonstrate the recoveries and concentrate quality that can be achieved with the tested material.

Three types of the mineral materials identified as MN Sulfide, MN Oxide, and SUP Sulfide were used in the 2017 test work. The test work results identified that a good copper concentrate grade of approximately 28% Cu or higher could be expected. The results appear to suggest the potential need of a regrind mill. As chalcopyrite tends to be harder and floats at a coarser size with associated gangues, the regrind is anticipated to improve the target mineral liberation and remove any entrained particles. The grindability test results from the three mineral samples show that the Bond ball work index ranged from 18.1 to 21.3 kWh/st, indicating that these materials should be very resistant to ball mill grinding. The samples tested in the 2017 test program may not be well representative of the mineralization of the deposit as estimated. Further test work on representative samples should be conducted.

25.3 Mining Methods

The Moonlight deposit is most suited to open pit bulk mining methods. Additional work is required to better understand stable open pit slopes and for understanding rock breaking requirements.

Additional potential for optimization of mining remains through use of one or more of the following:

- potential use of electrically powered shovels as opposed to diesel powered equipment
- trolley assist on mine ramps to reduce haul fuel requirements

- in pit crushing and conveying
- potential use of automated mining equipment to increase efficiency and reduce labour requirements.

It is noted that the open pit will require backfilling as per California mining regulations, and that the planned mining for the Moonlight deposit will not result in adequate waste rock to backfill to original ground levels.

25.4 RECOVERY METHODS

The 60,000 st/d processing plant will utilize three stages of crushing (gyratory, cone and HPGR crushing), ball mill grinding, and conventional flotation to produce a high-grade copper concentrate with associated precious metal credits. The processing plant will operate in two 12 h shifts per day, 365 d/a. The primary crushing plant availability will be 70%. Secondary crushing, tertiary crushing, grinding, and flotation plant availability will be 92%.

The crushed mineralized material will be conveyed to the grinding area and ground to 80% passing 110 µm in a ball mill grinding circuit. The ground material will be processed using copper rougher/scavenger flotation followed by copper rougher concentrate/scavenger regrinding. The reground copper rougher/scavenger flotation concentrate will then be further upgraded by three stages of cleaner flotation. The copper cleaner scavenger tailings will be retreated in the copper rougher flotation circuit, Copper rougher/scavenger flotation tailings will be delivered to the TMF. The third cleaner flotation concentrate, which will on average contain approximately 28% copper, will be thickened and then pressure-filtered before it is shipped to smelters.

The flowsheet and equipment proposed for the project have been widely used in mining industry and are expected that this equipment can be operated and maintained effectively in the local environment.

25.5 PROJECT INFRASTRUCTURE

The Property is accessible through a network of existing forestry service roads. High voltage power source is reasonable close to the Property. Proximity of small towns nearby would allow construction workers and operation staff to reside in towns nearly, thus eliminate the need for building permanent accommodations on site.

The gentle terrain in the proposed process plant site allows ancillary buildings and facilities to be located close to the process plant. This will help reducing the material movements between various facilities on site.

Major buildings at the plant site will include the mill building, administration building, truck shop complex, assay laboratory, primary crushing (gyratory crusher) building, secondary crushing (cone crusher) and tertiary crushing (high-pressure grinding roll [HPGR]) building, concentrate storage and concentrate loadout facility, substation, warehouse, and cold storage. The TMF is designed to accommodate over 370 million st of tailings, to be generated over the 17-year LOM. The design mill throughput rate is nominally 21.9 million st/a. The design will permit storage of approximately 315 million cu yd of tailings at an average tailings dry density of 87 lb/cu ft.

25.6 ENVIRONMENTAL

There are no significant issues identified in this study pertaining to environmental conditions, permitting or social/community impact. Limited baseline sampling shows that applicable standards for water quality are not exceeded in the main stems of water bodies draining pre-existing historical mining impacts. Acid-base Accounting (ABA) indicates that neither the tailings produced from flotation testing nor existing tailings are acid producing.

Permitting a new mine in the area is not precluded by existing legislation or land use regulations. A full environmental review under both the state and federal regulations is anticipated as land management and ownership is divided between private and federal entities. As with most mines, transportation, air quality, waste disposal, and water quality will likely constitute the primary areas of focus. California has a strict mining law that mandates pit backfilling of new open-pit mines if wastes are available.

The local economy is still dominated by resource use and extraction industries such as logging and ranching although recreation is a growing presence in the county. Community support for the Project will likely be mixed, with support from those residents interested in economic development and opposition from those who see no direct benefits. No serious outreach to the communities of interest has been undertaken so the full extent of support or opposition is not quantified.

25.7 CAPITAL AND OPERATING COST ESTIMATE

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is US$512.9 million. This total includes all direct costs, indirect costs, Owner's costs, and contingency. All costs are shown in US dollars and the accuracy range of the estimate is ±35%.

This estimate has been prepared with a base date of Q4 2017 and does not include any escalation past this date. Where applicable, the quotations used in this estimate were obtained in Q4 2017 and are budgetary and non-binding.

On average, the LOM on-site operating cost for the Project are estimated to be $7.77/st of material processed. The operating costs are defined as the direct operating costs including mining, processing, surface services and (G&A) costs.

The expected accuracy range of the operating cost estimate is +35%/-25%.

25.8 ECONOMIC ANALYSIS

A pre-tax economic model has been developed from the estimated costs and the open pit production schedule. The base case has an IRR of 16.4% and a NPV of US$237 million at an 8% discount rate for the 17-year mine life. The payback of the initial capital is within 4.87 years.

The economic evaluation conducted for the PEA, shows that the project is most sensitive to copper price and least sensitive (of the parameters evaluated) to capital costs.

The project has a negative net present value at a copper price of less than US$2.85/lb.

26.0 RECOMMENDATIONS

26.1 ESTIMATED IMPLEMENTATION COSTS

Table 26.1 outlines the estimated implementation costs of the recommendations described in this section.

Table 26.1 Estimated Implementation Costs of Selected Recommendations

Recommended Test Work or Study	Estimated Cost (US$)
Resource Development Drilling and Modeling	
Exploration Drilling	3,900,000
Resource Model Updates	200,000
Mining	
Preliminary Geotechnical Study	300,000
Pit Designs	150,000
Metallurgical Test Work	
Mineralogical Study and Flotation Optimization Tests	250,000
Crushability and Grindability	60,000
Bacterial Oxidation Heap Leaching	80,000
Tailings Sample Characterization	20,000
Recovery Methods	
Plant Design and Layout Optimization	200,000
Project Infrastructure	
Geotechnical Drilling Investigation	300,000
Construction Schedule Optimization and Equipment Modularization	50,000
Tailings Management Facility	
Site Investigation and Testing Programs	500,000
Environmental	
Air	150,000
Hydrology (surface water)	50,000
Water Quality	25,000
Groundwater Hydrology and Chemistry	150,000
Waste characterization	100,000
Meteorology	50,000
Biologic Baseline	50,000

26.2 GEOLOGY

It is recommended to undertake additional geologic interpretation to support the statistical trends in metals grades incorporated into the Mineral Resource estimates.

26.3 MINERAL RESOURCES

The QPs recommend that Crown Mining perform a work program to advance the Moonlight deposit to the prefeasibility stage. The program comprises work to improve confidence in the Mineral Resources, convert Inferred Mineral Resources to Indicated Resources, and Mineral Resources to Mineral Reserves. The following recommendations are made with respect to the Mineral Resource estimate:

- Complete 25 HQ infill core holes (17,000 ft) to confirm results from 1960s Placer-Amex programs which compose approximately 85% of the drillhole database.
- Complete 25 HQ core holes (22,000 ft) to target areas of Inferred Mineral Resource with higher copper grade in order to convert a portion of this material to Indicated Mineral Resource.
- Survey to tie Placer-Amex and Sheffield drillhole monuments to the new topography and NAD83 State Plane coordinate system.
- Collect additional specific gravity determinations to fully characterize the rock volumes conceived for mining.
- Audit the Sheffield portion of the database versus the assay certificates, rebuilding the database as, or where necessary.
- Check 100% of the copper assays in the Placer-Amex portion of the assay table against the hand-posted assays on the logs; replace Placer-Amex precious metal assays with results obtained by infill drilling.

An additional recommendation is that exploration and QA/QC programs for future work should conform to industry best practices, including using commercial database management software.

The estimated cost of the recommended drilling, assaying, surveying, SG characterization work and data base audit is presented in Table 26.1.

26.4 MINING METHODS

No geotechnical evaluation has been conducted in order to determine stable pit slopes. As such Tetra Tech recommends that future exploration drilling should include capturing geotechnical data such that a geotechnical evaluation can be completed prior to advancing the project engineering.

26.5 Metallurgical Test Work

Tetra Tech recommends further metallurgical test work to optimize process conditions and to establish design-related parameters on representative samples for the next stage of study. Following are the recommendations:

- Further metallurgical test work and mineralogical characterization should be conducted to optimize process conditions and to establish design-related parameters for the next stage of study. The test work should include variability testing of samples from different mineral zones and lithological areas. Also, the potential mill feeds for the initial years based on the preliminary mine plan should be tested to verify their metallurgical response to the developed processing conditions.
- Further study into the mineralization hardness should be conducted to confirm the crushability and grindability of the mineralization, including these samples' amenability to HPGR crushing.
- A preliminary study into the amenability of the mineralization to bacterial oxidation heap leaching is recommended.
- Preliminary evaluation of the tailings sample characterization, including ABA tests and metal leachability, should be conducted.

26.6 Recovery Methods

According to the results of next phase test work, further optimizations on plant design and layout are recommended. The costs associated with the optimizations will be part of the costs for the next phase of study.

26.7 Project Infrastructure

Geotechnical drilling investigation for infrastructure foundation studies are recommended for the proposed locations of the mineral processing facilities and ancillary buildings.

The opportunity of optimizing the cash flow by expediting the construction schedule and adapting modularization of the process plant should be evaluated further as part of the next phase of study.

26.7.1 Tailings Management Facility

The following tasks are recommended to advance the TMF design concept:

- A trade-off study of alternate tailings storage methods should be undertaken that includes consideration of thickened and dry stack approaches.

- The viability of dam construction using cyclone sands should be confirmed based on testing and characterisation of representative tailings samples.
- A subsurface geotechnical investigation including materials characterisation via field and laboratory testing should be performed to assess foundation conditions and potential construction materials. Geotechnical characterisation of tailings samples should be undertaken.
- Geochemical assessment of tailings, mine waste, and potential construction materials.
- The design of containment features should be developed based on seepage and stability assessments that consider material properties, site conditions, and regulatory requirements. Contaminant fate and transport modelling should be undertaken to support determination of containment requirements.
- Design of water management features, including diversion size and alignment, that incorporates seasonal climate and mine site water balance considerations.
- A geotechnical and environmental monitoring plan should be developed that includes consideration of monitoring instrument type and position, and locations of groundwater monitoring wells.
- The closure and reclamation plan should be developed in accordance with design guidelines and regulatory requirements.

26.8 ENVIRONMENTAL

Due to the limited nature of existing baseline environmental data for the project the next step recommendation is to conduct a scoping-level assessment of major environmental elements. The scoping assessment would allow project developers to identify environmental issues needing further study and initiate discussion with the regulatory community before investing in the more extensive characterizations necessary for permitting efforts. Future programs would be tailored to address these concerns.

A proposed scoping assessment would involve the following elements:

- Air: establish baseline air quality with emphasis on particulate matter in the 2.5 and 10 µm size fractions through the installation of a continuous dual PM monitor on the project. Data collection for a full calendar year is proposed. Equipment can be rented.
- Surface Water Hydrology: establish gauging stations above and below the project on potentially affected stream courses. Stream levels could be measured on a quarterly basis.
- Surface Water Quality: re-establish sampling sites on Moonlight and Lights Creeks to monitor existing stream geochemistry. Quarterly sampling in conjunction with hydrology is deemed adequate for this stage.

- Groundwater Hydrology: drill one monitoring well in the project outline to identify groundwater conditions and geochemistry. These data would be used to design a more thorough groundwater assessment to assess how development would affect existing groundwater pathways, flow conditions and geochemistry. If this program is conducted at the same time as proposed exploration drilling, significant savings may be obtained by developing selected bore holes as monitoring wells.
- Waste Characterization: characterize the geochemistry and metal leaching potential of non-mineralized bedrock, and tailings generated from additional metallurgical testing. Humidity cell testing, while anticipated, is not necessary at this stage of the project. A suggested program would include bulk and trace element chemistry, column leaching and acid base accounting.
- Meteorology: air modelling and water management will require site-specific meteorological data including air speed and direction, precipitation and evaporation. At this point in the project a survey of existing meteorological sites in the region should be undertaken to help scope the requirements for an on-site station at some point in the future.
- Biologic Baseline: existing literature and regional surveys should be reviewed in order to assess the potential for threatened or endangered species habitat in the project area.

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APPENDIX A

PRE- AND POST-TAX CASHFLOW

MINE PRODUCTION SCHEDULE	Total/ Average	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19 to Year 30
Mill feed ('000 st)	364,967			18,615	21,900	21,900	21,900	21,900	21,900	21,900	21,889	21,900	21,897	20,238	19,594	21,900	21,900	21,900	21,900	21,835		
Waste mined ('000 st)	285,879		5,000	12,479	18,976	15,141	23,086	23,098	2,030	18,459	23,111	23,087	23,103	24,762	25,406	15,664	12,931	9,600	6,008	3,937		
Total mined ('000 st)	650,846		5,000	31,094	40,876	37,041	44,986	44,998	23,930	40,359	45,000	44,987	45,000	45,000	45,000	37,564	34,831	31,500	27,908	25,772		
Strip ratio (waste:ore)	0.78			0.67	0.87	0.69	1.05	1.05	0.09	0.84	1.06	1.05	1.06	1.22	1.30	0.72	0.59	0.44	0.27	0.18		
Mill Feed Grades																						
Gold (oz/'000 st)	0.07			0.12	0.08	0.10	0.08	0.07	0.10	0.07	0.07	0.10	0.07	0.05	0.06	0.06	0.05	0.05	0.05	0.04		
Copper (%)	0.25			0.32	0.30	0.27	0.23	0.23	0.31	0.20	0.21	0.23	0.24	0.25	0.24	0.23	0.22	0.21	0.23	0.27		
Silver (oz/st)	0.07			0.09	0.07	0.08	0.05	0.06	0.07	0.05	0.06	0.07	0.08	0.10	0.07	0.07	0.07	0.08	0.09	0.11		
Copper Concentrate Production																						
Concentrate Ton (dry mass - '000 st)	2,763			181	200	184	152	154	206	136	142	156	165	157	144	157	149	144	154	180		
Concentrate Ton (wet mass - '000 st)	2,984			196	216	199	164	167	223	147	154	168	178	169	155	170	161	155	166	194		
Concentrate Grade - Copper % of dry mass	0.28			0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28		
Metal Content of Concentrates																						
Copper ('000 lb)	1,547,161			101,601	111,970	103,306	85,248	86,402	115,553	76,228	79,786	87,348	92,193	87,702	80,543	88,067	83,682	80,480	86,307	100,747		
Silver ('000 oz)	19,141			1,179	1,140	1,217	837	957	1,129	739	851	1,005	1,214	1,420	966	1,077	1,108	1,265	1,392	1,645		
ANNUAL OPERATING COSTS																						
Total Mining ($ million)	$849			$43	$51	$46	$52	$53	$33	$51	$57	$58	$58	$57	$59	$52	$50	$46	$42	$41		
Processing ($ million)	$1740			$89	$104	$104	$104	$104	$104	$104	$104	$104	$104	$96	$93	$104	$104	$104	$104	$104		
G&A ($ million)	$237			$14	$14	$14	$14	$14	$14	$14	$14	$14	$14	$14	$14	$14	$14	$14	$14	$14		
Total Operating Costs ($ million)	$2825			$145	$169	$164	$170	$172	$152	$169	$175	$176	$176	$168	$166	$170	$168	$165	$161	$159		
COPPER CONCENTRATE TREATMENT CHARGES																						
Metal Deductions																						
Copper ('000 lb)	55,265			3,629	4,000	3,690	3,045	3,086	4,128	2,723	2,850	3,120	3,293	3,133	2,877	3,146	2,989	2,875	3,083	3,599		
Silver ('000 oz)	2,506			165	181	167	138	140	187	123	129	142	149	142	130	143	136	130	140	163		
Payabel Metals Net of Deductions				0																		
Copper ('000 lb)	1,492,163			97,989	107,990	99,633	82,218	83,330	111,446	73,519	76,949	84,243	88,916	84,585	77,680	84,936	80,707	77,619	83,239	97,166		
Silver ('000 oz)	16,635			1,014	958	1,050	699	817	942	616	722	864	1,064	1,278	836	935	972	1,134	1,252	1,482		
Value of Payables				0																		
Copper ($ million)	$4700			$309	$340	$314	$259	$262	$351	$232	$242	$265	$280	$266	$245	$268	$254	$245	$262	$306		
Silver ($ million)	$299			$18	$17	$19	$13	$15	$17	$11	$13	$16	$19	$23	$15	$17	$17	$20	$23	$27		
Net Revenue from Sales																						
Copper ($ million)	$4700			$309	$340	$314	$259	$262	$351	$232	$242	$265	$280	$266	$245	$268	$254	$245	$262	$306		
Silver ($ million)	$284			$17	$16	$18	$12	$14	$16	$11	$12	$15	$18	$22	$14	$16	$17	$19	$21	$25		
Total Revenue from Sales ($ million)	$4985			$326	$357	$332	$271	$276	$367	$242	$255	$280	$298	$288	$259	$284	$271	$264	$284	$331		
OFF SITE COSTS																						
Off-site Unit Costs Copper ($ million)																						
Transport and Concentrate Loadout	$122			$8	$9	$8	$7	$7	$9	$6	$6	$7	$7	$7	$6	$7	$7	$6	$7	$8		
Ocean Freight	$108			$7	$8	$7	$6	$6	$8	$5	$6	$6	$6	$6	$6	$6	$6	$6	$6	$7		
Treatment Charges Copper	$175			$12	$13	$12	$10	$10	$13	$9	$9	$10	$10	$10	$9	$10	$9	$9	$10	$11		
Concentrate Loss	$4			$	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$		
Insurance	$5			$	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$		
Refining Charges Copper	$97			$6	$7	$6	$5	$5	$7	$5	$5	$5	$6	$5	$5	$6	$5	$5	$5	$6		
Refining Charges Silver	$5			$	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$		
Total Copper Concentrate Off-site Charges ($ million)	$516			$34	$37	$34	$28	$29	$38	$25	$27	$29	$31	$29	$27	$29	$28	$27	$29	$34		

CROWN MINING CORP.
MOONLIGHT PROJECT PRELIMINARY ECONOMIC ASSESSMENT
ECONOMIC MODEL in $million unless stated otherwise

MINE PRODUCTION SCHEDULE	Total/ Average	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19 to Year 30
NET PROJECT INCOME																						
Net Revenue from Concentrate Sales ($ million)	**$4468**			$292	$319	$297	$243	$248	$329	$217	$228	$251	$267	$259	$232	$254	$243	$237	$255	$298		
Project Income Value ($ million)	**$1643**			$147	$150	$133	$73	$76	$177	$47	$53	$75	$92	$91	$66	$84	$75	$72	$94	$139		
CAPITAL INVESTMENT $millions																						
Direct Costs	**-$314**	-$79	-$235	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$		
Indirect Costs	**-$188**	-$29	-$159	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$	$		
Sustaining Capital Costs	**-$97**	$	$	$	$	-$9	$	$	-$9	$	-$20	-$23	$	-$12	-$14	$	$	-$9	$	$		
Other Non-operational Costs	**-$193**	$	-$11	-$5	-$6	-$6	-$7	-$7	-$7	-$7	-$7	-$7	-$7	-$7	-$7	-$7	-$7	-$7	-$7	-$7	$16	-$76
Working Capital (US$)	**$**	$	$	-$36	$8	$15	$	$	$2	-$1	-$1	$	$	$1	$	$	$	$	$	$	$13	$
	-$792	-$108	-$405	-$42	$2	-$1	-$8	-$8	-$15	-$9	-$28	-$30	-$7	-$19	-$21	-$8	-$7	-$17	-$7	-$7	$29	-$76
SUMMARY OF OPERATING CAH FLOW - Pre-TAX		$																				
Net Annual Cash Flow ($ million)	**$851**	-$108	-$405	$105	$152	$132	$65	$68	$162	$38	$25	$45	$84	$72	$45	$76	$68	$56	$87	$132	$29	-$76
PRE-TAX CASH FLOW ANALYSIS																						
NPV 8% ($ million)	**$237**																					
IRR Pre-tax (%)	**16.4%**																					
Payback Period	**4.87**																					
SUMMARY OF OPERATING CAH FLOW - Post-TAX																						
Federal Taxes ($ million)	**-$78**	$	$	-$3	-$3	-$2	$	-$1	-$5	-$1	-$1	-$1	-$6	-$7	-$4	-$6	-$6	-$6	-$8	-$16	-$2	$
California Taxes ($ million)	**-$65**	$	$	-$6	-$6	-$5	-$1	-$1	-$8	$	$	-$1	-$5	-$5	-$3	-$4	-$3	-$3	-$5	-$8	$	$
Post-tax Cash Flows ($ million)	**$708**	-$108	-$405	$96	$143	$125	$63	$66	$149	$37	$24	$42	$74	$60	$38	$66	$59	$47	$74	$108	$27	-$76
PRE-TAX CASH FLOW ANALYSIS																						
NPV 8% ($ million)	**$179**																					
IRR Pre-tax (%)	**14.6%**																					
Payback Period	**5.13**																					

CHRIS JOHNS, P.ENG.

I, Chris Johns, P.Eng., of Kelowna, British Columbia, do hereby certify:

- I am a Geological Engineer with Tetra Tech Canada Inc. located at Suite 150, 1715 Dickson Avenue, Kelowna, British Columbia, V1Y 9G6.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Moonlight Deposit, Moonlight-Superior Copper Project, California, USA" dated March 2nd, 2018 (the "Technical Report").
- I am a graduate of Queen's University (B.Sc, Geological Engineering, 1994) and University of Alberta (M.Sc., Environmental Engineering, 1999). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#39423) and Alberta (#60174), and a Chartered Professional Engineer with the Institution of Engineers Australia. I have over 20 years experience in geological engineering and have been involved with tailings storage facility design from scoping study through feasibility and construction stage. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not completed a personal inspection of the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.14.1, 18.6, and 26.7.1 of the Technical Report.
- I am independent of Crown Mining Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of April 2018 at Kelowna, British Columbia.

"Original document signed and sealed by
Chris Johns, P.Eng."

Chris Johns, P.Eng.
Geological Engineer
Tetra Tech Canada Inc.

HASSAN GHAFFARI, P.ENG.

I, Hassan Ghaffari, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Director of Metallurgy with Tetra Tech Canada Inc. located at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Moonlight Deposit, Moonlight-Superior Copper Project, California, USA" dated March 2nd, 2018 (the "Technical Report").
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#30408). My relevant experience includes 27 years of experience in mining and plant operation, project studies, management, and engineering. As the lead metallurgist for the Pebble Copper/Gold Moly Project in Alaska, I am coordinating all metallurgical test work and preparing and peer reviewing the technical report and the operating and capital costs of the plant and infrastructure for both the scoping and prefeasibility studies. For the Ajax Copper-Gold Project in BC, I was the Project Manager responsible for the process, infrastructure and overall management of the 60,000 t/d mill. As well, I was the Project Manager responsible for ongoing metallurgical test work and technical assistance for the La Joya Project Copper/Silver/Gold Project in Durango, Mexico. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property that is the subject of this Technical Report was November 16th, 2017 for one day.
- I am responsible for Sections 1.1, 1.2, 1.10, 1.14, 1.15, 1.16.1, 1.18, 2.0, 3.0, 4.0, 5.0, 13.0, 18.0, 20.0, 21.1, 24.0, 25.2, 25.5, 25.6, 25.7, 26.1, 26.5, 26.7, 26.8, and 27 of the Technical Report.
- I am independent of Crown Mining Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of April 2018 at Vancouver, British Columbia.

"Original document signed and sealed by Hassan Ghaffari, P.Eng."

Hassan Ghaffari, P.Eng.
Director of Metallurgy
Tetra Tech Canada Inc.

JIANHUI (JOHN) HUANG, PH.D., P.ENG.

I, Jianhui (John) Huang, Ph.D., P.Eng., of Coquitlam, British Columbia, do hereby certify:

- I am a Senior Metallurgist with Tetra Tech Canada Inc. located at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Moonlight Deposit, Moonlight-Superior Copper Project, California, USA" dated March 2nd, 2018 (the "Technical Report").
- I am a graduate of North-East University, China (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals, China (M.Eng., 1988), and Birmingham University, United Kingdom (Ph.D., 2000). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#30898). My relevant experience includes over 35 years involvement in mineral processing for base metal ores, gold and silver ores, and rare metal ores. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not completed a personal inspection of the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.13, 17.0, 19.0, 21.2.3, 21.2.4, 21.2.5, 25.4, 26.6, and 27.0 of the Technical Report.
- I am independent of Crown Mining Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of April 2018 at Vancouver, British Columbia.

"Original document signed and sealed by Jianhui (John) Huang, Ph.D., P.Eng."

Jianhui (John) Huang, Ph.D., P.Eng.
Senior Metallurgist
Tetra Tech Canada Inc.

MARK HORAN, P.ENG.

I, Mark Horan, P.Eng., of North Vancouver, British Columbia, do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech Canada Inc. located at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Moonlight Deposit, Moonlight-Superior Copper Project, California, USA" dated March 2nd, 2018 (the "Technical Report").
- I have a BSc. Mining Engineering degree from the University of Witwatersrand, South Africa and a MSc. from Rhodes University, South Africa. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#170768). I have 18 years' experience including working in precious and base metal operations and in consulting. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property that is the subject of this Technical Report was on November 16th, 2017, for one day.
- I am responsible for Sections 1.12, 1.16.1, 1.17, 3.0, 15.0, 16.0, 21.2.1,21.2.2, 22.0, 25.3, 25.8, 26.4, and 27.0 of the Technical Report.
- I am independent of Crown Mining Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of April 2018 at Vancouver, British Columbia.

"Original document signed and sealed by
Mark Horan, P.Eng."

Mark Horan, P.Eng.
Senior Mining Engineer
Tetra Tech Canada Inc.

DONALD E. CAMERON, M.SC., SME

I, Donald E. Cameron, M.Sc., SME, of Harrison, Idaho, USA, do hereby certify:

- I am a Geologist with Cameron Resources Consulting LLC., located at 27357 S Highway 97, Harrison, Idaho 83833.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Moonlight Deposit, Moonlight-Superior Copper Project, California, USA" dated March 2nd, 2018 (the "Technical Report").
- I am a graduate of the University of Wisconsin, Madison (Bachelor of Arts, Geology, 1974) and University of Arizona, Tucson (Masters of Science, Geology, 1976). I am a Registered Member in good standing of the Society for Mining, Metallurgy and Exploration, Inc. (2012-Present). I have practiced my profession since 1976 and have been employed as exploration and mine geologist for copper, base and precious metals in many locations worldwide and for several companies, major and junior. My training and experience includes resource estimation. I have engaged in my consulting practice since 2012, principally in the areas of grade control and resource estimation for exploration projects and operating mines.
- As a result of my experience and qualifications I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property that is the subject of this Technical Report was from September 26-27, 2017.
- I am responsible for Sections 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 1.11, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 14.0, 23.0, 25.1, 26.1, 26.2, 26.3, and 27.0 of the Technical Report.
- I am independent of Crown Mining Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of April 2018 at Harrison, Idaho, USA.

"Original document signed by
Donald E. Cameron, M.Sc., SME"

Donald E. Cameron, M.Sc., SME
Principal
Cameron Resources Consulting LLC

Exhibit B

https://www.researchgate.net/publication/270457097_4500-year-old_mining_pollution_in_Southwestern_Spain_Long-Term_implications_for_modern_mining_pollution/link/561e73a408ae50795afefc10/download?_tp=eyJjb250ZXh0Ijp7ImZpcnN0UGFnZSI6InB1YmxpY2F0aW9uIiwicGFnZSI6InB1YmxpY2F0aW9uIn19

4500-year-old mining pollution in Southwestern Spain: Long-Term implications for modern mining pollution

Article *in* Economic Geology · May 2000
DOI: 10.2113/gsecongeo.95.3.655

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4,500-YEAR-OLD MINING POLLUTION IN SOUTHWESTERN SPAIN: LONG-TERM IMPLICATIONS FOR MODERN MINING POLLUTION

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Abstract

The Tinto river drains the Rio Tinto mining district, which comprises the world's largest known massive sulfide deposits; these orebodies have been mined from the third millenium BC to the present. The Tinto river is strongly acidic (pH, 1.5–2.5); during flood events, it transports a sandy material, including abundant detrital pyrite grains. A core drilled in the Holocene sediments of the Tinto estuary allows for investigation of recent and historical mining pollution. Two anomalous horizons have been recognized (0–1.3 m; 3–4 m). Both are characterized by very high metal content (100 times over the background) and by the presence of abundant clastic pyrite grains. The metal association (Pb, Ba, As, Cu, Zn, Sn, Tl, Cd, Ag, Hg, Au) is typical of that of the Rio Tinto pyritic ore. The upper horizon corresponds to the modern mining activity; the lower horizon has been dated at 2530 BC (^{14}C AMS calibrated age).

We show here that active mining occurred early (Copper Age) in the Rio Tinto area, resulting in a watershed-scale metal contamination. We also show that anthropogenic input of metals may be accumulated and immobilized during thousands of years in estuarine sediments.

Introduction

Mining activity is a major source of metal contamination by toxic metals released into surface waters. Renewed interest in the impact of mining followed the recent accident at Aznalcollar, Spain. On April 25, 1998, a tailings spill from the mine of Aznacollar, in the southern Iberian pyrite belt, released about 7×10^6 m^3 of sulfide sludge in a tributary of the Guadalquivir river (Van Geen and Chase, 1998), which drains the Donana national park, the most important wildlife reserve of UNESCO in Europa (Fig. 1). The sulfide sludge contained a mixture of acidic waters (pH, 2–3.5) and very fine sulfide material (<30 μm) dominated by pyrite, with about 2 percent Zn, 0.9 percent Pb, 0.6 percent As, 0.2 percent Cu (dry material) and abundant traces of other toxic metals such as Tl, Hg and Cd (Table 1). The flood wave was toxic for plankton, benthos, fish, and crab populations in the river. One week after the spill, extremely elevated Zn concentrations (0.6–1.2%) were found in the river sediments downstream from the mine over a distance of 40 km (Van Geen and Chase, 1998).

There is much archeological evidence of ancient mining in the Iberian pyrite belt; it is well known, for example, that the Rio Tinto orebodies have been mined at various times since the third millennium BC (Briard, 1976; Rothenberg and Blanco Freijero, 1980). The aim of this study was to investigate the impact of ancient mining, at a watershed-scale, and to compare ancient mining contamination with modern mining release. This should allow predictions of the long-term behavior of modern metal-contaminated sediments in this area.

The Rio Tinto Watershed

The southern Iberian pyrite belt, which belongs to the southern part of the Iberian Variscan orogenic belt, is the largest repository of volcanogenic massive sulfide deposits in the world. The pyrite belt includes more than 80 known deposits that are hosted in a Late Visean volcano-sedimentary sequence. These massive pyrite deposits contain (mined and reserves) about 32 Mt Zn, 13 Mt Cu and 11 Mt Pb (metal tonnages, Leistel et al., 1994).

The Rio Tinto massive sulfide district is the biggest in its class. The Rio Tinto district comprises more than 10^9 t of massive pyrite ore. These super-giant deposits have abundant base metal sulfides (Zn, Cu, Pb) and associated trace metals (Cd, As, Tl, Sn, Hg, Ag and Au; Table 1). The deposits have also had an extensive mining history. The Rio Tinto deposit has been mined since the Copper Age, then during Tartessian and Phoenician times (1200–500 BC), with greatest amount of activity taking place during the Roman period (Flores, 1981). Mining started again in the last part of the nineteenth century and has continued to the present. These successive mining activities, from the Copper Age until the present day, have exploited outcropping and near-surface pyritic orebodies, leaving wide volumes of pyrite-rich waste rocks and mining wastes.

Acid mine drainage resulting from the oxidation of pyrite is especially important in the Rio Tinto mining district. The headwaters of the Tinto river are in the area of intense mining, which includes wide stockpiles of pyrite-rich wastes and retention ponds of acid mine

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0361-0128/00/3055/655-8 $6.00

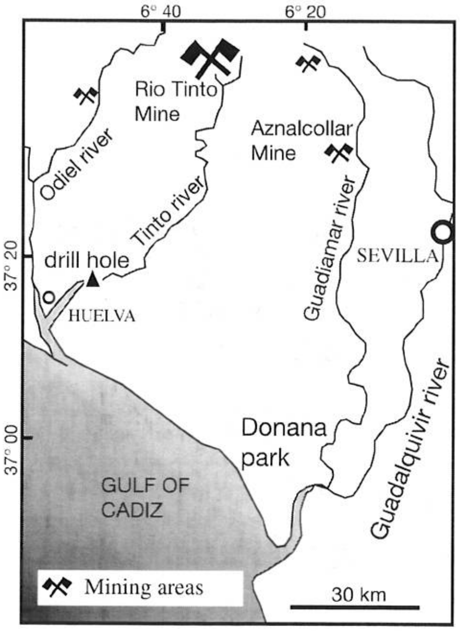

Fig. 1. Sketch map (southwest Spain) showing the location of the Rio Tinto and of the Aznalcollar mining areas and the location of the core drilled in the upper estuary of the Tinto river (Huelva ria).

drainage waters. The name of the Tinto river ("tinto " means "red wine" in Spanish) clearly refers to the uncommon brown-red color of its waters.

The Tinto river, which is 90 km long, remains strongly acidic (pH, 1.5–2.5) from its source zone, about 400 m elev, down to its estuary in the Ria of Huelva (Fig. 1). Its red-colored waters contain high sulfate and dissolved metal contents (Nelson and Lamothe, 1993; Elbaz-Poulichet et al., 1999). The mean discharge is relatively small—about 15 m^3/s, ranging from 1 to 100 m^3/s depending on seasonal variations, including dry periods and rainy periods with floods. The Tinto river sediments are gray sands, including quartz and slate elements and abundant detrital pyrite grains that are weakly weathered and slightly rounded.

Methods and Materials

A core was drilled (lat. 37°18'16", long. 6°48'10"), down to the bedrock, through the Holocene sediments of the upper part of the Rio Tinto estuary (Fig. 1). This zone corresponds today to a flood plain that is usually dry, being 1.15 m above the mean high-water level. The core, 7 cm diameter, is about 15 m in length. Core recovery was relatively good (92%) and the core material was only moderately disturbed and fragmented; consequently, there is no uncertainty in depth control. Core material was protected in a PVC sheath. It was sawed longitudinally in four parts for lithological, geochemical, and dating studies, and the last part was kept as reference. Lithology was studied both using optical microscopy and scanning electronic microscopy to investigate the sulfide phases. Twenty samples of core material (30–50 g dry material) were selected for major and trace elements analysis (Fig. 2).

Present sediments from the Tinto river were collected randomly within the uppermost 5 cm. They were dried before examination by SEM, then analyzed (30–50 g) for major and trace elements. The sulfide sludge from Aznalcollar was collected along the banks of the Guadiamar river one day after the spill.

The geochemical analyses for major (including S, CO_2,and organic C) and trace elements (including Cl and Hg) were done by X-RAL laboratories (Don Mills, Ontario, Canada) and at the Montpellier University, using XRF (X-ray fluorescence), NAA (neutron activation analysis), ICP-MS (inductively coupled plasma-mass spectrometry) and AA (Atomic Absorption) spectrophotometry.

The SEM investigations were performed with a Hitachi S-4500 instrument coupled with an energy-dispersive X-ray spectrometer (EDS); detection limits were about 0.1 percent, with a precision within 20 percent.

Activation mass spectrometry (AMS) radiocarbon dating was performed by Beta Analytic, Inc. (Miami). The analyzed sediment samples (35–75 g) contained enough organic carbon (0.5–1%) to ensure accurate analysis and all analytical steps went normally (graphitization and AMS radiocarbon counting); a charcoal fragment (4.3 g) was picked for complementary analysis. The conventional ^{14}C ages were calibrated to calendar years using the Pretoria Calibration Procedure based on tree-ring data as calibration curves (dendrocalibration); the calibrated ages are given BC ages with 95 percent probability.

The ^{210}Pb determinations were done on the uppermost 30 cm of the core in order to determine the chronology of pollu-

TABLE 1. Metal Contents of the Two Anomalous Horizons of the Core and Comparizon with the Normal Estuarian Sediments of the Core

	Zn	Cu	Pb	As	Cd	Sn	Ag	Tl	Hg	Au	Ba
Rio Tinto massive sulfide ore (avg)[1]	20,000	7,000	7,000	2,000	150	350	45	35	40	0.8	Unknown
Pyritic tailings of Aznalcollar (spill)	21,200	2,120	8,500	6,100	31	22	50	103	Unknown	0.06	70
Pyrite-rich sand from the Tinto river	3,200	950	1,200	3,900	57	20	14	24	12	0.07	2,900
Upper pyrite-rich sand horizon (core)	300	760	5,300	1,400	9	100	17	18	5.1	0.2	3,400
Lower pyrite-rich sand horizon (core)	240	400	2,500	900	6	45	10	12	3.0	0.1	1.600
Normal estuarian black mud (core)	67	24	7	12	<1	2	0.9	0.4	0.04	0.003	230

All values given in ppm

Metals concentrations in the sands from the upper part of the Tinto river, in Rio Tinto massive pyrite ore, and in pyrite sludge released from the Aznalcollar spill are given for comparizon with the anomalous horizons of the core

[1] Leistel et al. 1993

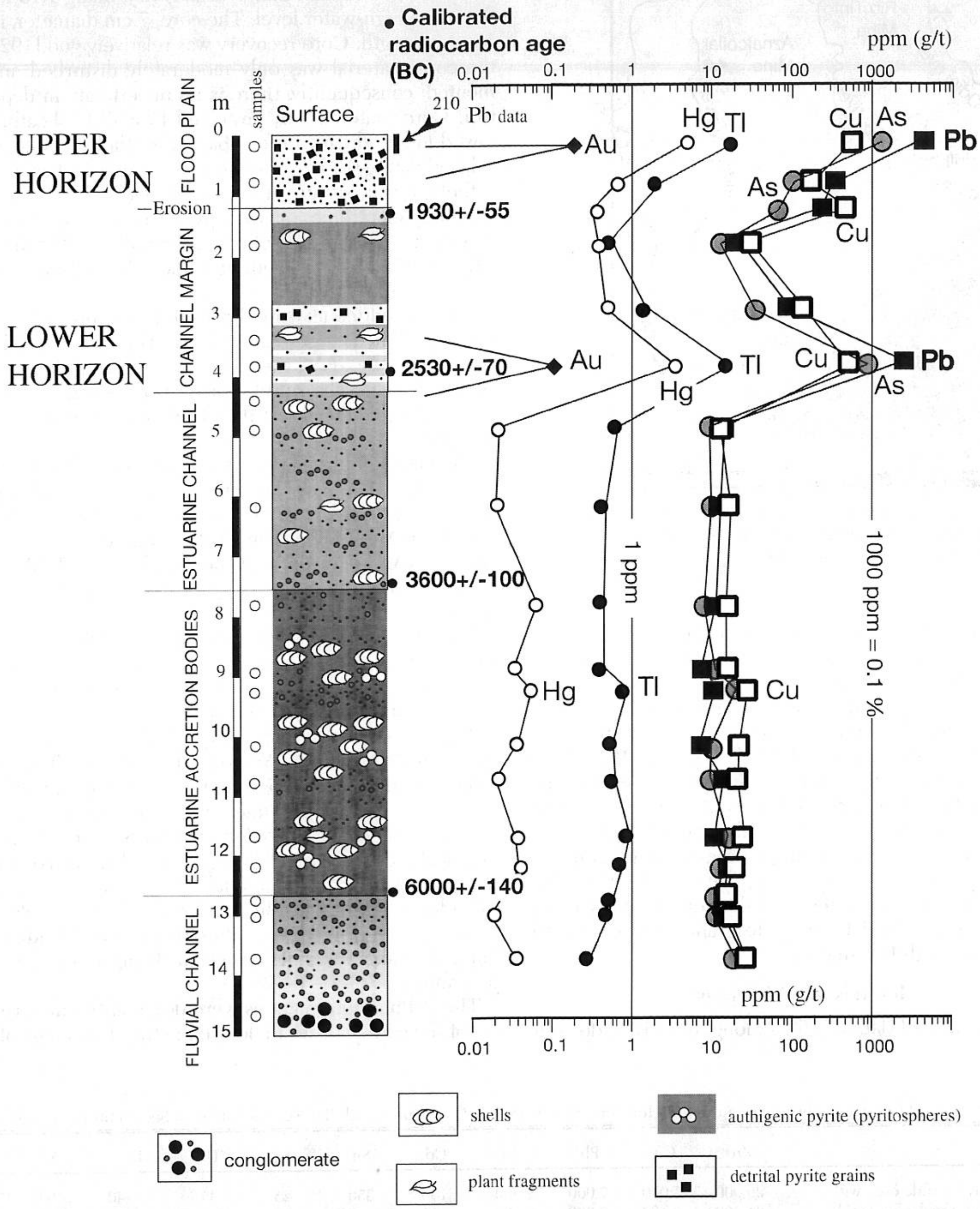

FIG. 2. Description, age dates and metal concentrations of the core. The lithostratigraphic sequence corresponds to an estuarine evolution, ending with an erosional hiatus (flood plain). The ^{14}C AMS radiocarbon dates (given as calibrated BC ages) agree with an Holocene transgressive cycle; the ^{210}Pb data from the uppermost part indicate present sediments. Two horizons show metal concentrations (ppm) that are two orders of magnitude over background. Both metal-contaminated horizons include abundant clastic pyrite grains.

tion associated with modern mining (Davis et al., in press); analyses were conducted at Florida State University by W. Burnett and associates.

Results

Lithostratigraphy

From bottom to top, the following materials are present in the core (Fig. 2): (1) coarse detrital sediments (fluvial channel and fluvial bar); (2) shelly and sandy black muds, including shell-rich horizons with authigenic pyritic nodules (estuarine accretion bodies); (3) muddy sands with shell fragments (estuarine channel); (4) alternating yellow sands and dark green muds (channel margins); and (5) yellow sands of the flood plain at the top of the core. This uppermost horizon results from flood deposits that may be strongly erosional, as suggested by the lithologic break and the sharp discontinuity with the underlying muddy horizon. Almost every year there is a major flood from the Tinto river, eroding and/or depositing up to 50 cm of sandy material on the flood plain.

Presence of metal-rich horizons

The trace metal contents in the Holocene estuarine sediments (Fig. 2, Table 1) are similar to averaged continental sediments (Taylor and McLennan, 1985). The highest contents are in organic carbon-rich sediments containing diagenetic pyrite (Fig. 3D) overgrowing plant debris or shell fragments. Against this normal geochemical background two remarkable horizons (0–1.3 m and 3–4 m) are characterized by metal concentrations that are two orders of magnitude higher than those of the other layers (Fig. 2). These horizons contain 2,500 to 5,300 ppm Pb and 900 to 1,400 ppm As, respectively. In both cases the same metal association is present, composed of high Pb, Ba, As-Cu, Zn-Sn, Tl-Cd, Ag, Hg-Au, in decreasing order of importance.

SEM observations (Fig. 3)

The two anomalous horizons are also remarkable for their mineralogic composition. They consist mainly of light yellow sands and silts, including abundant clastic pyrite grains (2–12 wt %). The pyrite grains are small and well sorted (20–50 μm); they correspond to angular fragments of subhedral pyrite grains (Fig. 3A2) that have been only slightly rounded, and which exhibit only rare dissolution pits and cracks. The only oxidized material consists of ochre fragments in the silt layers. EDS-SEM investigations suggest that galena is present as small accessory grains (1–5 μm), partly included in pyrite; rare gold inclusions (0.5 μm) are also present in pyrite. The high barium content is clearly explained by the presence of lamellar fragments of barite in the pyrite-rich sands. Cassiterite is present as small, perfectly euhedral crystals (10 μm), explaining the high Sn contents (40–100 ppm Sn).

The lower horizon (0.5–1.2% organic carbon) contains black plant fragments that often display woody cellular textures (Fig. 3C). These charcoal fragments are very small and well sorted (0.1–1.2 mm). A few small globules (30–500 μm) of vesicular glass, with smooth surfaces, are also present in this horizon (Fig. 3B). EDS-SEM analysis suggests they consist either of a Fe-Si glass, with traces of sulfur, or of a carbon-iron material with small contents of copper and sulfur (0.1–1%). These compositions, which differ from those of natural vesicular glasses, such as lavas, are similar to those of scorias and slags from metallurgical furnaces.

Dating results

The four ^{14}C calibrated ages (BC) obtained are consistent with the relative stratigraphic position of the analyzed samples (Fig. 2) : 6,000 ±140 yr for the base of the estuarine accretion bodies (12.5 m); 3,600 ±100 yr for the base of the estuarine channel (7.5 m); 2,530 ±70 yr for the lower metal-contaminated horizon (4 m) and 1,930 ±–55 yr for the floor of the uppermost metal-contaminated horizon (1.3 m).

The ^{210}Pb concentrations along the uppermost 30 cm of the core (Davis et al., in press) are strikingly constant and relatively high (8 ± –2 Bq/kg).

Discussion

Evolution of the Holocene depositional environment

The lithostratigraphic sequence and the ^{14}C ages correspond fairly well to the Holocene transgression that started in the Huelva area about 8,000 BC, filled up the estuary, and ended with a stabilization of the sea level about 3,000 BC (Borrego et al., 1999). The transgression is connected with a deglacial sea-level rise (Mannion, 1997). From ^{14}C radiodating, it appears the sedimentation rates of the estuarine sediments were between 1 and 7 mm/yr.

The two metal-contaminated horizons correspond to well-sorted sandy flood deposits. The lower horizon results from input of fluvial sands during a progradation stage in the estuarine system; the overlying muddy and shelly horizon corresponds to tidal sediments along channel margins. The upper horizon results from discontinuous input of fluvial sands over the surface of the flood plain—which is usually dry—during seasonal floods.

Geochemical and mineralogical evidence for metal contaminations from the Rio Tinto mineralization

The metals present in these two anomalous horizons reflect fairly well those of the Rio Tinto sulfide ore, including base and trace metals (Table 1). For example, the relatively high Au content of the pyritic horizons (0.1–0.2 ppm) is in agreement with the presence of gold in the Rio Tinto mineralization (0.5–1.5 ppm); SEM observations reveal that gold inclusions are present in the detrital pyrite grains. The abundance of barium, and the presence of barite detrital grains, may be explained by the fact that barite is a common gangue mineral of the sulfide ores. The relatively high Sn concentrations and the presence of cassiterite grains are in agreement with the presence of cassiterite in the Rio Tinto ore. The arsenic concentrations in detrital pyrite grains, which have been picked up from the core, range from 1 to 2 percent As, explaining the high arsenic contents of the pyrite-rich horizons.

However, the order of abundance is not exactly the same. For example, Zn and Cd concentrations are low compared to the other base metals (Pb, Cu) in the pyrite-rich sands. This may be explained either by the fact that Zn and Cd are relatively more easily soluble in surface waters or that sphalerite was not abundant in the transported pyritic material.

The pyrite grains from the two metal-contaminated hori-

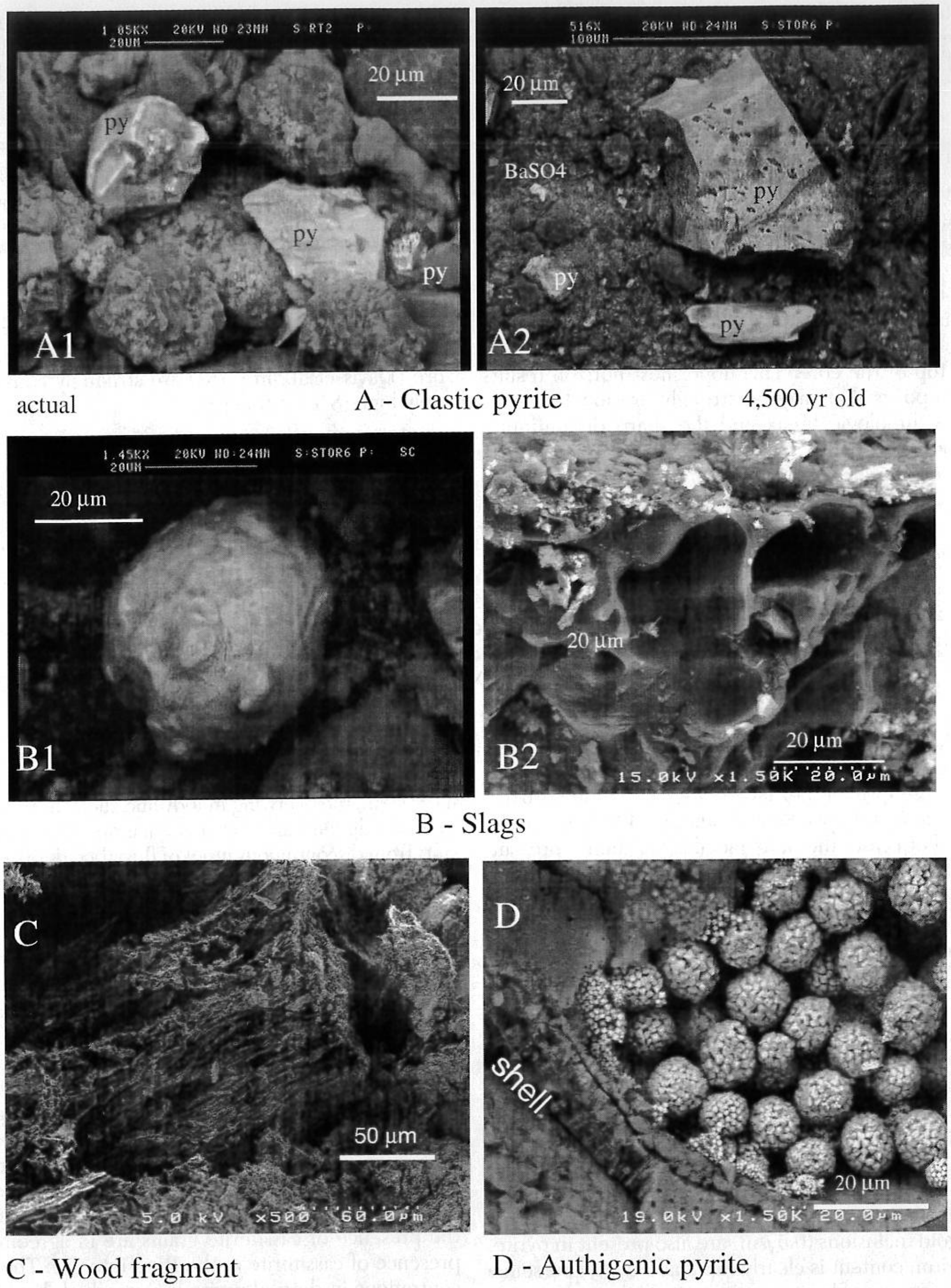

FIG. 3. Scanning electronic microscopic (SEM) images. A. (A1) Clastic pyrite grains, displaying subhedral to angular shapes, from the present sands of the Tinto river. (A2) Pyrite grains from the lower metal-contaminated horizon of the core (dated at 2530 ±70 BC) are similar in shape and size to the clastic pyrite grains from the present sands of the Tinto river (A1); note the presence of a barite fragment. B. (B1) Slag droplet from the lower metal-contaminated horizon; its composition is that of a Si-Fe glass with minor contents of sulfur (0.1–1% S). (B2) Vesiculated slag fragment, from the lower metal-contaminated horizon, showing a C-Fe-O composition with minor contents of copper, sulfur and silica (0.1–1%). The analytical data were performed using an energy-dispersive X-ray spectrometer (EDS). C. Wood fragments (charcoal) are relatively abundant (1%) in the lower metal-contaminated horizon (dated at 2530 ±70 BC) D. Authigenic pyrite crystallizing as agregates of pyritospheres in shell or plant fragments from the lower part of the core (black muddy estuarine sediments); note the difference in shape and size compared to the detrital pyrites (A1, A2).

zons are angular clastic pyrite grains (Fig. 3A2) that may be slightly rounded and corroded. They are clearly different in shape and size from the authigenic pyrite crystals and the spherulitic agregates of pyrite (Fig. 3D) that are present in the shelly black mud horizons of the core (Fig. 2). The only obvious source of pyrite in the catchment zone of the Tinto river is the Rio Tinto mining area. There are outcropping massive pyrite orebodies, with subhedral pyrite grains similar in shape and size to those of the anomalous horizons of the core, and huge stockpiles of pyrite-rich tailings and wastes from modern mining activity. The pyrite grains are very abundant in the present surface sands collected along the bed and the banks of of the Tinto river: in the immediate vicinity of the mine area, there are pyritic sands containing up to 60 wt percent pyrite, and downstream from the coring location, the estuarian sediments still contain 1 to 10 wt percent pyrite. The pyrite-rich sediments of the Tinto river display very high concentrations of toxic metals (0.5% As, 0.5% Pb, 0.3% Zn, 0.2% Cu; Table 1). These high metal contents can be ascribed to pyrite (this is the case for As) or to discrete Pb-Zn-Cu sulfide phases associated with or included in pyrite. The clastic pyrite grains from the surface sands along the Tinto river are similar in shape and size (Fig. 3A1) to those from the metal-contaminated horizons at depth in the core, providing evidence that pyrite grains may be transported by the Tinto river from the Rio Tinto mining zone to the estuarine zone. Considering the hydric flow during seasonal flood events and the average geometry of the Tinto river, the rate of sediment transport can be roughly calculated: the time for the transportation of the pyrite grains—from the source zone to the estuary—may be from 15 to 45 hours. Consequently, the pyrite grains may be deposited very quickly in the estuarine sediments without having suffered any weathering during their transportation. The same shape and size of pyrite grains characterize the pyritic sludge released within a few hours by the Aznalcollar tailings spill in the Donana national park, 40 km downsream (Fig. 1).

These geochemical and mineralogical observations are the first indication that the two anomalous horizons correspond to input of pyrite-bearing and metal-rich sands resulting from mining activity in the Rio Tinto source region.

Age of the upper metal-contaminated horizon

The impact of intensive modern mining activity that started 130 yr ago has been clearly recorded in shelf surface sediments of the Gulf of Cadiz (Van Geen et al., 1997). The upper metal-contaminated horizon of the core may correspond to this modern mining. The uppermost 30 cm of the core have high and constant ^{210}Pb concentrations; this means that the upper part of the upper contaminated horizon was deposited a short time ago, probably during recent flood events. However, considering the discontinuous sedimentary and/or erosional history of the flood plain of the upper estuary, we are not sure that this 1.3-m-thick pyrite-rich horizon corresponds in its entirety to the modern mining period. A ^{14}C AMS radiocarbon dating was performed on an ochre layer, just below the upper horizon (Fig. 2). The ochre has an age of 1930 ±50 BC (calibrated age). This is consistent with the chronostratigraphy of the core but indicates that the upper part of the Holocene sequence has been eroded before or during the deposition of the upper, metal-contaminated horizon.

Age of the lower metal-contaminated horizon

The lower horizon has been dated at 2530 ±70 BC (AMS ^{14}C calibrated age). The analyzed sample (pyrite-rich sand) contains 1 percent organic carbon. Tiny black fragments of charcoal are the only possible organic carbon source; there are no shell fragments or carbonate (<0.1%). Dating of a single charcoal material, picked up 10 cm below the first dated sample, has given a 500-yr older age (3015 ±70 yr BC), which could indicate derivation from a 500-yr-old tree ("old wood effect") or material derived from an older layer. Although the ancients may have been burning old wood in their furnaces, this is unlikely to have significantly affected the observed ^{14}C stratigraphy of the core. The logical progression of ^{14}C dates down the core suggests that resedimentation processes in the estuary have not resulted in major disturbances in in the chronostrtigraphy.

These ages correspond to the Copper Age in the western Mediterranean area and confirm that active mining started early in the Rio Tinto district.

The presence of small droplets and fragments of likely slags (vesicular glasses with Fe-Si or C-Fe compositions and up to 0.5% copper and sulfur) in the lower horizon (Fig. 3B), is compelling evidence for contemporaneous metallurgical activity. In the same way, the presence of tiny and well-sorted charcoal fragments may reflect the common use of small charcoal fragments during metallurgical treatments.

Copper Age mining and metallurgy in the Rio Tinto area

The oldest findings indicate that metallurgical activities in the region date from 2700 BC (Rothenberg and Blanco Freijero, 1980). Except for a few metal tools in some graves and scarce traces of mining excavations and ovens, there has been little evidence of important Copper Age mining activity in the Rio Tinto district. However, the Almerian Copper Age civilization (3000–2200 BC) is well known in eastern Andalusia, Spain, for the important development of copper mining and metallurgy (i.e., in the fortified site of Los Millares, Almeria). Similar activity was likely taking place in western Andalusia, notably in the Rio Tinto area (Briard, 1976). Unfortunately, the subsequent mining periods probably erased most of the Chalcolitic mining and metallurgical works. The Romans started their mining activity from the Tartessian-Phoenician works, and active mining today recovers gold (1–1.5 ppm) from the Roman mining wastes.

Conclusions

1. We show here a new record of watershed-scale impact of early mining, over a distance of about 100 km. A 4,500-yr-old (2530 BC) metal contamination, caused by Copper Age mining, has existed in southern Spain. Notwithstanding the recent accident at Aznalcollar, it is possible that long-term release of metals from ancient mining operations that have not received the benefit of modern remediation may be a more serious problem than the impact of much larger, modern-day operations.

2. Anthropogenic input of metals may remain immobilized for millennia in estuarine sediments. Most metals can be locked as sulfides in estuarine sediments where anoxic condi-

tions (organic matter, fast sedimentation rates) can enhance the formation of authigenic sulfides and/or prevent the oxidation of detrital sulfides. Considering the recent spill of pyrite tailings at Aznalcollar, these findings may have implications for modern mining. Part of the sulfide-rich material recently discharged into the Guadiamar river (Fig. 1) might remain stored in the Guadalquivir estuarine sediments for millenia. Precautions must be taken to prevent any change in the estuarine system, particularly oxidation (by draining, dredging, or erosion) of these potentially highly toxic or deleterious materials.

3. Sulfide grains can be quickly transported far away from their source zone, during flood events, without having suffered weathering. This kind of metal transportation in surface waters, often neglected, can be of great importance, locally.

Acknowledgments

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Exhibit C

https://www.sfgate.com/news/article/Iron-Mountain-a-study-in-perpetual-pollution-3092377.php

NEWS

Iron Mountain - a study in perpetual pollution

By **Jane Kay**, *EXAMINER ENVIRONMENTAL WRITER*
Nov 2, 1997

1997-11-02 04:00:00 PDT SHASTA COUNTY, CALIFORNIA -- KESWICK, Shasta County - Hidden in the southeastern foothills of the Klamath Mountains, abandoned piles of crushed ore and rock higher than a 70-story building leak rivulets of orange-and-green water into a tributary of the Sacramento River.

It's a virulent polluter, this acid drainage from Iron Mountain Mine nine miles north of Redding. The largest mine in California, it's been poisoning salmon and other downstream fish as far away as San Francisco for a century.

Scientists link higher amounts of copper in clams 200 miles away in San Francisco Bay to discharge from Iron Mountain.

The mine dumps an average of 180 pounds of copper daily to the Sacramento River near Lake Shasta. By comparison, all the factories and sewage treatment plants around the Bay together are allowed to dump only an average of 55 pounds a day, because copper is so toxic to plants and animals at the base of the food web.

Yet, some call Iron Mountain Mine a Superfund success story – thanks to the way the Environmental Protection Agency used the federal Comprehensive Environmental Response, Compensation and Liability Act of 1980 to reduce poisons flowing from this inactive mine that was one of the nation's worst toxic waste sites.

Three years ago, the U.S. EPA forced the mine's owners to build and operate a treatment plant for toxic runoff. The system, which geologists estimate will be needed for 3,000 years, neutralizes the acidic waters with lime and reduces the flow of toxic metals 80 percent.

Now the giant chemical company deemed responsible by the EPA for cleaning up this most concentrated acid mine drainage on earth is asking a federal court judge to rule that the government should bear the cost.

If it prevails, California taxpayers' share of the mine cleanup bill would total $9 billion, not counting inflation.

Company's legal strategy<

It's part of a new legal strategy from companies that have spent tens of millions of dollars fighting Superfund cleanup orders.

Rhone Poulenc, the multinational firm ordered to pay Iron Mountain cleanup costs, is one of dozens of corporations seeking to turn the nation's hazardous-waste cleanup program on its head. In the last five years, some federal judges have ruled in their favor.

The government had a role in the pollution, either by purchasing the mine's product or through acts of its own upstream, Rhone Poulenc says, and thus should pay for the cleanup.

U.S. District Court Judge David F. Levi ruled against some of company's most important motions in the case last week. Rhone Poulenc's lawyers haven't said whether they would appeal.

"This case has a long way to go," cautioned Tom Bloomfield, an attorney with the EPA's Region 9 in San Francisco.

Rhone Poulenc - 12 percent owned by the French government - seeks to recover $104 million it claims to have spent at Iron Mountain, including the cost of building the multimillion dollar treatment plant in 1994. It also wants the state to pay for perpetual cleanup at a cost of $3-4 million a year.

"They muddy the waters'<

Environmentalist Lois Gibbs - known as the "Mother of Superfund" because the law passed in the wake of her hometown activism at Love Canal in Niagara Falls - sees the flurry of counterclaims and defenses used by Rhone Poulenc as part of an emerging strategy around the country.

"These companies boggle the process," Gibbs said.

"They do that by hiring lots of lawyers. . .who find a whole lot of reasons why they couldn't pay. They muddy the waters. They know the suits that they file against the EPA and the state will go nowhere and are unlikely to win. They know it makes everybody nervous, and costs everybody money. Even though the suit has no merit, someone has to write up court documents. Someone needs to go to court and defend."

What happens in the Iron Mountain Mine case could open up routes for other companies fighting Superfund cleanup orders, U.S. Department of Justice and EPA lawyers predict.

"We're seeing it more and more," said Bloomfield of the EPA. "Companies are countersuing, and trying to make the federal or state government liable or lessen their own liability or persuade the government to pick a less expensive cleanup."

Richard Lazarus, a professor of environmental law at Georgetown University Law Center, says the tide appears to be turning away from government.

In the 1980s and early 1990s, the government successfully argued that the Superfund law holds current and previous owners and dumpers liable for cleaning up active and abandoned hazardous-waste sites, said Lazarus.

"One by one, the defendants lost virtually all of these cases," he said.

But having failed in the courts to change Superfund and having failed in Congress to amend it, the companies now are trying to argue that the government, by its actions at a site, should share in the liability, Lazarus said.

"In the last few years," said Lazarus, "industry has started to win."

A day in court<

In a hot Sacramento courtroom last July, eight lawyers in dark pin-striped suits lined up to present arguments for Rhone Poulenc.

On the other side, two Department of Justice lawyers and two EPA lawyers stood for the federal government. A deputy attorney general sat ready to protect the interests of the State of California.

Rhone Poulenc, which says it's spent $16 million on legal fees in the last seven years on this matter, has already raised at least 21 defenses and dozens of counterclaims to the EPA's demand that it pay for the cleanup.

On this day, Paul Galvani, a Boston lawyer for Rhone Poulenc, was arguing before Judge Levi that the chemical firm should be able to review and overturn EPA's past cleanup solutions at the mine.

The EPA is "spending our money like drunken sailors," said Galvani, who represents at least five other sites on the Superfund list for his firm, Ropes and Gray.

For Rhone Poulenc, Galvani and his platoon of assistants claim that the U.S. government should share cleanup costs. Among their reasons:

* EPA's solutions exacerbated the problem and cost too much.

* Shasta Dam, built by the U.S. Bureau of Reclamation in the 1940s, holds back water, preventing natural dilution of the mining waste.

* The government bought metals from the mine during World War II and were on the site, and thus had a role in the pollution.

The EPA responds that just being a consumer of the metals mined at Iron Mountain does not make the government responsible for the pollution. There would have to be some actual control over the mine or a sharing in profits.

EPA lawyers say Shasta Dam doesn't cause pollution and points to massive fish poisonings caused by Iron Mountain long before the reservoir was built.

"The defendants are always looking for defenses. As the case law develops, then they decide on which ones to move forward," said Kathleen Johnson, EPA's acting hazardous waste branch chief in the office of the regional counsel.

Peter Taft, a Los Angeles lawyer who has been involved with nine Superfund cases, including some of the nation's worst, recalled how fighting back in court paid off for four oil companies at the McColl site, a Fullerton chemical waste landfill dating from the 1950s.

Shell Oil, Arco, Texaco and Unocal won't know until November how the judge will apportion cleanup costs at McColl. But while the companies paid legal fees of $15.8 million over 13 years, the cleanup bill dropped from as much as $400 million for removing and incinerating toxic garbage to a $24 million remedy of leaving it in the ground, surrounding it with a slurry wall and capping it, Taft said.

"We spent a lot of money," Taft said, "but we saved more than we spent."

Inside the mine<

In the glaring sunlight up at Iron Mountain Mine, state biologist Dennis Heiman sniffs the stench seeping from the collapsed red mountain where miners once took copper, pyrite, gold and silver.

"Even I can smell that," said Heiman, inured to the acrid odor of mine drainage but reacting to sulfur fumes from the copper-rich turquoise pools. When rain water hits pyrite ore, it forms sulfuric acid, which leaches out copper and other metals.

Heiman has been coming to Iron Mountain for the Central Valley Regional Water Quality Control Board in Redding for seven years. His colleague, Jim Pedri, started in 1978.

Bureaucrats, including Harry Rectenwald, a state Fish and Game biologist, and Rick Sugarek, an EPA chemical engineer, have visited thousands of times.

From Redding, they can see what looks like a 100-story rock dump. That's Iron Mountain Mine. Waste piles, iron-stained rocks and scattered mining debris litter the 4,400 acres.

Hidden below the surface is a network of tunnels and chambers, now collapsed. The Richmond mine alone, one of four big ones, is like a buried city.

"Imagine yourself in downtown San Francisco, and you've got 20 office buildings between 10 and 20 stories high. That's what they've carved out underground," said Sugarek, EPA project manager at the mine. "After mining, they allowed the rocks to fall in. So you have a rubblized zone in the mountain that's 70 stories high and covers the footprint of the office buildings. The water filters through this broken up pyrite deposit, just like a big Mr. Coffee, and forms a highly concentrated mine drainage."

On the surface, lies the apparatus aimed at stemming the pollution - a sludge-drying plant, a treatment plant with a settling pond as big as an Olympic swimming pool and a sludge dump.

Mine's ownership history<

The state ordered its first enforcement action against the mine in 1976, the same year then-owner Stauffer Chemical was on the verge of selling to Sacramento businessman Ted Arman, who wanted the tailings for fertilizers.

"We were informed that they were going to sell it to a man who didn't have much money," said state biologist Pedri.

"We issued an order to Stauffer Chemical, telling them to clean up and abate the effects of acid mine drainage on the property. It's still in effect."

Arman bought it. "They didn't privy me to any of the environmental problems although we were a year in negotiations," he says. "That's where my "innocent landowner' defense comes. Stauffer didn't tell me until six months after I bought the property. Then I learned the environmental problems had gone on for years. I would never have bought it."

The EPA put the mine on its National Priorities List in 1983, calling it one of the worst hazardous waste sites in the nation.

But Arman didn't have the money to pay for cleanup. EPA lawyers cast an eye on the former owner, Stauffer Chemical.

In the meantime, Rhone Poulenc had purchased Stauffer Chemical and become its successor, thus - in EPA eyes - assuming responsibility for the mine.

EPA lawyer, Bloomfield, said, "One of the main principles of Superfund is that the polluter pays rather than the taxpayer. The company that benefitted from the polluting activity should be responsible for cleaning up."

Judge takes government's side<

Rhone Poulenc told the court it wasn't liable for pollution by those who mined the mountain, but in September, Judge Levi ruled for the government, declaring Rhone Poulenc the successor to the original mining company, Mountain Copper, which sold to Stauffer after creating the pollution.

On Wednesday, Levi ruled against Rhone Poulenc's claim that the government was partly responsible for the pollution because it built Shasta Dam and bought minerals in the 1940s.

Brian Spiller, spokesman for Stauffer Management Co., a company set up to manage the Iron Mountain cleanup on behalf of Rhone Poulenc, said the lawyers hadn't yet decided whether to appeal.

"The case has a long way to go yet," Spiller said, adding that Rhone Poulenc is not trying to use the courts to sink Superfund.

"It's not our attempt to stretch this out. We have to respond in the detail that the court requires. . . . It looks like a lot on paper. But it's typical of any environmental case under (the Superfund law)."

Effect on the salmon<

Downstream in the Sacramento River, nine miles below Iron Mountain Mine and 100 miles above the Sacramento courtroom, Fish and Game biologist Rectenwald crouches by the water, watching salmon circulate around gravel mounds, guarding their eggs in shallow, fast-moving riffles.

Rectenwald, who grew up fishing the American River, has been on the Iron Mountain case for 12 years. He's pleased about the treatment plant built in 1994. The system, operated by Rhone Poulenc under EPA order, removes up to 80 percent of the copper, zinc, cadmium and acids.

The EPA wants Rhone Poulenc to take care of most of the other 20 percent, and treat into perpetuity at a cost of about $3 million to $4 million a year.

Just a month ago, the EPA ordered Rhone Poulenc to build a $21 million dam and pond on a metal-rich creek, saying the treatment would effectively end all but 5 percent of the mine's polluted drainage flowing to the Sacramento River.

Even with current treatment, the Bay still gets the downstream dregs 200 miles away, said Larry Kolb, assistant executive officer of the San Francisco Bay Regional Water Quality Control Board.

"Sometimes we feel that we're issuing jaywalking tickets in the Bay Area when there are homicides occurring just up the river," said Kolb.

Copper at low levels kills species of single-celled algae, worms, insects, shrimp-like creatures and larvae of mussels and oysters that feed animals at the base of the food web, according to the San Francisco Estuary Institute.

Up on the Sacramento River, Rectenwald talks about the health of the river's salmon runs. Pollution from Iron Mountain, insufficient nesting gravel and a diversion dam that keeps the fish from reaching cold water are the biggest threats to the salmon's survival in the Upper Sacramento River.

Scientists say the metals can kill the young salmon. More subtly, they reduce growth and damage immune systems of adult fish, and cause them problems when they switch from fresh to salt water.

The U.S. National Oceanic and Atmospheric Administration says iron mountain affects the most valuable portion of the spawning ground and calls it "one of the most significant Superfund sites in terms of its impacts on natural resources."

Water project managers still fear fish kills if the Bureau of Reclamation can't release enough fresh water from Shasta Dam to dilute toxics from the mine. At times, operators dump three times the normal 5,000 cubic feet a second over 30 days - enough water to supply 2.4 million people a year - just to keep the fish alive.

Courtroom exasperation<

"We're trying to restore the river. We're trying to protect all these natural processes," Rectenwald said.

"We want to keep these fish alive here and in good health, and to keep their tissue clean from contaminants. Water users are making sacrifices to solve their environmental problems. The regular waste dischargers have taken care of their effluent. So now we want to see that Iron Mountain Mine is moving ahead."

During last summer's court sessions in Sacramento, Judge Levi expressed exasperation over "millions of documents that fill the corridors" - 75-page briefs, boxes of depositions and experts' reports and motions and answers to motions.

The administrative record alone is 125,000 pages, the largest ever in the EPA region - four states and Pacific Island territories. Tens of thousands of those pages were produced by Rhone Poulenc, opposing EPA actions, urging less onerous remedies or requesting delays.

"I wonder," said the judge, "if we gave everyone a shovel, we could just go up and clean up Iron Mountain Mine. It would save me a great deal of time." <

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By **Jane Kay**