Preliminary Economic Assessment NI 43-101 Technical Report

Moonlight-Superior Copper Project

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This Technical Report on the Moonlight- Superior Copper Project is submitted to US Copper Corp. and is effective December 16, 2024.

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ACRONYMS AND ABBREVIATIONS

µm	micron
AA	Atomic Adsorption
Ag	silver
ANFO	ammonium nitrate/fuel oil
ALS	ALS Chemex
As	arsenic
Au	gold
Boyles Brothers	Boyles Brothers Drilling
CDN	CDN Resource Laboratories Ltd.
CE	Categorical Exclusion
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimeter
CO	cobalt
CRM	certified reference materials
CSAMT	Controlled Source Audio-frequency Magnetotellurics
Cu	copper
CV	coefficient of variation
cy	cubic yard
DDH	diamond drill hole
EA	Environmental Assessment
EIS	Environmental Impact Statement
EM	electromagnetics
EW	electrowinning
FAAS	flame atomic absorption spectrometry
Fe	iron
Fugro	Fugro Airborne Surveys

G&A	general and administrative
g/L	grams per liter
g/t	grams per st
gph	gallons per hour
gpm/ft^2	gallons per minute per square foot
GPS	global positioning system
GRE	Global Resource Engineering Ltd.
Grv	Granitic intrusive
H_2SO_4	sulphuric acid
HCl	hydrochloric acid
HDPE	high density polyethylene
HF	hydrogen fluoride
$HClO_4$	perchloric acid
H_2SO_4	sulfuric acid
HGC	Heinrichs Geoexploration Company
HNO_3	nitric acid
hp	horsepower
IOCG	Iron Oxide Copper Gold
ICP	Inductively Coupled Plasma
ICP-AES	Inductively Coupled Plasma-Atomic Emission Spectroscopy
ID2	Inverse Distance squared
IP	induced polarization
IRR	Internal Rate of Return
kg	kilogram
kg/st	kilograms per st
kWh/st	kilowatt-hour per short ton
lb	pound
lb/st	pounds per short ton
LCD	Lights Creek District
LCS	Lights Creek Stock
lph/m^2	liters per hour per square metre
LREE	light rare earth element
MO	molybdenum
m^3/hr	cubic meters per hour
MIBC	methyl isobutyl carbinol
mL	milliliter
mm	millimeter
Mo	molybdenum
MRE	Mineral Resource Estimate
Mt	metric tonnes

MW	megawatts
N/A	not applicable
Nevoro	Nevoro Inc.
Nevoro Copper	Nevoro Copper Inc.
Ni	nickel
NI	National Instrument
NN	Nearest Neighbor
NPV	Net Present Value
NSR	net smelter return
OK	Ordinary Kriging
opt	ounces per ton
ORP	oxidation-reduction potential
oz	ounce
PAG	potentially acid generating
PAX	potassium amyl xanthate
PEA	Preliminary Economic Assessment
PG&E	Pacific Gas & Electric Company
Placer-Amex	American Exploration and Mining Co.
PLS	pregnant leach solution
ppm	parts per million
QA/QD	quality assurance/quality control
QP	Qualified Person
RM	Registered Member
ROM	run-of-mine
RQD	rock quality designation
SAG	semi-autogenous grinding
Sheffield	Sheffield Resources Inc.
SME	Society for Mining, Metallurgy and Exploration
SMPR	sulfide mining production rate
sq	square
st	short tons
Starfield	Starfield Resources Inc.
SX	solvent extraction
tpd	short tons per day
TSF	tailings storage facility
Union Assay	Union Assay Laboratory
US$, USD	U.S. dollar
US Copper	US Copper Corp.
V	volt
XRF	x-ray fluorescence

1 SUMMARY

US Copper Corp. ("US Copper") is a TSX-V-listed copper asset development company based in Toronto, ON. US Copper has retained Global Resource Engineering Ltd. ("GRE") to prepare an updated Mineral Resource Estimate (MRE) and Preliminary Economic Assessment (PEA) and National Instrument (NI) 43-101 Technical Report for the Moonlight-Superior Project (the "Project," the "Property," or the "Moonlight-Superior Project").

Practices consistent with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) (2014) were applied to the generation of this MRE/PEA.

1.1 Property Description and Ownership

The Moonlight-Superior Project is located approximately 10 miles northeast of Greenville, California and approximately 100 miles northwest of Reno, Nevada. The property consists of 270 unpatented claims totaling 5,578 acres, 36 patented lode claims totaling 736 acres, 162 acres of fee lands, with a total of approximately 6,056 acres when adjusted for claim overlap. The claims are shown on Figure 4-2, listed in Table 4-1: Claim Information, and summarized in Table 4-2.

US Copper acquired the claims/patents covering Superior and Engels from the Trustee in Starfield's bankruptcy on June 27, 2013, including a minor amount of exploration and office equipment and supplies, the stored core and the complete database held by Nevoro which comprehensively documents all known exploration activity on the property from 1960 to 2013. In 2016, US Copper optioned the Moonlight claims from Canyon Copper and finalized the purchase in 2018.

1.2 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 Lights Creek District (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 three miles east of the Moonlight deposit, and the Superior Mine is approximately 2.2 miles southeast of the Moonlight deposit. Both now form part of the US Copper 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, Lamb's 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 acres 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. In 2021, the name of Crown Mining Corp changed to the US Copper Corp. (US Copper). In 2021, US Copper conducted a core drilling program at Superior. In 2023, US Copper conducted RC drilling programs at Moonlight, Lamb's Ridge, and Engels.

1.3 Geology 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-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.

The LCD copper deposits are primarily hosted in the early Jurassic (178 Ma), multiphased, quartz monzonite Lights Creek Stock (LCS), which intrudes slightly older metavolcanic 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 seven square miles (sq miles). 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.

The Superior deposit lies within the Lights Creek Stock near the south-eastern margin and south of Engels. The deposit is hosted within the quartz monzonite. Disseminated copper mineralization at Superior lies within a roughly circular area about 610 meters in diameter. Disseminated mineralization consists of fine chalcopyrite and lesser bornite with typical grades of between 0.1% and 0.3% copper. Within this disseminated mineralization are tabular brecciated structures that were historically mined up to 244 meters along strike, 183 meters down dip and three to seven meters wide. Mineralization in the breccia-veins consists of magnetite-actinolite-minor quartz-siderite-bornite-chalcopyrite.

The geology and mineralization at Lamb's Ridge appear to be most similar to Superior and was characterized by Placer-Amex geologists as a porphyry system. The wide-spaced (330- to 660-foot [100- to 200-meter]) drilling indicates disseminated copper mineralization similar to that found at Superior;

however, no occurrences of the high-grade breccia-veins mined at Superior have been encountered in the drill holes. That said, the drilling that has been done defines significant copper mineralization with copper grades in 16-foot (5-meter) composites exceeding 0.3% copper (Cu) over 4,900 feet (1,500 meters) north-to south and 1,640 feet (500 meters) east to west.

The Engels deposit lies outside the Lights Creek Stock, immediately adjacent to its eastern margin in an area represented by both gabbroic-phase intrusives and roof-pendant metavolcanics. Mineralization in the Engels Mine area occurs in a 1,280-foot (390-meter) by 656-foot (200-meter) pipe-like zone. Mineralization is associated with brecciated zones that exhibit features of both an intrusion breccia and a hydrothermal breccia. The relationship of mineralization to zones of breccia and contacts between the quartz diorite and metavolcanic is evident. The disseminated copper minerals are often very abundant and locally coalesce. Copper grades exceeding 15% Cu have been encountered in several 6.5-foot (2-meter) core intercepts.

Copper mineralization at Engels is strongly oxidized to depths of 230 feet (70 meters). Assay analysis for sulfuric acid soluble copper in a portion of samples from the post 2004 drilling indicates copper oxides representing 90% of total copper within these depths. Copper oxide minerals consist primarily as malachite with lesser chryscolla and azurite. The principal sulfide minerals consist of bornite and chalcopyrite.

Placer-Amex, Sheffield, and Sheffield's successors recognized that there are at least two styles 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% Cu.

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. Veinlet-or-breccia-hosted mineralization dominates the northern part of the Moonlight deposit, where chalcocite-rich mineralization commonly grades more than 1% Cu. 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% Cu in the central and southern portions of the deposit.

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 short tons (st) at an average grade of 0.54% Cu. Sheffield drilled 15 shallow reverse circulation (RC) holes at Moonlight in 2007, which appear to support the deposit's potential for economic copper oxide mineralization.

1.4 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-chloritesericite 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-actinolitetourmaline-apatite. At Moonlight, copper mineralization is associated with tourmalinespecularite-chlorite-sericite. During an April 2015 field visit to the LCD, Sillitoe categorized Engels, Lambs Ridge, Superior, and Moonlight as IOCG deposits (Cole, 2015).

1.5 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 miles of the LCD. Soil sampling produced six >1,000 parts per million (ppm) copper-in-soil anomalies and several other anomalies of lower magnitude. This work identified several 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 feet in 409 drill holes. 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 conducted 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 (Fugro) 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 an exploration of the LCD.

1.6 Drilling

Between 1964 and 1975, Placer-Amex drilled 213,028.00 feet in 440 diamond drill holes, all using a combination of NX and BX cores. Drilling was primarily focused on eight areas containing anomalous copper in soils. The drilling included 149 holes at the Superior deposit, 28 holes on Lamb's Ridge, 10 at the Engels Mine, and 213 at the Moonlight deposit. Out of these four main deposits, Placer-Amex drilled 26 holes at Copper Mountain, one at Osmeyer Ridge, four at Blue Copper, and nine at Gossan Ridge.

It should be noted that there are 18 holes, including FG-01 to FG-18, totaling 6,897 feet at the north of Moonlight deposit and out of the property, drilled by Placer-Amex in the 1960s. These holes do not have collar and survey data in the US Copper Corp. database and were not used for MRE in this technical report.

In 2005 and 2006, Sheffield drilled 11,135 feet 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 feet; however, they also drilled 1,420 feet in 15 RC 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 feet in 12 holes at Engels, Nevoro Copper completed 2,603 feet in seven 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. Starfield, the successor to Nevoro, drilled an additional seven holes at Engels in 2009 and 2010 totaling 2,071.50 feet.

In 2021, US Copper Corp. drilled 5,872 feet in seven core holes at Superior. In 2023, US Copper Corp also drilled two RC holes at Lamb's Ridge totaling 820 feet, 15 RC holes at Engels totaling 3,990 feet, and 15 RC holes at Moonlight totaling 2,430.0 feet. All drilling programs in 2021 and 2023 were conducted to check and verify the previous work and to establish an oxide resource.

1.7 Sample Preparation, Analysis and Security

The copper deposits of the LCD have seen three major exploration campaigns separated by a 35-year hiatus. Placer-Amex explored the district from 1964 to 1970, drilling 209,764 feet of core. From 2005 through 2010, Sheffield and its successors, Nevoro Copper and Starfield, drilled 28,919.6 feet in 87 holes. From 2021 through 2023, US Copper Corp drilled 13,112.3 feet in 39 holes.

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-foot-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 nor Robert Wetzel's 2009 report discusses the details of sample handling, sample preparation, quality assurance/quality control (QA/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 the best practices of the time.

The Sheffield/Nevoro/Starfield programs were designed largely to support the credibility of the assay results reported by Placer-Amex. During 2006 to 2007, Sheffield conducted a re-assay program on 50 core samples from the Superior Historical 1960s. In 2009, Starfield conducted a re-assay program on 533 core samples from the Superior Historical 1960s. During these re-assay programs, samples were analyzed by American Assays Labs. From 2005 to 2010, there was no formal QA/QC program in place; however, there are a few data points in the US Copper database for some of the QA/QC programs. There are a few limited QA/QC programs during the 2005 to 2008 drilling campaigns at Moonlight conducted by Sheffield and Nevoro and also during the 2007 to 2010 drilling campaigns at Engels conducted by Nevoro and Starfield. Sheffield's 2005 to 2010 program appears to have been conducted according to current industry practices; QA/QC results for copper in these drilling campaigns are acceptable. Results from Nevoro's 2008 drilling are nearly identical to the twinned Placer-Amex holes.

During the US Copper Corp. drilling program, for RC holes, recovered cuttings were delivered to a rotary splitter for sample collection for RC holes. The drill contractors collected a sample of the split at the rig during drilling using a pre-labeled bag. Samples were collected at 5-foot intervals, except for the 2023 drilling campaign at Engels, in which samples were collected at 10-foot intervals. The 2021 and 2023 core and RC samples were submitted to the ALS-Chemex laboratory in Reno. The core and RC samples were sorted, dried, crushed, and pulverized at the ALS laboratory to 85% minus 75 microns (µm) using methodology WEI-21. Total copper was assayed by ALS methods Cu OG62 and CU AA62, which use four-acid digestion, and the copper content was determined either by ICP or AA. The QA/QC results for copper conducted by US Copper Corp during the 2021 and 2023 drilling programs are acceptable and follow the current industry best practice.

1.8 Data Verification

In 2013, an Independent Mineral Consultant (W.F. Tanaka) prepared the first NI 43-101 compliant resource estimate for the deposits of the Superior Project. Tanaka completed a data verification program for a significant portion of the historical drill hole database. Tanaka reviewed and examined the project's drill hole database, which contains assay, survey, and geological information for historical drill campaigns. Tanaka presented a summary description of the checks made on, and the corrections or adjustments made to the drill hole database. A detailed list of errors was provided to US Copper.

In addition to the above, a total of 366 assay intervals for the Engels drilling done by Sheffield, representing 19% of the total modern Engels database, were checked against the assay certificates for data entry errors in copper (3 methods), silver, gold, iron, and arsenic. A total of 51 errors were found, all confined to the iron assays. No other errors were found for the other elements. On the whole, the error rate discovered by Tanaka in the above comparisons corresponds to a 1.99% error rate. Tanaka mentioned that this error rate is acceptable for a database that was not previously subject to rigorous scrutiny.

In 2018, Tetra Tech prepared a technical report and preliminary economic assessment for the Moonlight deposit. Tetra Tech reviewed historical data for the Moonlight project and checked the accuracy of the database. 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 Tetra Tech, 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).

In 2021, US Copper conducted a verification of all Superior historical data (1960s) by re-assaying all core samples from the historical drilling, using the remaining 1/2 split core. In this program, a total of 448 samples were sent to the ALS for analysis. In 2024, GRE’s Qualified Person (QP) reviewed US Copper’s in-house QA/QC procedure and found no error.

In 2024, GRE’s QP, Dr. Hamid Samari, reviewed the database from the 1960s to 2023 drilling programs. The data, including collar, survey, assay, geology, original certificates, and QA/QC files, was provided to GRE in .csv and pdf formats.

GRE’s QP, Dr. Hamid Samari, reviewed all available historical data. Original assay certificates and QA/QC data for the 2005, 2006, 2007, and 2008 programs were the only available data from drilling programs prior to US Copper (1964-2016). For those drilling campaigns, pre-US Copper, GRE’QP did manual audit work on approximately 11% of original assay certificates with the database, including 3,012 intervals, and found no material errors. GRE’s QP also reviewed all existing QA/QC data and did not find any errors that could materially impact the MRE.

From 2021 to 2023, US Copper completed 13,112.3 feet containing 2,009 assay samples in 39 drill holes at Superior, Moonlight, Lamb's Ridge, and Engels. GRE performed an independent analysis of US Copper's data relevant to the 2021 to 2023 drilling programs, comparing the data with the provided assay certificates. About 40% of all original assay certificates for 18 holes, including 837 intervals, from the 2021 to 2023 drilling programs, were manually spot-checked with the database for accuracy, and no errors were found. GRE's QP also reviewed all existing QA/QC data for these drilling programs and did not find any errors that could materially impact the MRE.

1.9 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 of 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 kiloWatt-hours per short ton (kWh/st), indicating that these materials should be very resistant to ball mill grinding.

1.10 Mineral Resource Estimate

The mineral resource estimate for the Moonlight-Superior Property was completed by Terre Lane (GRE), Society for Mining, Metallurgy and Exploration (SME)-Registered Member (RM). Ms. Lane is a Qualified Person as defined by NI 43-101 and is independent of US Copper. Ms. Lane estimated the mineral resource for the Project using an inverse distance squared interpolant. Geostatistics and mineral resource estimation were done with Leapfrog EDGE®. Model visualization was done with Leapfrog Geo® software, and the mineral resources were constrained with a Lerch-Grossman pit optimization. The metals of interest at the Project are copper, silver, and gold. The Mineral Resource estimate reported here was prepared in a manner consistent with the "CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines" adopted by CIM Council on November 29, 2019. The mineral resources are classified as Measured, Indicated, and Inferred in accordance with "CIM Definition Standards for Mineral Resources and Mineral Reserves," prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014. Classification of the resources reflects the relative confidence of the grade estimates. The effective date of the mineral resource estimate reported herein is December 16, 2024.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources are that part of the mineral resource for which quantity and grade or quality are estimated based on limited geologic evidence and sampling, which is sufficient to imply but not verify grade or quality continuity. Inferred mineral resources may not be converted to mineral reserves. It is reasonably expected, though not guaranteed, that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.

The Engels (Northeast Area) and Superior (South Area) deposits have existing underground workings. For each of these areas, blocks falling withing the existing workings were given Cu, Ag, and Au grades of 0, although tonnage was left in the model in the event any backfilling or collapse occurred.

Resources are reported within an optimized pit shell for each project area and meet the test of reasonable prospects for economic extraction. For sulfide material, a 10.45 net smelter return (NSR) cutoff was chosen, and for oxide and transition material, a 0.16% Cu cutoff was chosen for reporting the mineral resource. The cutoff grades were calculated based on the parameters in Table 14-8.

Table 1-1 shows the Mineral Resource Estimate for the Project.

Table 1-1: Moonlight-Superior Project Mineral Resource Statement

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
Indicated										
Engels	Oxide	0.16	%	2.39	0.81	40,861	7.72	565,232	0.055	4,050
	Transition	0.16	%	7.52	0.50	79,941	4.75	1,093,948	0.042	10,194
	Sulfide	10.45	NSR/ton	8.32	0.46	76,750	5.83	1,415,487	0.056	13,585
Lambs Ridge	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	1.61	0.27	8,614	0.00	0	0.000	0
Moonlight	Oxide	0.16	%	1.35	0.36	10,244	3.77	154,364	0.128	5,460
	Transition	0.16	%	25.71	0.33	179,071	3.85	2,972,073	0.037	30,083
	Sulfide	10.45	NSR/ton	232.35	0.30	1,390,461	1.87	12,674,340	0.009	61,721
Copper Mountain	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	3.94	0.32	24,936	0.00	0	0.000	0
Superior	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	119.64	0.30	722,893	0.81	2,817,086	0.004	14,949
Total	Oxide	0.16	%	3.74	0.68	51,104	6.59	719,596	0.087	9,510
	Transition	0.16	%	33.23	0.39	259,012	4.20	4,066,021	0.042	40,277
	Sulfide	10.45	NSR/ton	365.86	0.30	2,223,654	1.58	16,906,913	0.008	90,255
Total				**402.83**	**0.31**	**2,533,771**	**1.85**	**21,692,531**	**0.012**	**140,042**
Inferred										
Engels	Oxide	0.16	%	0.15	1.18	3,740	11.91	55,046	0.010	48
	Transition	0.16	%	1.73	0.49	18,287	5.20	281,158	0.019	1,053
	Sulfide	10.45	NSR/ton	6.93	0.38	52,445	5.08	1,027,412	0.041	8,280
	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
Lambs Ridge	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	3.46	0.30	20,954	0.00	0	0.000	0
Moonlight	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	30.82	0.28	175,635	0.09	81,857	0.000	35
Copper Mountain	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	3.90	0.27	21,320	0.00	0	0.000	0
Superior	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	17.60	0.29	101,817	0.01	2,681	0.000	23
Total	Oxide	0.16	%	0.15	1.25	3,740	12.64	55,046	0.011	48
	Transition	0.16	%	1.73	0.53	18,287	5.58	281,158	0.021	1,053
	Sulfide	10.45	NSR/ton	62.71	0.30	372,171	0.61	1,111,950	0.005	8,338
Total				**64.59**	**0.31**	**394,199**	**0.77**	**1,448,154**	**0.005**	**9,440**

Notes:

1. The effective date of the Mineral Resource is December 16, 2024.
2. The Qualified Person for the Mineral Resource Estimate is Terre Lane of GRE.
3. Mineral resources are reported at a 0.16% Cu cutoff for oxide and transition material and at a 10.45 NSR cutoff for sulfide material. The oxide and transition cutoff is calculated based on a long-term copper price of US$4.00/lb; assumed combined operating costs of US$7.50/ton (process and G&A); metallurgical recovery of 75% for copper. The sulfide cutoff is calculated as the breakeven NSR, which is equal to the combined process and G&A costs for the sulfide material.
4. Mineral resources are captured within an optimized pit shell and meet the test of reasonable prospects for economic extraction by open pit. The optimization used the same mining costs of US$2.35/ton mined and a 45º pit slope.
5. Rounding may result in apparent differences when summing tons, grade, and contained metal content.
6. lb = pound; oz = ounce; ppm = parts per million

1.11 Mining Methods

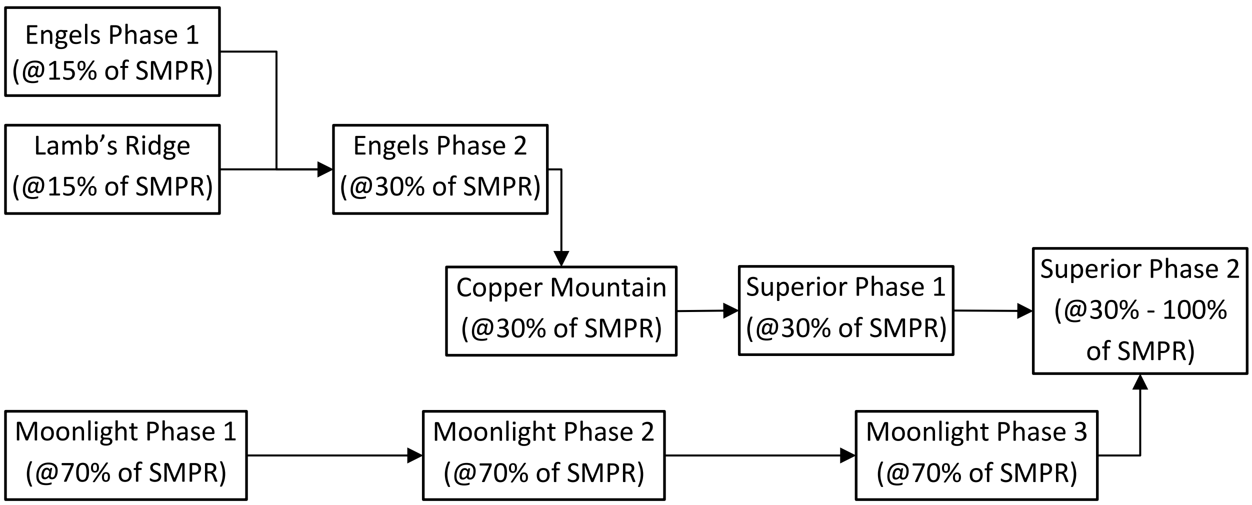
Mine plans for the resource area were designed and planned using conventional open pit mining methods. The open pit areas are suitable for phased designs. Ms. Lane of GRE used a single bench format consisting of 40-foot vertical benches with a horizontal 24-foot catch bench. Haul roads were designed with a minimum width of 112 feet and a maximum gradient of 10%. Haul ramps and roads have been designed to accommodate two-way traffic using 250-ton haul trucks, water diversion ditches, and safety berms. Minor sections were narrowed to a single lane of 70 feet.

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

- High-Grade Sulfide Mining Production Rate (SMPR): 60,000 tons per day (tpd)
- Mine Operating Days per Week: 7
- Mine Operating Weeks per Year: 52
- Mine Operating Shifts per Day: 2
- Mine Operating Hours per Shift: 12

All facilities needed for the project, including administrative offices, warehouse, ammonium nitrate/fuel oil (ANFO) storage, equipment shop, fuel station, plant, leach pad, and waste storage, will need to be constructed. Ms. Lane of GRE developed conceptual layouts for the project.

The schedule includes concurrent mining of pits and phases as follows:



For the pits and phases scheduled at 15% to 30% of the SMPR, the blasted rock would be loaded with 16-cubic yard (cy) capacity loaders into 105-ton capacity haul trucks. For the pits and phases scheduled at 70% to 100% of the SMPR, the blasted rock would be loaded with 29-cubic yard (cy) capacity hydraulic shovels into 250-ton capacity haul trucks. Mineralized high-grade sulfide material would be hauled to the primary crusher, mineralized low-grade sulfide material would be hauled to a stockpile, mineralized oxide material would be hauled to the leach pad, mineralized transition material would be hauled to either the leach pad or a temporary stockpile, and waste material would be hauled to the waste storage facilities.

1.12 Recovery Methods

The Superior and Moonlight resource have an oxide and transition cap that shows amenability to conventional acid heap leaching. This material is underlain by primary copper sulfides consisting of chalcopyrite with minor bornite, which show amenability to conventional sulfide flotation. Flotation tests indicate that copper recoveries above 90% should be achievable at a moderate primary grind size.

Additional test work, including a geometallurgical investigation, is recommended to define the expected ultimate metal recoveries for heap leaching and flotation.

1.13 Capital and Operating Costs

The capital cost estimate has been prepared for the PEA under the assumption of mill processing of sulfide mineralized material at a design rate of 60,000 tpd, and heap leaching of oxide and transition material at a design rate of 10,000 tpd. Project costs were estimated using cost data from Infomine (2024) and experience of senior staff. The estimate assumes that the project will be operated by the owner with purchased equipment.

The capital costs are summarized in Table 1-2.

Table 1-2: Moonlight-Superior Copper Project Capital Cost Summary

Item	Total ($millions)
Mine Equipment	$218.17
Process	$611.54
Infrastructure	$117.35
G&A	$100.72
Working	$36.12
Sustaining	$9.27
Contingency	$218.63
Total	**$1,311.80**

Operating costs are summarized in Table 1-3.

Table 1-3: Moonlight-Superior Copper Project Operating Cost Summary

Item	Total Operating Cost ($millions)	Unit Operating Cost	Unit
Mining	$899	$1.51	$/ton mined
Processing – Sulfides	$1,520	$5.24	$/ton processed
Processing – Oxides and Transition	$215	$8.74	$/ton processed
Rehandle	$85	$0.75	$/ton processed
G&A	$108	$0.34	$/ton processed
Contingency	$283	$0.90	$/ton processed
Total	**$3,111**		

1.14 Economics

Readers are advised that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under National Instrument 43-101. This 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 under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this preliminary economic assessment will be realized.

A multi scenario analysis method was used to analyze the economic performance of the project by varying the cutoff grades.

Ms. Lane of GRE evaluated the following options:

- Sulfide high-grade cutoffs of 15, 16, 17, 18, 19, and 20 NSR
- Oxide + transition material cutoff grades of 0.12%, 0.16%, 0.20%, 0.24%, and 0.28 % copper

After analyzing the economic results of all cases considered, Ms. Lane of GRE selected the 15 NSR high-grade sulfide cutoff and 0.28% copper oxide+transition cutoff as the base case as it results in the best overall economics.

Ms. Lane of GRE performed an economic analysis of the project by building an economic model based on the following assumptions:

- Copper price of $4.15/lb, based on using a weighted average of the 3-year trailing average copper price and the 1-year futures price, calculated as: 60% x 3-year trailing average price of $4.06/lb + 40% x 1-year futures price of $4.30/lb
- Silver price of $27.40/oz, based on using a weighted average of the 3-year trailing average silver price and the 1-year futures price, calculated as: 60% x 3-year trailing average price of $24.19/oz + 40% x 1-year futures price of $32.26/oz
- Gold price of $2,320/oz, based on using a weighted average of the 3-year trailing average gold price and the 1-year futures price, calculated as: 60% x 3-year trailing average price of $2,015/oz + 40% x 1-year futures price of $2,779/oz
- Sulfide material mineral recoveries of: 90.2% for copper, 80.4% for silver, and 71.0% for gold
- Heap leach mineral recoveries of: 75% for oxide material copper and 60% for transition material copper
- Leach recovery delay as follows: 60% of the final recovery during the first year on the heap, 30% recovered in the second year on the heap, and 10% recovered during the third year on the heap
- Copper 100% payable
- $.036/lb Cu from the heap leach cathode premium
- $160/ton transportation and off-site charges
- $3 million cost up front to purchase back royalties
- All costs input to the model are in US dollars.
- Sales and use taxes are not included in the model

Table 1-4 presents the key economic results for the project.

Table 1-4: Moonlight-Superior Copper Project Key Economic Results

Economic Measure	Value
After Tax NPV @ 7% (millions)	$1,075
Internal Rate of Return (IRR)	23%
Initial Capital (millions)	$956
Payback Period (year)	5.3
All-in Sustaining Cost ($/lb Cu Produced)	$2.51

The project economics shown in the PEA are favorable, providing positive Net Present Value (NPV) values at varying copper grades, copper prices, capital costs, and operating costs. 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 under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this preliminary economic assessment will be realized.

1.15 Recommendations

The QPs recommend the following Phase 1 items and budget (inclusive of contingency) to advance the Moonlight-Superior Copper project towards production (Table 26-1).

Table 1-5: Moonlight-Superior Copper Project Estimated Costs to Complete the Phase 1 Work Program

Exploration Cost Area	Total
Exploration Drilling	$5,000,000
Metallurgical Testing	$400,000
Permitting	$500,000
Total	**$5,900,000**

A comprehensive metallurgical test program is recommended to fully evaluate the potential of heap leach treatment for oxide and transition materials. This program should include bottle roll leach tests in conjunction with column leach tests. The variables that should be examined include grade, resource spatial distribution, mineralogy, and particle size. Additionally, these tests should include both conventional acid leaching and bioleaching. Additionally, flotation testing should be conducted on the sulfide materials examining variables including grade, resource spatial distribution, mineralogy, grind size and locked cycle flotation cleaning tests.

For exploration, the QPs recommend a drilling program on the order of 5,000 to 10,000 feet to outline additional resources.

A Phase 2 program would be contingent upon positive results from the Phase 1 program, and the scope of the Phase 2 program are conditioned on the results of the Phase 1 program. For the purposes of conceptual level planning, it is assumed that a Phase 2 program would consist of a nominal $25 million program that would include an expanded exploration drill program to upgrade resources to reserves and engineering and economics studies that would result in a Pre-feasibility Study.

The QPs recommend further engineering evaluation of different projects sizes and the optimization of mine plans.

The QPs recommend the evaluation and incorporation of existing and/or future technologies to improve sustainability and reduce environmental impacts of the Project.

Baseline studies are recommended to support the preparation of permitting documents. Baseline studies should include fauna and flora, archeology, human component, paleontology, and landscape.

Development of other preliminary engineering studies that will support early preparation of an Environmental Impact Statement are recommended. The following studies should be conducted to support infrastructure designs:

- Seismic study
- Hydrology and hydrogeology

- Geomorphology and geological risk
- Geotechnical studies
- Condemnation drilling

The QPs recommend additional evaluation of the potential for potentially acid generating (PAG) material, metal leaching, and groundwater mobilization of contaminants.

2 INTRODUCTION

2.1 Issuer and Terms of Reference

This technical report has been prepared for US Copper ("US Copper"). US Copper is a TSX-Venture-listed copper asset development company based in Toronto, ON. US Copper has retained Global Resource Engineering Ltd. ("GRE") to prepare a Mineral Resource Estimate (MRE) and Preliminary Economic Assessment (PEA) and subsequent NI 43-101 Technical Report for the Moonlight-Superior project (the "Project", the "Property" or the "Moonlight-Superior Project").

Practices consistent with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) (2014) were applied to the generation of this MRE/PEA.

The Moonlight-Superior Project is located approximately 10 miles northeast of Greenville, California and approximately 100 miles northwest of Reno, Nevada. The property consists of 270 unpatented claims totaling 5,578 acres, 36 patented lode claims totaling 736 acres, 162 acres of fee lands, with a total of approximately 6,056 acres when adjusted for claim overlap. The claims are shown on Figure 4-2, listed in Table 4-1: Claim Information, and summarized in Table 4-2

2.2 Sources of Information

A portion of the background information and technical data presented in this report was obtained from the following documents:

- Technical Report and Resource Estimate on the Moonlight Copper Property, Plumas County, California for Sheffield Resources Ltd., by Orequest Consultants Ltd. And Giroux Consultants Ltd., April 2007 (Orequest and Giroux, 2007)
- Technical Report and Resource Estimate for the Superior Project, Plumas County, California, prepared for Crown Gold Corporation by William F. Tanaka, November 2014 (Tanaka, 2014)
- Technical Report and Preliminary Economic Assessment for the Moonlight Deposit, Moonlight-Superior Copper Project, California, USA, prepared for Crown Mining Corp. by TetraTech, March 2018 (TetraTech, 2018)

The information contained in current report Sections 4 through 14 was largely presented in, and in some cases, is excerpted directly from, the reports listed above. GRE has reviewed this material in detail and finds the information contained herein to be factual and appropriate with respect to guidance provided by NI 43-101 and associated Form NI 43-101F1.

Additional information was requested from and provided by US Copper. In preparing Sections 9 through 13 of this report, the authors have relied in part on historical information including exploration reports, technical papers, sample descriptions, assay results, computer data, maps and drill logs generated by previous operators and associated third party consultants. Historical documents and data sources used during the preparation of this report are cited in the text, as appropriate, and are summarized in current report Section 27.

2.3 Qualified Persons and Personal Inspection

The Qualified Persons responsible for this report are Dr. Hamid Samari, Ms. Terre Lane, and Dr. J. Todd Harvey, all of GRE.

Dr. Samari, is a QP geologist with more than 25 years of professional experience as a consulting geologist and has contributed to numerous mineral resource projects, including more than twenty gold, silver, and polymetallic resources throughout the southwestern United States and South America over the past seven years. Dr. Samari is highly experienced in exploration geology and managing exploration programs, including geological and stratigraphic modeling, field mapping, design drilling and sampling, aerial photos and satellite image interpretation, geophysical data interpretation, interpretation of subsurface deposits, and preparing 3D geologic models, structural geological modeling, and multiple structural and physical geology aspects. He has also worked on different types of precious metals in North and South America. Dr. Samari is specifically responsible for Sections 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 4.1, 4.2, 4.3, 5, 6, 7, 8, 9, 10, 11, 12.1, 12.2, 12.3, 12.4, 12.5, 25.1, 25.2, 26.1.1, 26.1.2, and 27 of the Technical Report.

Ms. Lane, SME-RE, is a QP mining engineer with more than 35 years of experience. Her career has included full charge management of feasibility studies, mine and process engineering, and project development for surface and underground greenfield mines, and brownfield expansions. She has experience with a range of minerals, including base and precious metals, coal, potash, beryllium, uranium, talc, and sand and gravel; and she has managed projects throughout the world including the U.S., Canada, Mexico, India, Ireland, Russia, China, Chile, Bolivia, Peru, Costa Rica, Africa, and New Zealand. She has experience in most underground mining methods, from shrinkage stoping and cut and fill, to room and pillar, to block cave. Ms. Lane’s professional experience includes conceptual and detailed engineering, project optimization, project development, construction, start-up, and operations. She has directed engineering studies for numerous mine development projects and has managed engineering and feasibility study budgets as high as $25M/year. She has been responsible for underground exploration programs in foreign countries. She is an expert at resource estimation and mine design and has completed several hundred projects using all modeling techniques; estimates included: statistical analysis, geo-statistical analysis, inverse distance estimation, Kriging, single stage and multiple Indicator Kriging, geometallurgical modeling, and estimation of error. Ms. Lane is a Mining and Metallurgical Society of America (MMSA) Qualified Professional in Ore Reserves and Mining and she is a SME Registered Member. Ms. Lane is responsible for Sections 1.1, 1.10, 1.11, 1.13, 1.14, 1.15, 2, 3,, 12.7, 14, 15, 16, 18, 19, 20, 21, 22, 23, 24, 25.3, 25.4, 25.5, 26.1.4, 26.1.6, and 26.2.

Dr. Harvey, PhD, SME-RE, is a QP process engineer with over 25 years of experience in mining, renewable energy, and technology. Dr. Harvey is a Qualified Person under the Society of Mining Engineers (SME) Registered Member accreditation. Dr. Harvey’s background includes conventional gold recovery processes and refractory gold pretreatment via pressure oxidation, stirred tank BIOX, heap bio-oxidation, and roasting circuit design. Conventional base metal process design including polymetallic flotation, conventional oxide heap leaching, heap bioleaching and stirred tank bioleaching. Dr. Harvey has performed consulting for several companies in the field of process design and optimization, due diligence, and financial modeling. He possesses significant international experience, having lived in West Africa and South Africa and conducted a variety of projects in multiple countries. Dr. Harvey has extensive experience designing, performing, and analyzing metallurgical test work including mineralogy, crushing, grinding, gravity separation, filtration/thickening, flotation, CIL, heap leaching (gold/copper/zinc), refractory ore treatment (bioleaching – heap/tank, autoclaves, roasting), SX/EW, and tailings treatment. He has authored over 20 peer-reviewed technical papers and numerous studies and has presented at a variety of international conferences. He holds patents related to bioheap leaching biofuels production. Dr. Harvey is responsible for Sections 1.9, 1.12, 12.6, 13, 17, and 26.1.3.

Larry Breckenridge, P.E., is an Environmental Engineer with over 20 years of experience in mining environmental engineering including hydrogeology, geochemistry, water resource development, and environmental management. His work includes a diverse array of projects, including evaluation the geochemical conditions of a gold mine in Armenia, modeling contaminant transport at an inactive uranium mill in Texas, evaluating the hydrologic containment of an unlined tailings storage facility at a Guatemalan gold mine, and creating a geochemical model of a mining pit lake in Venezuela. He is an expert in the management and mitigation of Acid Rock Drainage (ARD) and other water quality impacts from hard-rock mining. He is also an expert in mine water balances and groundwater modeling using a variety of modern programs. Mr. Breckenridge has written numerous mine closure plans and is skilled in the economic assessment of environmental liabilities. Mr. Breckenridge fulfills the definition of a Qualified Person in accordance with Canadian NI 43-101 standards. Mr. Breckenridge is responsible for Sections 4.4, 4.5, 20, and 26.1.5.

GRE's QP, Dr. Hamid Samari, conducted an on-site inspection at the project site from 7 to 8 August 2024, accompanied by US Copper's geologist, Mr. Justin Claiborne. The GRE's QP, Dr. Hamid Samari, conducted this field visit mainly to check exploration programs and to conduct field checks, including the validation and accuracy of collar coordinates, geological maps, and geological logging, and to take a few core and pulp samples for assay checking.

2.4 Units of Measure

All currency amounts are stated in US dollars (US$, USD). Quantities are generally stated in U.S. Imperial units, including short tons (st, t), miles (mi) or feet for distance, acres for area, percentage (%) for copper grades, and troy ounces per st (oz/t, opt, oz/st) for gold and silver grades.

3 RELIANCE ON OTHER EXPERTS

During the preparation of this report, the authors relied in good faith on information and agreements provided by US Copper regarding property ownership, mineral tenure, mineral rights, permitting, environmental liabilities, and property agreements as described in Sections 4 and 5 of this report. An independent verification of land title and tenure was not performed. Relevant information was provided to GRE from Justin Claiborne on December 12, 2024, and December 15, 2024.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location and Ownership

The bulk of the following property description was taken from the 2013 NI 43-101 report prepared by Tanaka, Independent Mineral Consultant Consultants for US Copper Corp. (Crown Gold Corp.), and the 2018 NI 43-101 report prepared by Tetra Tech for US Copper Corp. (Crown Mining Corp.).

The Moonlight-Superior Project is located approximately 10 air miles northeast of the town of Greenville in Plumas County, California, which is approximately 100 miles northwest of Reno, Nevada (The property consists of 270 unpatented claims totaling 5,578 acres, 36 patented lode claims totaling 736 acres, 162 acres of fee lands, with a total of approximately 6,056 acres when adjusted for claim overlap. The claims are shown on Figure 4-2, listed in Table 4-1: Claim Information, and summarized in Table 4-2.

Figure 4-1).

The project location is shown on the Moonlight Peak and Kettle Rock 7.5' USGS topographic maps. The Latitude at the approximate center of the property is 40°13'36" N and the Longitude is 120°48'11" W or UTM coordinates of 686,855E and 4,455,250N (NAD 27 CONUS). The property lies within Sections 1, 2, 11 12, 13,14& 24 T27N R10E, Sections 4,5,6 7 ,8,9,17&18 T27N, R11E, Sections 35 & 36 T28N, R10E and Section 31&32 T28N, R11E in Plumas County, California.

The property consists of 270 unpatented claims totaling 5,578 acres, 36 patented lode claims totaling 736 acres, 162 acres of fee lands, with a total of approximately 6,056 acres when adjusted for claim overlap. The claims are shown on Figure 4-2, listed in Table 4-1: Claim Information, and summarized in Table 4-2.

Figure 4-1: Regional Location Map

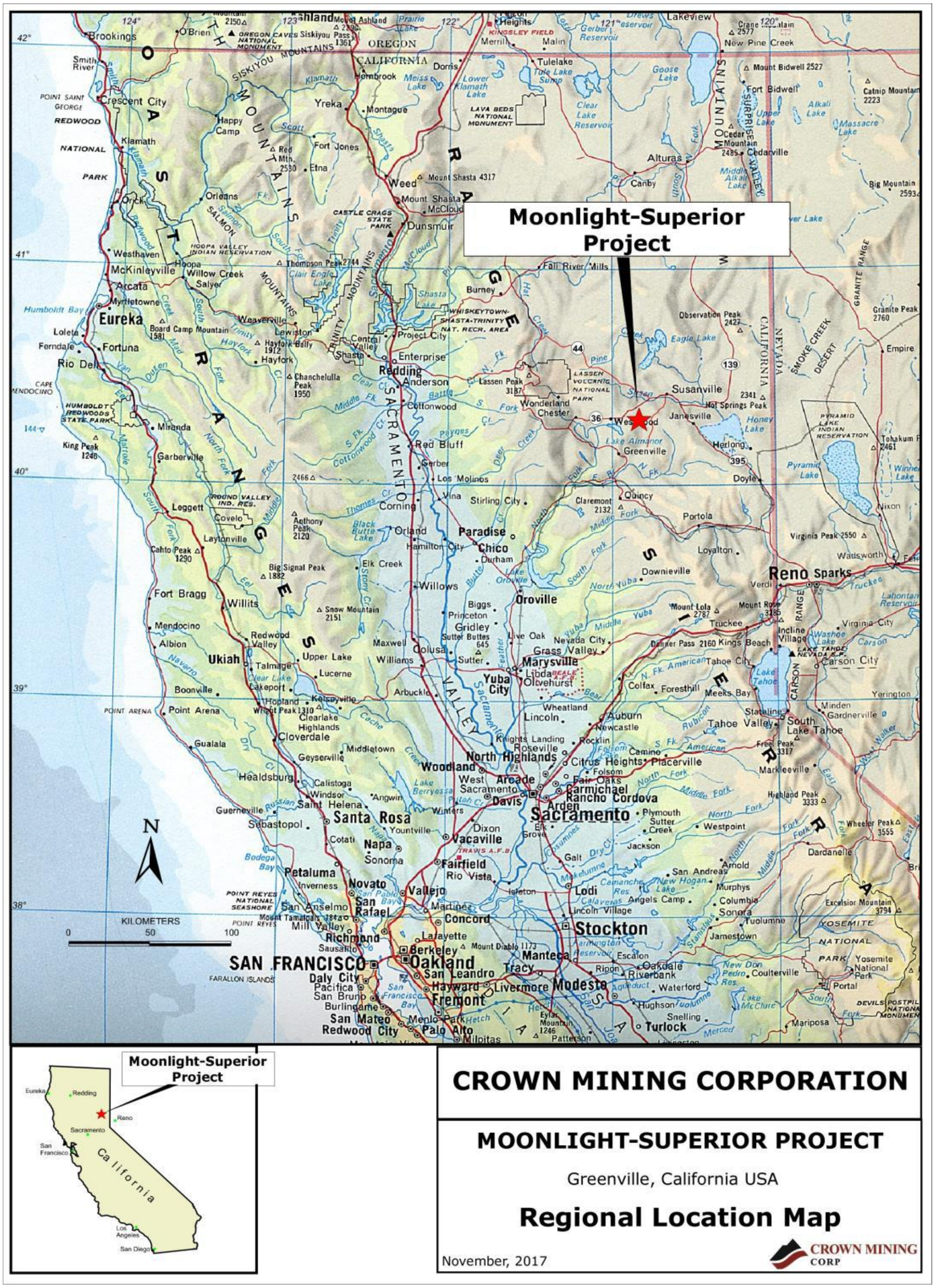
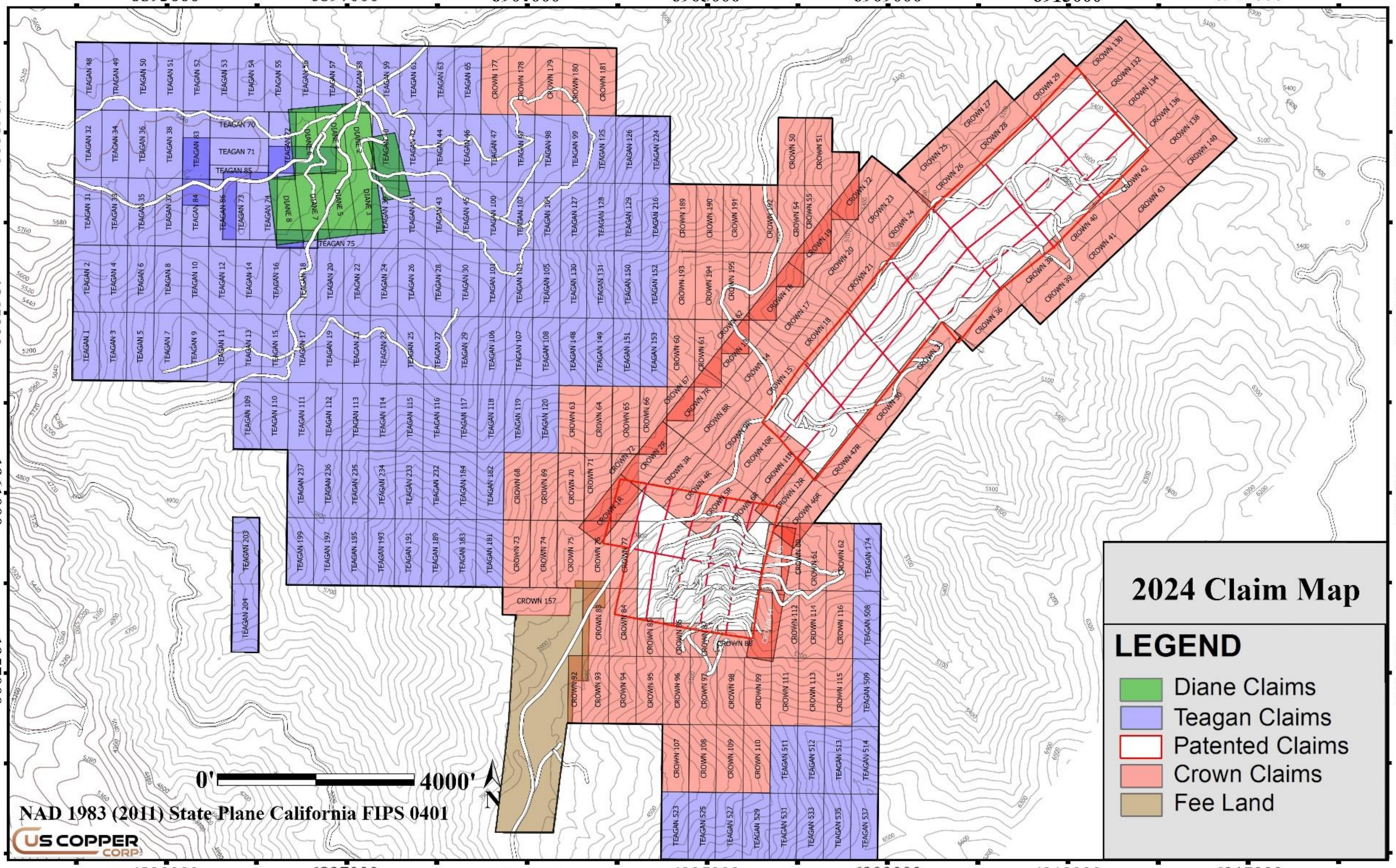


Figure 4-2: Moonlight-Superior Project Property Map



Source: US Copper, 2024

Table 4-1: Claim Information

Claim Name	Claim No.	Meridian Township Range Section	Serial Number	CAMC	Areas (acres)	Area (hectares)	Book	Date Located	Next Payment Due Date	County
Crown	1R	21 0270N 0110E 007	CA101378195	312269	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	2R	21 0270N 0110E 007	CA101378196	312270	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	3R	21 0270N 0110E 008	CA101378197	312271	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	4R	21 0270N 0110E 008	CA101378198	312272	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	5R	21 0270N 0110E 008	CA101378199	312273	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	6R	21 0270N 0110E 008	CA101378200	312274	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	7R	21 0270N 0110E 008	CA101378996	312275	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	8R	21 0270N 0110E 008	CA101378997	312276	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	9R	21 0270N 0110E 008	CA101378998	312277	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	10R	21 0270N 0110E 008	CA101378999	312278	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	11R	21 0270N 0110E 008	CA101379000	312279	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	12R	21 0270N 0110E 008	CA101379126	312280	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	46R	21 0270N 0110E 008	CA101379127	312281	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	47R	21 0270N 0110E 008	CA101379128	312282	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	13	21 0270N 0110E 008	CA101847453	311873	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	14	21 0270N 0110E 008	CA101847454	311874	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	15	21 0270N 0110E 008	CA101847455	311875	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	16	21 0270N 0110E 008	CA101847689	311876	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	17	21 0270N 0110E 008	CA101847690	311877	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	18	21 0270N 0110E 008	CA101847691	311878	20.66	8.36	2015	11/17/2015	9/2/2025	Plumas
Crown	19	21 0270N 0110E 005	CA101847692	311879	20.66	8.36	2015	11/16/2015	9/2/2025	Plumas
Crown	20	21 0270N 0110E 005	CA101847693	311880	20.66	8.36	2015	11/16/2015	9/2/2025	Plumas
Crown	21	21 0270N 0110E 005	CA101847694	311881	20.66	8.36	2015	11/15/2015	9/2/2025	Plumas
Crown	22	21 0270N 0110E 005	CA101847695	311882	20.66	8.36	2015	11/16/2015	9/2/2025	Plumas
Crown	23	21 0270N 0110E 004	CA101847696	311883	20.66	8.36	2015	11/16/2015	9/2/2025	Plumas
Crown	24	21 0270N 0110E 004	CA101847697	311884	20.66	8.36	2015	11/15/2015	9/2/2025	Plumas
Crown	25	21 0270N 0110E 004	CA101847698	311885	20.66	8.36	2015	11/15/2015	9/2/2025	Plumas
Crown	26	21 0270N 0110E 004	CA101847699	311886	20.66	8.36	2015	11/15/2015	9/2/2025	Plumas
Crown	27	21 0270N 0110E 004	CA101847700	311887	20.66	8.36	2015	11/15/2015	9/2/2025	Plumas
Crown	28	21 0270N 0110E 004	CA101847701	311888	20.66	8.36	2015	11/15/2015	9/2/2025	Plumas
Crown	29	21 0270N 0110E 004	CA101847702	311889	20.66	8.36	2015	11/15/2015	9/2/2025	Plumas

Claim Name	Claim No.	Meridian Township Range Section	Serial Number	CAMC	Areas (acres)	Area (hectares)	Book	Date Located	Next Payment Due Date	County
Crown	30	21 0270N 0110E 008	CA101847703	311890	20.66	8.36	2015	11/14/2015	9/2/2025	Plumas
Crown	33	21 0270N 0110E 009	CA101847706	311893	20.66	8.36	2015	11/14/2015	9/2/2025	Plumas
Crown	36	21 0270N 0110E 009	CA101847709	311896	20.66	8.36	2015	11/13/2015	9/2/2025	Plumas
Crown	38	21 0270N 0110E 004	CA101848056	311898	20.66	8.36	2015	11/13/2015	9/2/2025	Plumas
Crown	39	21 0270N 0110E 009	CA101848057	311899	20.66	8.36	2015	11/13/2015	9/2/2025	Plumas
Crown	40	21 0270N 0110E 004	CA101848058	311900	20.66	8.36	2015	11/13/2015	9/2/2025	Plumas
Crown	41	21 0270N 0110E 003	CA101848059	311901	20.66	8.36	2015	11/13/2015	9/2/2025	Plumas
Crown	42	21 0270N 0110E 003	CA101848060	311902	20.66	8.36	2015	11/13/2015	9/2/2025	Plumas
Crown	43	21 0270N 0110E 003	CA101848061	311903	20.66	8.36	2015	11/13/2015	9/2/2025	Plumas
Crown	44	21 0270N 0110E 017	CA101848062	311904	20.66	8.36	2015	11/16/2015	9/2/2025	Plumas
Crown	45	21 0270N 0110E 017	CA101848063	311905	20.66	8.36	2015	11/16/2015	9/2/2025	Plumas
Crown	50	21 0270N 0110E 005	CA101645762	313406	20.66	8.36	2016	6/23/2016	9/2/2025	Plumas
Crown	51	21 0270N 0110E 005	CA101645763	313407	20.66	8.36	2016	6/23/2016	9/2/2025	Plumas
Crown	54	21 0270N 0110E 005	CA101645766	313410	20.66	8.36	2016	6/11/2016	9/2/2025	Plumas
Crown	55	21 0270N 0110E 005	CA101645767	313411	20.66	8.36	2016	6/23/2016	9/2/2025	Plumas
Crown	56	21 0270N 0110E 005	CA101645768	313412	20.66	8.36	2016	6/23/2016	9/2/2025	Plumas
Crown	58	21 0270N 0110E 008	CA101645770	313414	20.66	8.36	2016	6/9/2016	9/2/2025	Plumas
Crown	59	21 0270N 0110E 008	CA101645771	313415	20.66	8.36	2016	6/9/2016	9/2/2025	Plumas
Crown	60	21 0270N 0110E 008	CA101646897	313416	20.66	8.36	2016	6/23/2016	9/2/2025	Plumas
Crown	61	21 0270N 0110E 008	CA101646898	313417	20.66	8.36	2016	6/9/2016	9/2/2025	Plumas
Crown	62	21 0270N 0110E 008	CA101646899	313418	20.66	8.36	2016	6/9/2016	9/2/2025	Plumas
Crown	63	21 0270N 0110E 007	CA101646900	313419	20.66	8.36	2016	6/23/2016	9/2/2025	Plumas
Crown	64	21 0270N 0110E 007	CA101646901	313420	20.66	8.36	2016	6/23/2016	9/2/2025	Plumas
Crown	65	21 0270N 0110E 007	CA101646902	313421	20.66	8.36	2016	6/23/2016	9/2/2025	Plumas
Crown	66	21 0270N 0110E 008	CA101646903	313422	20.66	8.36	2016	6/23/2016	9/2/2025	Plumas
Crown	67	21 0270N 0110E 008	CA101646904	313423	10.3	4.17	2016	6/23/2016	9/2/2025	Plumas
Crown	68	21 0270N 0110E 007	CA101646905	313424	20.66	8.36	2016	6/9/2016	9/2/2025	Plumas
Crown	69	21 0270N 0110E 007	CA101646906	313425	20.66	8.36	2016	6/9/2016	9/2/2025	Plumas
Crown	70	21 0270N 0110E 007	CA101646907	313426	20.66	8.36	2016	6/9/2016	9/2/2025	Plumas
Crown	71	21 0270N 0110E 007	CA101646908	313427	20.66	8.36	2016	6/23/2016	9/2/2025	Plumas
Crown	72	21 0270N 0110E 007	CA101646909	313428	20.66	8.36	2016	6/23/2016	9/2/2025	Plumas
Crown	73	21 0270N 0110E 018	CA101646910	313429	20.66	8.36	2016	6/9/2016	9/2/2025	Plumas
Crown	74	21 0270N 0110E 018	CA101646911	313430	20.66	8.36	2016	6/9/2016	9/2/2025	Plumas

Claim Name	Claim No.	Meridian Township Range Section	Serial Number	CAMC	Areas (acres)	Area (hectares)	Book	Date Located	Next Payment Due Date	County
Crown	75	21 0270N 0110E 018	CA101646912	313431	20.66	8.36	2016	6/9/2016	9/2/2025	Plumas
Crown	76	21 0270N 0110E 018	CA101646913	313432	20.66	8.36	2016	6/8/2016	9/2/2025	Plumas
Crown	77	21 0270N 0110E 017	CA101646914	313433	20.66	8.36	2016	6/8/2016	9/2/2025	Plumas
Crown	80	21 0270N 0110E 017	CA101646917	313436	20.66	8.36	2016	6/8/2016	9/2/2025	Plumas
Crown	81	21 0270N 0110E 017	CA101648085	313437	20.66	8.36	2016	6/8/2016	9/2/2025	Plumas
Crown	82	21 0270N 0110E 017	CA101648086	313438	20.66	8.36	2016	6/8/2016	9/2/2025	Plumas
Crown	83	22 0270N 0110E 018	CA101648087	313439	20.66	8.36	2016	6/7/2016	9/2/2025	Plumas
Crown	84	21 0270N 0110E 017	CA101648088	313440	20.66	8.36	2016	6/7/2016	9/2/2025	Plumas
Crown	85	21 0270N 0110E 017	CA101648089	313441	20.66	8.36	2016	6/7/2016	9/2/2025	Plumas
Crown	86	21 0270N 0110E 017	CA101648090	313442	20.66	8.36	2016	6/7/2016	9/2/2025	Plumas
Crown	87	21 0270N 0110E 017	CA101648091	313443	20.66	8.36	2016	6/8/2016	9/2/2025	Plumas
Crown	88	21 0270N 0110E 017	CA101648092	313444	20.66	8.36	2016	6/8/2016	9/2/2025	Plumas
Crown	92	22 0270N 0110E 018	CA101648322	313445	10.3	4.17	2016	6/7/2016	9/2/2025	Plumas
Crown	93	22 0270N 0110E 018	CA101648323	313446	20.66	8.36	2016	6/7/2016	9/2/2025	Plumas
Crown	94	21 0270N 0110E 017	CA101648324	313447	20.66	8.36	2016	6/7/2016	9/2/2025	Plumas
Crown	95	21 0270N 0110E 017	CA101648325	313448	20.66	8.36	2016	6/7/2016	9/2/2025	Plumas
Crown	96	21 0270N 0110E 017	CA101648326	313449	20.66	8.36	2016	6/7/2016	9/2/2025	Plumas
Crown	97	21 0270N 0110E 017	CA101648327	313450	20.66	8.36	2016	6/24/2016	9/2/2025	Plumas
Crown	98	21 0270N 0110E 017	CA101648328	313451	20.66	8.36	2016	6/24/2016	9/2/2025	Plumas
Crown	99	21 0270N 0110E 017	CA101648329	313452	20.66	8.36	2016	6/24/2016	9/2/2025	Plumas
Crown	107	21 0270N 0110E 020	CA101648334	313457	20.66	8.36	2016	6/7/2016	9/2/2025	Plumas
Crown	108	21 0270N 0110E 017	CA101649559	313458	20.66	8.36	2016	6/24/2016	9/2/2025	Plumas
Crown	109	21 0270N 0110E 017	CA101649560	313459	20.66	8.36	2016	6/24/2016	9/2/2025	Plumas
Crown	110	21 0270N 0110E 017	CA101649561	313460	20.66	8.36	2016	6/24/2016	9/2/2025	Plumas
Crown	111	21 0270N 0110E 017	CA101783500	318358	20.66	8.36	2018	5/14/2018	9/2/2025	Plumas
Crown	112	21 0270N 0110E 017	CA101783501	318359	20.66	8.36	2018	5/14/2018	9/2/2025	Plumas
Crown	113	21 0270N 0110E 017	CA101783502	318360	20.66	8.36	2018	5/14/2018	9/2/2025	Plumas
Crown	114	21 0270N 0110E 017	CA101783503	318361	20.66	8.36	2018	5/14/2018	9/2/2025	Plumas
Crown	115	21 0270N 0110E 017	CA101783504	318362	20.66	8.36	2018	5/14/2018	9/2/2025	Plumas
Crown	116	21 0270N 0110E 017	CA101783505	318363	20.66	8.36	2018	5/14/2018	9/2/2025	Plumas
Crown	130	21 0270N 0110E 003	CA101766335	319521	20.66	8.36	2018	11/16/2018	9/2/2025	Plumas
Crown	132	21 0270N 0110E 003	CA101765930	319523	20.66	8.36	2018	11/16/2018	9/2/2025	Plumas
Crown	134	21 0270N 0110E 003	CA101765932	319525	20.66	8.36	2018	11/16/2018	9/2/2025	Plumas

Claim Name	Claim No.	Meridian Township Range Section	Serial Number	CAMC	Areas (acres)	Area (hectares)	Book	Date Located	Next Payment Due Date	County
Crown	136	21 0270N 0110E 003	CA101765934	319527	20.66	8.36	2018	11/16/2018	9/2/2025	Plumas
Crown	138	21 0270N 0110E 003	CA101765936	319529	20.66	8.36	2018	11/15/2018	9/2/2025	Plumas
Crown	140	21 0270N 0110E 003	CA101765938	319531	20.66	8.36	2018	11/15/2018	9/2/2025	Plumas
Crown	157	21 0270N 0110E 018	CA105287895		20.66	8.36	2021	11/28/2021	9/2/2025	Plumas
Crown	177	21 0270N 0110E 006	CA105287915		20.66	8.36	2021	11/17/2021	9/2/2025	Plumas
Crown	178	21 0270N 0110E 006	CA105287916		20.66	8.36	2021	11/17/2021	9/2/2025	Plumas
Crown	179	21 0270N 0110E 006	CA105287917		20.66	8.36	2021	11/17/2021	9/2/2025	Plumas
Crown	180	21 0270N 0110E 006	CA105287918		20.66	8.36	2021	11/17/2021	9/2/2025	Plumas
Crown	181	21 0270N 0110E 006	CA105287919		20.66	8.36	2021	11/17/2021	9/2/2025	Plumas
Crown	189	21 0270N 0110E 005	CA105287927		20.66	8.36	2021	11/18/2021	9/2/2025	Plumas
Crown	190	21 0270N 0110E 005	CA105287928		20.66	8.36	2021	11/18/2021	9/2/2025	Plumas
Crown	191	21 0270N 0110E 005	CA105287929		20.66	8.36	2021	11/18/2021	9/2/2025	Plumas
Crown	192	21 0270N 0110E 005	CA105287930		20.66	8.36	2021	11/18/2021	9/2/2025	Plumas
Crown	193	21 0270N 0110E 005	CA105287931		20.66	8.36	2021	11/18/2021	9/2/2025	Plumas
Crown	194	21 0270N 0110E 005	CA105287932		20.66	8.36	2021	11/18/2021	9/2/2025	Plumas
Crown	195	21 0270N 0110E 005	CA105287933		20.66	8.36	2021	11/18/2021	9/2/2025	Plumas
Diane	1	21 0270N 0100E 001	CA101451661	264419	20.66	8.36	BK58P	9/1/1994	9/2/2025	Plumas
Diane	2	21 0270N 0100E 001	CA101493668	264420	20.66	8.36	BK58P	9/1/1994	9/2/2025	Plumas
Diane	3	21 0270N 0100E 001	CA102520548	264421	20.66	8.36	BK58P	9/1/1994	9/2/2025	Plumas
Diane	4	21 0270N 0100E 001	CA101302163	264422	20.66	8.36	BK58P	9/1/1994	9/2/2025	Plumas
Diane	5	21 0270N 0100E 001	CA101350106	264423	20.66	8.36	BK58P	9/1/1994	9/2/2025	Plumas
Diane	6	21 0270N 0100E 001	CA101332052	264424	20.66	8.36	BK58P	9/1/1994	9/2/2025	Plumas
Diane	7	21 0270N 0100E 001	CA101339186	264425	20.66	8.36	BK58P	9/1/1994	9/2/2025	Plumas
Diane	8	21 0270N 0100E 001	CA101333570	264426	20.66	8.36	BK58P	9/1/1994	9/2/2025	Plumas
Teagan	1	21 0270N 0100E 011	CA101317612	283131	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	2	21 0270N 0100E 002	CA101317613	283132	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	3	21 0270N 0100E 011	CA101317614	283133	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	4	21 0270N 0100E 002	CA101317615	283134	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	5	21 0270N 0100E 011	CA101317616	283135	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	6	21 0270N 0100E 001	CA101317617	283136	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	7	21 0270N 0100E 012	CA101317618	283137	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	8	21 0270N 0100E 001	CA101317619	283138	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	9	21 0270N 0100E 012	CA101317620	283139	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas

Claim Name	Claim No.	Meridian Township Range Section	Serial Number	CAMC	Areas (acres)	Area (hectares)	Book	Date Located	Next Payment Due Date	County
Teagan	10	21 0270N 0100E 001	CA101317621	283140	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	11	21 0270N 0100E 012	CA101317622	283141	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	12	21 0270N 0100E 001	CA101317623	283142	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	13	21 0270N 0100E 012	CA101317624	283143	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	14	21 0270N 0100E 001	CA101317625	283144	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	15	21 0270N 0100E 012	CA101317626	283145	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	16	21 0270N 0100E 001	CA101317627	283146	20.66	8.36	2005	11/11/2004	9/2/2025	Plumas
Teagan	17	21 0270N 0100E 012	CA101318822	283147	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	18	21 0270N 0100E 001	CA101318823	283148	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	19	21 0270N 0100E 012	CA101318824	283149	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	20	21 0270N 0100E 001	CA101318825	283150	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	21	21 0270N 0100E 012	CA101318826	283151	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	22	21 0270N 0100E 001	CA101318827	283152	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	23	21 0270N 0100E 012	CA101318828	283153	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	24	21 0270N 0100E 001	CA101318829	283154	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	25	21 0270N 0110E 007	CA101318830	283155	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	26	21 0270N 0110E 006	CA101318831	283156	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	27	21 0270N 0110E 007	CA101318832	283157	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	28	21 0270N 0110E 006	CA101318833	283158	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	29	21 0270N 0110E 007	CA101318834	283159	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	30	21 0270N 0110E 006	CA101318835	283160	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	31	21 0270N 0100E 002	CA101318836	283161	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	32	21 0270N 0100E 002	CA101318837	283162	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	33	21 0270N 0100E 002	CA101318838	283163	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	34	21 0270N 0100E 002	CA101318839	283164	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	35	21 0270N 0100E 001	CA101318840	283165	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	36	21 0270N 0100E 001	CA101318841	283166	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	37	21 0270N 0100E 001	CA101318842	283167	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	38	21 0270N 0100E 001	CA101320022	283168	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	39	21 0270N 0100E 001	CA101320023	283169	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	40	21 0270N 0100E 001	CA101320024	283170	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	41	21 0270N 0110E 006	CA101320025	283171	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	42	21 0270N 0110E 006	CA101320026	283172	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas

Claim Name	Claim No.	Meridian Township Range Section	Serial Number	CAMC	Areas (acres)	Area (hectares)	Book	Date Located	Next Payment Due Date	County
Teagan	43	21 0270N 0110E 006	CA101320027	283173	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	44	21 0270N 0110E 006	CA101320028	283174	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	45	21 0270N 0110E 006	CA101320029	283175	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	46	21 0270N 0110E 006	CA101320030	283176	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	47	21 0270N 0110E 006	CA101320031	283177	20.66	8.36	2005	11/10/2004	9/2/2025	Plumas
Teagan	48	21 0270N 0100E 002	CA101320032	283178	20.66	8.36	2005	11/8/2004	9/2/2025	Plumas
Teagan	49	21 0270N 0100E 002	CA101320033	283179	20.66	8.36	2005	11/8/2004	9/2/2025	Plumas
Teagan	50	21 0270N 0100E 001	CA101320034	283180	20.66	8.36	2005	11/8/2004	9/2/2025	Plumas
Teagan	51	21 0270N 0100E 001	CA101320035	283181	20.66	8.36	2005	11/8/2004	9/2/2025	Plumas
Teagan	52	21 0270N 0100E 001	CA101320036	283182	20.66	8.36	2005	11/7/2004	9/2/2025	Plumas
Teagan	53	21 0270N 0100E 001	CA101320037	283183	20.66	8.36	2005	11/7/2004	9/2/2025	Plumas
Teagan	54	21 0270N 0100E 001	CA101320038	283184	20.66	8.36	2005	11/7/2004	9/2/2025	Plumas
Teagan	55	21 0270N 0100E 001	CA101320039	283185	20.66	8.36	2005	11/7/2004	9/2/2025	Plumas
Teagan	56	21 0270N 0100E 001	CA101320040	283186	20.66	8.36	2005	11/7/2004	9/2/2025	Plumas
Teagan	57	21 0270N 0100E 001	CA101320041	283187	20.66	8.36	2005	11/7/2004	9/2/2025	Plumas
Teagan	58	21 0270N 0100E 001	CA101320042	283188	20.66	8.36	2005	11/7/2004	9/2/2025	Plumas
Teagan	59	21 0270N 0100E 001	CA101511222	283189	20.66	8.36	2005	11/7/2004	9/2/2025	Plumas
Teagan	60	21 0270N 0110E 006	CA101511223	283190	20.66	8.36	2005	11/7/2004	9/2/2025	Plumas
Teagan	61	21 0270N 0110E 006	CA101511224	283191	20.66	8.36	2005	11/7/2004	9/2/2025	Plumas
Teagan	63	21 0270N 0110E 006	CA101511225	283193	20.66	8.36	2005	11/8/2004	9/2/2025	Plumas
Teagan	65	21 0270N 0110E 006	CA101511226	283195	20.66	8.36	2005	11/8/2004	9/2/2025	Plumas
Teagan	69	21 0270N 0100E 001	CA101511228	283199	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	70	21 0270N 0100E 001	CA101511229	283200	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	71	21 0270N 0100E 001	CA101511230	283201	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	72	21 0270N 0100E 001	CA101511231	283202	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	73	21 0270N 0100E 001	CA101511232	283203	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	74	21 0270N 0100E 001	CA101511233	283204	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	75	21 0270N 0100E 001	CA101511234	283205	20.66	8.36	2005	11/9/2004	9/2/2025	Plumas
Teagan	83	21 0270N 0100E 001	CA101314555	283389	20.66	8.36	2005	1/15/2005	9/2/2025	Plumas
Teagan	84	21 0270N 0100E 001	CA101314556	283390	20.66	8.36	2005	1/16/2005	9/2/2025	Plumas
Teagan	85	21 0270N 0100E 001	CA101314557	283391	20.66	8.36	2005	1/15/2005	9/2/2025	Plumas
Teagan	86	21 0270N 0100E 001	CA101314558	283392	20.66	8.36	2005	1/16/2005	9/2/2025	Plumas
Teagan	87	21 0270N 0100E 001	CA101314559	283393	20.66	8.36	2005	1/16/2005	9/2/2025	Plumas

Claim Name	Claim No.	Meridian Township Range Section	Serial Number	CAMC	Areas (acres)	Area (hectares)	Book	Date Located	Next Payment Due Date	County
Teagan	88	21 0270N 0100E 001	CA101314560	283394	20.66	8.36	2005	1/15/2005	9/2/2025	Plumas
Teagan	89	21 0270N 0100E 001	CA101314561	283395	20.66	8.36	2005	1/15/2005	9/2/2025	Plumas
Teagan	97	21 0270N 0110E 006	CA101734370	284706	20.66	8.36	2006	10/11/2005	9/2/2025	Plumas
Teagan	98	21 0270N 0110E 006	CA101734371	284707	20.66	8.36	2006	10/11/2005	9/2/2025	Plumas
Teagan	99	21 0270N 0110E 006	CA101734372	284708	20.66	8.36	2006	10/11/2005	9/2/2025	Plumas
Teagan	100	21 0270N 0110E 006	CA101734373	284709	20.66	8.36	2006	10/11/2005	9/2/2025	Plumas
Teagan	101	21 0270N 0110E 007	CA101734374	284710	20.66	8.36	2006	10/11/2005	9/2/2025	Plumas
Teagan	102	21 0270N 0110E 006	CA101734375	284711	20.66	8.36	2006	10/11/2005	9/2/2025	Plumas
Teagan	103	21 0270N 0110E 007	CA101735466	284712	20.66	8.36	2006	10/11/2005	9/2/2025	Plumas
Teagan	104	21 0270N 0110E 006	CA101735467	284713	20.66	8.36	2006	10/11/2005	9/2/2025	Plumas
Teagan	105	21 0270N 0110E 007	CA101735468	284714	20.66	8.36	2006	10/11/2005	9/2/2025	Plumas
Teagan	106	21 0270N 0110E 007	CA101735469	284715	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	107	21 0270N 0110E 007	CA101735470	284716	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	108	21 0270N 0110E 007	CA101735471	284717	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	109	21 0270N 0100E 012	CA101735472	284718	20.66	8.36	2006	10/11/2005	9/2/2025	Plumas
Teagan	110	21 0270N 0100E 012	CA101735473	284719	20.66	8.36	2006	10/11/2005	9/2/2025	Plumas
Teagan	111	21 0270N 0100E 012	CA101735474	284720	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	112	21 0270N 0100E 012	CA101735475	284721	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	113	21 0270N 0100E 012	CA101735476	284722	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	114	21 0270N 0110E 007	CA101735477	284723	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	115	21 0270N 0110E 007	CA101735478	284724	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	116	21 0270N 0110E 007	CA101735479	284725	20.66	8.36	2006	10/13/2005	9/2/2025	Lassen
Teagan	117	21 0270N 0110E 007	CA101735480	284726	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	118	21 0270N 0110E 007	CA101735481	284727	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	119	21 0270N 0110E 007	CA101735482	284728	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	120	21 0270N 0110E 007	CA101735483	284729	20.66	8.36	2006	10/13/2005	9/2/2025	Plumas
Teagan	125	21 0270N 0110E 006	CA101735486	284734	20.66	8.36	2006	10/25/2005	9/2/2025	Plumas
Teagan	126	21 0270N 0110E 006	CA101736621	284735	20.66	8.36	2006	10/25/2005	9/2/2025	Plumas
Teagan	127	21 0270N 0110E 006	CA101736622	284736	20.66	8.36	2006	10/21/2005	9/2/2025	Plumas
Teagan	128	21 0270N 0110E 007	CA101736623	284737	20.66	8.36	2006	10/21/2005	9/2/2025	Plumas
Teagan	129	21 0270N 0110E 006	CA101736624	284738	20.66	8.36	2006	10/21/2005	9/2/2025	Plumas
Teagan	130	21 0270N 0110E 007	CA101736625	284739	20.66	8.36	2006	10/25/2005	9/2/2025	Plumas
Teagan	131	21 0270N 0110E 007	CA101736626	284740	20.66	8.36	2006	10/21/2005	9/2/2025	Plumas

Claim Name	Claim No.	Meridian Township Range Section	Serial Number	CAMC	Areas (acres)	Area (hectares)	Book	Date Located	Next Payment Due Date	County
Teagan	148	21 0270N 0110E 007	CA101512291	285680	20.66	8.36	2006	4/10/2006	9/2/2025	Plumas
Teagan	149	21 0270N 0110E 007	CA101512292	285681	20.66	8.36	2006	4/10/2006	9/2/2025	Plumas
Teagan	150	21 0270N 0110E 007	CA101512293	285682	20.66	8.36	2006	4/23/2006	9/2/2025	Plumas
Teagan	151	21 0270N 0110E 007	CA101512294	285683	20.66	8.36	2006	4/23/2006	9/2/2025	Plumas
Teagan	152	21 0270N 0110E 008	CA101512295	285684	20.66	8.36	2006	4/23/2006	9/2/2025	Plumas
Teagan	153	21 0270N 0110E 008	CA101512296	285685	20.66	8.36	2006	4/23/2006	9/2/2025	Plumas
Teagan	174	21 0270N 0110E 017	CA101512299	285706	20.66	8.36	2006	4/30/2006	9/2/2025	Plumas
Teagan	181	21 0270N 0110E 018	CA101512300	285713	20.66	8.36	2006	4/25/2006	9/2/2025	Plumas
Teagan	182	21 0270N 0110E 018	CA101512301	285714	20.66	8.36	2006	4/25/2006	9/2/2025	Plumas
Teagan	183	21 0270N 0110E 018	CA101512302	285715	20.66	8.36	2006	4/25/2006	9/2/2025	Plumas
Teagan	184	21 0270N 0110E 007	CA101512303	285716	20.66	8.36	2006	4/25/2006	9/2/2025	Plumas
Teagan	189	21 0270N 0110E 018	CA101513463	285753	20.66	8.36	2006	5/30/2006	9/2/2025	Plumas
Teagan	191	21 0270N 0110E 018	CA101513465	285755	20.66	8.36	2006	5/30/2006	9/2/2025	Plumas
Teagan	193	21 0270N 0100E 013	CA101513466	285756	20.66	8.36	2006	5/30/2006	9/2/2025	Plumas
Teagan	195	21 0270N 0100E 013	CA101513469	285759	20.66	8.36	2006	5/30/2006	9/2/2025	Plumas
Teagan	197	21 0270N 0100E 013	CA101512306	285721	20.66	8.36	2006	5/6/2006	9/2/2025	Plumas
Teagan	199	21 0270N 0100E 013	CA101512308	285723	20.66	8.36	2006	5/6/2006	9/2/2025	Plumas
Teagan	203	21 0270N 0100E 013	CA101513459	285727	20.66	8.36	2006	5/6/2006	9/2/2025	Plumas
Teagan	204	21 0270N 0100E 013	CA101513460	285728	20.66	8.36	2006	5/6/2006	9/2/2025	Plumas
Teagan	216	21 0270N 0110E 005	CA101355403	286546	20.66	8.36	2006	10/14/2006	9/2/2025	Plumas
Teagan	224	21 0270N 0110E 005	CA101855211	285898	20.66	8.36	2006	6/19/2006	9/2/2025	Plumas
Teagan	232	21 0270N 0110E 007	CA101733186	286055	20.66	8.36	2006	7/23/2006	9/2/2025	Plumas
Teagan	233	21 0270N 0110E 007	CA101733187	286056	20.66	8.36	2006	7/23/2006	9/2/2025	Plumas
Teagan	234	21 0270N 0100E 012	CA101733190	286062	20.66	8.36	2006	7/23/2006	9/2/2025	Plumas
Teagan	235	21 0270N 0100E 012	CA101733182	286049	20.66	8.36	2006	7/23/2006	9/2/2025	Plumas
Teagan	236	21 0270N 0100E 012	CA101733183	286050	20.66	8.36	2006	7/23/2006	9/2/2025	Plumas
Teagan	237	21 0270N 0100E 012	CA101733184	286051	20.66	8.36	2006	7/23/2006	9/2/2025	Plumas
Teagan	508	21 0270N 0110E 017	CA101656361	293354	20.66	8.36	2008	8/19/2008	9/2/2025	Plumas
Teagan	509	21 0270N 0110E 017	CA101656362	293355	20.66	8.36	2008	8/19/2008	9/2/2025	Plumas
Teagan	511	21 0270N 0110E 017	CA101656870	293357	20.66	8.36	2008	8/20/2008	9/2/2025	Plumas
Teagan	512	21 0270N 0110E 017	CA101656871	293358	20.66	8.36	2008	8/20/2008	9/2/2025	Plumas
Teagan	513	21 0270N 0110E 017	CA101656872	293359	20.66	8.36	2008	8/20/2008	9/2/2025	Plumas
Teagan	514	21 0270N 0110E 017	CA101656873	293360	20.66	8.36	2008	8/20/2008	9/2/2025	Plumas

Claim Name	Claim No.	Meridian Township Range Section	Serial Number	CAMC	Areas (acres)	Area (hectares)	Book	Date Located	Next Payment Due Date	County
Teagan	523	21 0270N 0110E 020	CA101656878	293369	20.66	8.36	2008	8/17/2008	9/2/2025	Plumas
Teagan	525	21 0270N 0110E 020	CA101656879	293371	20.66	8.36	2008	8/17/2008	9/2/2025	Plumas
Teagan	527	21 0270N 0110E 020	CA101656880	293373	20.66	8.36	2008	8/16/2008	9/2/2025	Plumas
Teagan	529	21 0270N 0110E 020	CA101656881	293375	20.66	8.36	2008	8/16/2008	9/2/2025	Plumas
Teagan	531	21 0270N 0110E 020	CA101656882	293377	20.66	8.36	2008	8/16/2008	9/2/2025	Plumas
Teagan	533	21 0270N 0110E 020	CA101656883	293379	20.66	8.36	2008	8/16/2008	9/2/2025	Plumas
Teagan	535	21 0270N 0110E 020	CA101656884	293381	20.66	8.36	2008	8/16/2008	9/2/2025	Plumas
Teagan	537	21 0270N 0110E 020	CA101656885	293383	20.66	8.36	2008	8/16/2008	9/2/2025	Plumas

In acquiring the property from Starfield, Crown Gold assumed the original terms of the lease agreement for the 36 patented lode and 162.13 acres (65.61 hectares) of fee lands claims made between Sheffield and the California-Engels Mining Company signed April 24, 2006. Those terms are presented below:

Sheffield Inc. (ASI) entered an “Exploration Permit with option to Lease and Purchase” (the “Agreement) with California-Engels Mining Company (California – Engels). The optioned block consists of six fee property claims (162.12 acres) and 36 patented lode mineral claims (735.98 acres); full details are shown in Appendix A. The terms of the agreement as follows have been provided to the Author by the management of Sheffield:

> Exploration Permit: Sheffield must pay US$20,000 on signing of the Agreement to initiate the Exploration Lease which has a term of 120 days for completion of due diligence studies and selection of lands to be included in the Mining Lease. The Exploration Permit terminated when Sheffield notified California-Engels of its decision as to include all optioned lands in a Mining Lease.
>
> Mining Lease: Sheffield paid US$1000 to initiate the Mining Lease and upon acceptance by the TSX-Venture Exchange American Sheffield issued 50,000 Sheffield common shares to California-Engels. (money has been paid and shares were issued) On each anniversary of the acceptance during the currency of the Mining Lease Sheffield will pay California-Engels US$20,000 and will on each of the first two anniversaries issue to that company 100,000 Sheffield common shares. (all monies owing to November, 2013 have been paid and all shares owed have been issued).
>
> In the event Sheffield completes a bankable feasibility study on the California-Engels properties or begins construction of a mill for commercial production of mineral products from the property, Sheffield will in the first instance of each event issue to California-Engels 200,000 Sheffield common shares.
>
> Sheffield will during the currency of the Mining Lease perform a minimum of US$25,000 annually or work on the property and will pay any land taxes assessed against the property.
>
> Purchase of Property: Sheffield may, at a time of its selection and before commencement of commercial mining on the properties, purchase the California-Engels properties that are subject to this agreement by paying to California-Engels at Sheffield’s election either US$10 million or issuing one million shares of Sheffield common shares. Sheffield has the right to make payment in cash or shares at its sole discretion.
>
> California-Engels reserves for itself the rights to timber on the property and the right to manage said timber as a tree farm. Said timber management activities may not interfere with Sheffield’s exploration or mining activities. In the event Sheffield notifies California-Engels that the timber must be removed to make room for Sheffield’s activity, California-Engels must remove the timber, or Sheffield may harvest the timber on behalf of California-Engels and recover Sheffield’s costs by deducting them from the proceeds of the sale of the timber.

California-Engels also reserves for itself the rights to specified dumps of broken rock which may be sold to third parties or used in maintaining the roads on the property.

California-Engels further reserves for itself a 2% Net Smelter Return Royalty capped at US$25,000,000.

On purchase of the property the annual payments increase to US$60,000 and are deductible from future royalty payments”

In summary, the total area of the Crown Gold claim block, minus the area covered by the overlapping claims, is approximately 6,056 acres (2,450 hectares) and is summarized in Table 4-2.

Table 4-2: Mineral Claims Summary

Claims	Number of Claims	Acres	Hectares
Unpatented claims (22.06 acre basis)	270	5,578	2,257
Patented claims	36	736	298
Fee lands		162	65
Subtotal patented and unpatented	306	6,476	2,620
Approximate overlap with patented claims	28	-219	-89
Approximate overlap with unpatented claims	12	-201	-81
totals	**266**	**6,056**	**2,450**

4.2 Permitting and Environmental Liabilities

Exploration on Federal lands requires a permit to conduct exploration except for sampling of rocks and soils by hand and activities that create no land disturbance. The three levels of permits reflect increasing disturbance:

- No permitting is necessary for surface exploration on the patented mining claims on the Superior Project.
- Sampling of rocks and soils by hand would require no permit. Activities that create no land disturbance would also be permitted.
- The lowest level is Categorical Exclusion (CE). This is the least intense disturbance and requires some public notification. Track mounted auger drilling and no new road clearing would fit in this category according to United States Forest Service (USFS) personnel. A lead time of three to four months would be required to grant this level of permit.
- Environmental assessment (EA) requires an in-depth study with 30 days for public comment, plus additional time for appeal. Drilling with an RC rig using water, new road construction, etc., would require this level of permit. USFS personnel suggest that one year may be required to receive a permit. Studies on archaeology and sensitive plant species would be required prior to disturbance.
- Environmental Impact Statement (EIS) is the highest permit level and would be required for mine development. Several aspects should be factored into timing of exploration plans.
- The time needed to issue permits is governed by available USFS personnel resources or for the company to hire an outside approved consultant to complete the work.

During the dry season, the threat of forest fires may limit access to the area.

Exploration and mining can be conducted year-round, due to the established road and its proximity to infrastructure. Additional claims may be needed to support all future exploration or mining operations including facilities and potential waste disposal areas. Potential processing plant sites may have to be located closer to water. Controlling the mineral rights under valid lode claims will not fully entitle the company to develop a mine. Permitting will need to be carefully planned and executed to be sustainable in the community and this area of California.

California is often perceived as having a restrictive regulatory environment in regard to mining operations. Historically mining operations have been permitted even when there were legitimate social or environmental concerns. Specific examples of successful permitting in California include:

- The open pit mines at Carson Hill and Jamestown were permitted and operated to their economic limit in very close proximity to residential and commercial development.
- Approval was required by three separate counties and the federal government for the open pit Mclaughlin Mine. It was permitted and operated until reserves were exhausted in a geologic environment with high levels of toxic metals.
- The Sutter Creek and Washington-Niagara Mines have recently received permits to conduct mining and milling operations. Underground development is proceeding at both operations.
- Equinox Gold Corp, a publicly listed gold producer, operates two open pit mining operations in southern California- the Mesquite mine and the Castle Mountain mine.
  - Mesquite is an open pit, run-of-mine heap leach gold mine located in Imperial County, California, USA, 16 miles west of the state border with Arizona and 24 miles north of the border with Mexico. Mesquite has produced more than five million ounces of gold since it commenced operations in 1986, with annual gold production averaging approximately 125,000 ounces over the last 10 years.
  - Castle Mountain is an open pit heap leach gold mine in San Bernadino County that is currently working on an expansion of their current activities. Phase 2 expansion will be contained within existing approved mine boundary but requires modifications to the Mine and Reclamation Plan and an updated Environmental Impact Statement due to increased land disturbance within mine boundary, and increased water use.
- MP Materials Corp. is an American rare-earth materials company headquartered in Las Vegas, Nevada, and is listed on the NYSE. MP Materials owns and operates the Mountain Pass mine in San Bernadino County, the only operating rare earth mine and processing facility in the United States. It is an open pit mining operation with a 24-year remaining mine life.

4.3 Water Rights

The PEA assumes that water rights can be acquired for mine operations, but this must be confirmed in subsequent study phases.

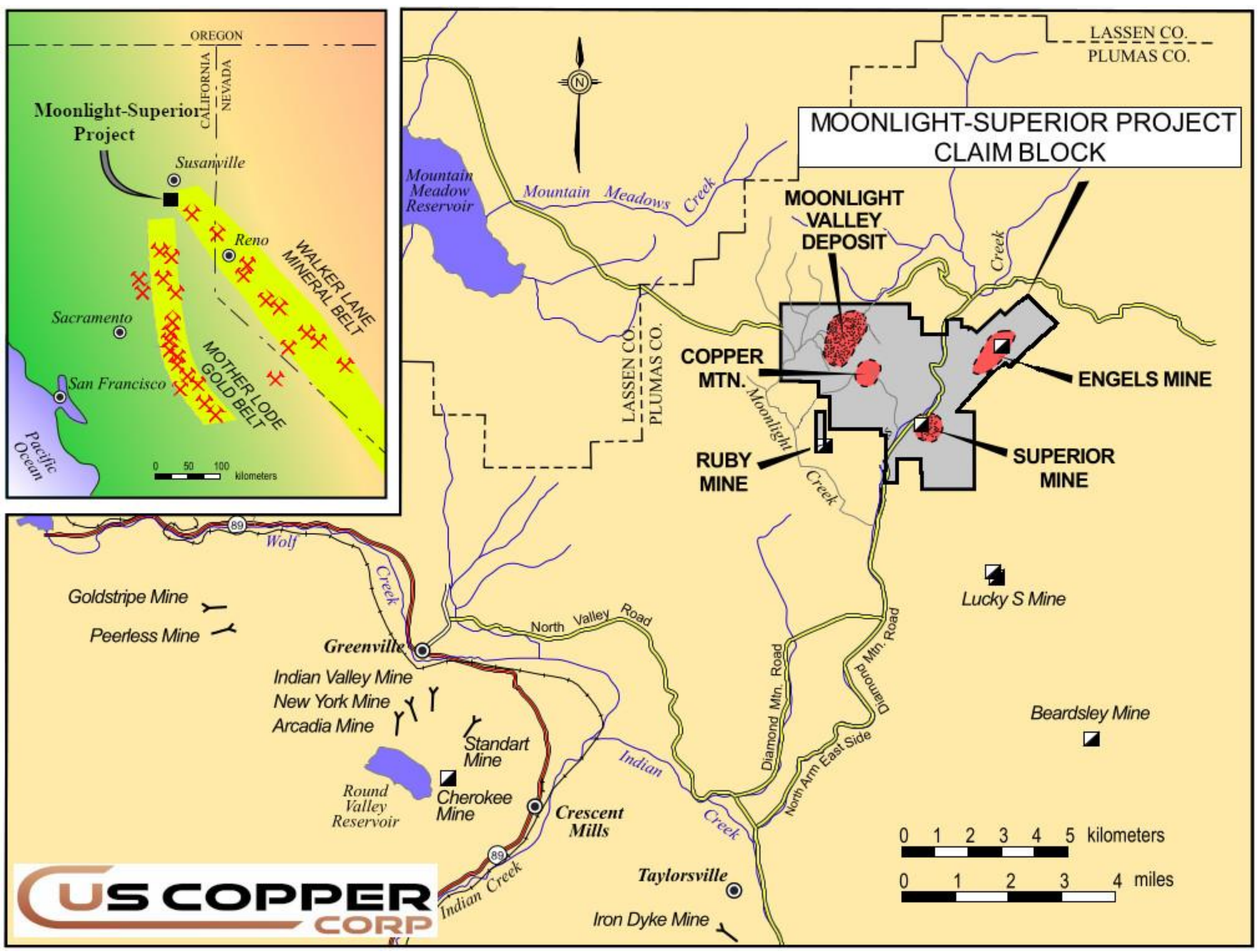
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access and Climate

The bulk of the following property description was taken from the April 2013 NI 43-101 report of the Moonlight-Superior Project prepared by Tanaka, Independent Mineral Consultant Consultants, for US Copper Corp. (Crown Gold Corp.).

The property can be accessed from the Reno Nevada International airport by US Interstate 395 north for approximately 5.3 miles, then Quincy/Crescent Mills Road (State Highway 70) northwest for approximately 86 miles to the town of Greenville California, then by the North Valley Road and Diamond Mountain Road northeast for approximately 18 miles to the project location. Figure 5-1 presents the access and cultural features surrounding the Moonlight-Superior Project property.

Figure 5-1: Project Property Location Map



The climate is defined by hot summers to a maximum of 100 °F (38 °C) and cold, windy winters with lows to -9 °F (-23° C). Precipitation is moderately light with average rainfall of 30 inches (76 centimeters [cm]) and average snowfall of approximately 11.5 feet (3.5 meters). 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 during drilling phases. The Mountain Meadows Reservoir is located approximately six miles to the west-northwest of the property which could supply water for all advanced exploration activities on the property.

5.2 Local Resources and Infrastructure

The paved Diamond Mountain Road from Greenville provides direct access to the Superior adit. The remainder of the claim blocks are accessible via a series of gravel roads, many of which are actively used by logging companies operating east of the company's claim block. The access is fair across the current project ground using active forestry roads and drill access roads completed by Placer in the 1960s and 1970s.

No homes are located on the property. The nearest ranch and home is located approximately 3 mi (5 km) west-southwest on the secondary access road off Highway 36.

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, which runs through the town of Westwood, approximately 15 road miles (24 road km) to the west of the property. International air services are located in Reno, Nevada, approximately 79 miles (127 km) southeast of Susanville. The closest deep-water port is Sacramento, which is located approximately 150 miles (241 km) to the southwest.

There is a very large, highly trained mining-industrial workforce available in Northern Nevada. Supplies and services for mining companies to conduct full exploration and mining development projects are available at Carlin, Elko, Winnemucca, and Reno. There are also additional workforce resources in the nearby towns of Quincy and Greenville.

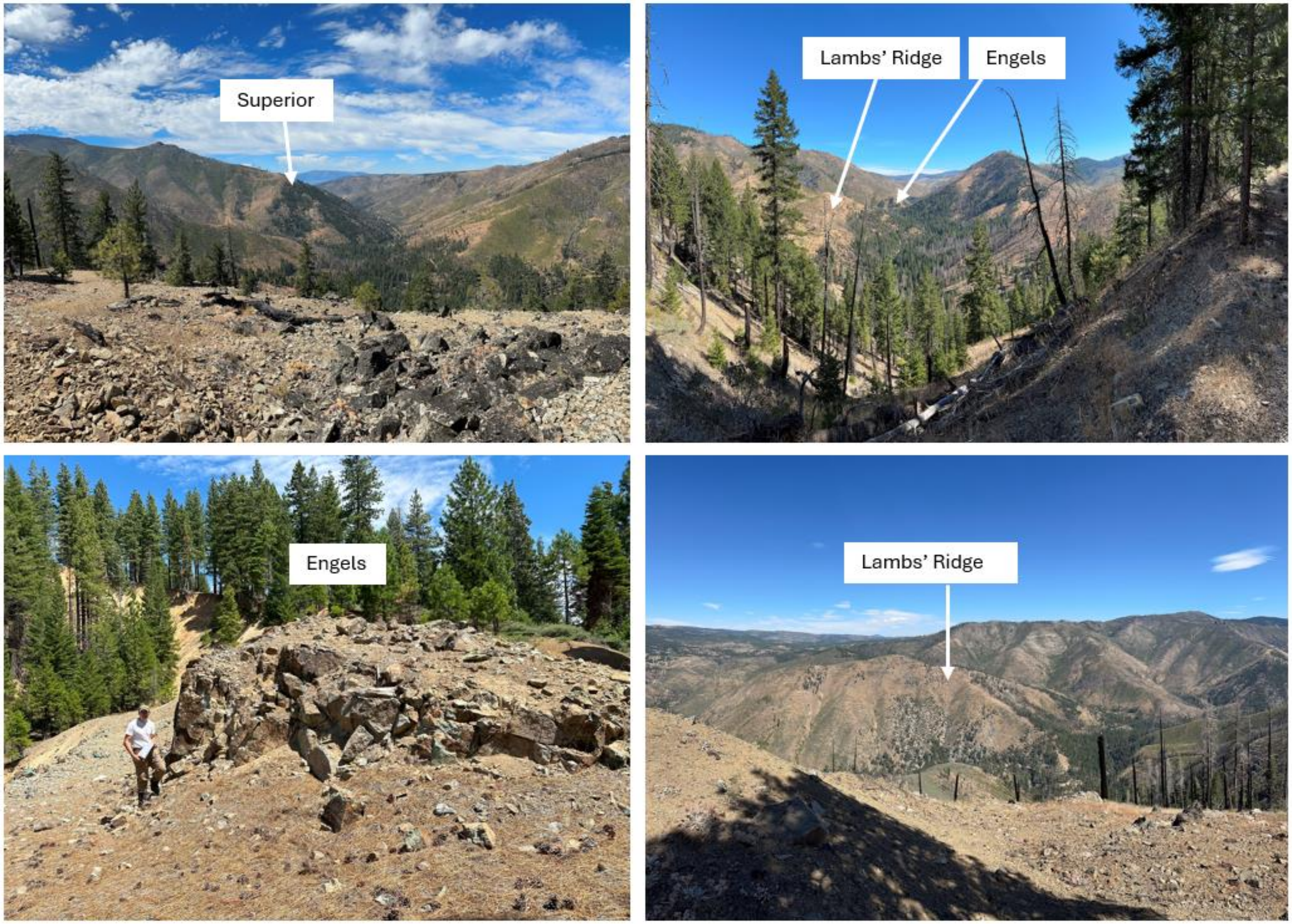
Exploration and mining could be conducted year-round, due to the established roads and the project's proximity to the nearby towns. Exploration in winter will incur additional costs for regular snow removal.

The property has sufficient surface rights for future exploration or mining operations although there is likely to be a requirement to lease nearby flat land available within a six-mile (10-km) radius for including potential waste disposal areas and tailings storage facilities.

5.3 Physiography

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 5,518 feet (1,682 meters) to a high of approximately 6,420 feet (1,957 meters). There are a few bedrock exposures on the property (Figure 5-2).

Figure 5-2: Physiographic General View of the Deposit with the Project Area



Source: GRE,2024

Note: Superior deposit due south of the Engels deposit and Lambs Ridge looking west from Engels

6 HISTORY

6.1 Historical Exploration, Development, and Ownership

This property history is taken and modified from "Technical Report and Resource Estimate for the Superior Project, Plumas Co. California" by William F. Tanaka, 15 November 2013 and 2018 NI 43-101 report prepared by Tetra Tech for US Copper Corp. (Crown Mining Corp.).

Henry A. Engels and his sons acquired the Superior Mine in 1880 and discovered the Engels Mine in 1883.

Since 1930, activity in the Lights Creek District (LCD) has largely been limited to exploration. Newmont Mining explored the area in 1953 and 1954 and completed a preliminary aerial geologic map of the Lights Creek area. Phelps Dodge conducted some investigations in the early 1960s. Lessees mined a few thousand tons of ore from the Superior in the early 1960s. This ore was shipped directly to the smelter, and reportedly often ran more than 10 % copper (Cu) and 4 ounces per ton (opt) silver (Ag).

In 1961, Amex (predecessor to Placer-Amex, Placer Dome, Barrick) decided to pursue a general investigation of the LCD. Reconnaissance surveys were conducted in 1962 and 1963, and extensive stream sediment and soil sampling surveys were conducted in 1964 and 1965. The Superior, Moonlight Valley, Lamb's Ridge, Engels Mine, Warren Creek and Blue Copper areas all showed plus 1,000 parts per million (ppm) Cu anomalies in soils.

Amex signed a sublease on the California-Engels property in July 1964 and began drilling at the Superior in September 1964. Drilling at Superior was completed in January 1967, and the results indicated a considerable tonnage of low-grade disseminated copper. The first hole in the Lamb's Ridge soil geochemical anomaly was drilled in December 1964, and the first claims in Moonlight Valley were staked in December 1964. The first holes in the Engels Mine and Warren Creek anomalies were drilled in September and October 1965.

Extensive errors were discovered in the drill core assays conducted by the Amex lab at the Golden Sunlight project in Montana, and the process of re-assaying all the pulps at Union Assay in Salt Lake was begun in October 1967. This re-assaying was completed in April 1968 and included third party QC assays by Hawley and Hayley and the Amex lab in Vancouver.

Drilling continued in 1968, 1969, and 1970. A total of 219,914.87 feet (67,033.1 meters) of drilling had been completed in the Lights Creek district from 1964 to 1970 by Placer-Amex drilling.

Preliminary metallurgical investigations were begun and the first of many deposit modeling and economic evaluations was begun in 1968. Computer models, a resource estimate, economic evaluation, permitting inquiries, and a summary report were completed in February 1972.

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 acres 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.

Sheffield began drilling to confirm and enhance the previously indicated historical resource at Moonlight Valley in December 2005. In 2005, 1,837 feet (560 meters) of HQ core drilling were completed. A total of 9,298 feet (2,834 meters) of core drilling were completed in 2006, with 7,618 feet (2,322 meters) completed at Engels. Sheffield completed 7,614 feet (2,320.7 meters) of core drilling at Engels in 2007.

Sheffield Resources was acquired by Nevoro Copper in July 2008. Nevoro completed 4,075.8 feet (1,242.3 meters) of core drilling at Engels in 2008. Additional unpatented lode claims were staked by Nevoro in 2007 (33 total), 2008 (23 total), and 2011 (12 total). The 2011 staking program was designed to cover any un-staked corners and fractions present between the patented and unpatented lode claims at Engels and Superior.

In 2009, Starfield Resources Inc. acquired Nevoro Inc., the parent company of Nevoro Copper Inc. and conducted a total of 2,071.5 feet (631.4 meters) of drilling at Engels in 2009 and 2010. They also contracted a property-wide airborne geophysical survey conducted by Fugro Airborne Surveys. Starfield dropped the unpatented claims encompassing the Moonlight deposit in 2012.

US Copper acquired the claims/patents covering Superior and Engels from the Trustee in Starfield's bankruptcy on June 27, 2013, including a minor amount of exploration and office equipment and supplies, the stored core and the complete database held by Nevoro which comprehensively documents all known exploration activity on the property from 1960 to 2013. In 2016, US Copper optioned the Moonlight claims from Canyon Copper and finalized the purchase in 2018.

6.2 Historical Mineral Resource Estimates

Historical resource estimates were compiled for the Engels, Superior, and Sulfide Ridge areas by Placer-Amex in the early 1970s. These estimates predate NI 43-101 guidelines and none of the following are regarded by the Author as compliant with current National Instrument 43-101 standards for reporting of resources and reserves. The Author here stresses that any reporting of Resource or Reserve categories referred to by Placer-Amex cannot be regarded as corresponding to current CIM definitions. Furthermore, US Copper is not treating these historic estimates as current mineral resources or mineral reserves.

Historical Resource Estimates for Engels:

- Placer-Amex determined in the 1970s that there may still be a small open pit potential of approximately two million tons grading 0.65% Cu remaining in the pillars and immediate area along strike.
- Additional indicated and inferred resources of 19 million tons averaging 0.63% Cu were reported by Placer-Amex that were not considered amenable to open pit mining methods at the time of the work.

- Placer-Amex also reported a small tonnage, 68,000 tons of 2% Cu remaining in the shaft level sill pillar.
- In 2013, Tanaka developed the resource estimate for Engels and reported 2.5 million tons at 1.05% Cu (Inferred).

Historical Resource Estimates for Superior:

- Preliminary "potential ore reserves" for Superior were estimated by Placer-Amex in 1967 ("*Preliminary Evaluation of Superior Pit, Lights Creek*", W.D. Baker, April 1967) of 54 million tons grading 0.60% Cu at an unspecified Cu cutoff.
- Preliminary computerized "ore reserves" for Superior were estimated by Placer-Amex of 43 million tons grading 0.559% Cu with a 0.3% Cu cutoff.
- In 1971-72, Placer-Amex completed further computer designed resource estimates using a 0.25% cutoff and reported "minable reserves within a smoothed ultimate pit" using the inverse distance to the 5th power as a block estimator, of 39 million tons grading 0.41% Cu with a strip ratio of 1.2:1 (Placer-Amex, 1972).
- In 2013, Tanaka developed the resource estimate for Superior and reported 54 million tons at 0.41 % Cu (Inferred).

Historical Resource Estimates for Lamb's Ridge:

Preliminary "potential ore reserves" for Lamb's Ridge were estimated by Placer-Amex in 1967 (Baker) of 100 million tons grading 0.45% Cu at an unspecified Cu cutoff.

Historical Resource Estimates for Moonlight:

- A number of Mineral Resource estimates were generated by Placer-Amex, as shown in Table 6-1.

Table 6-1: Historical Placer-Amex Moonlight Mineral Resource Estimates

Year	Tons	Grade (Cu%)	Cutoff (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 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 feet (6 meters) 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. The Gillette oxide mineral Resource Estimate is shown in Table 6-2.

Table 6-2: Historical Gillette Moonlight Oxide Mineral Resource Estimate

Area	No. of Holes	Area (square feet)	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	

- 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, et al., 2007).

Table 6-3: Historical Sheffield Moonlight Mineral Resource Estimate

Cutoff (Cu%)	Tons > Cutoff (tons)	Grade > Cutoff		
		Cu (%)	Au (oz/st)	Ag (oz/st)
Moonlight Indicated Resource Grade-Tonnage Table				
0.20	161,570,000	0.324	0.003	0.009
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

- In 2018, Tetra Tech prepared a technical report for Moonlight and reported mineral resources for the Moonlight copper deposit, with Donald Cameron as QP, as shown in Table 6-4 for Indicated and Inferred Mineral Resources. The cutoff for the reported resources was a US$6.25 NSR (NSR = 44.08*Cu + 0.348*31.10348*Ag).

Table 6-4: Historical Tetra Tech Moonlight Mineral Resources as of December 15, 2017

Class	Tons ('000s 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

6.3 Historical Production

Operations began in 1880 and continued until 1930. The main period of operation was between 1915 and 1930. Operations were suspended in 1930 due to a significant fall in the copper price in response to the Great Depression.

The total reported production from the Engels and Superior Mines was approximately 160 million pounds of copper, 23,000 ounces of gold, and 1.9 million ounces of silver recovered from 4.7 million short 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 0.5 opt Ag and 0.005 opt gold (Au).

7 GEOLOGICAL SETTING AND MINERALIZATION

The following description of the regional and local geology, lithology, structure, mineralization, and alteration specific to the Property was prepared by William F. Tanaka, Independent Mineral Consultant in 2014 for Superior, Lamb's Ridge, and Engels, and by Tetra Tech for Moonlight in 2018. Presented here is an excerpt from the 2014 and 2018 NI 43-101 Technical Reports issued by William F. Tanaka and Tetra Tech for US Copper Corp.

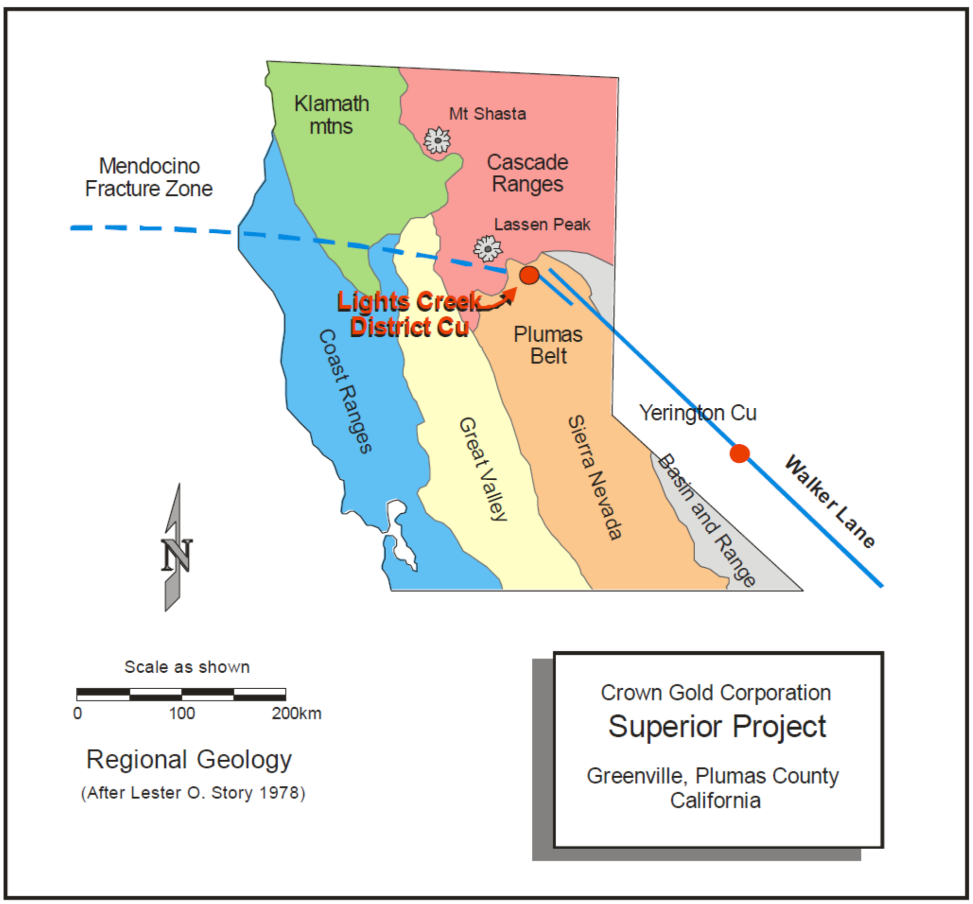
Dr. Hamid Samari of GRE has reviewed this information and associated supporting documentation in detail, together with a field visit to the property, and finds the discussion and interpretations presented herein to be reasonable and suitable for use in this report.

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 miles southeast of the Property. Numerous small mines and copper showings are located between the Walker Mine and the LCD. The Walker Lane has hosted some of the largest precious and base metal mines in the western US including the Yerington District about 160km southeast of Lights Creek, estimated to host the potential for a 20-billion-pound copper resource (Tetra Tech, 2012). Greenschist facies Mesozoic metavolcanic rocks with a general NNW strike and southwest dip have been intruded by the late Jurassic to Early Eocene Lights Creek Stock (LCS) in the Lights Creek District.

7.2 Property Geology

A section of Jurassic weakly metamorphosed dacites, andesites, basalts, and associated volcaniclastics are exposed in the Moonlight-Superior Project area. These metavolcanics are part of a 5.6-miles (9-km) thick section of early Mesozoic metavolcanic rocks that are exposed in a northwest trending belt about 50 miles (80 km) long. The metavolcanics in the project area have a fairly uniform regional northwest strike and moderate southwest dip. The sequence above the Lights Creek Stock intrusive contact in the Moonlight Valley area is dominantly made up of a complex of andesitic flows that have been characterized as keratophyres.

The metavolcanics are intruded by Jurassic-to-Cretaceous plutonic rocks of varying composition in and around the Plumas Copper Belt.

Work by Anderson (1931) and Storey (1978) suggest there are five distinct batholithic differentiates in the Lights Creek area. According to Storey (1978) “These are from oldest to youngest:

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

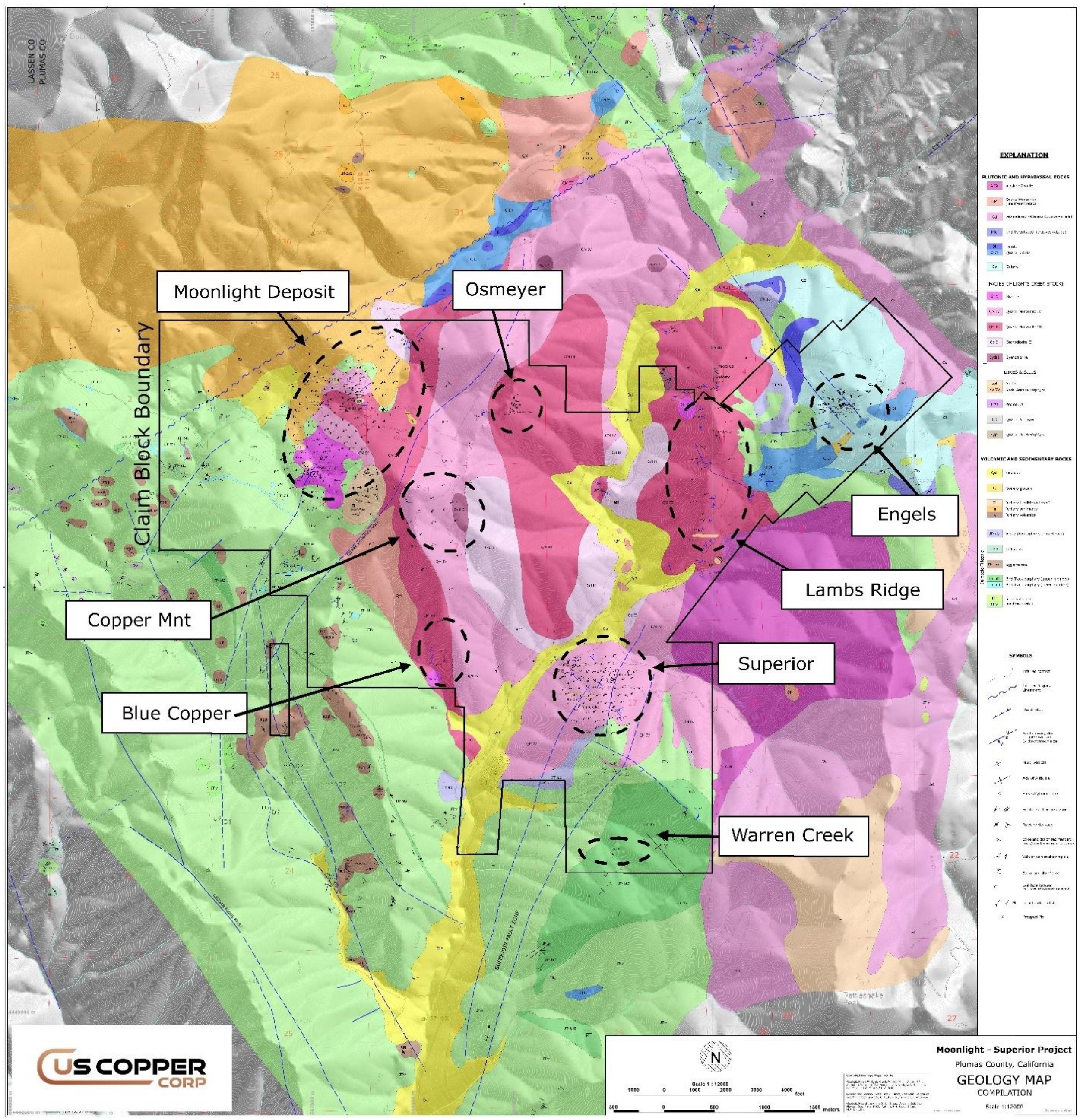
The quartz monzonite is the most heterogeneous in the overall make-up of any of the segregated intrusive bodies.” Several of these phases are shown in the simplified Placer-Amex map (Figure 7-2) as various shades of pink.

The Lights Creek Stock refers to the quartz monzonite listed above, which is the ore host at the Moonlight, Superior, and Lamb’s Ridge deposits. Surface exposures and drill intersections indicate the stock is dome-shaped with gently dipping flanks and probably underlies a much larger area than the outcrop at shallow depths. The stock appears to have domed the overlying metavolcanic with steeper dips on the flanks and flatter dips over the top of the intrusive.

The Lights Creek Stock varies considerably in texture and composition, and both Sheffield and Placer have noted that the quartz monzonite tends to be finer-grained with a more porphyritic texture near the contact with metavolcanics and less potassium feldspar-rich and more equigranular with depth and towards the center of the quartz monzonite stock.

The Lights Creek stock is a roughly circular fine to medium grained quartz monzonite to granodioritic tourmaline-rich intrusive, approximately seven sq miles (18 square km) in area, believed to represent a differentiated satellite of the Sierra Nevada batholith. Coarse-grained granodioritic Sierra Nevadan batholithic rocks are exposed a few kilometers to the east of Moonlight.

Figure 7-2: Simplified Property Geology Map



Source: Placer-Amex, updated by US Copper Corp., 2024.

7.3 Deposit Geology and Mineralization

Within the property, four main deposits, including Superior, Lamb's Ridge, Engels, and Moonlight have been explored. The first stage of exploration began in 1883 at the Superior Mine and later for the other deposits (see chapter six of this Technical Report). Existing drill hole data from the 1960s or before at these four main deposits and operation data from the Superior Mine have provided excellent data on geology, mineralization, and type of deposit across the property.

Most of the mineralization in the Lights Creek District appears to be related to the tourmaline-rich Lights Creek Stock or related dikes. While the Engels deposit lies just outside the stock, in the surrounding gabbroic-phase intrusive and metavolcanics, narrow dikes of granitic composition with abundant tourmaline have been noted. These dikes are interpreted to be late stage differentiates of the Lights Creek Stock and often display pegmatitic textures.

Mineralization in the LCD has been characterized as of the porphyry copper type. Placer, however, recognized that the deposits had characteristics which were not typical of porphyry copper deposits and lacked many of the typical features. Storey (1978) noted, “Typical porphyry copper-type alteration zonation as illustrated by Lowell and Guilbert is nonexistent.” Some of the early disseminated mineralization at Moonlight and Superior show some similarity to the diorite model porphyries common in British Columbia.

Many copper deposits which had previously been classified as porphyry copper type have now been re-characterized as belonging to the iron oxide copper type (IOCG). There is evidence that all these four main deposits could be included in this group.

A number of deposits have been classified as belonging to the iron oxide copper type. These deposits range in age from Precambrian to Tertiary and include 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.

7.3.1 Geology and Mineralization at Superior Deposit

The Superior deposit lies within the Lights Creek Stock near the southeastern margin and south of Engels (Figure 7-2). The deposit is hosted within the quartz monzonite, however exposures of more mafic units interpreted to be rafted zenoliths from the intruded host rock have been observed near the southern extent of exposure.

The mineralization at Superior is hosted in the Lights Creek Quartz Monzonite and minor generally flat-lying diabase dikes. The quartz monzonite is generally more equigranular and less potassium feldspar-rich than that observed at Moonlight.

Alteration at Superior includes both silicification and potassic alteration. As at Engels, magnetite appears to be a significant alteration mineral as well. Also in common with Engels, there is very little pyrite observed at Superior.

There are significant copper oxides deposited on the exposed surfaces of the underground workings at Superior. These appear to be the result of oxidation and re-deposition from weathering dating from the period of active mining (Photo 7-1).

Photo 7-1: Copper Oxides on the Surface of Underground Working

Source: GRE 2024.

Both disseminated and associated copper mineralization with magnetite and tourmaline veinlets are seen at Superior. Disseminated copper mineralization at Superior, revealed by drilling and exposure in underground workings, lies within a roughly circular area about 2,000 feet (610 meters) in diameter. This mineralization consists of finely disseminated chalcopyrite and lesser bornite. This disseminated mineralization typically runs 0.1 to 0.3% Cu, and copper minerals are typically associated with tourmaline. Within this disseminated mineralization are more than ten tabular brecciated structures (veins) that were mined up to 800 feet (244 meters) along strike, 600 feet (183 meters) down dip, and 10 to 23 feet (3 to 7 meters) wide (Photo 7-2).

Photo 7-2: Copper Sulfide Mineralization at Superior Deposit, Disseminated and Associated with Magnetite Tourmaline Veinlets

Source: GRE 2024.

Sulfide copper mineralization, mostly chalcopyrite and bornite, is associated with magnetite tourmaline veinlets (Photo 7-3). These veinlets mostly filled the fractures, which cut the host rock in different directions. Copper sulfide minerals within the magnetite tourmaline veinlets are mostly aligned parallel to the veinlets and sometimes cut the veinlets (Photo 7-3 B) showing a younger (post) mineralization phase.

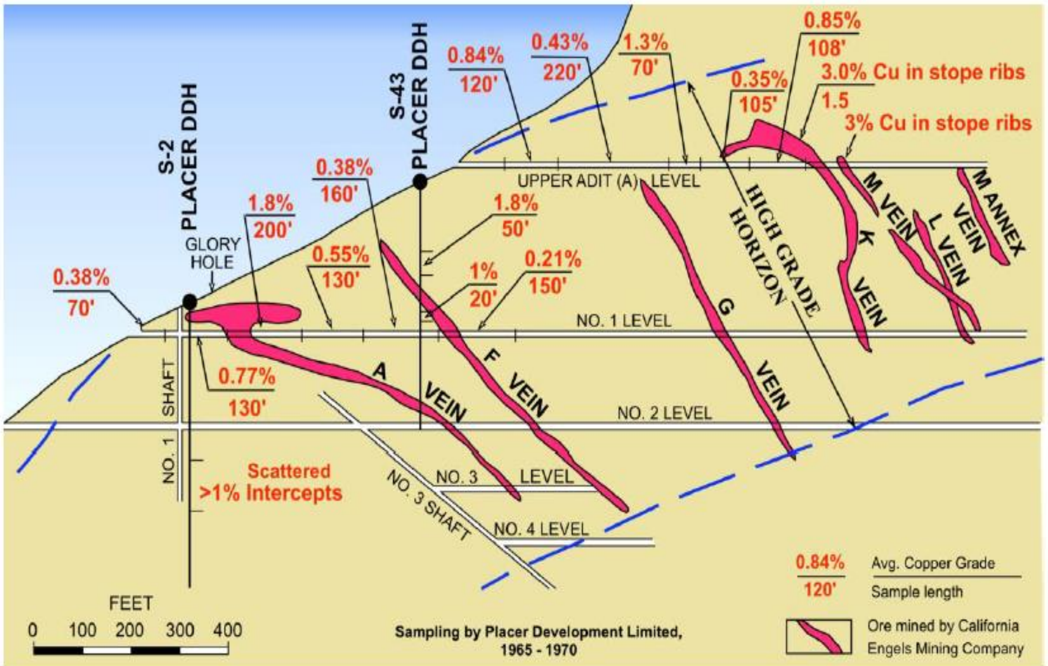
Photo 7-3: Sulfide Mineralization, which is Associated with Magnetite Tourmaline Veinlets at Superior, A) at Surface, B) in Core Hole S21-7

Source: GRE 2024.

There are two predominant trends to the breccia-veins. Veins trend north-south and dip to the east and there are a number of essentially flat lying veins. Mineralization in the breccia-veins consists of magnetite-actinolite-minor quartz-siderite-bornite-chalcopyrite. The historic mill feed from these stopes averaged about 2.2% Cu. These veins and the stockworks between them define a high-grade core to the Superior deposit.

Figure 7-3 presents a schematic cross section through Superior showing the distribution of the breccia veins as indicated by the stopes (magenta) as well as selected underground sampling results.

Figure 7-3: Schematic East-West Cross Section through Superior Showing the Distribution of Breccia Veins within the Mass of Disseminated Mineralization



Source: OreQuest, 2007

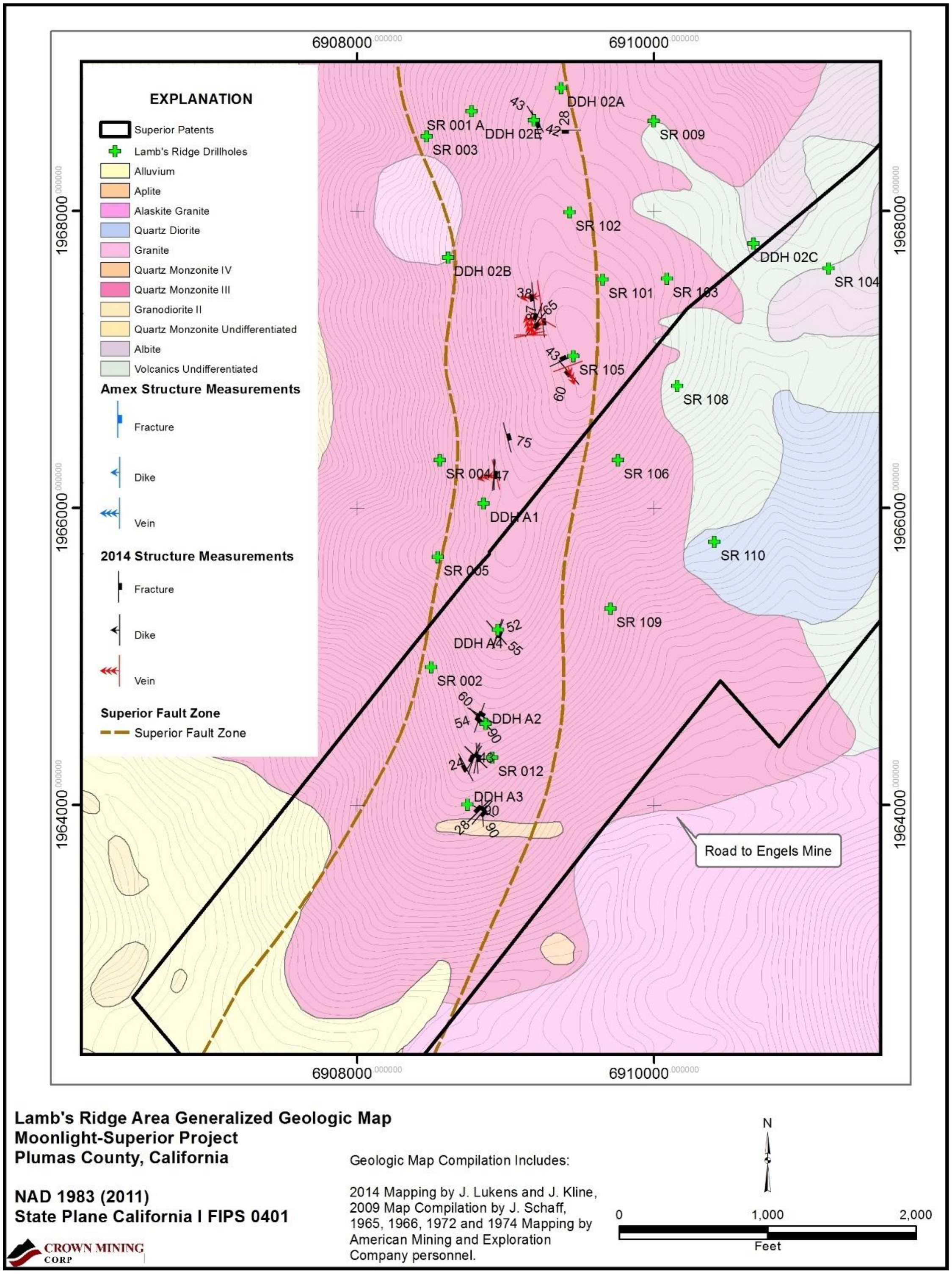
Historic mining at Superior focused on the chalcopyrite rich breccia veins. The surrounding body of disseminated copper mineralization, ignored as uneconomic in the past was subsequently defined from work completed by Placer-Amex. They drilled approximately 96 drill holes representing approximately 54,134 feet (16,500 meters) of diamond drilling (including 3,822 feet [1,165 meters] of rotary drilling) from 1964 to 1968.

7.3.2 Geology and Mineralization at Lamb Ridge Deposit

There is very little documentation regarding the geology of the Lamb's Ridge deposit. There is little outcrop visible. What can be interpreted comes largely from the very widely-spaced 28 core holes completed by Placer-Amex. US Copper Corp. also drilled two RC holes at Lamb's Ridge in 2023, totaling 820 feet (250 meters). A geological map of the Lamb's Ridge, Engels area, prepared in 1965 by Placer-Amex shows Lamb's Ridge to be hosted within the quartz monzonite of the Lights Creek Stock on the basis of small, scattered outcrops. Small prospect shafts and pits dating from the early 20th century provide additional scattered points of reference (Figure 7-4).

The geology and mineralization at Lamb's Ridge appears to be most similar to Superior and was characterized by Placer-Amex geologists as a porphyry system. The wide-spaced (328- to 656-foot [100- to 200-meter]) drilling indicates disseminated copper mineralization similar to that found at Superior; however, no occurrences of the high-grade breccia-veins mined at Superior have been encountered in the drill holes. That said, the drilling that has been done defines significant copper mineralization with copper grades in 16.5-foot (5-meter) composites exceeding 0.3% Cu over 4,921 feet (1,500 meters) north to south and 1,640 feet (500 meters) east to west.

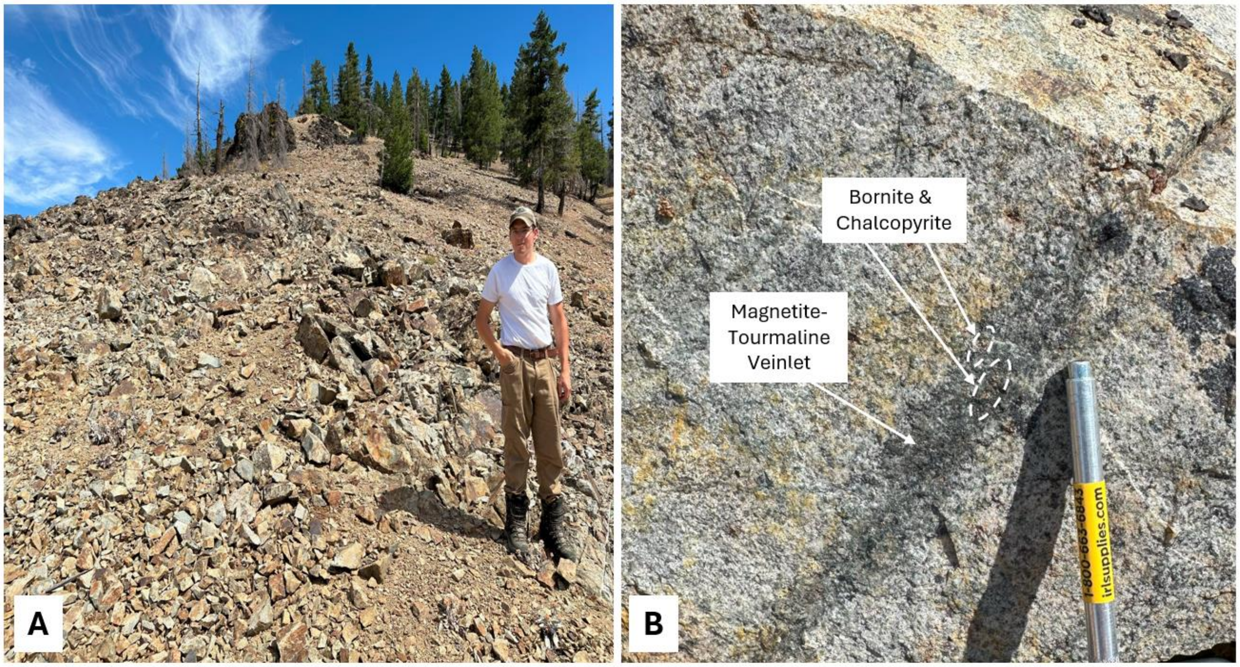
Figure 7-4: Geological Map of Lamb's Ridge



Source: US Copper Corp, 2014

Geologic data gathered by GRE's QP during the site visit confirms the similarity of geology and mineralization at Lamb's Ridge to Superior. The quartz monzonite here is also the primary host rock, and except for disseminated copper mineralization, copper sulfide minerals such as chalcopyrite are associated with magnetite and tourmaline veinlets (Photo 7-4).

Photo 7-4: A) Exposed Quartz Monzonite, B) Association of Copper Sulfide Minerals with the Magnetite-Tourmaline Veinlet at Lambs Ridge



Source: GRE 2024.

7.3.3 Geology and Mineralization at Engels Deposit

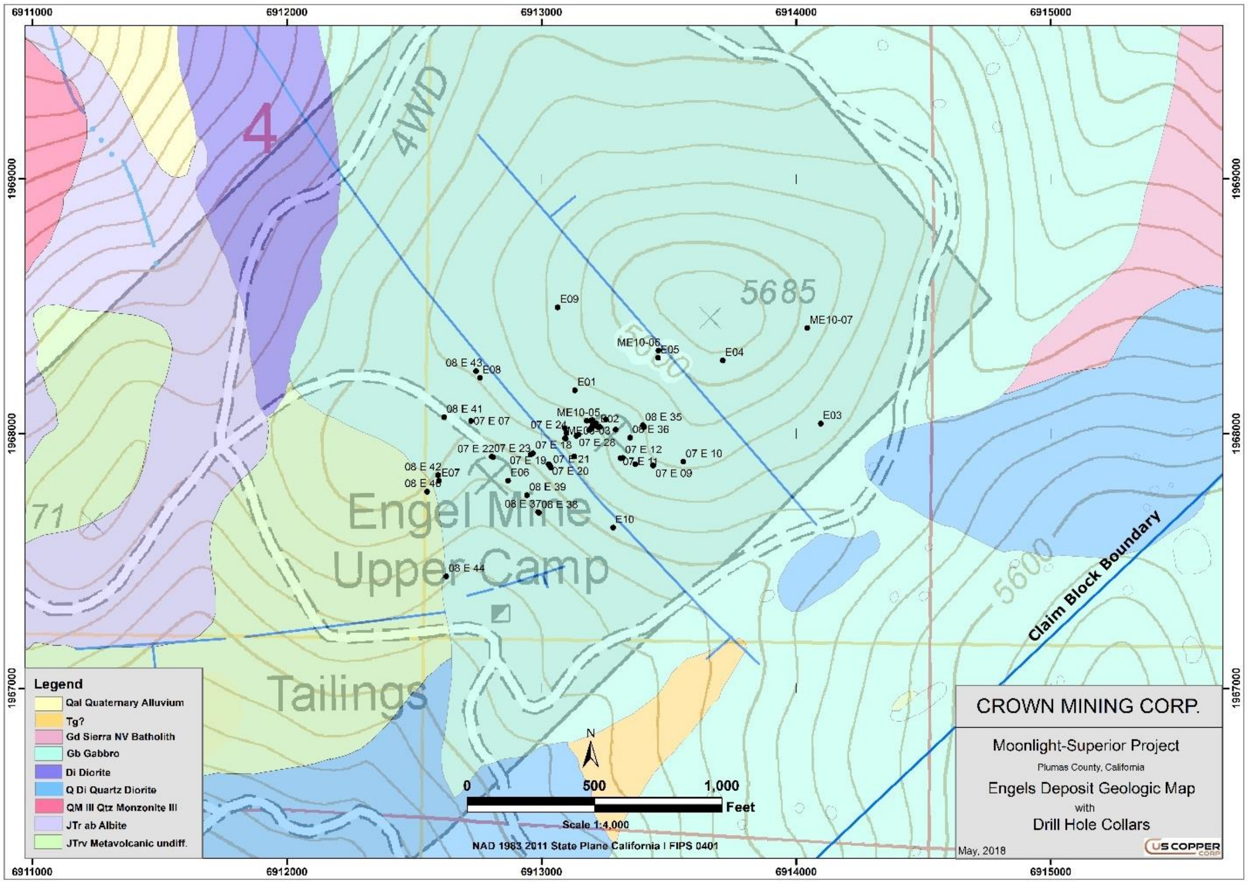
The Engels deposit lies outside the Lights Creek Stock, immediately adjacent to its eastern margin in an area represented by both gabbroic-phase intrusive and roof-pendant metavolcanics (Figure 7-5).

Engels is a structurally-controlled tabular shear-zone hosted deposit striking north-east and dipping steeply to the southeast. Mineralized widths range from 16.5 feet (5 meters) to over 66 feet (20 meters). The historically mined total strike length for the main ore shoot ranges from 328 feet (100 meters) to 820 feet (250 meters), while a narrower ore shoot to the south along strike was mined at lengths from 66 feet (20 meters) up to 197 feet (60 meters). The vertical extent mined is approximately 1,903 feet (580 meters).

Mineralization in the Engels Mine area occurs in a 1,280-foot (390-meters) by 656-foot (200-meter) pipe-like zone. Mineralization is associated with brecciated zones that exhibit features of both an intrusion breccia and a hydrothermal breccia.

A diorite or quartz diorite has intruded a pendant of plagioclase phenocryst-rich metavolcanic in a complex mass of dikes and dikelets. The fine-grained matrix of the metavolcanic has often been altered to biotite in the mine area. Primary mineralization consists of zones of silica + magnetite +- biotite hornfels alteration with varying amounts of disseminated bornite and chalcopyrite. This mineralization exhibits metasomatic textures and is most intense at or near the numerous contacts of the quartz diorite and metavolcanic (Figure 7-5).

Figure 7-5: Geology Map of Engels



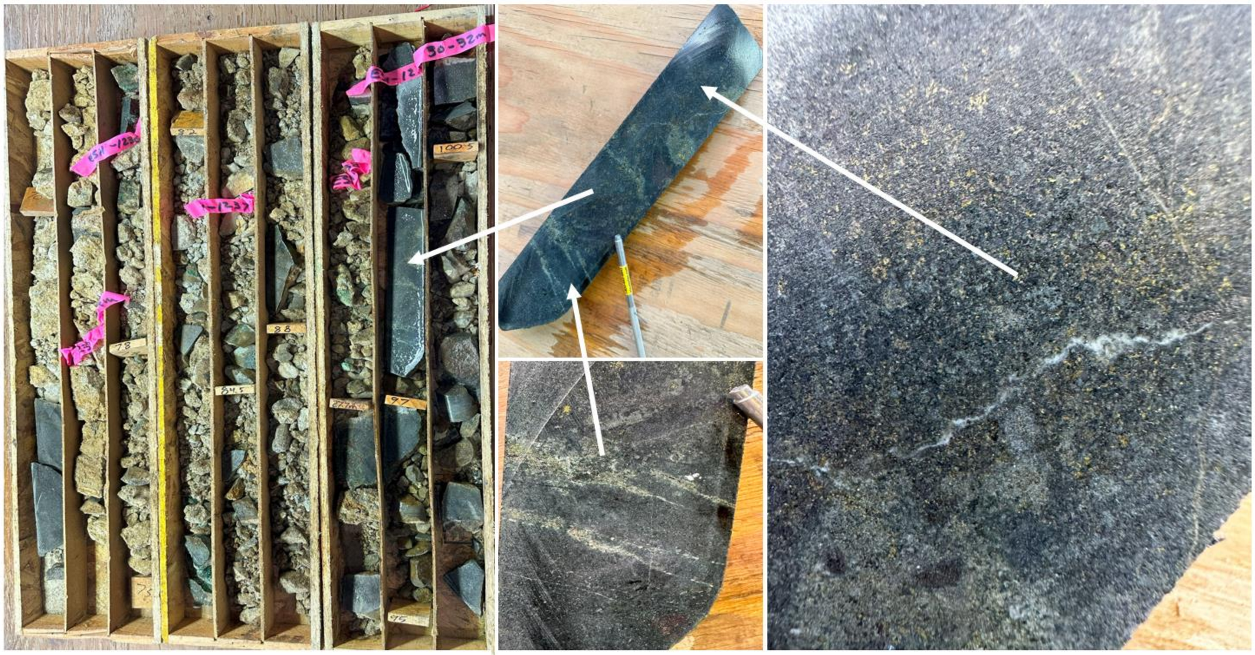
Source: Brady and Knowlton, 1982, Updated by US Copper Corp, 2014

The disseminated copper minerals are often very abundant and locally coalesce. Copper grades exceeding 15% Cu have been encountered in several 6.5-foot (2-meter) core intercepts. The relationship of mineralization to zones of breccia and contacts between the quartz diorite and metavolcanic is evident in surface exposures (Photo 7-5).

Calc-silicate minerals, especially epidote and locally garnet, are also present. The specific gravity varies widely. Magnetite or sulfide-rich rock often has a specific gravity of more than 2.8.

Much of the copper mineralization at Engels is strongly oxidized to a depth of 230 feet (70 meters) (Photo 7-6). Assay analysis for sulfuric acid soluble copper in a portion of samples from the modern (post 2004) drilling indicates copper oxides representing 90% of total copper within these depths.

Photo 7-5: Veinlet and Disseminated Mineralization at Engels



07 E 32, 67-80 ft   07 E 32, 80-90 ft   07 E 32, 90-101.5 ft   Close up view from Gabbro, veinlet and disseminated sulfide within the rock.

Source: GRE, 2024

Photo 7-6: Significant Copper Oxides at Engels Deposit

Source: GRE, 2024

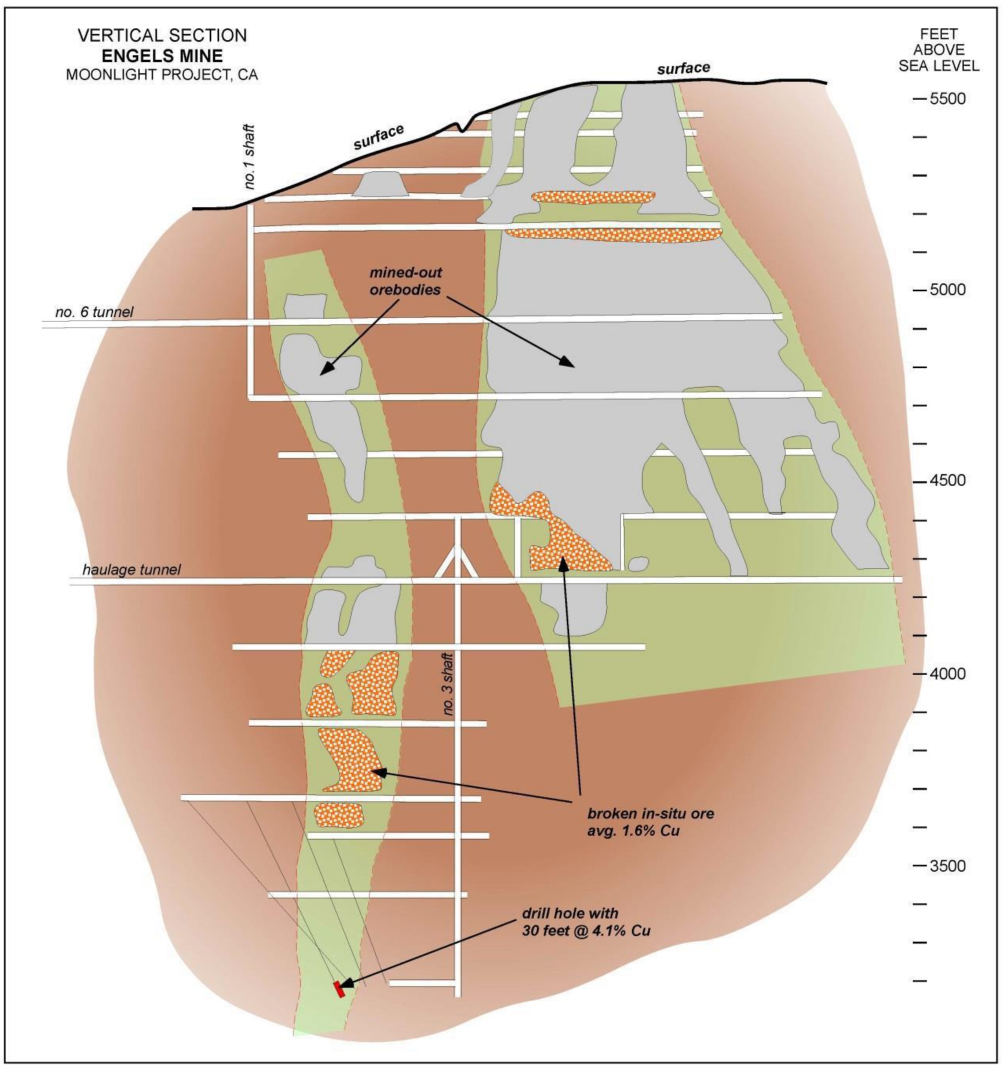
Copper oxide minerals consist primarily as malachite with lesser chryscolla and azurite and in copper-bearing limonites and clays. Electron microprobe work indicates some copper occurs as replacement of potassium in biotite. Typical oxidized copper-bearing silica hornfels show a specific gravity of 2.5. Very strongly weathered metavolcanic and diorite typically show a specific gravity of 2.3.

The principal sulfide minerals consist of bornite, and chalcopyrite hosted in a hornblende gabbro body. Younger quartz diorite and quartz monzonite bodies are associated with the gabbro and are considered to have played an important role in the placement of the copper mineralization.

The deposit appears to splay to the northwest in the upper 656 feet (200 meters), with widths increasing upward. Figure 7-6 presents a long section of the Engels deposit as it was mined. Source: OreQuest, 2007 data Placer-Amex, 1966

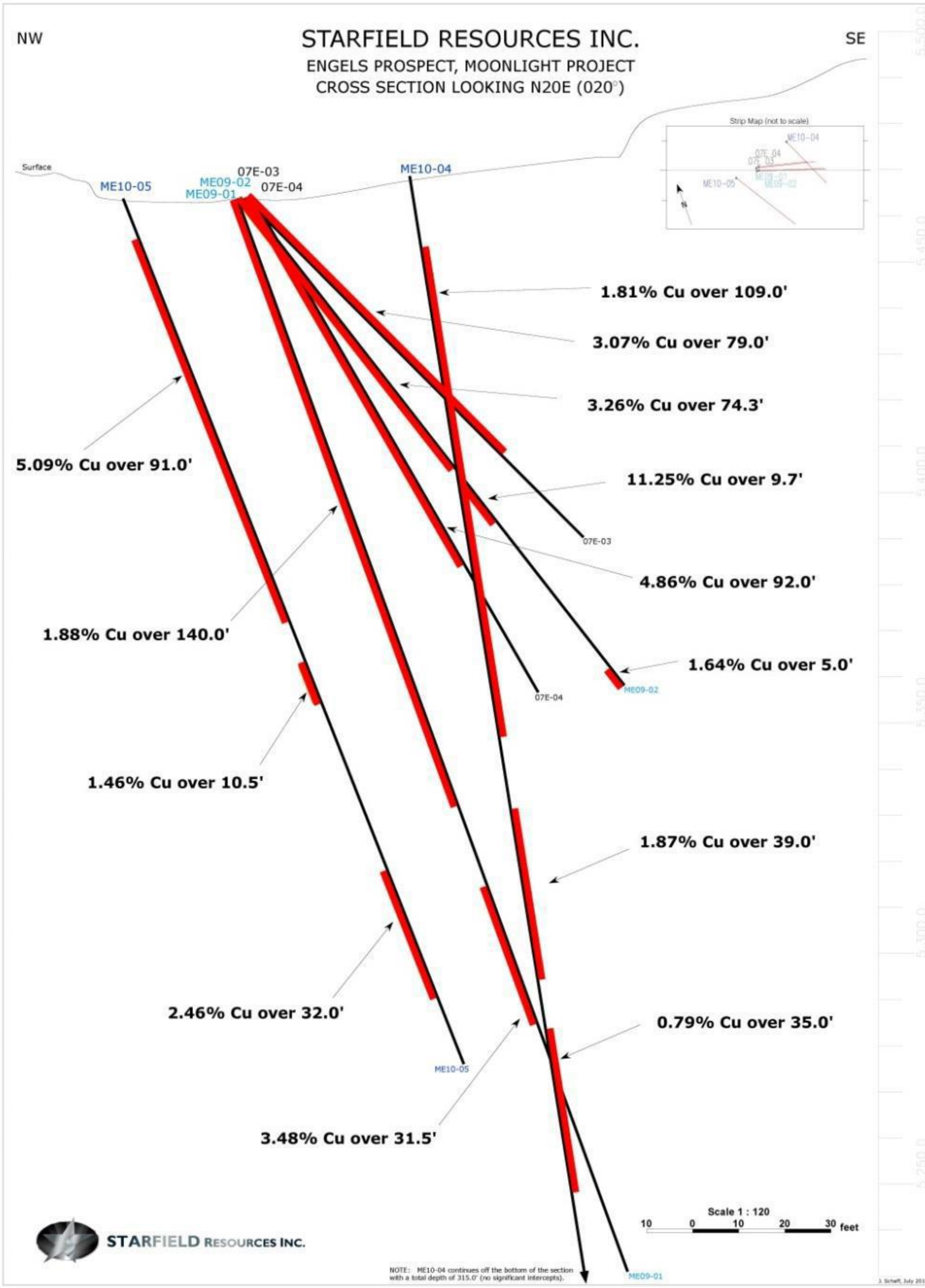
Figure 7-7 presents a cross section through Engels showing the composited drill hole intercepts from the Starfield drilling in 2009 and 2010. These intercepts indicate that significant material of the tenor historically mined underground remains within 328 feet (100 meters) of the surface. These drill holes are included in the resource estimate prepared for this report and inform the estimate for Engels.

Figure 7-6: Long section of Engels Showing the Historic Mining



Source: OreQuest, 2007 data Placer-Amex, 1966

Figure 7-7: Engels Northwest-Southeast Drill Hole Cross Section Presenting Summarized Results of 2009-2010 Drilling

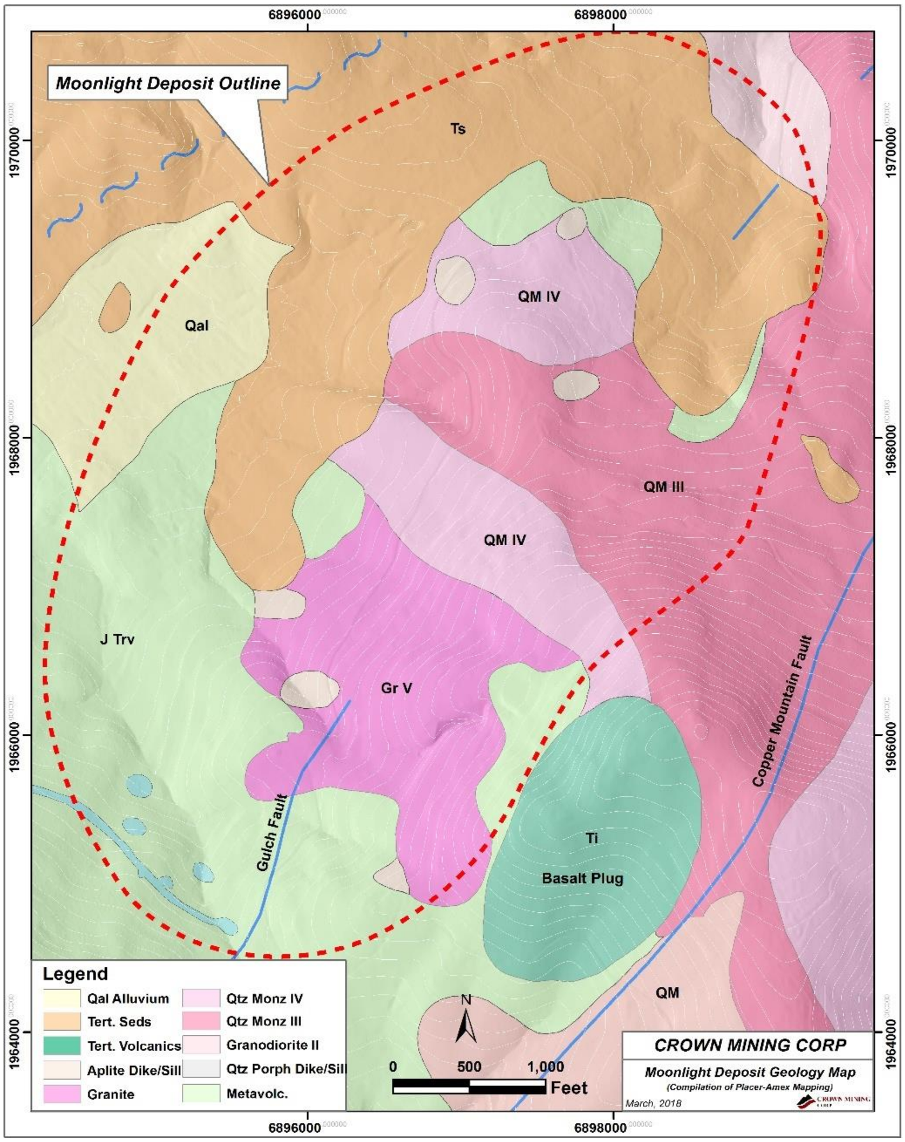


Source: Starfield, J Schaff, 2009

7.3.4 Geology and Mineralization at Moonlight Deposit

Figure 7-8 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 LCS 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 7-8: Geology Map of Moonlight



Source: OreQuest, 2007, updated by US Copper Corp, 2018.

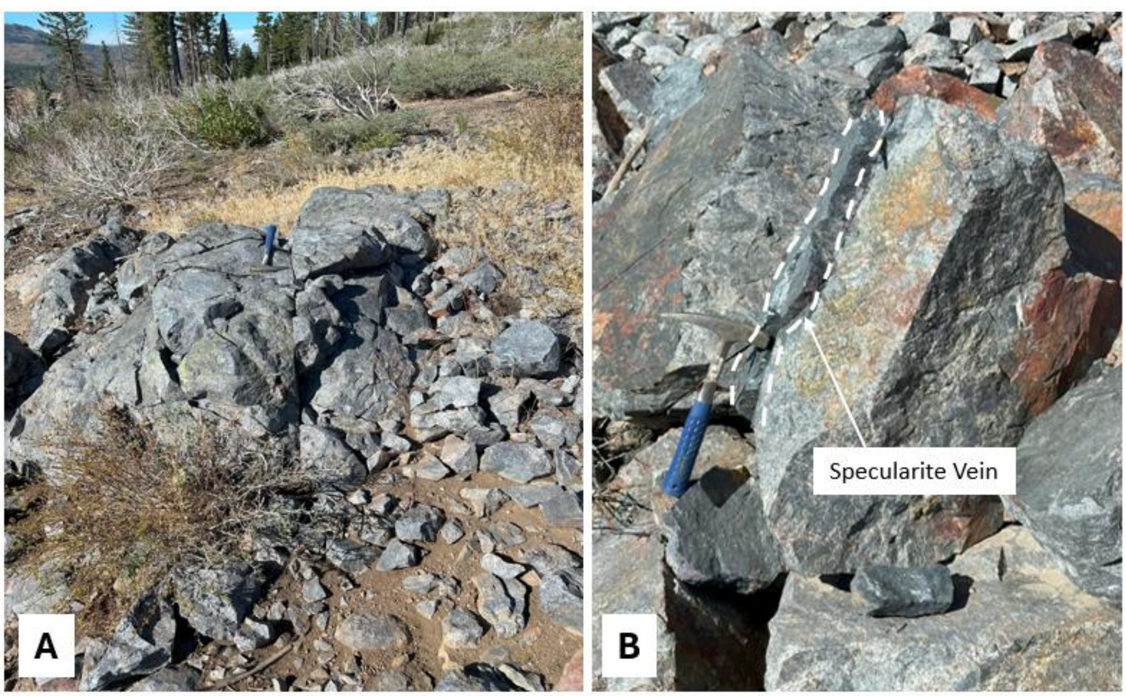
Although most of the Moonlight deposit area is covered by alluvium, GRE'QP checked a few outcrops in this area. The Granitic intrusive (Grv) at Moonlight shows the same evidence of mineralization at Superior and Lambs Ridge, including magnetite tourmaline veinlets and copper oxide minerals at the surface (Photo **7**-**7**).

Photo 7-7: A) Magnetite Veinlets in Granitic Intrusive (Gr V), and B) Copper Oxide in Granitic Intrusive

Source: GRE 2024.

The metavolcanic unit does not show any visible evidence of mineralization on the surface. No magnetite veinlet was seen in this unit. Specularite is an abundant iron oxide in the metavolcanic unit at Moonlight (Photo 7-8.

Photo 7-8: A) General View of Metavolcanic unit (J Trv), and B) Specularite Vein in Metavolcanic unit at Moonlight



Source: GRE 2024.

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% Cu. 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.

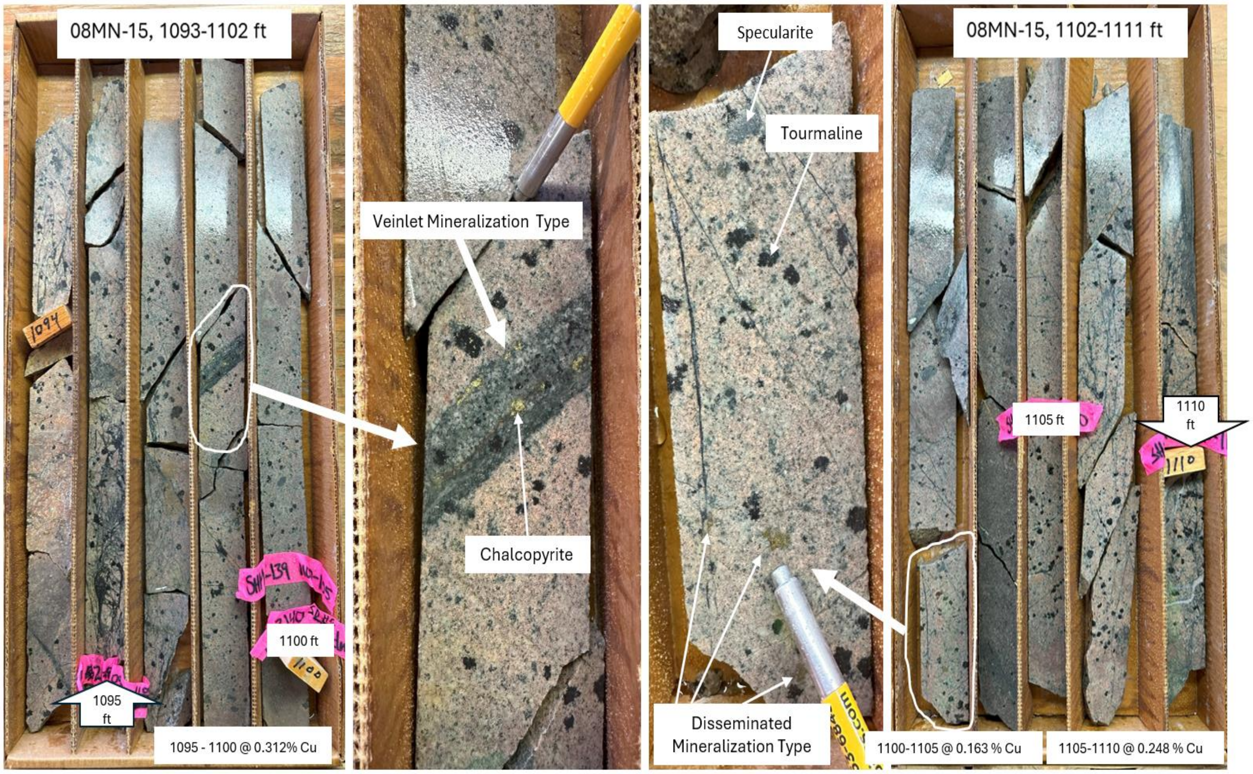
The copper sulfides show a vertical zonation, with chalcocite or digenite predominating in the upper levels of the deposit. With increasing depth, bornite predominates and chalcopyrite appears. Bornite is often observed to rim or cut chalcopyrite. Bornite and chalcopyrite may also be cut by chalcocite 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% Cu. 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% Cu 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.

GRE's QP inspected a few historic core samples at Moonlight, including some intervals from holes 05-MN-1, 06-MN-12, 08-MN-15, and 08-MN-20, for which all of the intervals were logged as quartz monzonite. The inspection confirmed all types of Mineralization at Moonlight described by R. G. Wetzel, including disseminated and veinlets copper mineralization types (Photo 7-9).

Photo 7-9: A View Showing Disseminated and Veinlet Mineralization Types at Moonlight inspected in Hole 08MN-15



Source: GRE 2024

7.4 Structural Control

Structural preparation has been important in localizing mineralization in the LCD; however, structures which host mineralization typically show little apparent displacement and individual structures can typically be traced for less than 50 feet (15 meters) and rarely up to 656 feet (200 meters) either along strike or dip. Mineralization is preferentially located in stockwork zones with fractures of multiple orientations or at the intersection of structures and lithologic contacts.

The structures which host the mineralization at the Ruby Mine in the LCD and the Walker Mine 15 miles (25 km) to the southeast, strike about N20W and dip steeply to the northeast. These mineralized zones parallel the trend of the Plumas copper belt and the Walker Lane.

N10E steep to moderately east dipping structures host significant portions of the mineralization mined in the past at the Superior Mine. Similar trending fracture zones are observed to host copper mineralization throughout the district including the Moonlight and Engels areas.

Northwest striking gently southwest dipping fracture zones are observed to host significant copper mineralization throughout the district as well.

A very significant portion of the copper mineralization is also truly disseminated and not associated with fractures or veinlets. This disseminated mineralization is typically associated with 0.08- to 0.4-inch (2- to 10- millimeter [mm]) blebs of tourmaline.

8 DEPOSIT TYPES

The Engels deposit is characterized as a shear zone-hosted, structurally-controlled, tabular breccia body(s) hosted within mafic units of the Lights Creek Stock as well as the metavolcanic rock into which the stock was intruded. Copper and silver mineralization at Engels appears to be associated with late-stage differentiates of the more felsic units of the Lights Creek Stock.

The Superior, Lamb's Ridge, and Moonlight deposits were historically classified as porphyry copper deposits with associated silver and, to a lesser extent, gold. Porphyry copper deposits provide more than 50% of the world's copper from over 100 producing mines.

The accepted geological model described for copper porphyry deposits is based largely on occurrences in Arizona and Chile. This model describes porphyry copper deposits as cylindrical, stock-like composite bodies having elongate outcrops 1-mile by 1.2-mile (1.5-km by 2-km) in diameter and containing an outer shell of medium to coarse-grained equigranular rock with a porphyritic core of similar composition.

The most common ore hosts are quartz monzonite to granodiorite felsic plutonic rocks. In addition, a second population of deposits occurs in more mafic intrusive rocks of syenitic to dioritic composition.

The model also describes a zonal pattern to alteration first documented by Lowell and Guilbert in 1970 (Lowell, et al., 1970), who suggested that four alteration halos were often present roughly centered on the porphyry stock:

- The Potassic Zone – this zone was always present and characterized by secondary potassium feldspar (K-spar), biotite and/or chlorite replacing primary K-spar, plagioclase and mafics. Minor sericite may be present.
- Phyllic Zone - not always present and characterized by vein quartz, sericite and pyrite with minor chlorite, illite and rutile replacing the K-spar and biotite.
- Argillic Zone – was not always present. It is identified by the clay minerals kaolinite and montmorillonite with minor disseminated pyrite. Plagioclase is strongly altered, K-spar unaffected and biotite is chloritized.
- Propylitic Zone –always present and contains chlorite, calcite and minor epidote. The mafic minerals are highly altered while the plagioclase is less altered.

At depth all zones are thought to coalesce into a single, large K-spar-quartz- chlorite-sericite unit.

Placer-Amex recognized that the deposits of the Lights Creek district had many characteristics which 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; Cole, 2015).

Many copper deposits which had previously been classified as porphyry copper-type have now been re-characterized as belonging to the iron oxide copper-type. There is considerable evidence that the porphyry-like Lights Creek deposits could be included in this group.

The IOCG group represents a very wide distribution of deposits in terms of age, size, mineralogy and metals present; however, the characteristics listed below 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 is 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 Lights Creek deposits show all of these characteristics. A number of deposits have been classified by various authors as belonging to the iron oxide copper type 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% Cu mineralization and there is potential to discover additional plus 2% Cu mineralization in the Lights Creek district.

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-[cobalt]Co-[nickel]Ni-[arsenic]As-[molybdenum]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 EXPLORATION

The following section is partly based on Tanaka (2014), Wetzel (2009), Placer-Amex (1972), and recent information from US Copper Corp.

9.1 Pre-US Copper

9.1.1 Soil and Rock Geochemical Sampling

Beginning in 1963 and continuing into 1965, Placer-Amex conducted a series of stream sediment, soil and rock geochemical surveys for copper and associated elements. Soil sampling was initially done on 300-foot 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 Lamb's Ridge) and several more of lesser magnitude. Follow-up sampling on 100-foot centers was carried out over most of the anomalous areas. This work identified a number of exploration targets in the district. 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. The composited results of this work are shown in Figure 9-2. The placers geochemistry results were later confirmed by standard channel sampling by Sheffield and then by US Copper.

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. US Copper verified Sheffield’s results with its own channel sampling.

The first soil sample map was updated and contoured using both the 1960s and 2005-2008 soil samples data by John Schaff from Starfield in 2009, that version was redrafted in 2018 by Schaff (Figure 9-3).

Placer-Amex also conducted a series of rock geochemical sampling for copper within the project area between 1963 through 1965. This program continued by Shefield from 2005 through 2007, and recently by US Copper Corp with 33 rock samples in 2021. The programs covered almost all exposed rocks within the property, including Superior, Lamb’s Ridge, Engels, Osmeyer, Moonlight, Copper Mountain, Blue Copper, and a few areas south of Superior and at the southwestern part of the property, totaling 432 rock samples. Figure 9-4 shows the final results from the geochemistry programs. This data was first drafted by John Schaff from Starfield in 2009, then was redrafted in 2018 by Schaff.

Figure 9-1 shows the results of the soil sampling campaign and identifies the named anomalies.

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. The composited results of this work are shown in Figure 9-2. The placers geochemistry results were later confirmed by standard channel sampling by Sheffield (Wetzel, 2009) and then by US Copper.

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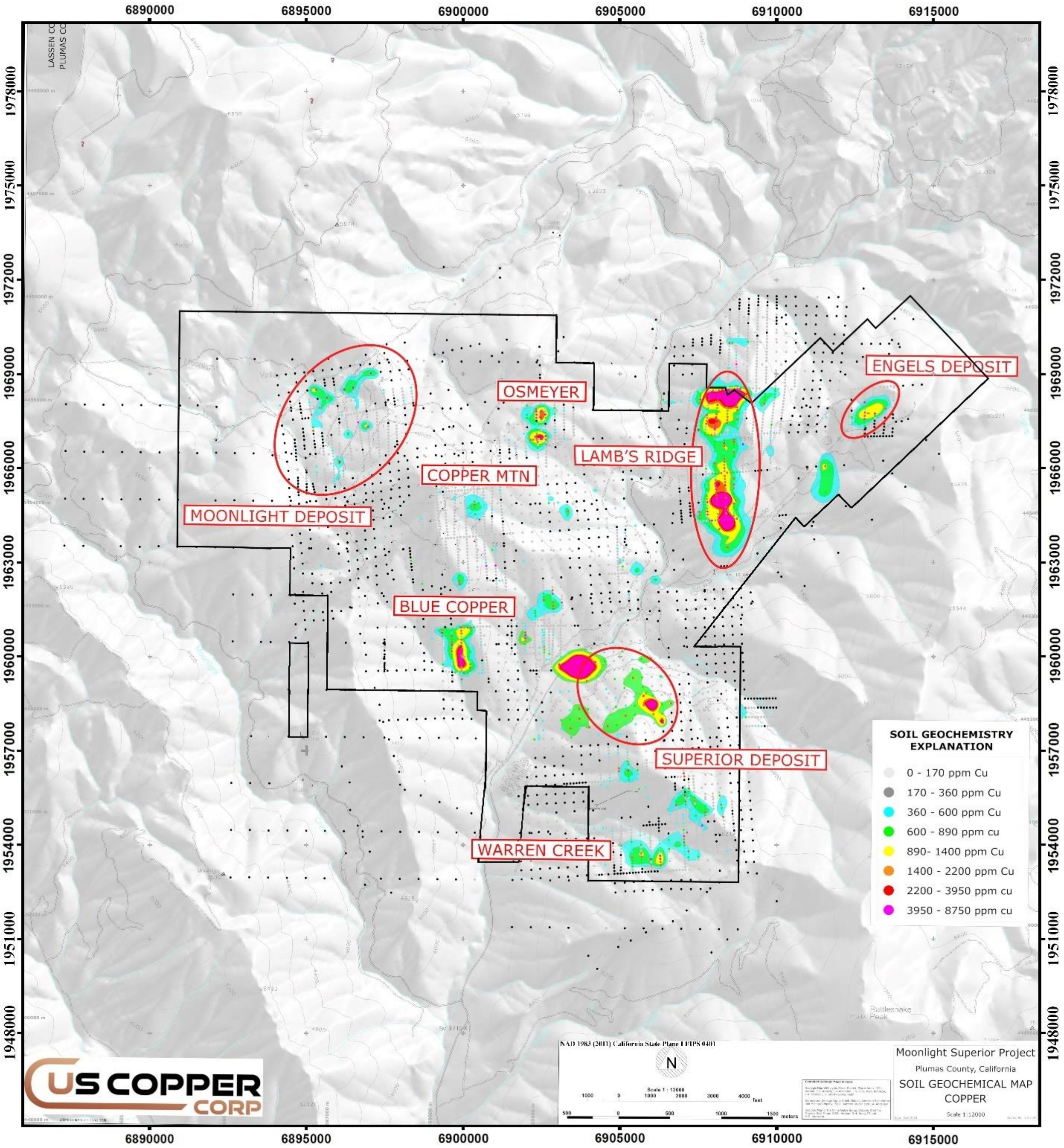
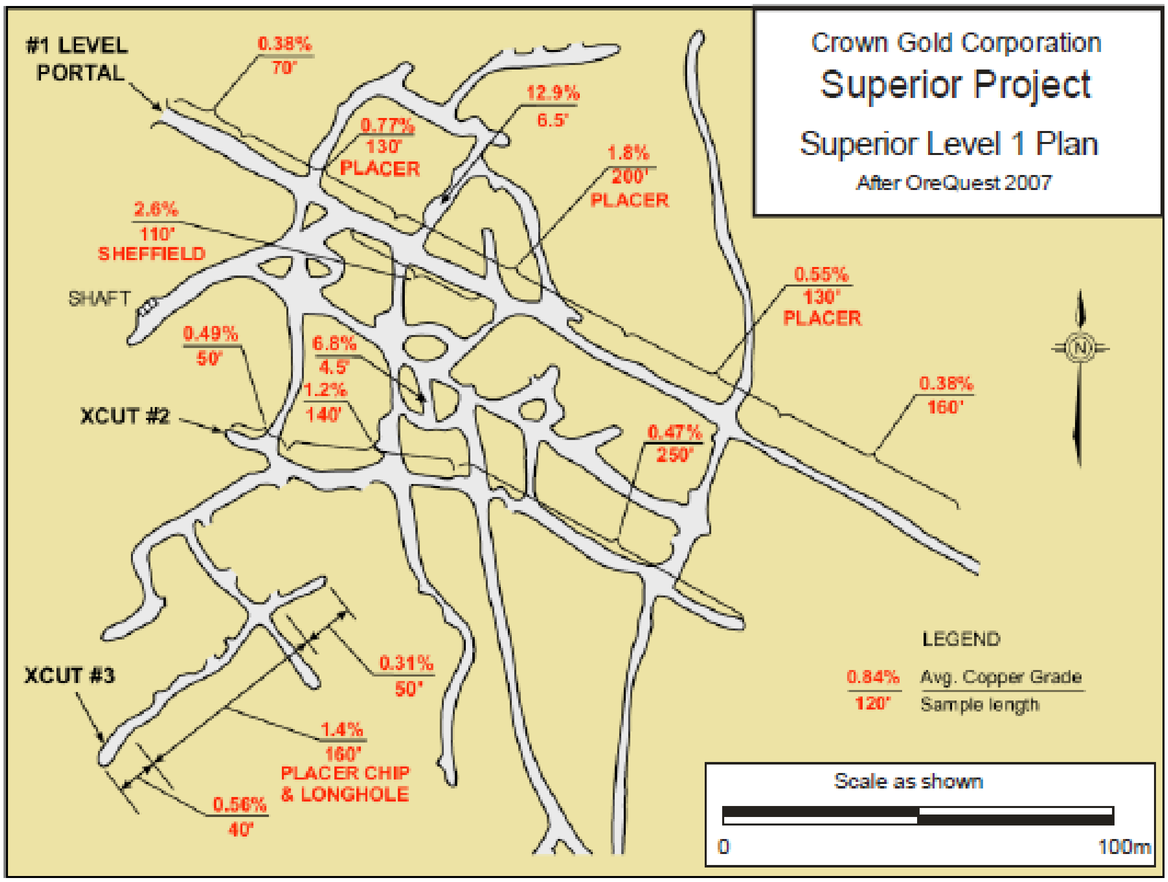
Figure 9-1: Soil Geochemistry Exploration Map for Copper



Source: Data Placer-Amex, 1966, Nevoro, 2009, US Copper Corp., 2016.

Figure 9-2: Superior Level 1 Plan showing composited Placer-Amex chip/channel sampling



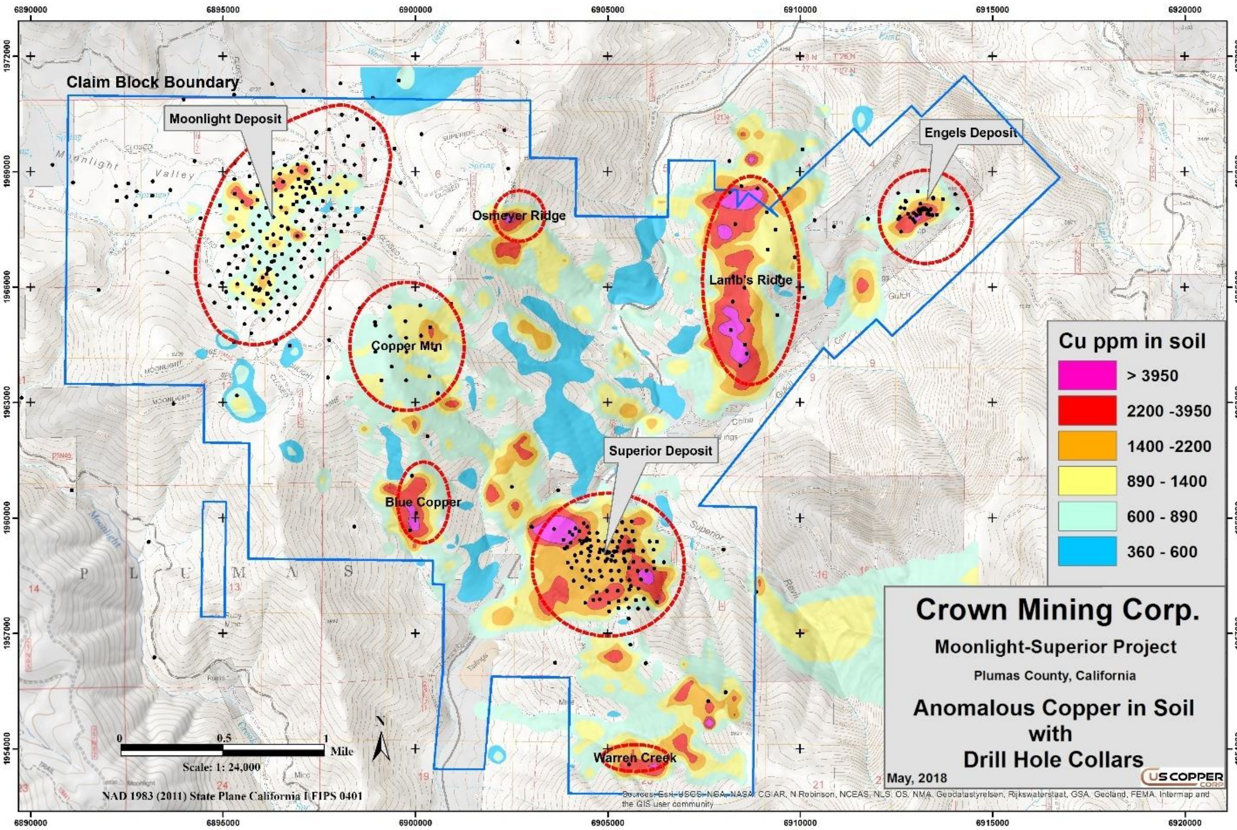
Source: OreQuest, 2007

Table 9-1: Summary of Sheffield Underground Sampling at Superior

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	0.2	0.042	8.9
113	1 Level Underground Samples	2.88	2.43	0.028	39.8

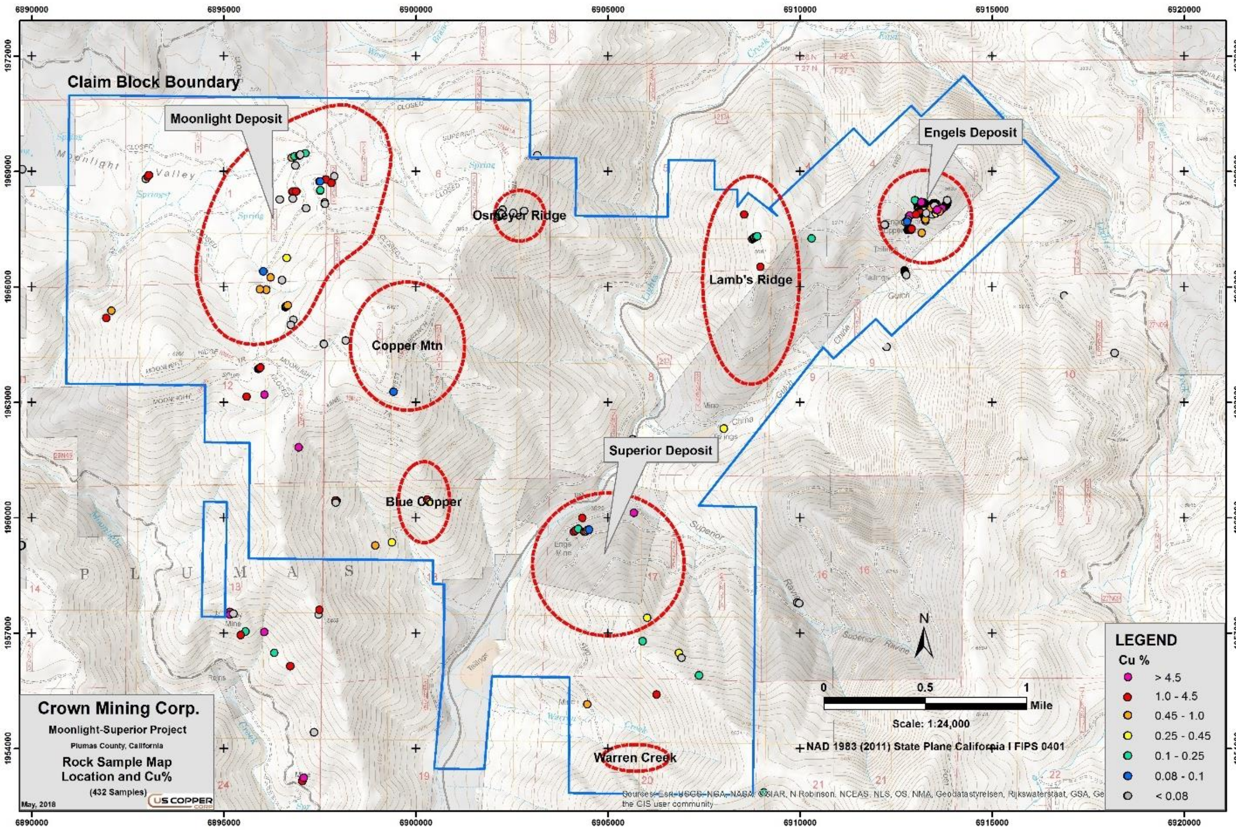
Source: Tanaka, 2013 g/t = grams per short ton

Figure 9-3: The Last Updated Soil Geochemistry Exploration Map for Copper



Source: Data Placer-Amex, 1966, Nevoro, 2009, US Copper Corp., 2018

Figure 9-4: Rock Geochemistry Exploration Map for Copper



Source: Data Placer-Amex, 1965, Nevoro, 2009, US Copper Corp., 2018

9.1.2 Source: US Copper Corp. 2018 Geological Mapping

In 1953 and 1954, 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 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.

In 1970, Placer- Amex prepared the geology map of the LCD based on petrographic analysis by J.D. Juilliard. Geological names were modified from California Division of Mines and Geology Westwood Map Sheet (1:250,000) 1960.

9.1.3 Geophysics

In 1965, Placer-Amex initiated a ground-based Induced Polarization (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 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, Garry Carlson of Gradient Geophysics reviewed the existing geophysical data and recommended an airborne magnetic and electromagnetics (EM) survey, a Deep IP-Resistivity survey, and a Controlled Source Audio-frequency Magnetotellurics (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 Fugro report stated:

> The airborne magnetic EM and radiometric survey has provided a great deal of geophysical data that can be used to improve the geological mapping in the area. The magnetic data from the survey clearly show several structural breaks. Where these are found in proximity to known mineralization they should be followed up on the ground as they may lead to other concentrations of mineralization along their structure. The magnetic data also see differences with the Lights Creek intrusive complex. This may be helpful in mapping the intrusive and gaining a better understanding of its emplacement and the effect on mineralization. The high magnetic response in the Engels mine area makes interpretation difficult and the conductive Tertiary metavolcanics in the Moonlight area overpower any weaker response from the thin veins or disseminated sulfides. Only the Superior Mine site shows a strong conductive response. This response is so powerful that if there were other similar sources in the area, they should be visible provided that they are crossed by the flight path. The depth slice data and apparent resistivity grids are helpful in differentiating between the rock types in the area. Resistive and conductive areas outlined on the interpretation map should be examined with respect to local geology to better define the extent of rock units or alteration in the area. Areas of known mineralization should be examined, and a note taken of any coincidence of geophysical parameters. If other locations show a similar coincidence of these parameters ground follow-up is recommended to determine if any of these areas may also possess mineralization. As more ground data are accumulated the responses in the airborne survey should be re-examined to see if there are other indications of mineralization targets.

9.2 US Copper

Exploration conducted by US Copper Corp. includes drilling and rock sampling. US Copper Corp. drilled seven core holes totaling 5,872 feet (1,789.8 meters) at Superior in 2021, 15 RC holes totaling 3,990 feet (1,260.2 meters) at Engels in 2023, and two RC holes totaling 820 feet (250 meters) at Lamb’s Ridge in 2023, and 15 RC holes totaling 2,430 feet (740.6 meters) at Moonlight in 2023.

US Copper Corp. also took 33 surface rock samples from Lamb’s Ridge, Superior, Engels, Ruby Mine, and Quigly South for geochemistry analysis.

9.3 Exploration Potential

9.3.1 Within Existing Deposits and Potentials

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), Tanaka (2014), Tetra Tech (2018) but also includes information gleaned from geophysical reports detailing IP-Resistivity surveys conducted over the soil sampling anomalies, and also the opinion of GRE's QP.

9.3.1.1 Superior

Placer-Amex soil sampling produced a concentric symmetrical 1,000-foot by 1,400-foot (305-meter by 305-meter) >2,000 ppm Cu anomaly with a 600-foot by 800-foot (183-meter by 244-meter) core carrying >5,000 ppm Cu. 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 Cu 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 feet (79.2 meters) of 1.83% Cu and 0.9 opt Au and 220 feet 67 meters) of 1.63% Cu and 0.64 opt Au in underground workings.

In 2021, US Copper Corp drilled seven core holes totaling 5,872.3 feet (1,789.9 meters) within the Superior deposits. All seven holes have intersections with high grade copper (>0.5 %), including 14 intervals in hole S21-1 (61.5 feet [18.7 meters]), 14 intervals in hole S21-2 (67.9 feet [20.7 meters]), eight intervals in hole S21-3 (32.9 feet [10 meters]), 13 intervals in hole S21-4 (67.0 feet [20.4 meters]), three intervals in hole S21-5 (12.3 feet [3.75 meters]), 15 intervals in hole S21-6 (69.3 feet [21.1 meters]), and 13 intervals in hole S21-7 (59.8 feet [18.2 meters]), totaling 370.7 feet (113 meters) of high grade intervals.

Any additional drilling should investigate the possible existence of other high-grade structurally-controlled segregations of high grade to the northeast, southwest, and at depth.

9.3.1.2 Lamb's Ridge

At Lamb's Ridge, soil sampling produced a >1,000 ppm Cu anomaly that extends 6,000 feet (1,828.8 meters) in a north-south direction, averages 1,000 feet (304.8 meters) in width and contains localized high-grade zones that carry >5,000 ppm Cu. Placer-Amex drilled a total of 28 holes at Lamb's Ridge. Wetzel (2009) noted that outcrop and talus are very abundant along Lamb's Ridge and speculated that copper from fractures may have been overrepresented in the sieved samples. Lamb's Ridge drilling is very widely spaced, with intervals of between 300 and 1,000 feet (91.4 and 304.8 meters), 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).

In 2023, US Copper Corp drilled two RC holes totaling 2,690.3 feet (820 meters) in the Lamb's Ridge area. There are only three intervals totaling 15 feet (4.6 meters) at >0.3% Cu and only one interval of 5 feet (1.5 meters) at 1.125% Cu in hole 23LRRC01. Hole 23LRRC02 shows six intervals totaling 30 feet (9.1 meters) of 0.3% to 0.4% Cu.

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.

9.3.1.3 Engels

Over the Engels Mine, a symmetrical, concentric, >500 ppm Cu in soil anomaly extends about 1,800 feet (548.6 meters) in a northeast direction, is about 1,000 feet (304.8 meters) wide, and contains a core that carries >5,000 ppm Cu. 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 Cu contour. Core recoveries were typically poor, but holes E-2 and E-7 carried >0.5% Cu. Sheffield/Nevoro drilled 44 core holes in 2007 and 2008. In 2009, Wetzel observed that >0.3% Cu mineralization encountered in Sheffield/Nevoro's drilling coincided with the 1,000 ppm Cu contour.

In 2023, US Copper Corp drilled 15 RC holes totaling 3,990 feet (1,216 meters) within the Engels deposit. The result shows there are 35 intervals totaling 665 feet (202.7 meters) of 0.5% to 0.99% Cu, 12 intervals totaling 235 feet (71.6 meters) of 1% to 1.49% Cu, and 14 intervals totaling 255 feet (77.7 meters) of >1.5% Cu.

Since drilling at Engels is tightly confined to the immediate vicinity of the historically mined volume, more targets along-strike or down dip need to be tested.

9.3.1.4 Moonlight

Placer-Amex soil sampling produced a very irregular, <500 ppm Cu anomaly that measures approximately 4,500 feet (1,371.6 meters) in a north-northeast direction and 3,500 feet (1,066.8 meters) in a west-northwest direction. There are numerous internal lows and local highs up to 5,000 ppm Cu within the anomaly. Wetzel (2009) stated that the anomalously high zones do not usually coincide with the location of near surface >0.5% Cu 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 southwest underneath the roof pendant meta-volcanic rocks. The Ruby Mine, located approximately 1.5 mi (2.4 km) 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% Cu, 0.06 opt Au, and 6.75 opt Ag. 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 (0.8 km) south of Moonlight hit 20 feet (6.1 meters) of 3.4% Cu 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.

In 2023, US Copper Corp drilled 15 RC holes totaling 2,430 feet (740.6 meters) at Moonlight. The result shows there are 42 intervals totaling 210 feet (64 meters) of 0.5% to 0.99% Cu, six intervals totaling 30 feet (9.1 meters) of 1% to 1.49% Cu, seven intervals totaling 350 feet (10.6 meters) of >1.5% Cu.

9.3.1.5 Warren Creek

Soil sampling produced an irregular 3,500- by 2,500-foot (1,066.8- by 760-meter) >1,000 ppm Cu anomaly in the Warren Creek drainage that contains localized highs >3,000 ppm Cu. 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-foot (461.8-meter) hole, DDH-01A, at Warren Creek. Wetzel (2009) reports that Placer-Amex drilled at least four holes, including a 2,000-foot (609.6-meter) deep hole. The authors have not completely resolved this discrepancy but assume that Wetzel was including the US series holes drilled 2,500 feet (762 meters) northeast of DDH-01A. It should be noted that in the records reviewed by the authors, there is no 2,000-foot (609.6-meter) deep drill hole in the Warren Creek area. Wetzel also states that >200-foot (61-meter) intercepts of 0.1 to 0.2% Cu mineralization were encountered in the metavolcanic rocks but that no high-grade copper was intersected either in the meta-volcanic rocks or in the underlying Lights Creek quartz monzonite.

9.3.1.6 Blue Copper

At Blue Copper, soil sampling produced an 1,800-foot (548.6-meter), north-south geochemical anomaly carrying >1,000 ppm Cu. 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 now known if the Blue Copper drilling tested those anomalies.

9.3.1.7 Copper Mountain

The Copper Mountain geochemical anomaly, an irregular 2,000- by 200-foot (609.6- by 61-meter) >500 ppm Cu anomaly is located approximately 2500 feet (762 meters) southeast of the Moonlight deposit. A few >2,000 ppm Cu highs are present within the 500 ppm Cu contour. Placer-Amex drilled 14,226 feet (4,336.1 meters) in 25 holes over a 3,500- by 4,000-foot (1,066.8- by 1,219.2-meter) area at Copper Mountain. A number of encouraging intercepts were encountered. These include 210 feet (64 meters) of 0.39% Cu in CM-11, 390 feet (118.9 meters) of 0.36% Cu in CM-12, and an average of 0.224% copper from 490 to 1,965 feet (149.4 to 598.9 meters) in CM-29. All drilling at Copper Mountain is vertical. As at Lamb’s Ridge, the drill holes are widely separated, with spacing ranging from 300 feet (91.4 meters) to over 700 feet (213.4 meters). With the exception of CM-29, the drilling depths average <500 feet (152.4 meters). The 1966 IP survey identified several chargeability anomalies that were recommended for drilling (Ludwig 1966).

9.3.1.8 Osmeyer Prospect

At the Osmeyer Prospect, 4,000 feet (1,219,2 meters) east of the Moonlight deposit, soil sampling produced an irregular 1,500- by 600-foot (457.2- by 182.9-meter) >1,000 ppm Cu 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 feet (90.2 meters). From 10 to 110 feet (3 to 33.5 meters), the hole averaged 0.15% Cu. The remainder of the hole carried consistent low-grade (<0.05% Cu) mineralization. CM-22, located 300 feet (91.4 meters) west of the anomaly, was drilled to a depth of 200 feet (61 meters) and intercepted consistent low-grade (<0.1% Cu) 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.

9.3.2 New Targets

A review of the data shows that several areas with copper anomalies resulting from the previous soil and rock sampling programs still need to be tested by more detailed exploration work, such as preparing large-scale geology and structural geology maps, additional surface sampling, and drilling. This technical report presents these areas as new targets for future exploration plans.

A fault zone controlled mineralization at Superior with at least two main northeast-trending faults. At Moonlight, two other northeast-trending faults, the Gulch and Copper Mountain faults, have controlled mineralization. All these northeast-trending faults seem terminated to the southwest by the Gossan Ridge fault and other parallel northwest-trending faults. GRE'QP is of the opinion that mineralization at Engels has probably been controlled by the northeast-trending faults. The extension of the Superior fault zone has likely affected this area and controlled mineralization, which should be further defined. The trace of the Main Fault Projection was mapped along the Engels deposit. Although mineralization at Engels has northeast-trending, more detailed field data should be gathered to confirm whether the northwest-trending faults are associated with mineralization or not. Morphotectonic evidence shows that in the middle of the property, between Superior and Lamb's Ridge, a few anomalies of copper are associated with a northwest-trending inferred fault.

Since the project area is located at the northeast of the N20W trending Plumas Copper belt, interpreted as an extension of the north-northwest trending Walker Lane structural lineament, both northeast and northwest-trending faults are likely associated with mineralization, an important subject that should be tested through large-scale structural mapping within the property. In this regard, GRE'QP considered areas for the future exploration program, associated with northwest-trending faults.

Structurally, these main normal faults, especially those striking northeast and with their second-order faults, have prepared several extension areas along or within themselves as conduits for hydrothermal fluids and, finally, the emplacement of copper deposits.

Considering copper anomalies due to previous soil and rock sampling, a geological structural map of the project area, and drilling results from previous exploration campaigns, at least 11 new targets can be defined within the property (Figure 9-6):

Figure **9**-**5** presented the location of existing deposits and new targets in association with soil anomalies and geological structures. GRE's QP believes the distribution of existing deposits has been controlled mainly by northeast-trending faults and their second-order faults and fractures. These faults have been terminated at the southwest and northeast of the project area by northwest-trending faults such as the Gossan Ridge (in the southwest) and the Main Fault Projection (in the northeast).

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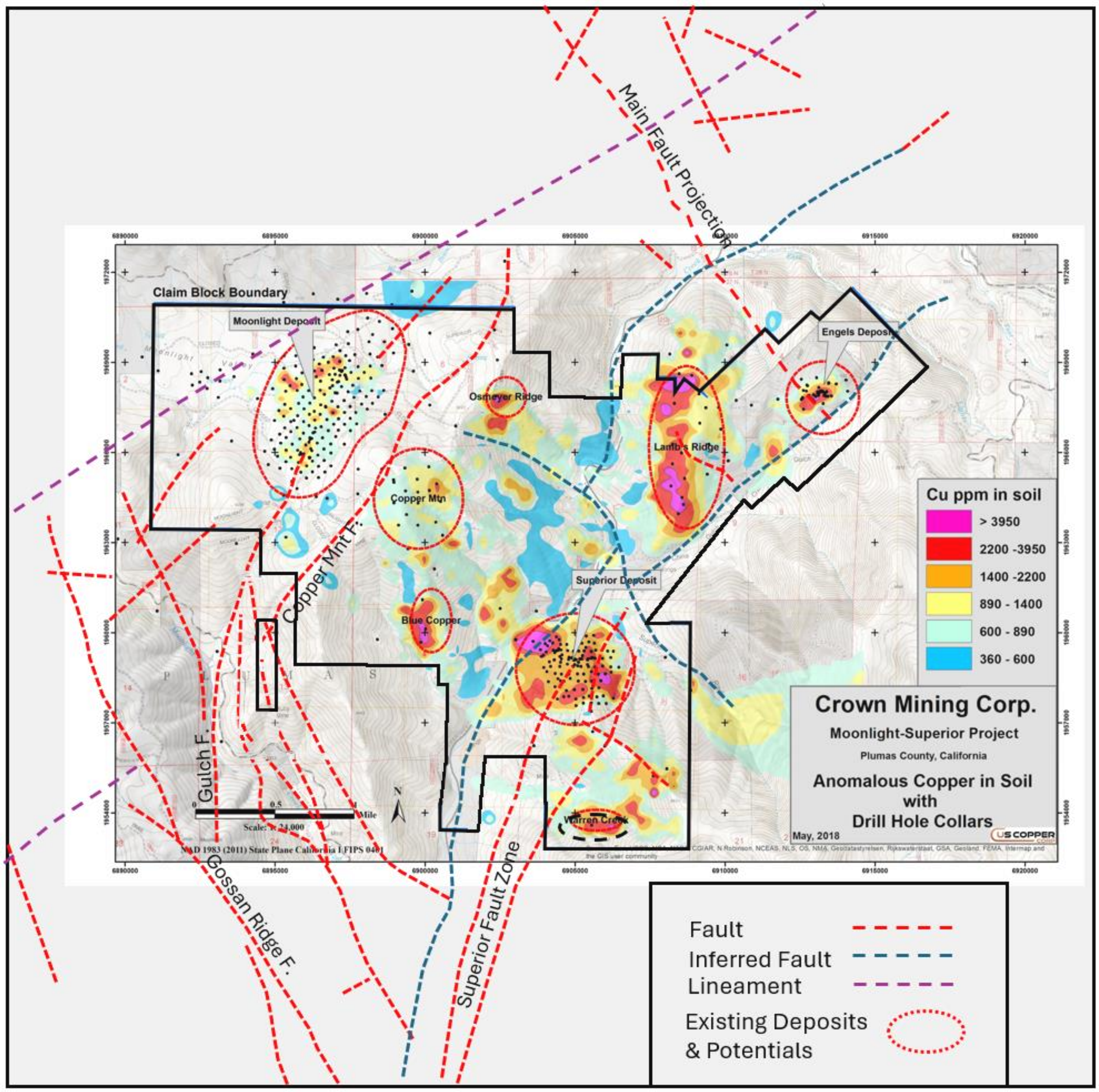
Considering copper anomalies due to previous soil and rock sampling, a geological structural map of the project area, and drilling results from previous exploration campaigns, at least 11 new targets can be defined within the property (Figure 9-6):

Figure 9-5**: Existing Deposits and New Targets Associated with Soil Anomalies and Geological Structures**

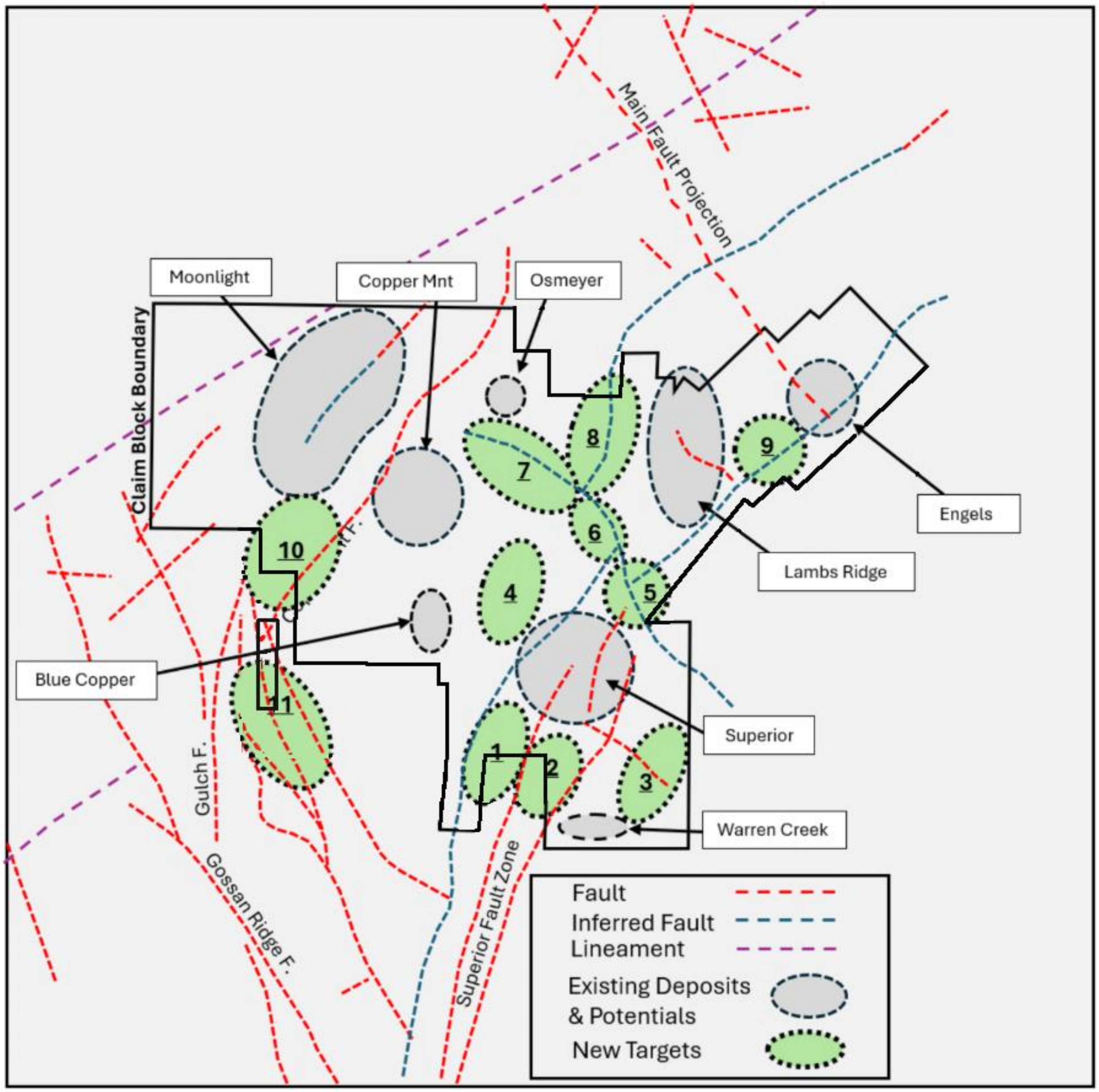


Source: GRE, 2024 (soil anomalies map is taken from US Copper Corp. dated 2018.

- Targets No. 1 and No.2: southwest of Superior, within fault zones.
- Target No.3: northeast of Warren Creek, with high copper anomalies. Probably, this target has the same trend as targets No. 1 and No.2.
- Target No.4: northwest of Superior, between Superior and Blue Copper with copper anomaly.
- Targets No.5: northeast of Superior and at the junction of two or three lineaments.
- Target No.6: north of Superior and at the junction between two lineaments and with copper anomaly.
- Target No.7: south of Osmeyer Ridge and along one lineament with copper anomalies.

- Target No.8: northeast of Target No.7 and along one lineament with copper anomalies.
- Target No.9: southwest of Engels and along one lineament with copper anomalies.
- Target No.10: southwest of Moonlight, within a fault zone with copper anomaly.
- Target No.11L south of target No.10, with a fault zone with northwest trending. This area is covered by Jurassic-Triassic undifferentiated volcanic and sedimentary rocks. This target is considered for testing whether the northwest-trending faults are associated with mineralization.

Figure 9-6: Simplified Model of Existing Deposits and New Targets



Source: GRE 2024.

9.4 QP Comments on Item 9 "Exploration"

The exploration programs completed to date are appropriate to the style of the deposit and prospects.

GRE's QP believes the distribution of deposits has been controlled mainly by northeast-trending faults, fractures, and their second-order faults. Most deposits are packed between northeast-trending faults and aligned through these faults.

Since only a few areas of mineralization have been discovered so far, the GRE's QP believes that a detailed land survey geophysical method such as CSAMT and large-scale structural geology mapping can reveal new areas with a high potential for precious metals for future drilling. Evaluation of mineralization along the northwest-trending faults is an important topic that needs to be explored. If the results are positive, several potentials will be added to the project area.

10 DRILLING

10.1 Introduction

In 1964, Placer-Amex contracted Boyles Brothers Drilling (Boyles Brothers) for the company's LCD drilling program. Beginning in 1964 and continuing through 1975, Boyles Brothers drilled 105,042 feet (32,016.8 meters) in 213 holes at the Moonlight deposit (Placer-Amex 1972). In 2005 and 2006, Sheffield drilled 14 HQ core holes (11,135 feet [3,393.9 meters]) 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. In 2007, Sheffield drilled 1,420.0 feet (432.8 meters) in 15 shallow RC holes designed primarily to test the copper oxide potential at the Moonlight deposit. Lang Drilling was contracted to complete the RC program. All reverse circulation holes except two returned significantly lower copper values than adjacent Placer-Amex core holes. Sheffield was acquired by Nevoro Copper in July 2008. Nevoro Copper completed seven vertical core holes totaling 2603.5 feet (793.5 meters) at the Moonlight deposit. The 2008 drill holes were twinned to select Placer-Amex core holes.

The Superior Project's drilling consists of historic drilling done by Placer-Amex from 1962 to 1970, including 69,007 feet (21,033.3 meters) in 149 core holes.

Ruen Drilling Incorporated, a California-licensed company based in Clark Fork, Idaho, conducted the historic core drilling at Engels from 1966 through 1967. HQ (2.5-inch [63.5-mm]) core was recovered from the collar of virtually all the core holes. In that campaign, 10 holes totaling 4,774.5 feet (1,455.3 meters) were drilled. From 2007 through 2010, Nevoro and Starfield drilled 13,761.3 feet (4,194.4 meters) in 51 holes at Engels.

Placer-Amex conducted the historic drilling at Lamb's Ridge from 1965 through 1970, including 12,621 feet (3,846.9 meters) in 28 holes.

In 2021, US Copper added seven core holes consisting of 5,872 feet (1,789.8 meters) of drilling at Superior. In 2023, US Copper Corp added 15 RC holes totaling 2,430 feet (740.7 meters) at Moonlight, two RC holes totaling 820 feet (249.9 meters) at Lamb's Ridge, and 15 RC holes consisting of 3,990 feet (1,216.2 meters) of drilling at Engels. Since 2021, US Copper has added 39 holes totaling 13,112.3 feet (3,996.6 meters) within the property.

Since 2016, US Copper staff reviewed the historical database and corrected several obvious errors upon inspection. Validation of the older drilling is currently underway.

The current database contains 566 drill holes amounting to 255,059.9 feet (77,742.3 meters) of drilling and containing 28,357 assayed intervals for Cu, Au, and Ag. Drilling spanned the period of 1964 through 2023, as summarized on Table 10-1. All of this drill hole information has been used to estimate grade in the block model.

Since the last technical report dated 2018, 39 holes, totaling 13,112.3 feet (3996.6 meters) of drilling, have been completed.

Table 10-1: A List of Drilling from 1964 to 2023

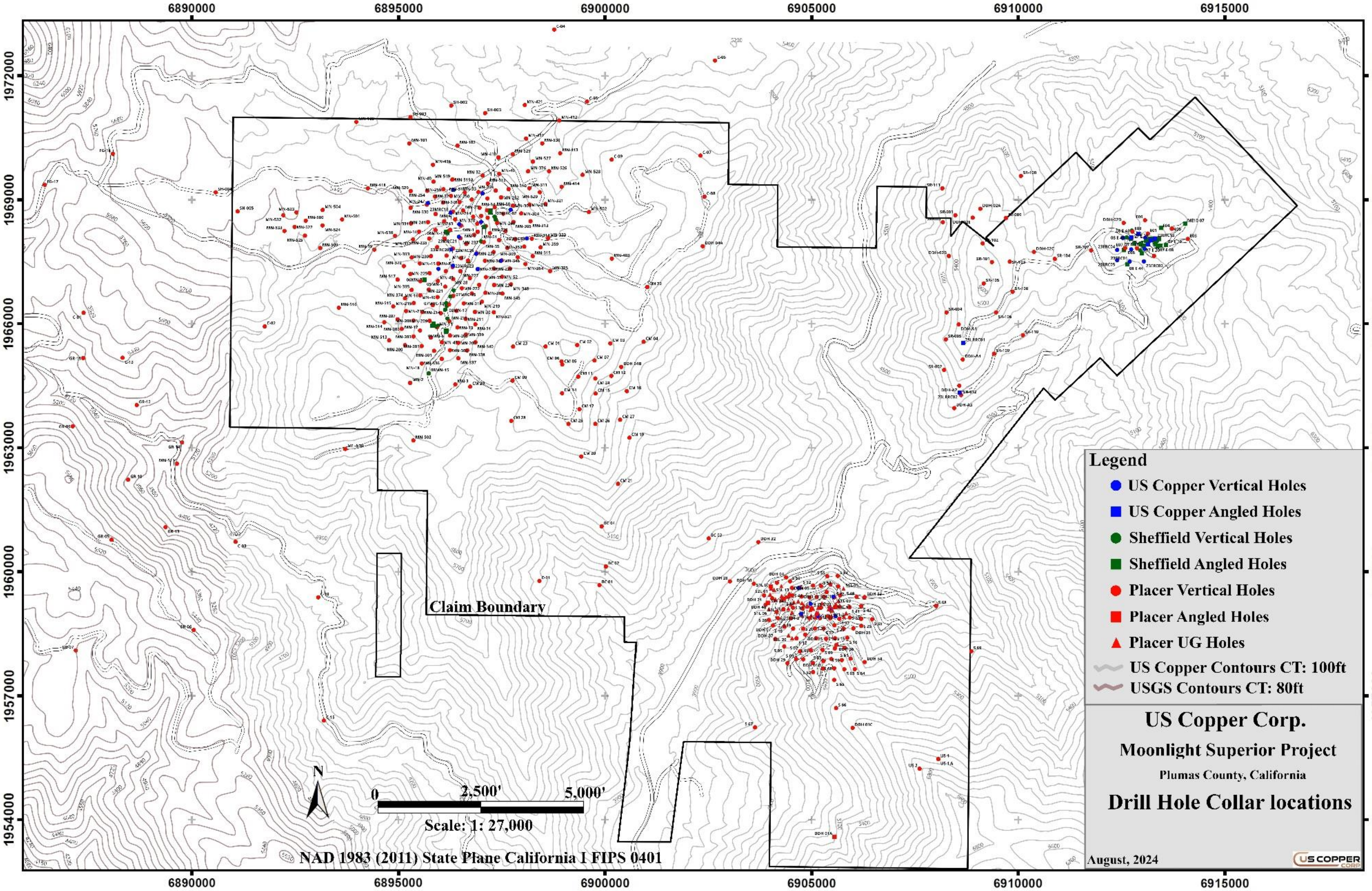
Main Prospect	Sub-Prospect	Year	Core		RC/Rotary		All Holes		Company
			# of Holes	Footage	# of Holes	Footage	# of Holes	Footage	
Moonlight	Moonlight	N/A	5	2,034			5	2,034.0	Placer-Amex
		1966	12	7,795.2			12	7,795.2	Placer-Amex
		1967	146	70,757.0			146	70,757.0	Placer-Amex
		1968	14	3,227			14	3,227.0	Placer-Amex
		1968	2	2,336.0			2	2,336.0	Placer-Amex
		1969	8	4,346.0			8	4,346.0	Placer-Amex
		1970	25	13,944.0			25	13,944.0	Placer-Amex
		1975	1	603.0			1	603.0	Placer-Amex
		2005	2	1,838.6			2	1,838.6	Sheffield
		2006	12	9,296.3			12	9,296.3	Sheffield
		2007			15	1,420.0	15	1,420.0	Sheffield
		2008	7	2,603.5			7	2,603.5	Nevoro
		2023			15	2,430.0	15	2,430.0	US Copper
	Copper Mountain	N/A	2	937.0			2	937.0	Placer-Amex
		1967	17	8,291.0			17	8,291.0	Placer-Amex
		1968	1	200.0			1	200.0	Placer-Amex
		1970	4	2,770.0			4	2,770.0	Placer-Amex
		1972	1	400.0			1	400.0	Placer-Amex
		1973	1	1,965.0			1	1,965.0	Placer-Amex
	Osmeyer Ridge	1965	1	296.0			1	296.0	Placer-Amex
Moonlight Total			**261**	**133,640**	**30**	**3,850**	**291**	**137489.54**	
Superior	Superior	N/A	51	17,351.00			51	17,351	N/A
		N/A	13	4,647.00			13	4,647	Placer-Amex
		1964	9	4,577.70			9	4,578	Placer-Amex
		1965	9	5,008.50			9	5,009	Placer-Amex
		1966	56	31,727.10			56	31,727	Placer-Amex
		1967	7	3,005.00			7	3,005	Placer-Amex
		1969	2	1,702.00			2	1,702	Placer-Amex
		1970	2	989.00			2	989	Placer-Amex
		2021	7	5,872.30			7	5,872	US Copper
Superior Total			**156**	**74,880**			**156**	**74879.6**	
Lamb's Ridge	Lambs Ridge	1965	9	3,549.00			9	3,549	Placer-Amex
		1966	6	1,805.00			6	1,805	Placer-Amex
		1967	1	150.00			1	150	Placer-Amex
		1968	1	500.00			1	500	Placer-Amex
		1969	5	3,329.00			5	3,329	Placer-Amex
		1970	6	3,288.00			6	3,288	Placer-Amex
		2023			2	820	2	820	US Copper
Lamb's Ridge Total			**28**	**12,621**	**2**	**820**	**30**	**13441**	
Engels	Engels	1966	1	334.00			1	334.00	Placer-Amex
		1967	9	4,440.50			9	4,440.50	Placer-Amex
		2007	32	7,613.90			32	7,613.90	Sheffield

Main Prospect	Sub-Prospect	Year	Core		RC/Rotary		All Holes		Company
			# of Holes	Footage	# of Holes	Footage	# of Holes	Footage	
		2008	12	4,075.90			12	4,075.90	Nevoro
		2009	3	574.00			3	574.00	Starfield
		2010	4	1,498			4	1,498	Starfield
		2023			15	3,990	15	3,990	US Copper
Engels Total			**61**	**18,536**	**15**	**3,990**	**76**	**22525.8**	
Blue Copper	Blue Copper	1966	1	269			1	269	Placer-Amex
		1967	3	1,614			3	1,614	Placer-Amex
Blue Copper Total			**4**	**1,883**			**4**	**1,883**	
Gossan Ridge	Gossan Ridge	1970	9	4,841			9	4,841	Placer-Amex
Gossan Ridge Total			**9**	**4,841**			**9**	**4,841**	
Total			**519**	**246,399.9**	**47**	**8,660**	**566**	**255,059.9**	

Of the original 566 drill holes, 519 are reported to be diamond drill holes (DDH), and the remaining (47 holes) are reverse circulation (RC). Figure 10-1 is a plot of all 566 drill hole collars at the project area superimposed over the topography map. Figure 10-2 shows a 3D map of all drill hole locations across the property. Figure 10-3 shows east-west cross-sections from all four primary deposits: Superior, Engels, Lamb's Ridge, and Moonlight. In Figure 10-4, only intervals with a Cu grade of more than 0.5 are present.

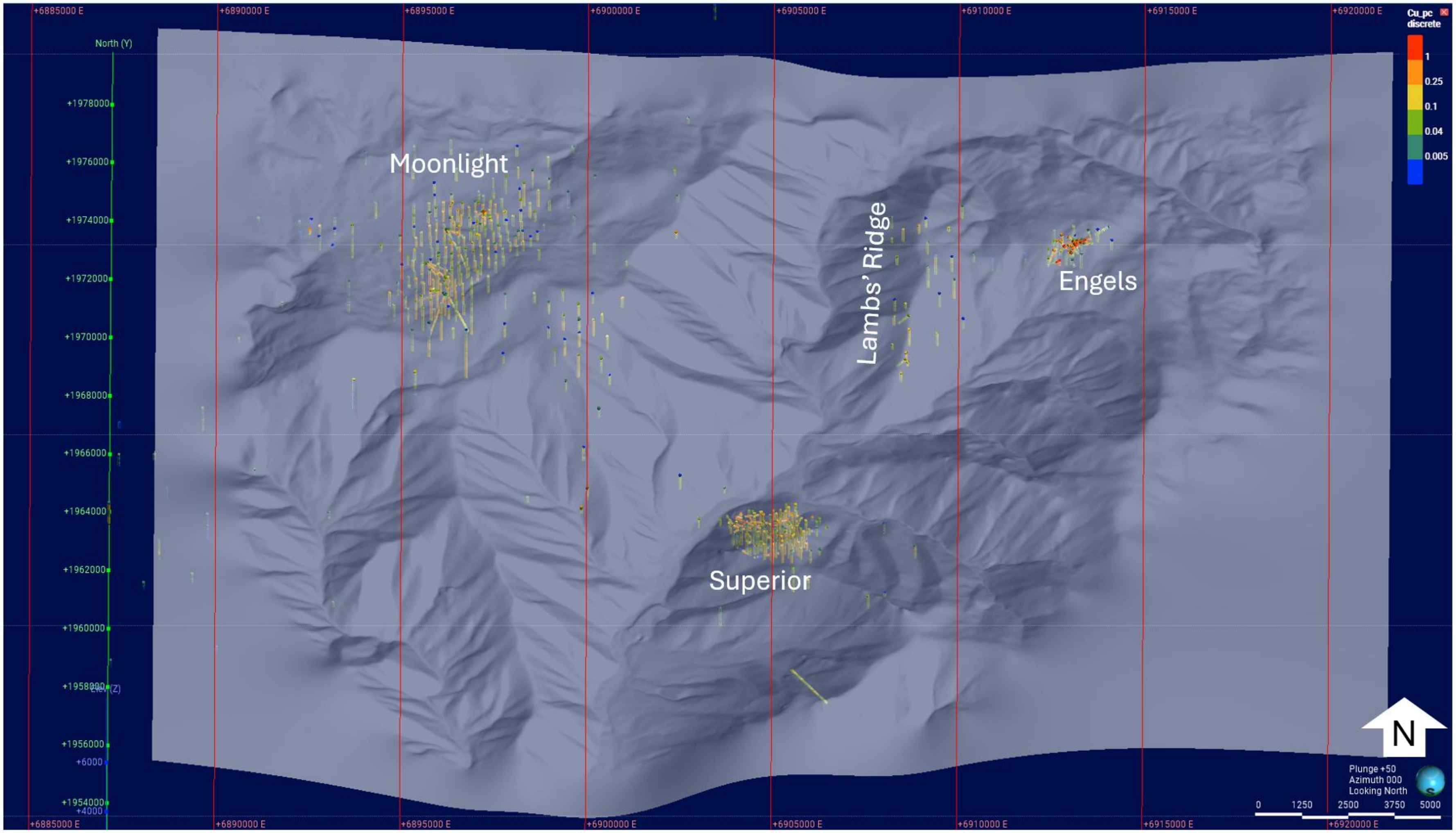
It should be noted that there are 18 holes, including FG-01 to FG-18, totaling 6,897 feet (2,102.2 meters) at the north of Moonlight deposit and out of the property, drilled by Placer-Amex in the 1960s. These holes have assay data for 650 sample intervals with a 10-foot interval sample. Of 650 samples, 321 samples have Cu%=0, 319 samples have Cu% 0.01-0.09, nine samples have Cu% 0.1-0.24, and one sample has Cu%=1.85. Since they are located out of the property, they are not listed in the US Copper holes and are not used for MRE.

Figure 10-1: Drill Collars for all Drilling Programs (1964-2023) Over Topography Map



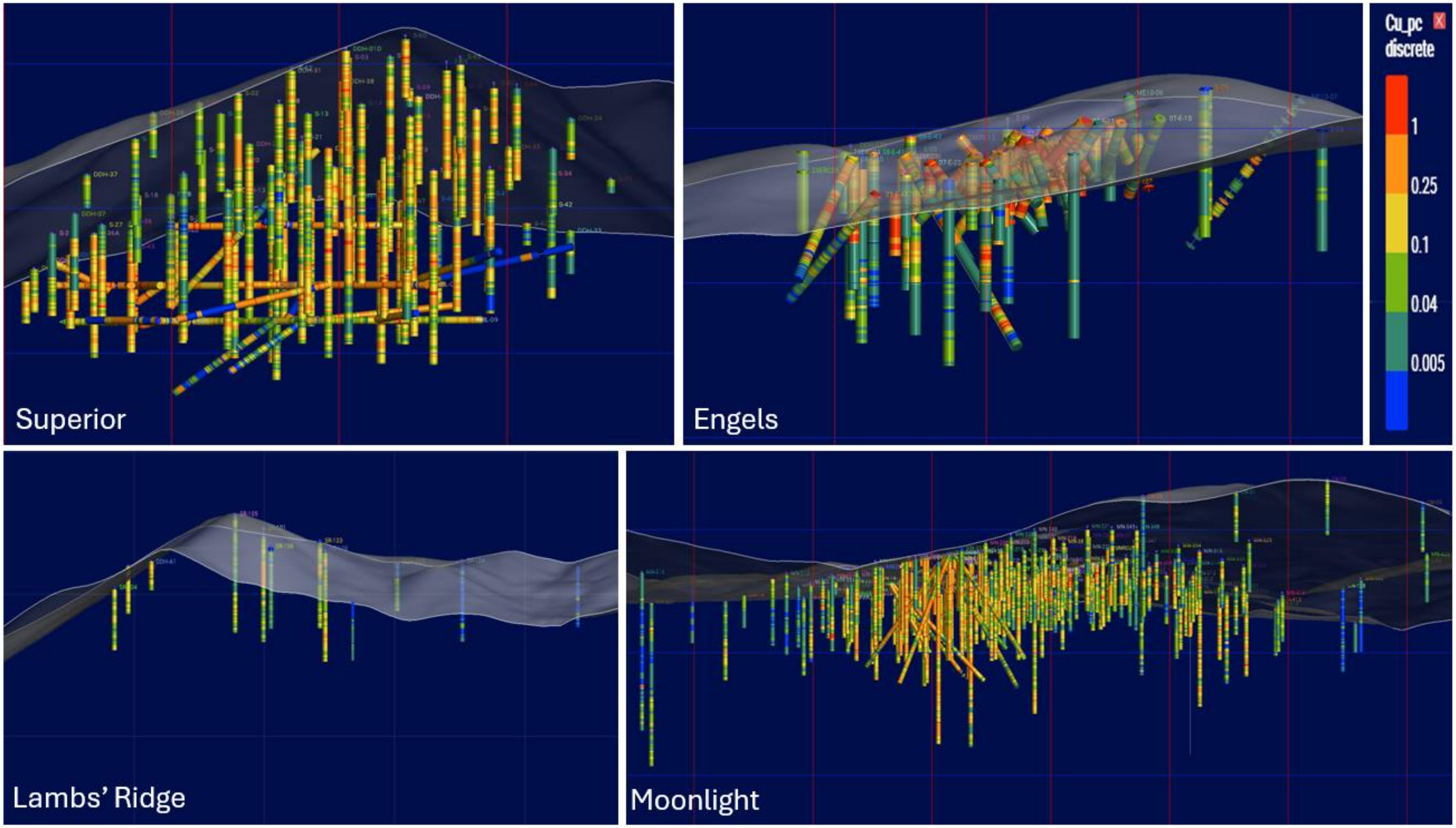
Source: US Copper, 2024

Figure 10-2: Drill Holes Locations for all Drilling Programs (1964-2023)



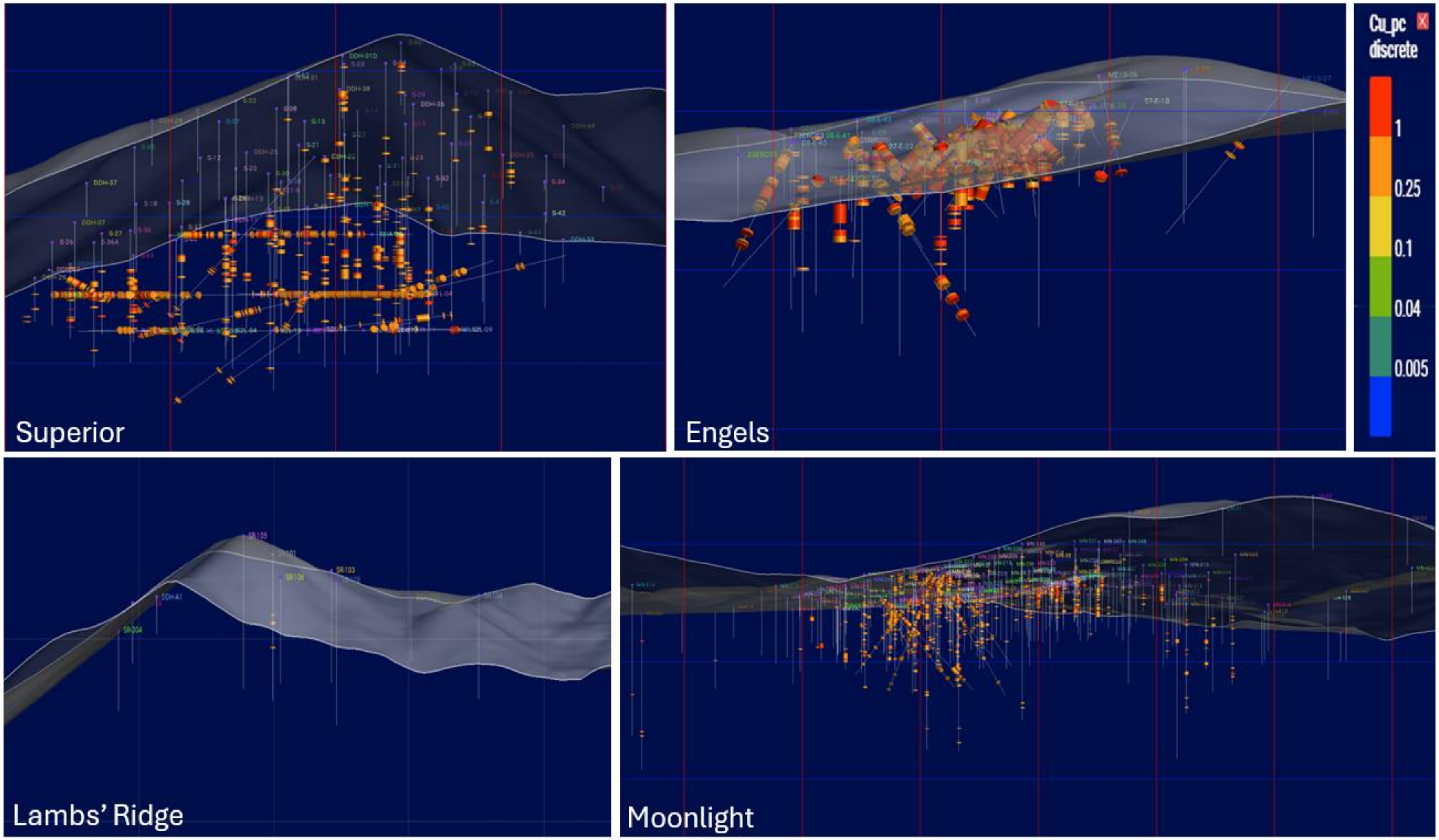
Source: GRE, 2024

Figure 10-3: East-West Cross Sections from Drill Holes with Their Assays for all Drilling Programs (1964-2023), Considering All Cu Grades (View to the North)



Source: GRE, 2024

Figure 10-4: East-West Cross Sections from Drill Holes with Their Assays for all Drilling Programs (1964-2023), Considering Only Intervals With Cu>0.5% (View to the North).



Source: GRE, 2024

10.2 Drill Methods

10.2.1 Pre-US Copper

At the drill site, the drill contractor's staff placed the core in wooden or cardboard boxes with appropriate footage blocks. Historic drilling for Placer-Amex was done by Boyles Brothers, a respected drilling contractor acquired by Layne Christiansen. Boyles typically drilled 6.5 to 39.04 feet (2 to 12 meters) at the collar of the hole with a rock bit and then set casing. NX (2.15-inch [54.7-mm]) core was recovered to a depth of 98 to 197 feet (30 to 60 meters) and then the hole was completed using BX (1.65-inch [42-mm]) core (Wetzel 2009).

In contrast to the NX and BX core favored by Placer-Amex in the 1960s, Sheffield consistently drilled HQ core except for hole 08MN-15, which was reduced to NQ core below 800 feet (243.8 meters). At the end of every shift, either the drill contractor's staff or a Sheffield/Nevoro geologist transported the core to a fenced and locked core logging facility in Crescent Mills, California.

The location and orientation of the existing drill holes were controlled to some extent by topography and access for surface drilling and the extent and availability of underground workings for underground drilling. They do not appear to be ideally oriented for the current understanding of the fabric of mineralization, particularly Engels and Superior. Lamb’s Ridge is an exception, largely because drilling is too sparse to identify any mineralization fabric.

10.2.2 US Copper

In 2021, US Copper drilled seven HQ core holes with an average depth of 840 feet (256 meters) at Superior, which was drilled by Timberline. The initial azimuth and dip of the drill holes were set by project geologists (John Gartner, Justin Claiborne). Timberline operated two twelve-hour shifts per day, seven days a week, until project completion due to the Dixie fire. Core was collected from the morning shift for logging at the company core facility. The Core had been drilled using both 5-foot (1.5-meter) and 10-foot (3-meter) rods; the core was placed into cardboard boxes after recovery.

In 2023, US Copper drilled 15 RC holes at Moonlight, two RC holes at Lamb’s Ridge, and 15 RC holes at Engels. US Copper had contracted Alaska Midnight Sun Drilling Inc. to perform the drilling operation using a tracked MPP Grasshopper RC rig. The initial azimuth and dip of the RC holes were set by project geologists (John Cleary, Justin Claiborne). Drilled material was collected out of the cyclone and mixed in the splitter. The material was split until a 10-pound sample was collected from the splitter; the remaining material was used to prepare the chip tray for chip logging. Samples and chips were collected in 5-foot (1.5-meter) intervals.

10.3 Logging Procedure

10.3.1 Pre-US Copper

Logging for the 2005-2008 work had been recorded into Excel files, with the 2005 through early 2008 work being done in meters and later in feet. Intervals every two meters or five feet had the alteration, lithology, rock quality designation (RQD), and recovery recorded. In addition, wider intervals were recorded to describe the overall description of the core, along with noting the position of notable structures and features. These logs were completed by geologists Rob Wetzel, W. Rowe, R. Gonzales, and W. Mitchell.

John Schaff completed the logging for the 2009 and 2010 drilling at Engels on a paper log describing the alteration, structure, and lithology in a graphics log, along with a written description of the interval local lithology and alteration.

10.3.2 US Copper

During the 2021 US Copper Superior project, the core was collected daily from Timberline and brought to the core facility for logging. Core logging was completed by project geos (Justin Claiborne and John Gartner). Core logging intervals were set based on lithology and alteration, with the depth, core angles, and description of important features being noted in the respective interval. Logging was completed on an official logging sheet and later input into an Excel file. Core logging and photographs were taken prior to cutting the core. Sample intervals for assaying were marked after logging for core cutting. Sample intervals were designated around every five feet depending on the local lithology, alteration, and estimated grade of the sample so as to not misrepresent the grade of the sample.

During the 2023 US Copper RC project at Engels, Moonlight, and Lamb's Ridge, drilled material was collected out of the cyclone and mixed in the splitter. The material was split until a 10-pound sample was collected from the splitter; the remaining material was used to prepare the chip tray for chip logging. Samples and chips were collected in 5-foot (1.5-meter) intervals. Preliminary logging took place in the field by Justin Claiborne and John Cleary as the drill completed each 5-foot (1.5-meter) rod string. The chip trays and preliminary notes were sent to the company core facility for more detailed logging under a microscope. Log entries were directly inserted into GeoSequal.

10.4 Recovery

10.4.1 Pre-US Copper

According to the Tanaka Technical Report dated 2013, Placer's BX drilling typically showed 95% recovery overall and lower recoveries in softer copper-bearing zones.

10.4.2 US Copper

During the 2021 US Copper Superior program, RQD and fracture counts were recorded for the core. Core recovery was calculated by measuring all the pieces of core between each depth block that are over 5 inches (127 mm) long and comparing the measurement to the interval length. Drill holes S21-3 and S21-4 were positioned around a fault, and as such, the core was more highly broken and had a lower recovery value. The average recovery value of the core above 5 inches (127 mm) in length between the seven drill holes is 66%.

10.5 Collar Survey

10.5.1 Pre-US Copper

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.

10.5.2 US Copper

Following the completion of the drill holes during the 2021 and 2023 drill programs, drill hole collars were surveyed using a Trimble Geo7x handheld GPS. Coordinates for each site were taken over a 5 to 10-minute or greater period. These points were later post-processed on GPS Pathfinder using base stations in Susanville and Quincy. Drill hole Collars were resurveyed during each surveying event to validate the accuracy of prior coordinates. The 2021 surveying events used an external antenna along with the Geo7x. The estimated accuracy after post-processing with this equipment set up and base stations is between a few feet and a few inches of the collar.

10.6 Down-Hole Surveys

10.6.1 Pre-US Copper

Downhole surveys were completed by the drilling contractor for some of the drill holes from 2005 to 2010.

10.6.2 US Copper

US Copper's 2021 Superior Drill hole program done by Timberline included downhole surveys for all seven of the drill holes. The downhole survey was completed using a Reflect EZ Trac survey equipment at intervals between 50 and 100 feet (15.2 and 30.5 meters).

US Copper's 2023 Work at Moonlight, Engels, and Lamb's Ridge did not include downhole surveys, as the work included mostly shallow drill holes to test the oxide resource at the prospects.

10.7 QP Comments on Section 10 Drilling

In the opinion of the QP, drilling was conducted in accordance with industry-standard practices. The drilling, as performed, provides suitable coverage of the zones of copper mineralization. Collar and downhole survey methods used generally provide reliable sample locations. Drilling methods provide good core recovery. Logging procedures provide consistency in descriptions.

The collected sample data adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits. Drill orientations are generally appropriate for the mineralization style for the bulk of the deposit area.

The QP considers that the quantity and quality of the logging, collar, and downhole survey data are sufficient to support Mineral Resource estimation.

No factors were identified with the data collection from the drill programs that could significantly affect Mineral Resource and Mineral Reserve estimation.

11 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

The data below was taken from Tanak's (2013) and Tetra Tech's (2018) technical reports. New data from the 2021 and 2023 drilling campaigns were taken from the US Copper database and the project reports.

11.1 Sampling

11.1.1 Pre-US Copper

Sample preparation and analyses completed for the historic drilling were done by a large, professional international mining company, Placer Dome, or its predecessor companies or its wholly-owned US subsidiaries Placer Dome/Amex.

For DDH's, the drill core has been cut in half, with half being sampled and sent to a lab and the remaining half stored for future reference and use. For the Pre-US Copper drilling programs, the cores were sampled at 10 to 5-foot intervals.

The 2006 underground sampling was completed in the old Superior underground workings. Select samples are taken to characterize a certain type or mineralogy, often high grade. Grabs are numerous pieces of material collected at random but not necessarily representative of grade in place. If meterage is shown, then the samples are chip-channels that should give a good representation of grade across the stated thickness. Only chip-channel samples were used in the averages and were 10-foot (3.1-meter) chip-channel samples that typically weighed 13 to 18 pounds (lbs) (6 to 8 kilograms [kg]).

The Superior underground workings contained some of the old 1966 to 1972 Placer drill core stored in cardboard boxes. Although the boxes were in poor condition, labels and intervals were sufficiently preserved to allow for a re-sampling of a number of intervals and therefore a comparison of Placer sampling and Sheffield sampling. In the 2006 Sheffield sampling of the old core, the remaining 1/2 split of core from Placer was sawed into two ¼ pieces and one of the ¼ pieces was sent to the lab.

Thirty samples of Placer core were collected in this manner. Two of the Placer core intervals sampled contained less core so the entire remaining 1/2 split was bagged and sent to the lab for analysis. Although there are some differences in the individual sample intervals of the core analyses, the overall core average was nearly identical at 0.37% Cu.

11.1.2 US Copper

For RC holes, recovered cuttings were delivered to a rotary splitter for sample collection for RC holes. The drill contractors collected a sample of the split at the rig during drilling using a pre-labeled bag. Samples are collected at 5-foot (1.5-meter) intervals and typically range from 10 to 15 pounds (4.5 to 6.8 kg) in weight, except for the 2023 drilling campaign at Engels, in which samples were collected at 10-foot (3-meter) intervals.

In 2021, US Copper drilled seven core holes at Superior. The drill core has been cut in half, with half being sampled and sent to a lab and the remaining half stored for future reference and use. For the US Copper drilling program, the cores were also sampled at 5-foot (1.5-meter) intervals.

11.2 Analytical Procedures

11.2.1 Pre-US Copper

The original core analyses for the Lights Creek District drilling were assayed at Placer's Golden Sunlight gold project in Montana. This lab was set up to treat the gold ores from the deposit, so the company's analytical techniques were well developed for precious metals procedures. During the late summer-to-early-fall of 1967, Placer-Amex determined that there was a problem with the soluble copper analyses being completed at their Golden Sunlight gold mine. Consequently, they began a program of re-assaying the entire core at an independent lab, Union Assay in Salt Lake City. The re-assaying, using chemical analyses, was completed by the spring of 1968. There are no records to indicate why Placer determined that the original analyses were incorrect, most of the results from the Golden Sunlight assayers no longer exist. Results used for grade determination did not include any of the original analyses, only the copper assays produced at Union Assay.

No reports or data detailing the methods of sample preparation, full analytical methods used, or quality control measures utilized by Placer-Amex were available to the writer for review and verification. It is encouraging to note that Placer-Amex must have had some system in place to determine that there was a problem with the original analyses completed at the Golden Sunlight mine to justify the re-assay of thousands of core samples. Full details of sample security of samples as required in NI 43-101 were not commonly provided in the internal company documents discussing the previous work. There is no reason to suspect any irregularities or question the results of the old sampling as the results contained in this report were collected by a reputable major mining company.

The 2005 to 2010 core samples were submitted to the ALS-Chemex laboratory, now ALS USA Inc. in Reno, NV. ALS USA is accredited by the Standards Council of Canada as conforming with requirements of CAN-P-1579, CAN-P-4E (ISO/IEC 17025:2005). At the ALS laboratory the core samples were sorted, dried, crushed and pulverized to 85% minus 75 microns (µm) using methodology WEI-21. Total copper was assayed by ALS methods Cu OG62 and CU AA62. These methods use a four-acid digestion by hydrogen fluoride (HF), nitric acid (HNO_3), perchloric acid ($HClO_4$), and hydrochloric acid (HCl) of the sample and the copper content is determined either by ICP (Cu OG62) or by AA (Cu AA62).

Sulfuric acid soluble copper was assayed by ALS method Cu-AA05. In the Cu-AA05 procedure the sample is leached at room temperature with 5% sulfuric acid and then agitated for an hour. The solution is analyzed by atomic absorption spectroscopy to determine the acid soluble copper concentration of the sample.

Gold was assayed by ALS method Au-AA23, which is a fire assay fusion of a 30-gram aliquot with an AA finish. The other elements were determined using ALS method ME-ICP- 61 in which the sample is digested in a four-acid leach and the elemental concentrations are determined by ICP-Atomic Emission Spectroscopy (AES). Once the results of the assays were received, they were posted on the digital drill logs.

11.2.2 US Copper

The 2021 and 2023 core and RC samples were submitted to the ALS-Chemex laboratory in Reno. At the ALS laboratory the core and RC samples were sorted, dried, crushed and pulverized to 85% minus 75 microns using methodology WEI-21. Total copper was assayed by ALS methods Cu OG62 and CU AA62. These methods use a four-acid digestion by HF, HNO3, HClO4 and HCl of the sample and the copper content is determined either by ICP (Cu OG62) or by AA (Cu AA62).

11.3 QA/QC

11.3.1 Pre-US Copper

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 during 1960s.

During 2006 to 2007, Sheffield conducted a re-assay program on 50 core samples from the Superior Historical 1960s. In 2009, Starfield conducted a re-assay program on 533 core samples from the Superior Historical 1960s. During these re-assay programs, samples were analyzed by American Assays Labs. The QA/QC program data for these two programs are presented in Table 11-1.

Table 11-1: Sheffield and Starfield (QA/QC Programs on Superior Historical Core Samples, Re-assay Programs)

Year	Re-assay Program	Core Samples	RC & Core Samples	Blank	Standard	Coarse Duplicate	Pulp Duplicate	% Of Total Assays
Historic (1960s) Superior	2006	51	51	2	3	N/A	N/A	9.8
	2009	533	533	27	26	N/A	46	18.6

During these campaigns, the material used for the blanks was marble, so no more information about the blank samples is available. Blank samples were inserted at approximately a rate of four and five blanks per 100 samples for the re-assay programs in 2006 and 2009, respectively.

No information is available about the standards used in the 2006 re-assay program. For the 2009 re-assay program, Standards were purchased from CDN Resource Laboratories Ltd. (CDN) of Delta, BC., and inserted at approximately a rate of five standards per 100 samples for the re-assay program in 2009.

Two certified reference materials (CRMs) obtained from CDN were used during the re-assay program in 2009, containing medium and high grades of copper up to a maximum of 1.038%. Table 11-2 Shows the seven CRMs used and their expected values and standard deviation for copper.

Table 11-2: List of Certified Reference Materials Used in the Re-assay Programs on Superior Historical Core Samples by Starfield

Certified Reference Material	Year	Certified Value %	- 3 Std dev	+ 3 Std dev
SH LG	2009	0.501	0.423	0.579
SH HG		1.038	0.912	1.164

From 2005 to 2010, there was no formal QA/QC program in place; however, there are a few data points in the US Copper database for some of the QA/QC programs that are presented in Table 11-3.

There are a few limited QA/QC programs during the 2005 to 2008 drilling campaigns at Moonlight conducted by Sheffield and Nevoro and also during the 2007 to 2010 drilling campaigns at Engels conducted by Nevoro and Starfield.

Table 11-3: Pre-US Copper (QA/QC Programs between 2005 to 2010)

Year	Rotary/RC Samples	Core Samples	RC & Core Samples	Blank	Standard	Coarse Duplicate	Pulp Duplicate	% Of Total Assays
2005 Moonlight	N/A	275	275	16	15	N/A	N/A	11.3
2006 Moonlight	N/A	1,391	1,391	16	79	N/A	N/A	6.8
2007 Moonlight	283	N/A	283	N/A	N/A	N/A	N/A	N/A
2008 Moonlight	N/A	515	515	7	28	N/A	N/A	6.8
2007 Engels	N/A	1,165	1,165	31	102	N/A	N/A	11.4
2008 Engels	N/A	753	753	12	35	N/A	N/A	6.2
2009 Engels	N/A	121	121	12	3	N/A	N/A	12.4
2010 Engels	N/A	199	199	5	5	N/A	N/A	5

During these campaigns, the material used for the blanks was Marble or from the Lab., no more information is available about the blank samples. Blank samples were inserted at approximately a rate of two and three blanks per 100 samples for Moonlight and Engels respectively.

Standards were purchased from CDN and inserted at approximately a rate of 6 standards per 100 samples for both Moonlight and Engels drilling programs.

Seven CRMs, obtained from CDN, were used during the 2005 to 2010 drilling programs, covering low, medium, and high grades copper to a maximum of 1.947% copper.

Table 11-4 shows the seven CRMs used and their expected values and standard deviation for copper.

Table 11-4: List of Certified Reference Materials Used in the 2005-2010 Drilling Programs

Certified Reference Material	Year	Certified Value %	- 3 Std dev	+ 3 Std dev
CDN-CGS-1	2005-2010	0.596	0.509	0.683
CDN-CGS-4		1.947	1.761	2.133
CDN-CGS-5		0.155	0.137	0.173
CDN-CGS-7		1.01	0.8	1.22
CDN-HG HLHC		5.07	4.26	5.88
SH LG		0.501	0.423	0.579
SH HG		1.038	0.912	1.164

11.3.2 US Copper

In 2021, US Copper conducted a re-assay program on the Superior historical 1960s core samples. Since the remaining Superior cores from the 1960s were Bx - BQ sized, US Copper used the remaining 1/2 split cores. The QA/QC program data is presented in Table 11-5

Table 11-5: US Copper (QA/QC Programs on Superior Historical Core Samples, a Re-assay program)

Year	Rotary/RC Samples	Core Samples	RC & Core Samples	Blank	Standard	Coarse Duplicate	Pulp Duplicate	% Of Total Assays
Historic (1960s) Superior	N/A	448	448	27	21	24	0	16

For this program, only one type of blank sample was used, which was purchased from CDN (CDN-BL-10). Blank samples were inserted at a rate of six samples per 100 samples.

The coarse duplicate samples were used for this program at a rate of five samples per 100 samples. Three CRMs obtained from CDN were used during this re-assay program, covering medium and high grades copper to a maximum of 1.587% Cu and inserted at approximately a rate of about five standards per 100 samples (Table 11-6).

Table 11-6: List of Certified Reference Materials Used in the Re-assay Programs on Superior Historical Core Samples by Us Copper

Certified Reference Material	Year	Certified Value %	- 3 Std dev	+ 3 Std dev
CDN-CGS-26	2021	1.58	1.37	1.79
CDN-CGS-41		1.71	1.56	1.86
CDN-CGS-45		0.747	0.66	0.834

US Copper implemented formal QA/QC programs for the 2021 to 2023 drill campaigns. These included the submission of blanks, CRMs, field duplicates, and the completion of a check assay program. A summary of the QA/QC sample numbers by year is included in Table 11-7 .

Table 11-7: List of Blank, CRM, and Duplicate samples Used in the 2021-2023 Drilling Programs

Year	Rotary/RC Samples	Core Samples	RC & Core Samples	Blank	CRM	Coarse Duplicate	Pulp Duplicate	% Of Total Assays
2021 Superior	N/A	1,159	1,159	66	75	66	N/A	17.8
2023 Moonlight	486	N/A	2,009	15	13	15	N/A	2.1
2023 Lambs Ridge	163	N/A	163	2	6	2	N/A	6.1
2023 Engels	199	N/A	199	13	17	N/A	N/A	15.0

For the 2021 Superior, 2023 Moonlight, 2023 Lamb's Ridge, and 2023 Engels drilling programs, blank samples were inserted at a rate of approximately six, one, one, and seven per 100 samples, respectively. Only one blank (CDN-BL-10) was used, which was purchased from CDN.

For these drilling programs no pulp duplicated was used.

For the 2021 core program, the coarse duplicate samples were marked every 20th sample and were prepared from the lab reject of the original ½ core. 2023 Moonlight and Lamb's Ridge duplicates were field duplicates. For the 2021 and 2023 drilling programs, coarse duplicate samples were taken at a rate of six, one, and one per 100 samples for the 2021 Superior, 2023 Moonlight, and 2023 Lamb's Ridge drilling programs, respectively. No Coarse duplicate sample was selected for the 2023 Engels drilling program.

Three CRMs obtained from CDN were used during the 2021 to 2023 drilling programs (Table 11-8). CRMs were inserted at a rate of approximately six, one, four, and nine per 100 samples for the 2021 Superior, 2023 Moonlight, 2023 Lamb's Ridge, and 2023 Engels drilling programs, respectively.

Table 11-8: List of Certified Reference Materials Used in the 2021-2023 Drilling Programs

Certified Reference Material	Year	Certified Value %	- 3 Std dev	+ 3 Std dev
CDN-CGS-26	2021-2023	1.58	1.37	1.79
CDN-CGS-41		1.71	1.56	1.86
CDN-CGS-45		0.747	0.66	0.834

11.4 Database

11.4.1 Pre-US Copper

The data available for historical data are the Excel files of collar, survey, lithology, and assay files. A review of the database showed there are historical paper data, including assay files and it is not known when exactly these paper files were scanned and digitized for import into the database.

11.4.2 US Copper

Several different methods for geologic and geotechnical data entry has been performed to enter the data into the database. In 2021 data entry personnel enter geology and geotechnical data into a series of Excel templates with extensive pick lists and validation rules. The logging geologist checked the digital file against the paper original and signed off on a printed copy of the captured data. The original paper capture forms were filed by drill hole. Since 2022, geologic logging was performed using the GeoSequel logging tool, GeoSequel Logger, where the geology was entered directly into the database. Each logging table has limited columns and selections for each column, which correspond with the previously described Excel template.

Assay data is imported as text upon receipt from the laboratory, retaining the original laboratory codes. Text is translated to numeric values within the database. Assay results are not reported from the database, until they have been QA/QC vetted. Assay results for blanks and standards are compared with expected results via database queries. Each result is graphed and validated by the senior geologist. After QA/QC validation, assays are assigned a passing or failing designation. Upon failure of the standard or blank, the failed standards and blanks and their corresponding assay batch is re-run until the results pass validation. The most recent assays for each failed batch were then utilized in the database.

QA/QC data is continuously reviewed as it is imported into the database. Comprehensive QA/QC reports are generated by a senior geologist and reviewed by senior staff at the end of each drilling campaign.

11.5 Sample Security

11.5.1 Pre-US Copper

Very little is known about the sample security pre-US Copper.

Tanaka in 2013 mentioned that he could evaluate the sample security procedures for the Placer-Amex drilling; however, given the prominence of the company involved, and the reputation of the drilling company used, he was prepared to accept the assay values produced with some limitations. Tanaka also considered the sample security appropriate for the drilling commissioned by Sheffield.

11.5.2 US Copper

From 2021 to 2023, US Copper maintained formal chain-of-custody procedures during all segments of sample transport.

RC samples were collected at the drill site and put into a single bin until the hole was completed. Upon abandonment of the hole, the drillers delivered the bin to a secure location in Cresent Mill, where the US Copper ‘s storage/core shack is located, for storage until the samples were picked up by the ALS pickup\delivery truck. When the ALS driver received the bin, the QA/QC samples were given to the driver. A list of the samples, including the QA/QC samples, was emailed to ALS. Upon arrival at the lab, the samples were placed in the receiving area and then inventory of the samples was completed. Upon completion of the inventory, the list was sent back to US Copper geologists to verify all samples were received at the lab.

The DDH samples were delivered to the core shack (in the same location as the RC samples) at the end of each shift. At the core shack, the core was logged, photographed, then cut by core saw. The samples were placed in one bin per hole and stored for pick up by ALS delivery staff. The procedure for ALS pick-up was the same as for the RC samples. The core shed was always locked except when active logging or cutting was being performed. The core shed was secured with locked gates and a security system to stop unauthorized entry.

11.6 QP Comments on Section 11 Sample Preparation, Analyses, And Security

GRE’s QP Dr. Hamid Samari reviewed all sample preparation procedures, analytical procedures, and security measures for the historical data and measures employed by US Copper for the drilling campaign between 2021 to 2023.

GRE’s QP believes that although previous practices and procedures prior to US Copper’s purchase of the property may not have met current best industry practices, based upon the reputation of the drilling company used, the information of a problem among assays and conducted a re-assay (see analytical procedure in this capture), and finally the re-assay program in 2021 conducted by US Copper on the remaining core samples from the Superior historical drilling programs are enough to accept the assay values.

In the opinion of GRE’s QP Dr. Hamid Samari, the sampling preparation, security, and analytical procedures used during 2021 and 2023 are consistent with generally accepted industry best practices and are, therefore, adequate for Mineral Resource estimation.

In the opinion of GRE's QP, Dr. Hamid Samari:

- A detailed review of field practices and sample collection procedures should be performed on a regular basis to ensure that the correct procedures and protocols are being followed.
- Review and evaluation of laboratory work should be an on-going process, including occasional visits to the laboratories involved.
- Sample collection, preparation, analysis, and security for the RC and core drill programs follow industry-standard copper deposit methods.
- Standards, blanks, and duplicates including one standard, one duplicate, and one blank sample should be inserted every 20 interval samples, as is common within industry standards. This standard procedure was considered during the 2021 Superior and 2023 Engels drilling programs and was not followed for the 2023 Moonlight and Lamb's Ridge drilling program. This standard should be considered and continued for future drilling programs.
- QA/QC program results do not indicate any problems with the analytical programs (refer to discussion in Section 12).
- Data is subject to validation, which includes checks on surveys, collar coordinates, and assay data. The checks are appropriate and consistent with industry standards (refer to discussion in Section 12).

GRE's QP Dr. Hamid Samari is of the opinion that the quality of the copper analytical data is sufficiently reliable to support Mineral Resource estimation without limitations on Mineral Resource confidence categories.

No factors were identified with any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

12 DATA VERIFICATION

12.1 External Data Verification

12.1.1 Independent Mineral Consultant (Tanaka)

In 2013, Independent Mineral Consultant (W.F. Tanaka) prepared the first National Instrument 43-101 compliant resource estimate for the deposits of the Superior Project. Tanaka completed a data verification program of a significant portion of the historical drill hole database. Tanaka reviewed and examined the project's drill hole database that contained assay, survey, and geological information for historical drill campaigns.

Tanaka presented a summary description of the checks made on, and corrections or adjustments made to the drill hole database. A detailed list of errors was provided to US Copper.

In addition to the above, a total of 366 assay intervals for the Engels drilling done by Sheffield representing 19% of the total modern Engels database, was checked against the assay certificates for data entry errors in copper (3 methods), silver, gold, iron, and arsenic. A total of 51 errors were found, all confined to the iron assays. No other errors were found for the other elements. The erroneous iron values were determined to arise from the spreadsheet supplied by the laboratory which was directly loaded into the database without checking. The correct values were present on the write-protected assay certificates provided in PDF format.

On the whole, the error rate discovered by Tanaka in the above comparisons corresponds to a 1.99% error rate. Tanaka mentioned that this error rate is acceptable for a database not previously subject to rigorous scrutiny. As a final adjustment, all assay values designated as below the detection limit for the assay method employed was set at ½ of the detection limit.

12.1.2 Tetra Tech

In 2018, Tetra Tech prepared a technical report and preliminary economic assessment for the Moonlight deposit, with Donald Cameron as QP. Tetra Tech reviewed historical data for the Moonlight project and checked the accuracy of the database.

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 Tetra Tech, Sheffield drilling programs substantially complied with current Exploration Best Practices recommended by Canadian Institute for Mining, Metallurgy and Petroleum (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.

Tetra Tech notes that Placer-Amex-era drill holes have not been 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 the technical report (Tetra Tech, 2018). Based on these findings, copper and silver assays from Placer-Amex drill campaigns are also suitable for use in Mineral Resource estimation.

12.2 Data Verification by Global Resource Engineering

The drilling data was submitted to GRE for a final review in July 2024. GRE's QP, Dr. Hamid Samari, conducted an independent database review from the 1960s to 2023 drilling programs. The data, including collar, survey, assay, geology, original certificates, and QA/QC files, was provided to GRE in .csv and pdf formats.

12.2.1 Pre-US Copper Data (1964-2016)

Pre-US Copper drilling programs, prior to 2016, include 527 holes containing 26,348 assays for 241,947.6 feet (73,745.6 meters) of drilling, only have a few data available on the QA/QC programs. For drillholes from this period the database contains collar, survey, assay, and geology information.

GRE's QP, Dr. Hamid Samari, reviewed all available historical data. Original assay certificates and a few QA/QC data for 2005, 2006, 2007, and 2008 are the only available data from drilling programs prior to US Copper (1964-2016).

For those drilling campaigns, pre-US Copper, GRE'QP did manual audit work on about 11% of original assay certificates for 68 holes, including 3,012 intervals, from 2005 to 2010 and found no material errors.

GRE's QP also reviewed all existing QA/QC data and did not find any errors that could affect the MRE.

12.2.2 US Copper Database (2021-2023)

From 2021 to 2023, US Copper completed 13,112.3 feet (3,996.6 meters) of drilling containing 2009 assay samples in 39 drill holes at Superior, Moonlight, Lamb's Ridge, and Engels.

The current data was provided to GRE in .csv format, including collar, survey, assay, geology for the entire database, and US Copper's in-house QA/QC files.

GRE independently analyzed US Copper's data relevant to the 2021 to 2023 drilling programs, comparing the data with the provided assay certificates. About 40% of all original assay certificates for 18 holes, including 837 intervals, from the 2021 to 2023 drilling programs were manually spot-checked with the database for accuracy, and no errors were found.

12.2.3 Verification of Pre-US Copper Analytical Quality Control Data

Pre-US Copper's in-house QA/QC data from the 1964 to 2016 drilling programs were submitted to GRE for review and verification. The data included in-house QA/QC data for re-assay programs on Superior historical data (the 1960s) conducted by Sheffield in 2006 and Starfield in 2009 and for 2005, 2007, 2008, 2009, and 2010 Moonlight and Engels drilling programs and showed satisfactory blank, standard, and duplicate results. No duplicate samples were used for drilling campaigns 2005, 2007, 2008, and 2009. All samples were analyzed at American Assay Labs, and their certificates are available.

Sheffield, Nevoro, and Starfield conducted the QA/QC programs for these data. There is no record of which specific blank was used in each drilling campaign. In some data it was mentioned that marble was used as blank samples. The Certified Reference Materials were mostly purchased from CDN Resource Laboratories Ltd. (CDN).

12.2.3.1 Re-assay Program

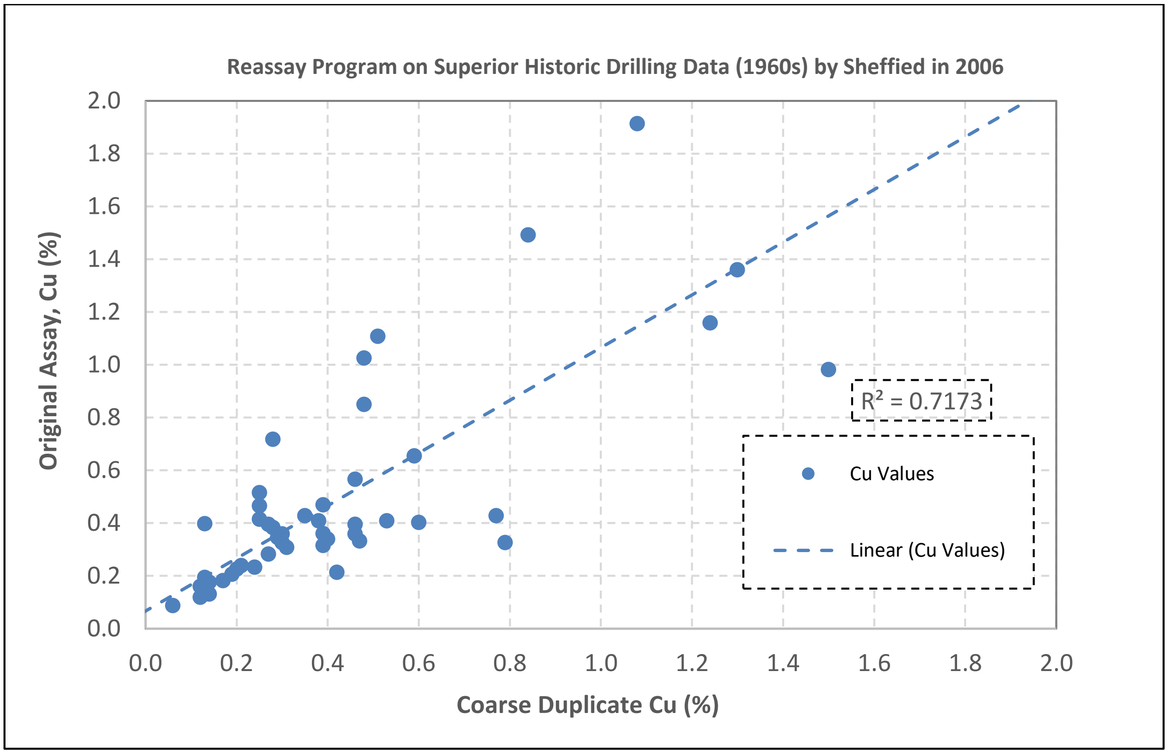
12.2.3.1.1 Sheffield

For the 2006 re-assay program, Sheffield used only two blanks for all 50 core samples selected from the Superior historical samples (1960s). The assay results for both of them evaluated 100 ppm Cu, which is under the threshold (5X DL) and an acceptable result. Generally, the QA/QC blank sample insertion rates for this program nearly followed the accepted industry standards, which is one blank sample for every 20 interval samples.

Three standards were used for this re-assay program. Two of the three standards are not listed. The only standard used was a CDN (std SH HG), and its assay result was 1.02% Cu, which is acceptable. No duplicate sample was used for this program.

Sheffield complied 51 samples out of four drill holes from the Superior historical data (1960s). Figure 12-1 is a scatterplot of the original sample versus the re-assays values for copper. The result does not show a high correlation between the original and the duplicate assays because the mineralization at Superior is also associated with the veinlets, and half or quarter-core samples do not always show the same assay results. Considering this subject, GRE’s QP is of the opinion that the result is acceptable for this re-assay program.

Figure 12-1: Scatterplot of Original Cu Assay and Re-assay, Sheffield in 2006



Source: GRE, 2024

12.2.3.1.2 Starfield

For the 2009 re-assay program, Starfield used 27 blank, 26 standards, and 46 pulp samples for all 533 core samples selected from the historical Superior samples (the 1960s) and showed satisfactory results.

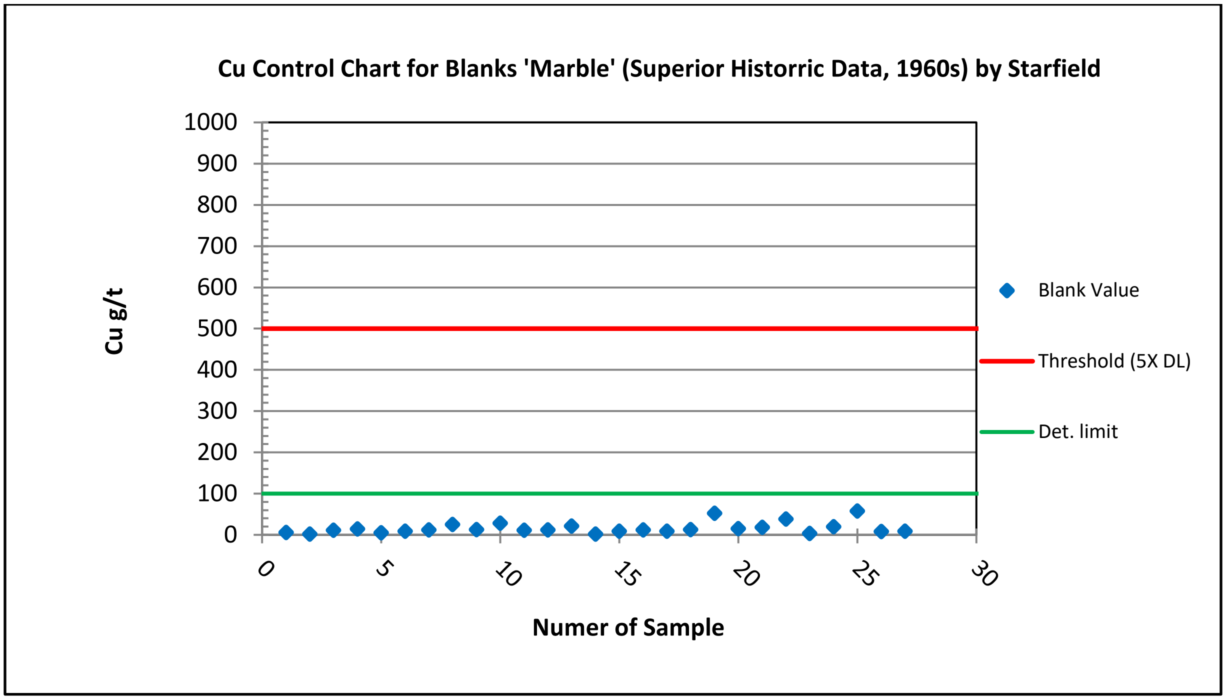
The blank samples used in this program were Marble, and they were inserted at a rate of one blank per 20 core samples, which is common within industry standards (Figure 12-2).

In the 2009 re-assay program, commercially prepared CRM samples for copper were inserted into the sample stream at a rate of one standard per 20 sample assays for all 533 core samples (Figure 12-3 and Figure 12-4).

Starfield considered 46 pulp duplicates for this program for all 533 core samples. Duplicate samples were prepared the same way as all assay samples and were assayed at American Assay Labs.

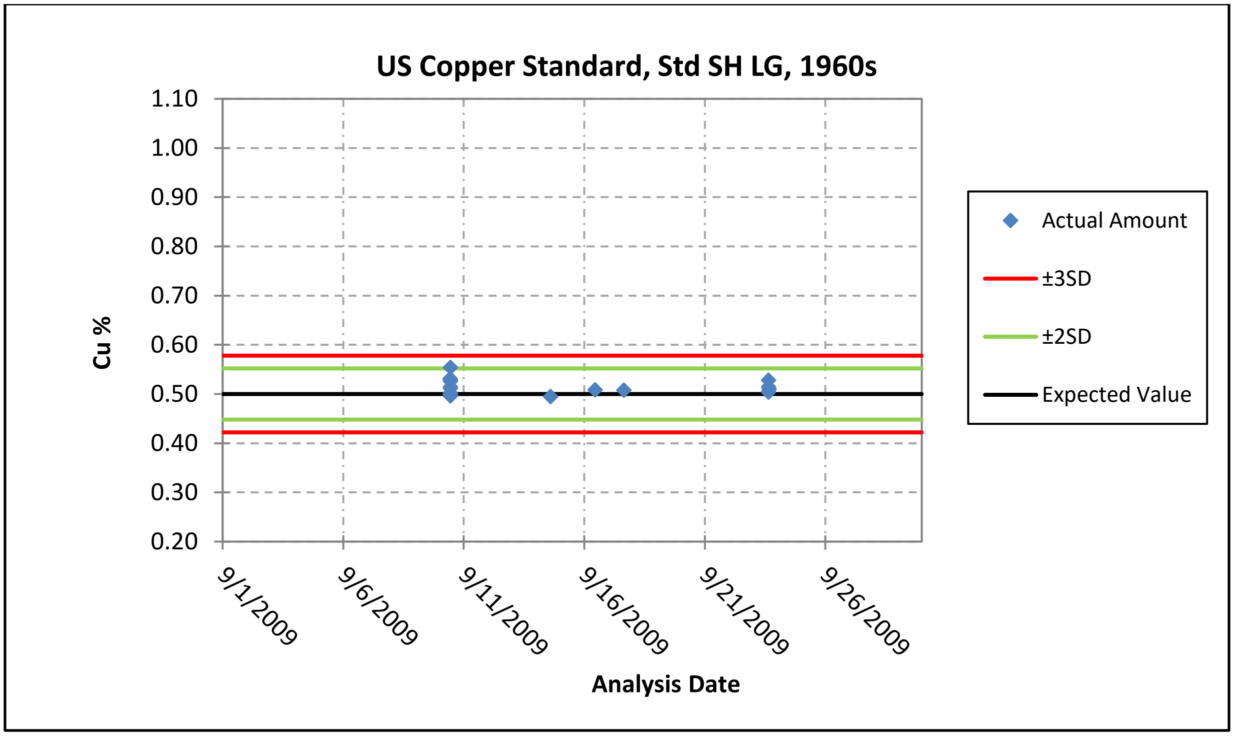
The Q-Q plot for copper effectively indicates no scatter, with R2 values of 0.9868 for pulp duplicates (Figure 12-5).

Figure 12-2: Blank Results from the 2009 Re-Assay Program, Starfield 2009



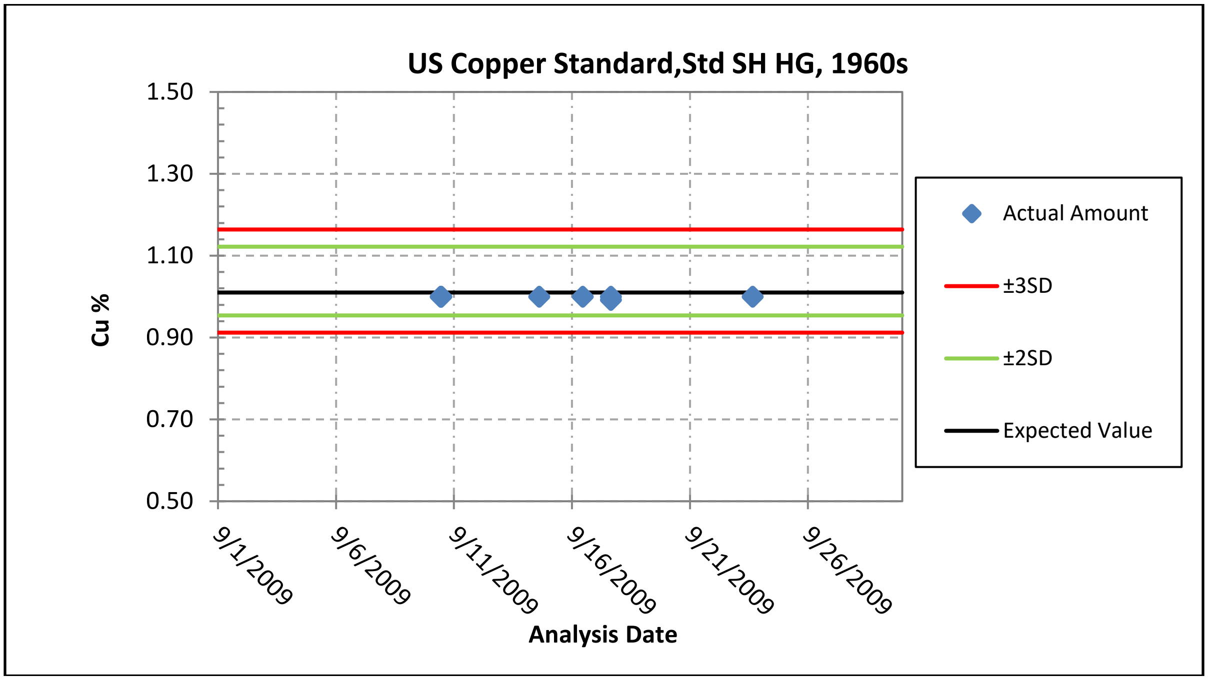
Source: GRE, 2024

Figure 12-3: SH LG Results from the 2009 Re-Assay Program, Starfield 2009



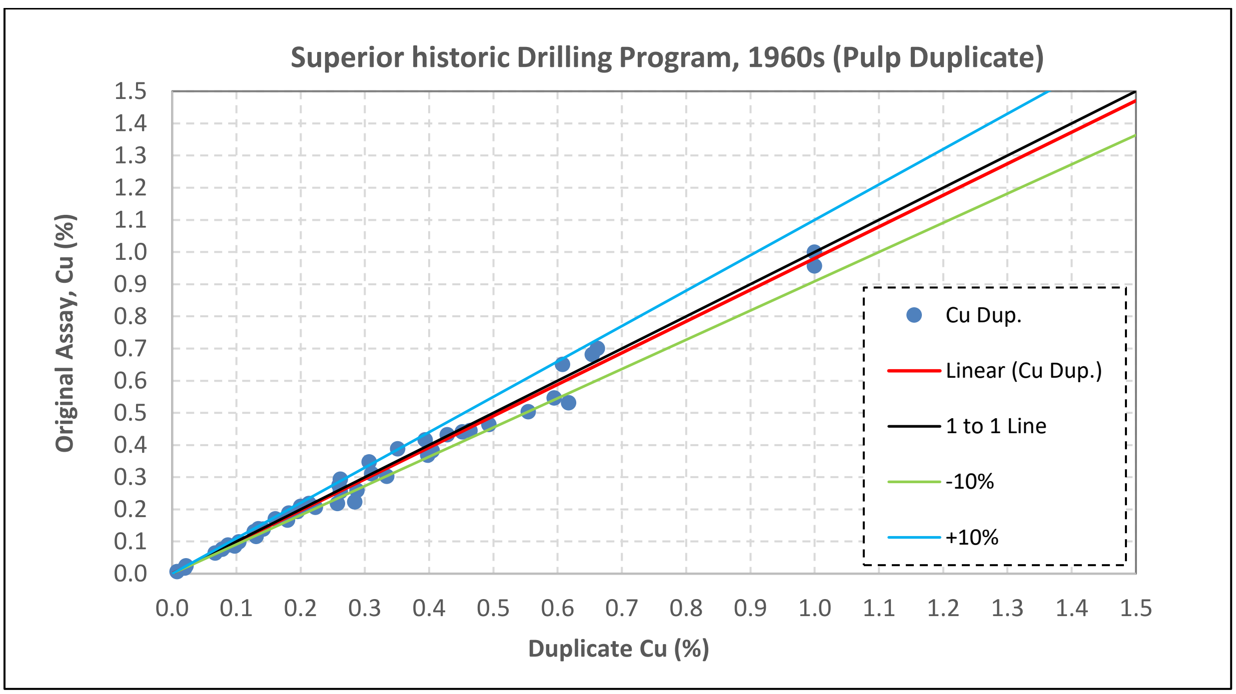
Source: GRE, 2024

Figure 12-4: SH HG Results from the 2009 Re-Assay Program, Starfield 2009



Source: GRE, 2024

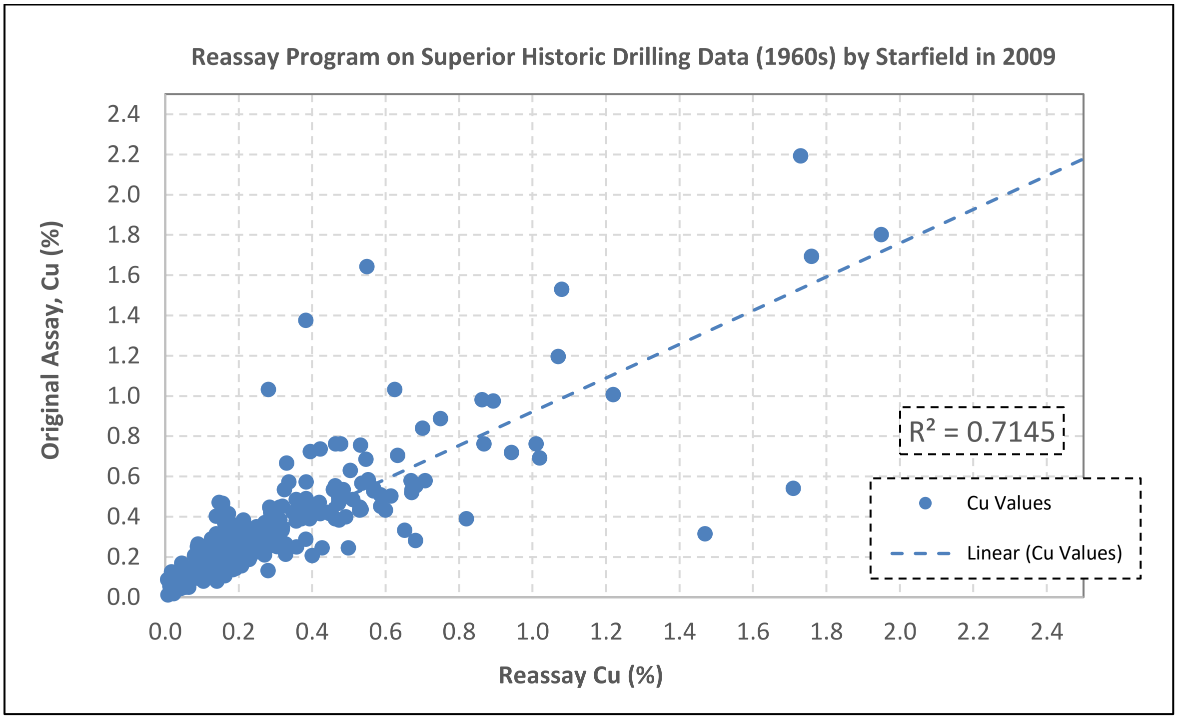
Figure 12-5: Pulp Duplicate Samples for the 2009 Re-assay Program, Starfield 2009



Source: GRE, 2024

In 2009, Starfield compiled 533 core samples out of 13 drill holes from the Superior historical data (1960s) for a re-assay program. US Copper provided GRE with this data. Only 321 samples out of 533 samples have the same sample intervals as their original assays and were considered to be reviewed. Figure 12-6 is a scatterplot of the original sample versus the re-assay values for copper. In this program, the results also do not show a high correlation between the original and the re-assays, especially for the samples with high-grade copper. Similar to the re-assay program conducted by Sheffield in 2006, this program also confirmed that high-grade mineralization zones are mainly associated with the veinlets, which do not have a homogenous distribution of mineralization along the core samples.

Figure 12-6: Scatterplot of Original Cu Assay and Re-assay, Starfield in 2009



Source: GRE, 2024

12.2.3.2 Drilling Program

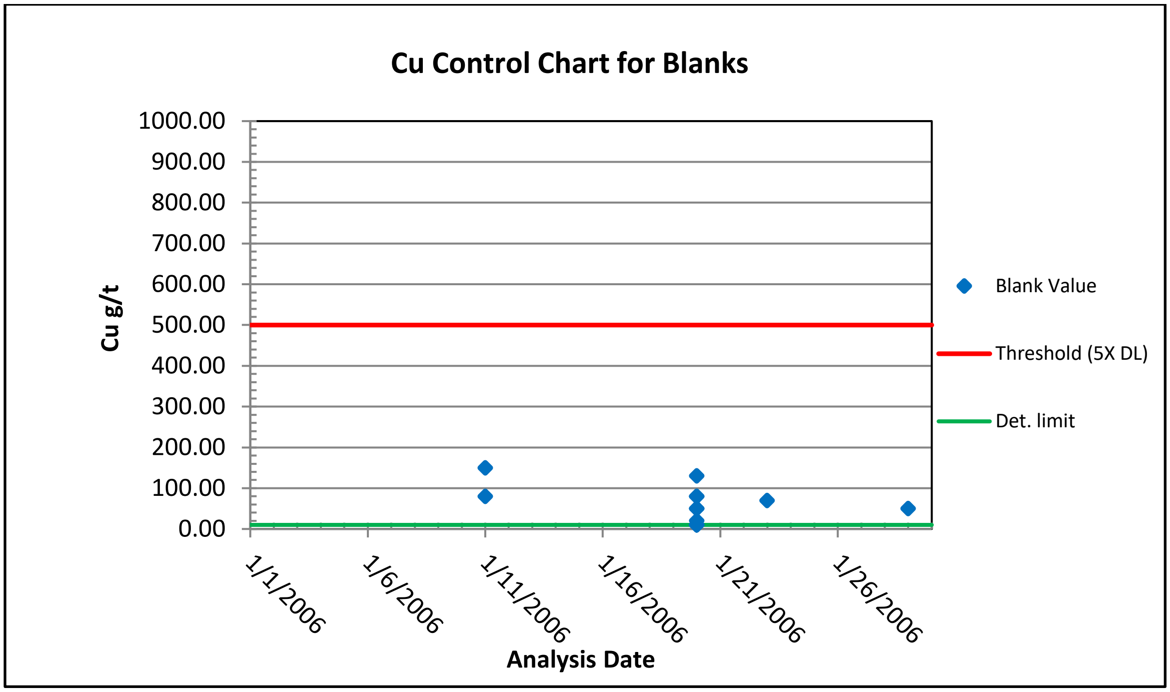
12.2.3.2.1 Blank Samples

Blank samples were used for Moonlight and Engels were marble. For Moonlight there are 16 blank samples for 2005, 16 blank samples for 2006, and seven blank samples for the 2008 drilling campaigns, totaling 39 blank samples for the 2005, 2006, and 2008 drilling programs. For Engels there are 31 blank samples for 2007, 12 blank samples for 2008, 12 blank samples for 2009, and five blank samples for the 2010 drilling programs, totaling 60 blank samples for the 2007, 2008, 2009, and 2010 drilling programs.

Figure 12-7 to Figure 12-13 show the assay results of the blanks for copper prepared by GRE. The revaluation results show only three samples for holes Engels 2009 registered above the 3X detection limit. It seems they are core samples not blank samples and a mislabeling probably happened for these three samples and there was no contamination during the lab analysis.

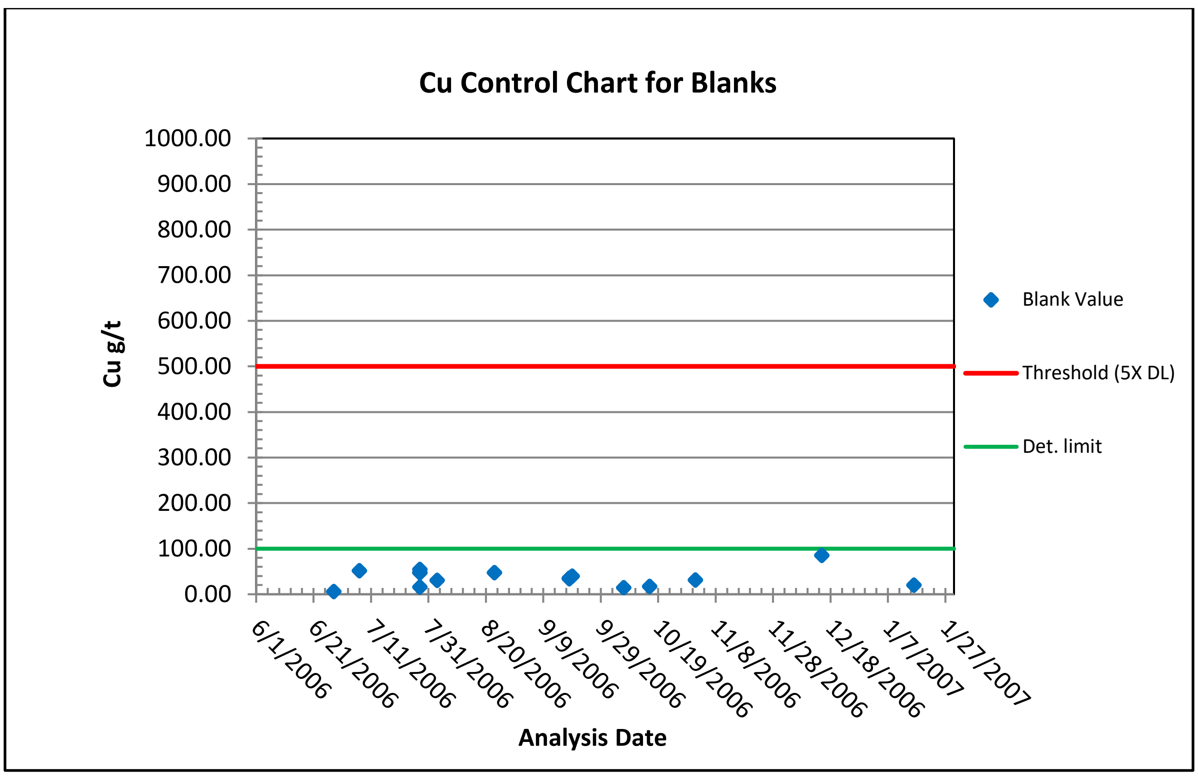
Generally, the QA/QC blank sample insertion rates except for campaigns 2005 Moonlight and 2009 Engels fall below general accepted industry standards. For future exploration campaigns, one blank sample should be inserted every 20 interval samples, as is common within industry standards.

Figure 12-7: 2005 Drilling Campaign, Blank Results, Moonlight



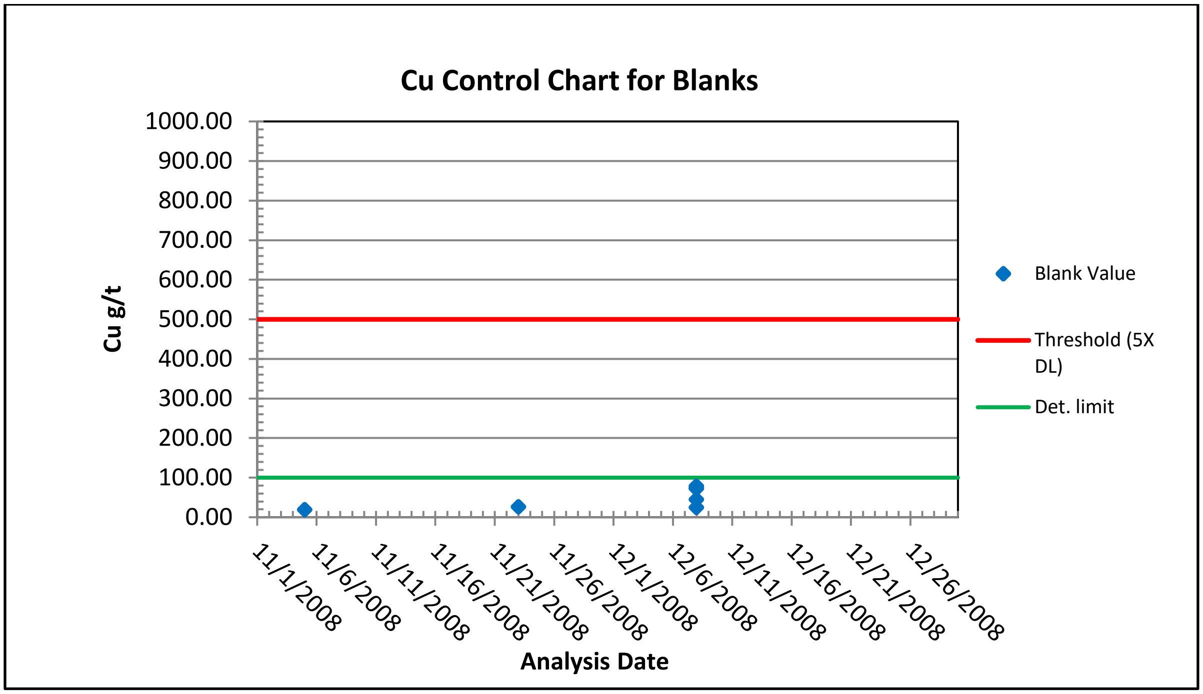
Source: GRE, 2024

Figure 12-8: 2006 Drilling Campaign, Blank Results, Moonlight



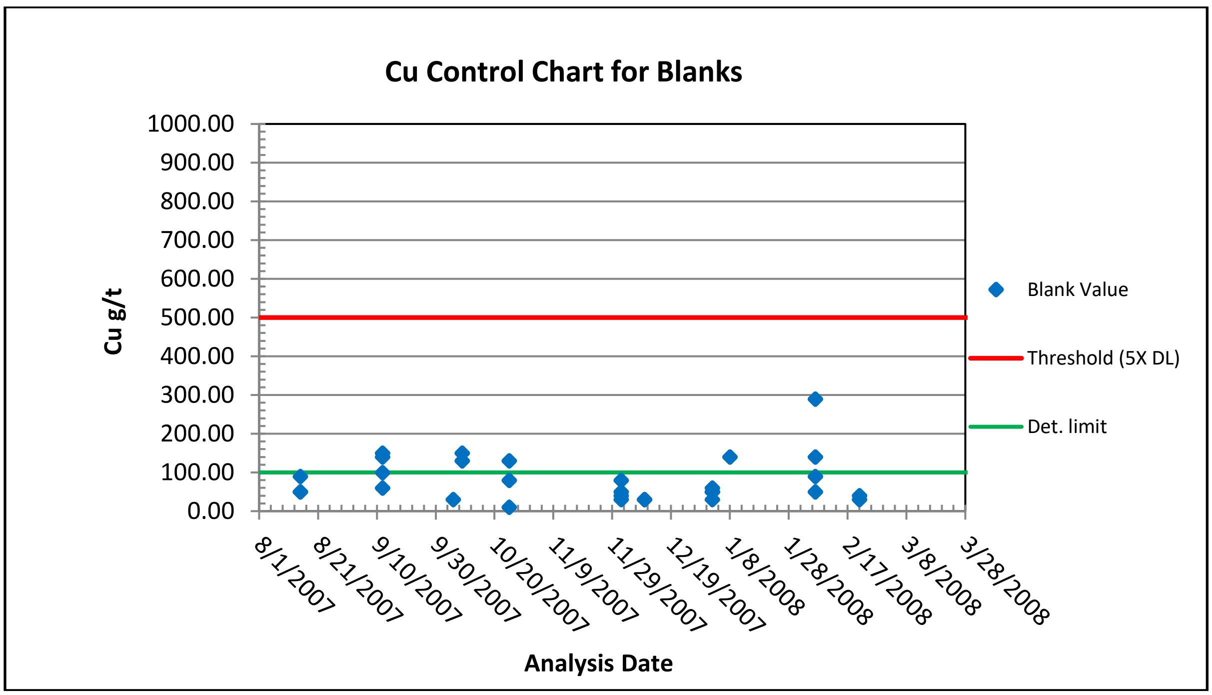
Source: GRE, 2024

Figure 12-9: 2008 Drilling Campaign, Blank Results, Moonlight



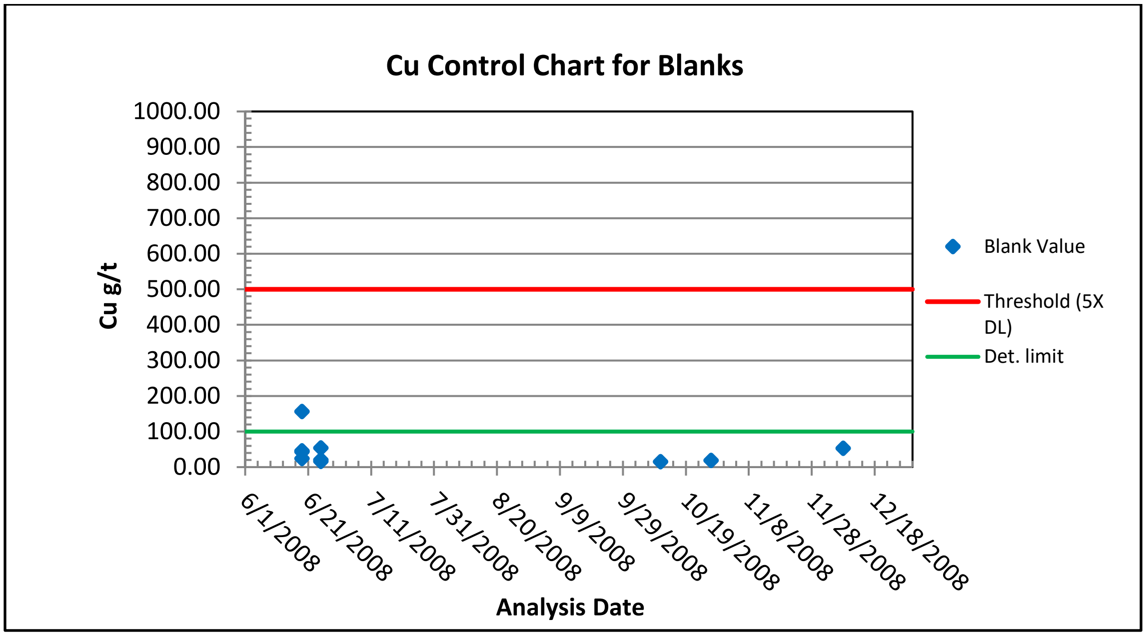
Source: GRE, 2024

Figure 12-10: 2007 Drilling Campaign, Blank Results, Engels



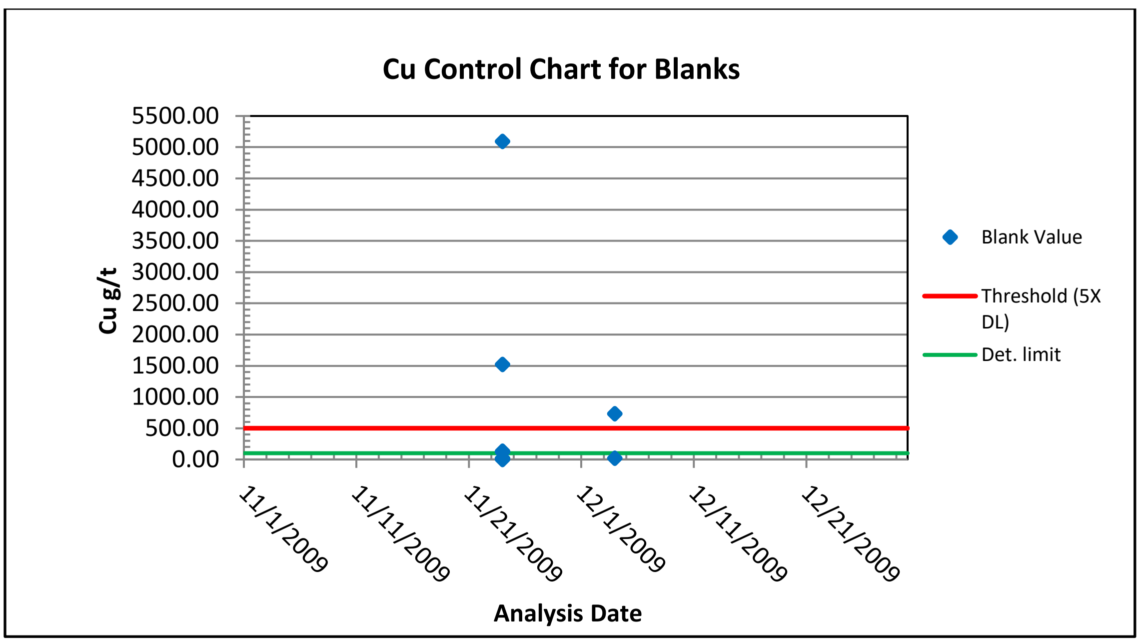
Source: GRE, 2024

Figure 12-11: 2008 Drilling Campaign, Blank Results, Engels



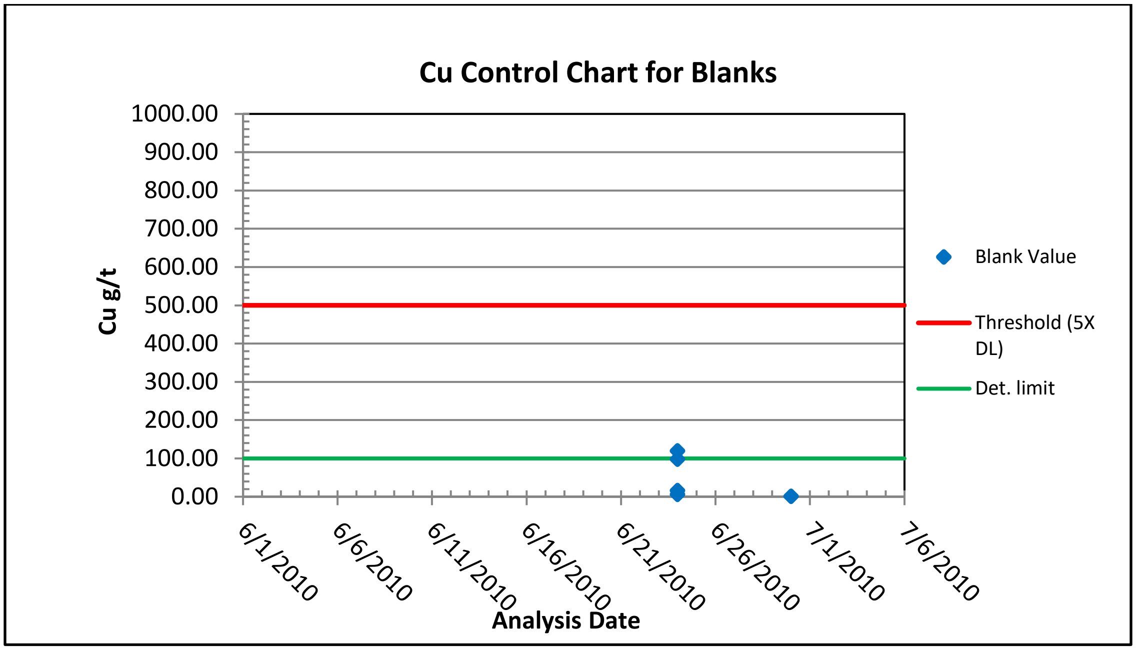
Source: GRE, 2024

Figure 12-12: 2009 Drilling Campaign, Blank Results, Engels



Source: GRE, 2024

Figure 12-13: 2010 Drilling Campaign, Blank Results, Engels



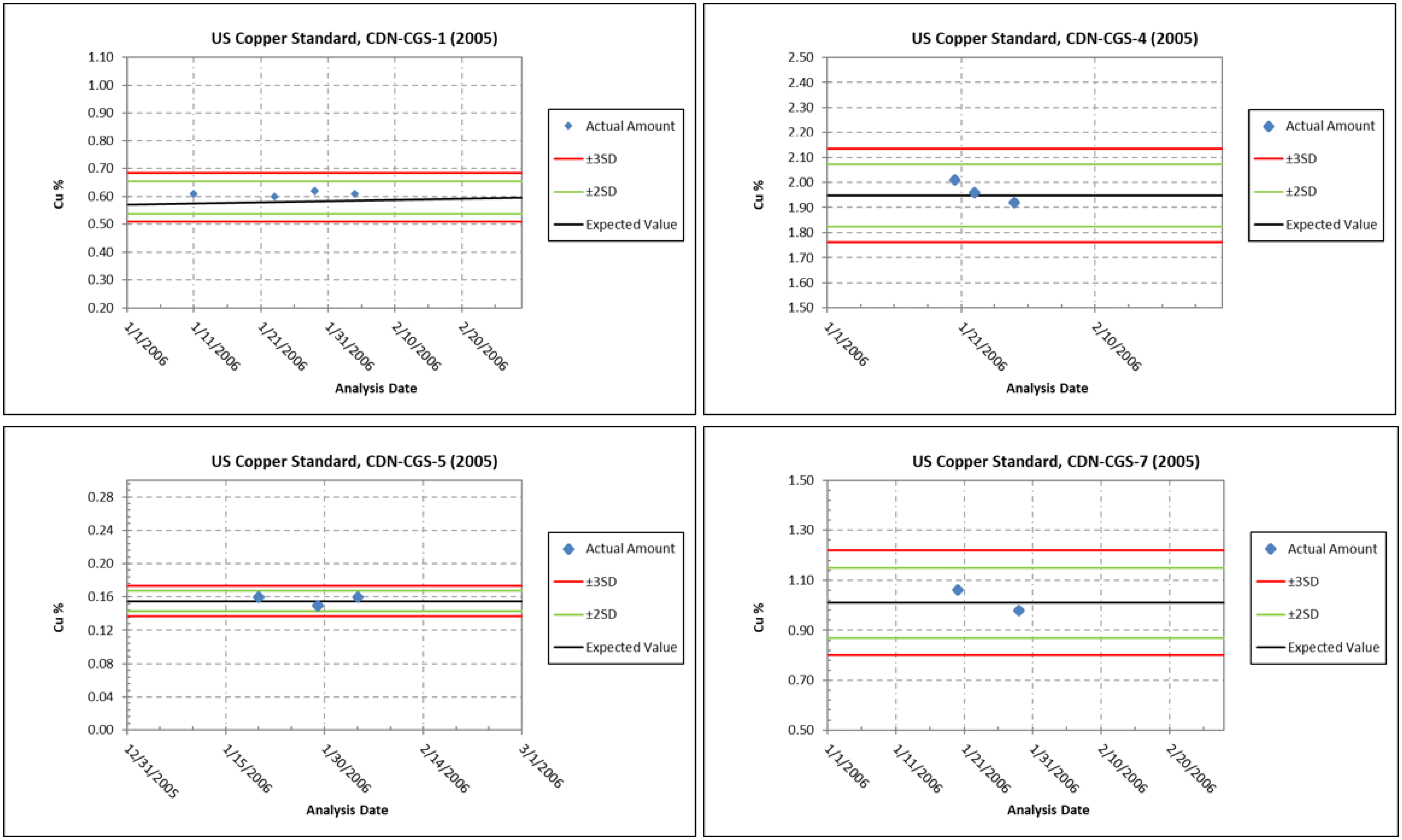
Source: GRE, 2024

1.1.1.1.2 Certified Reference Materials Analysis

Commercially prepared CRM samples for copper were inserted into the sample stream for Moonlight drilling programs in 2005, 2006, and 2008 at a rate of one standard per 18 sample assays for all 2128 core samples. For Engels drilling programs, CRM samples for copper were inserted into the sample stream in 2007, 2008, 2009, and 2010 at a rate of one standard per 11, 21, 40, and 40 sample assays, respectively, for all 2238 core samples.

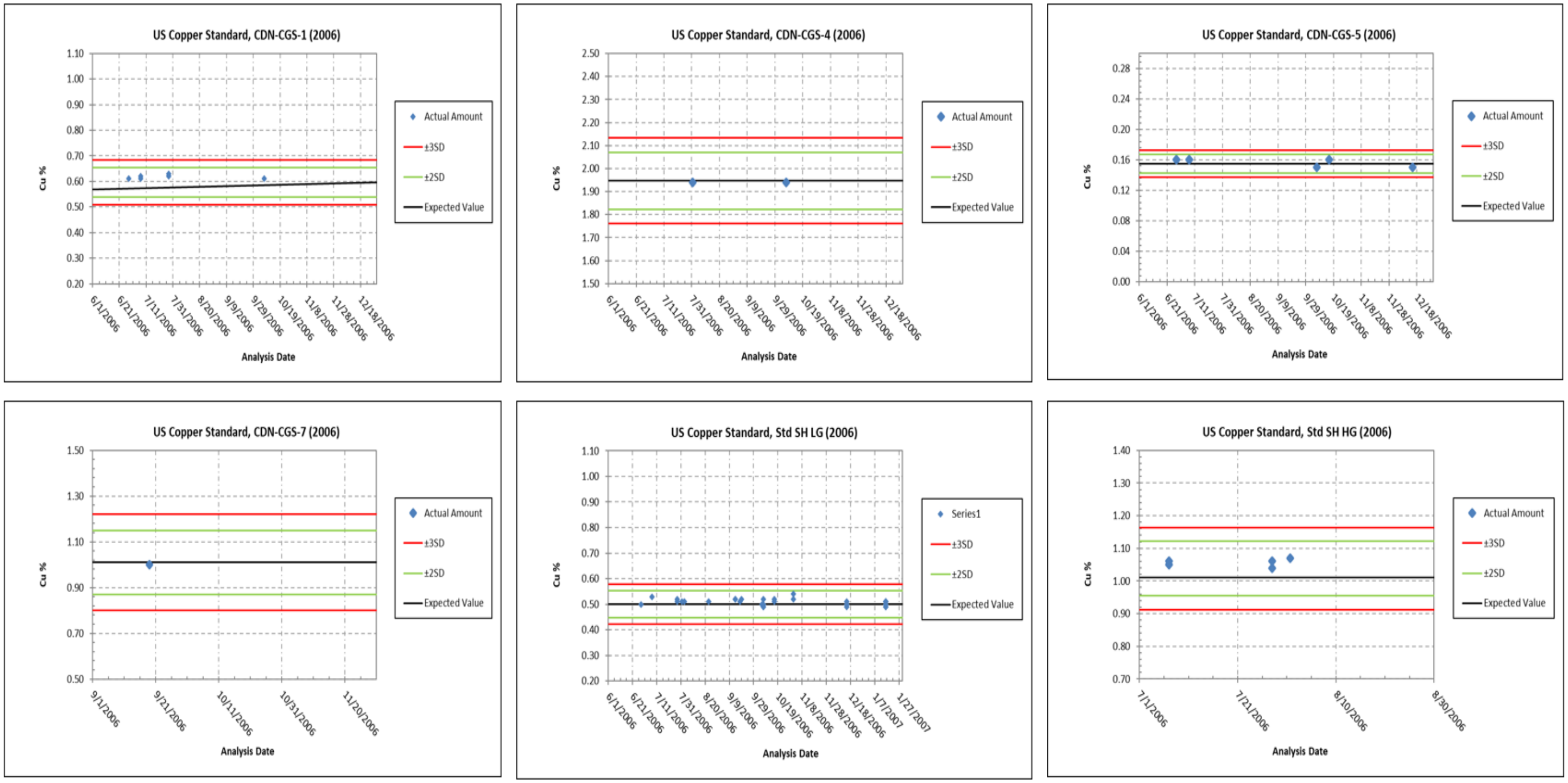
An analysis of CRMs charts for low, medium, and high copper grades showed no obvious errors or bias (see Figure 12-14 through Figure 12-20).

Figure 12-14: CDN-CGS-1, 4, 5, and 7 from the 2005 Moonlight Drilling Campaign



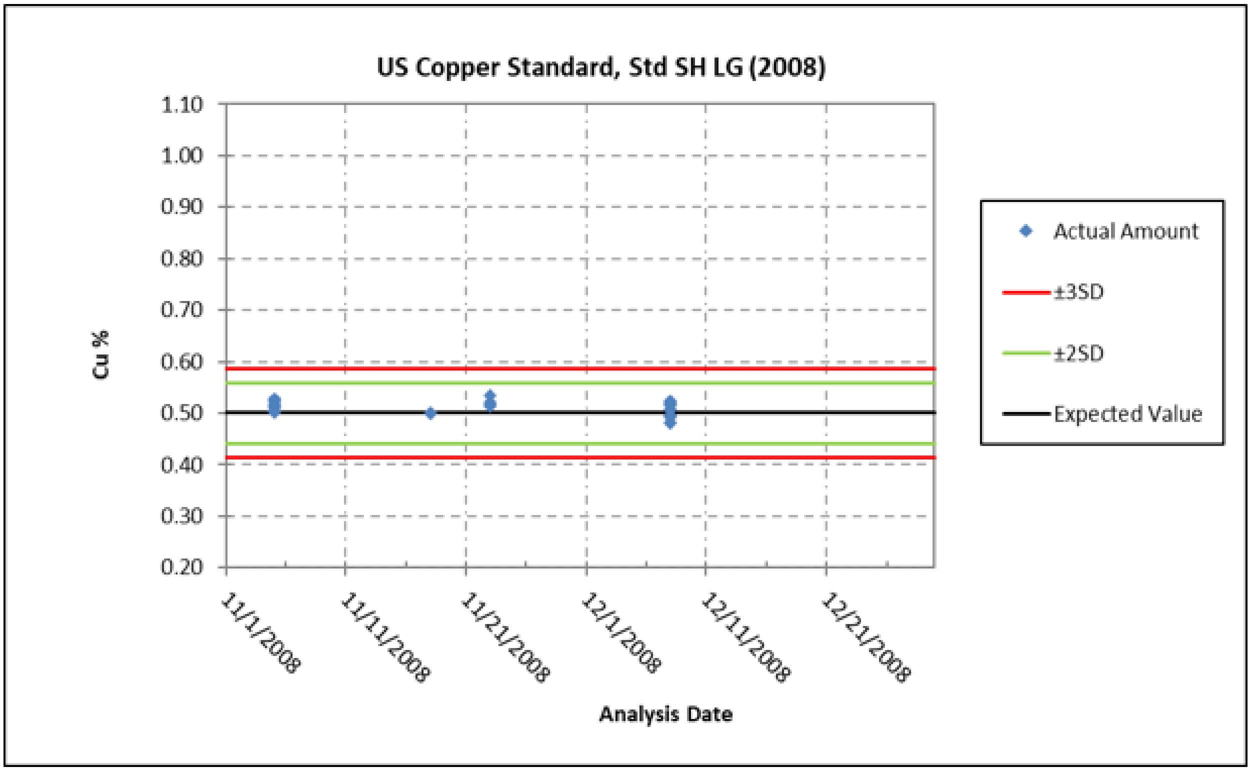
Source: GRE, 2024

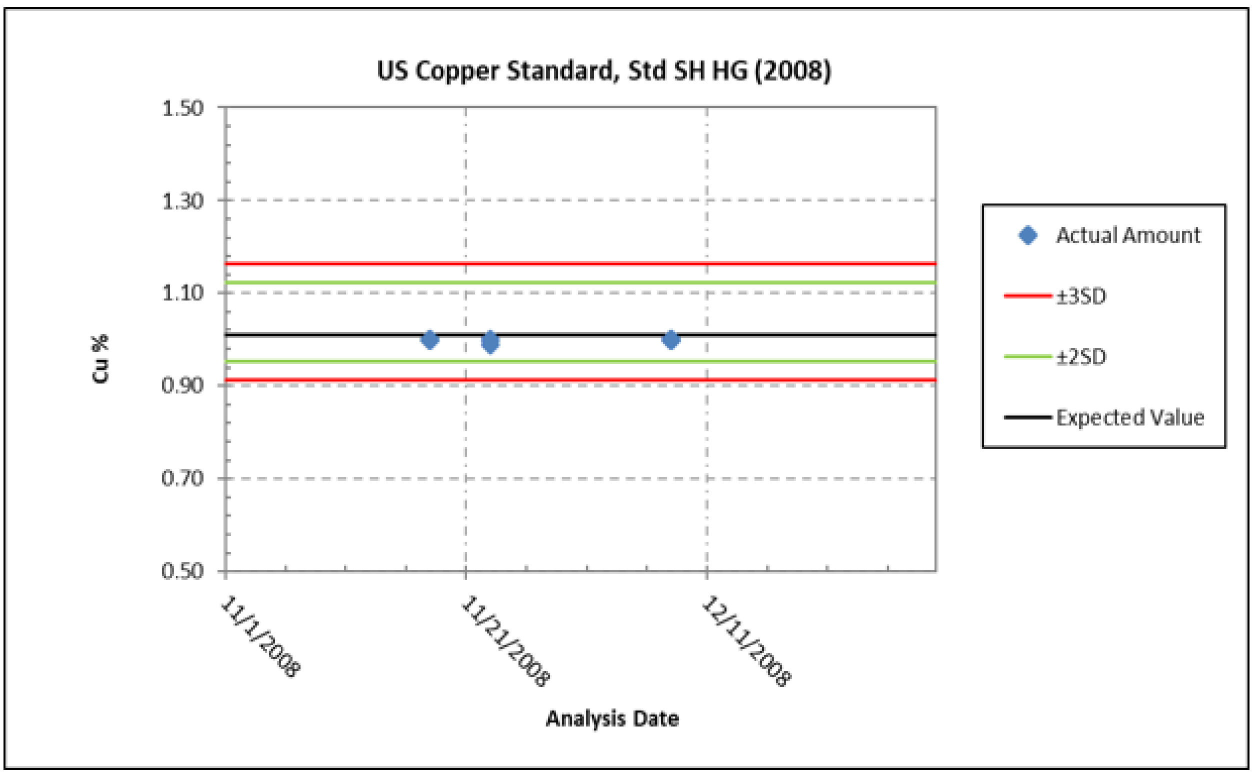
Figure 12-15: CDN-CGS-1, 4, 5, 7, SH LG, and SH HG from the 2006 Moonlight Drilling Campaign



Source: GRE, 2024

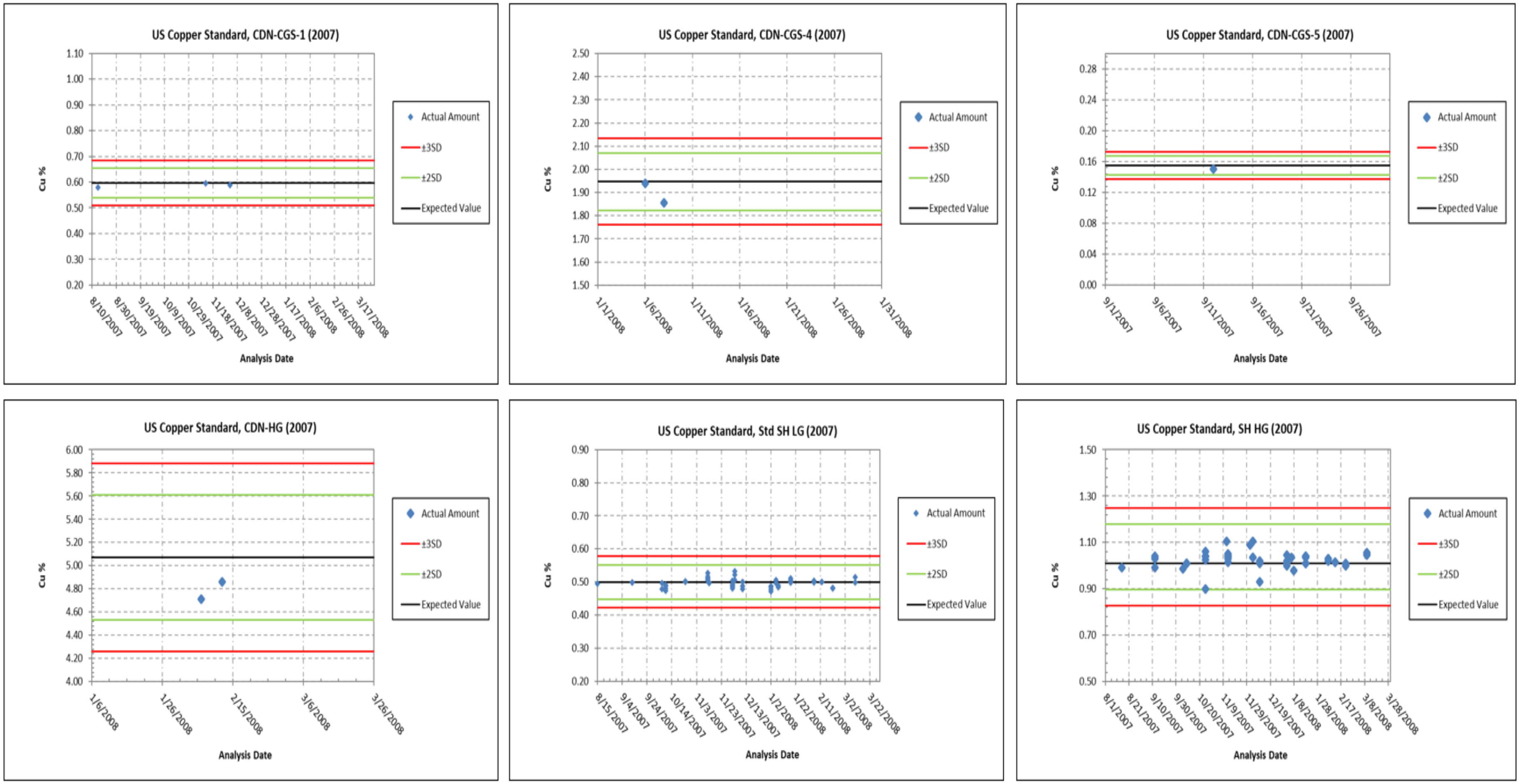
Figure 12-16: SH LG, and SH HG from the 2008 Moonlight Drilling Campaign





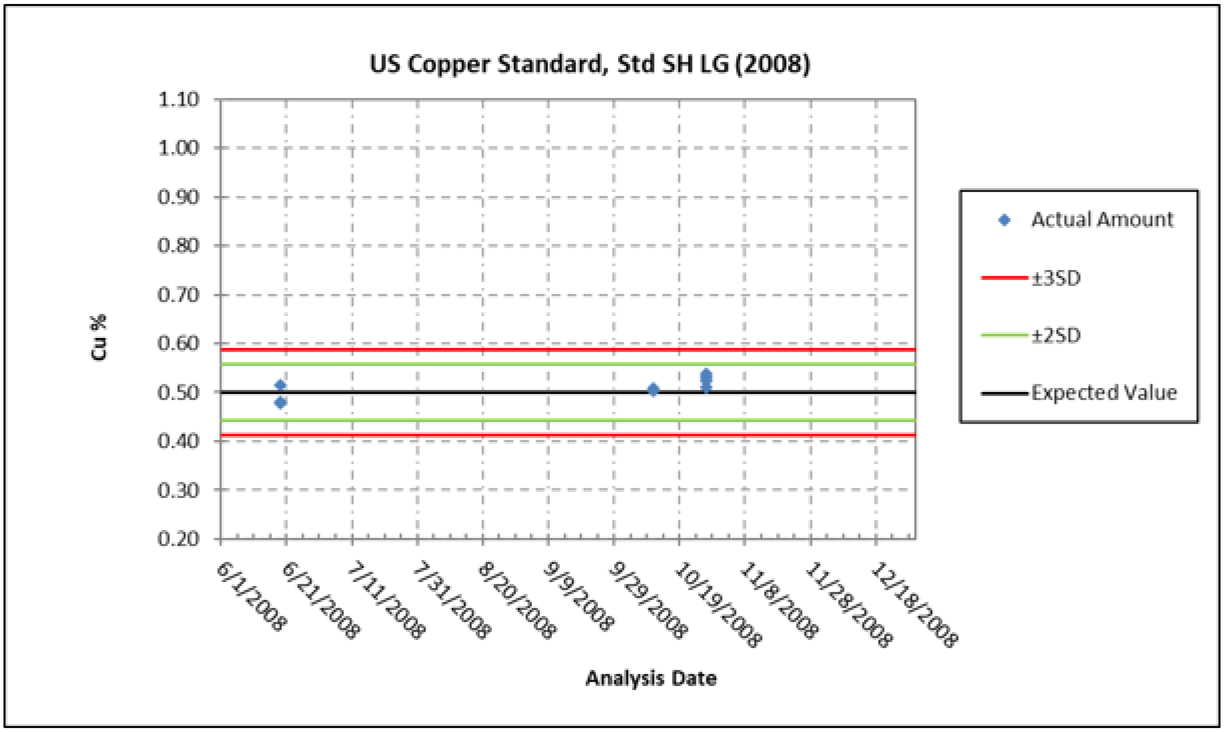
Source: GRE, 2024

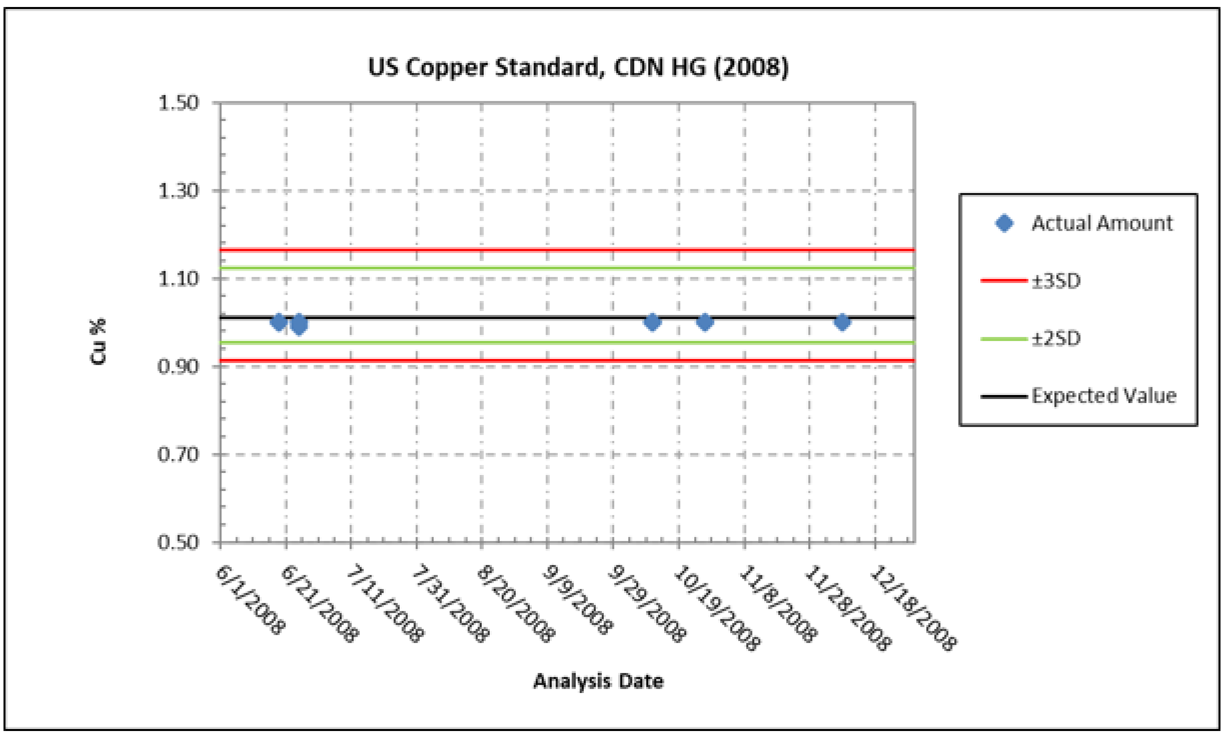
Figure 12-17: CDN-CGS-1, 4, 5, 7, CDN-HG, SH LG, and SH HG from the 2007 Engels Drilling Campaign



Source: GRE, 2024

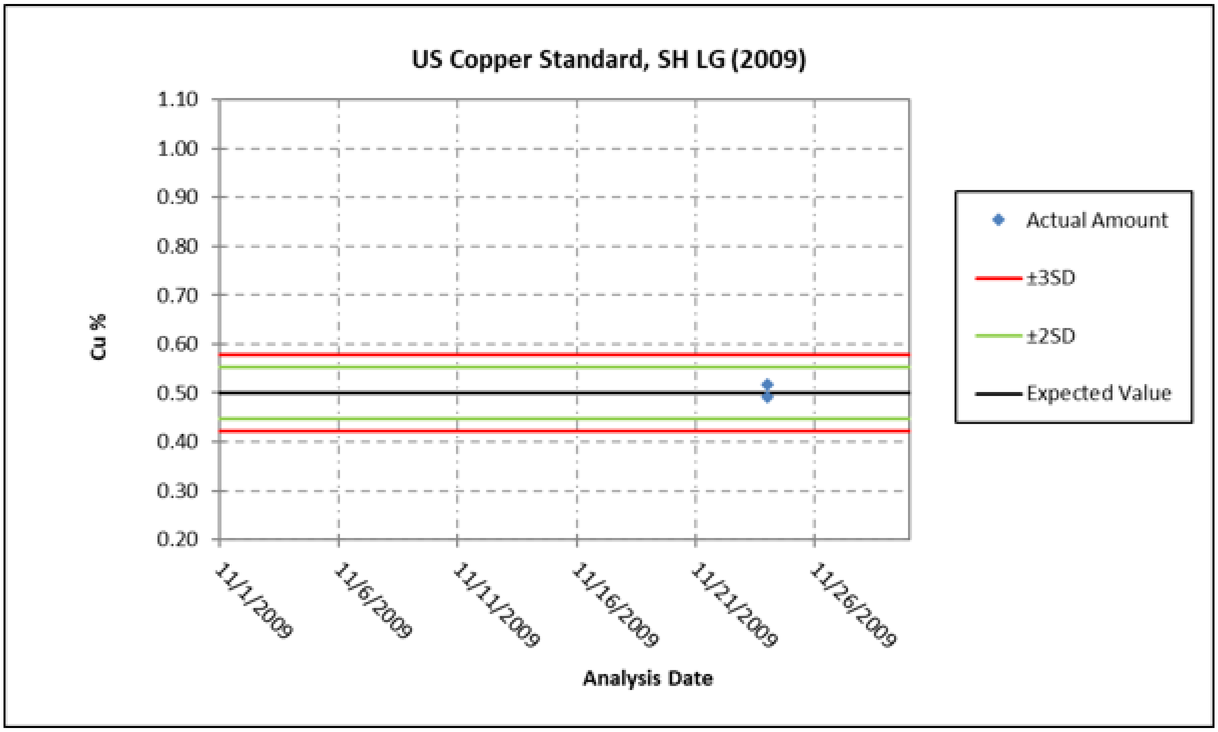
Figure 12-18: SH LG and SH HG from the 2008 Engels Drilling Campaign

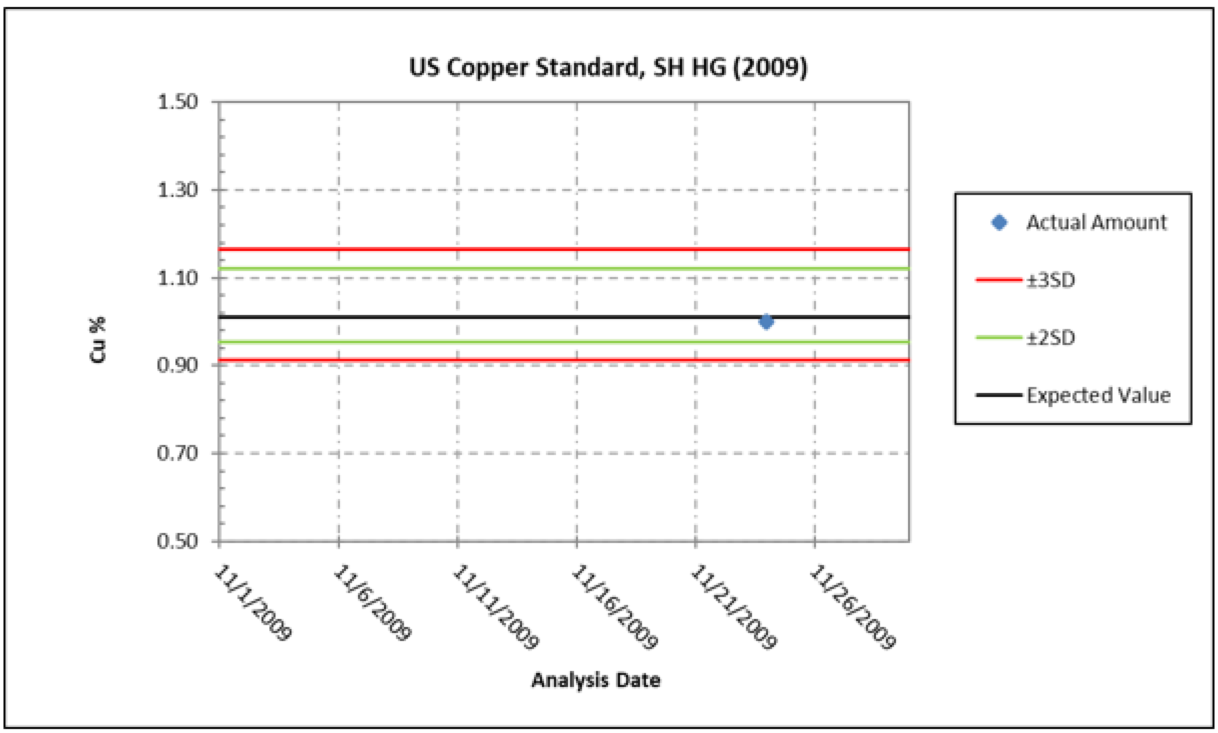




Source: GRE, 2024

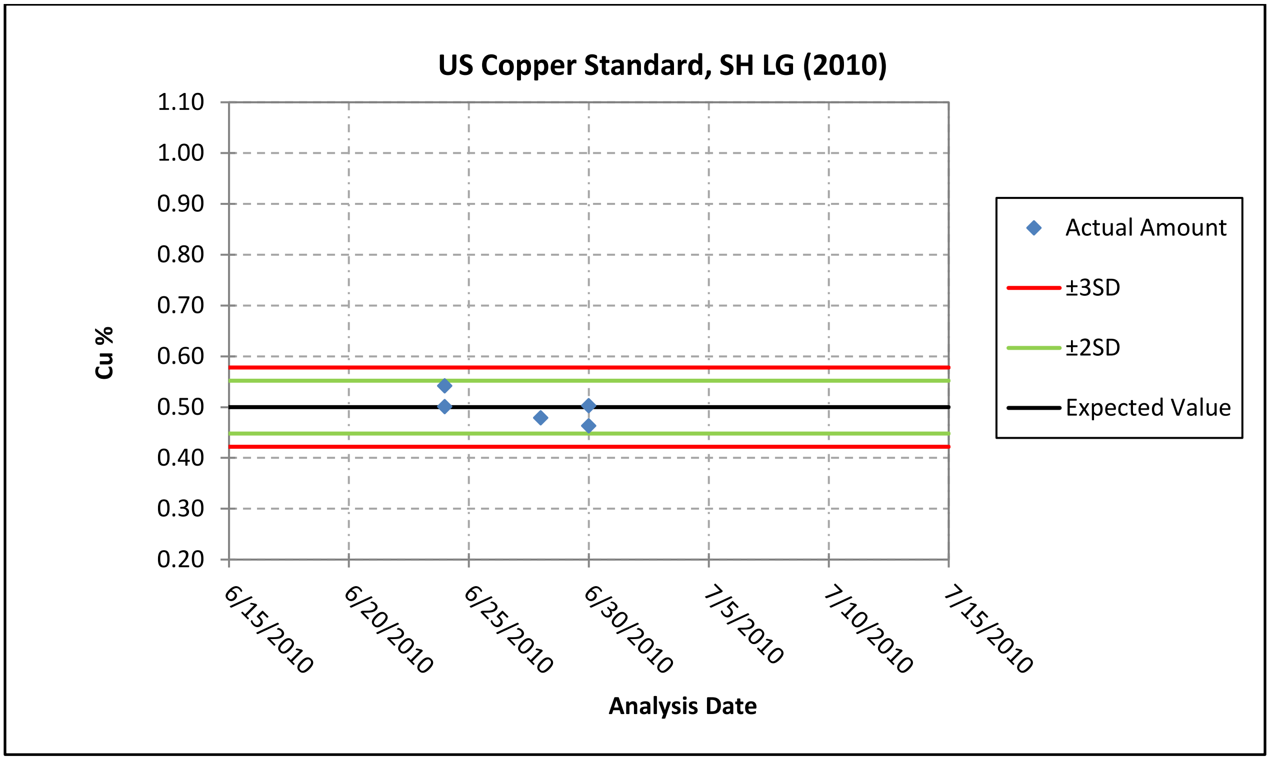
Figure 12-19: SH LG and SH HG from the 2009 Engels Drilling Campaign





Source: GRE, 2024

Figure 12-20: SH LG from the 2010 Engels Drilling Campaign



Source: GRE, 2024

12.2.4 Verification of US Copper Analytical Quality Control Data

12.2.4.1 Verification of the Superior Historical Data (1960s)

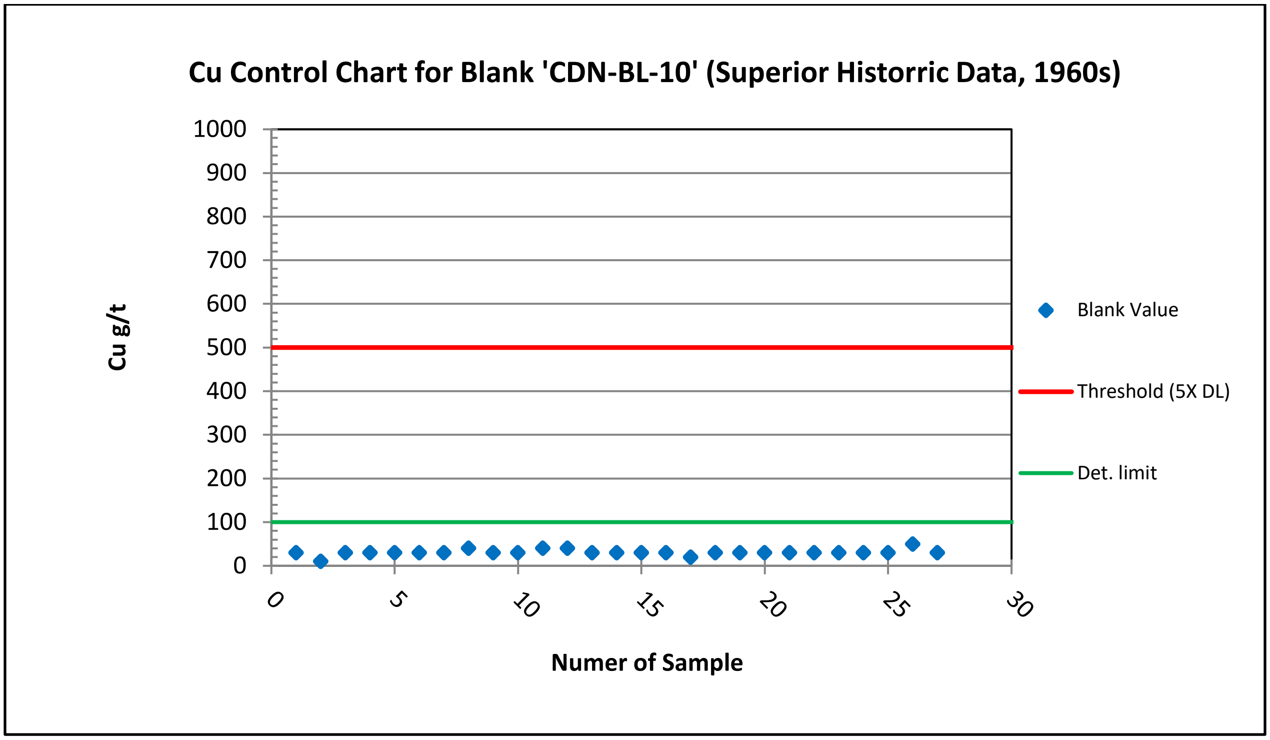
US Copper conducted a verification of some of the Superior historical data (1960s) by re-assays of 451 core samples from the historical drilling in 2021, using the remaining 1/2 split core. In this program a total of 451 core samples were sent to the American Assay Labs for analysis.

This program used blank samples, the CDN-10 blank. A total of 27 blank samples were inserted in the sample stream.

Figure 12-21 shows the assay results of the blanks. The revaluation results show zero samples registered above the detection limit. Thus, there was no contamination during the lab analysis.

Generally, the QA/QC blank sample insertion rates for this program followed general accepted industry standards, which is one blank sample for every 20 interval samples.

Figure 12-21: Blank Results for the 2021 CU Copper Re-assay Superior Historic Data



Source: GRE, 2024

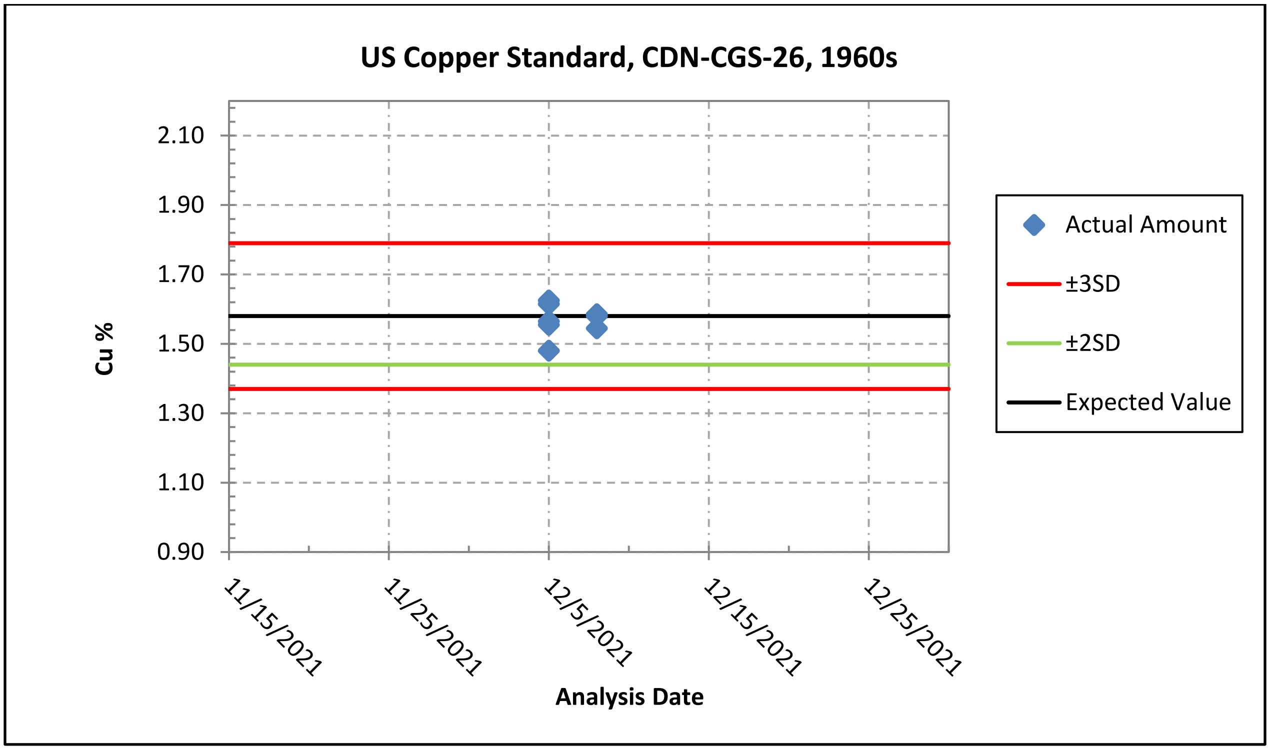
US Copper prepared CRMs CDN-CGS-26, CDN-CGS-41, and CDN-CGS-45 for this program. In total CRMs for copper were inserted into the sample stream at a rate of one standard per 20 sample assays for all 451 core samples, which follows the industry standard.

An analysis of CRMs charts copper showed no obvious errors or bias (see Figure 12-22 through Figure 12-24).

For this program, US Copper considered 24 coarse duplicates for all 451 core samples. Duplicate samples were prepared the same way as all assay samples and were assayed at ALS.

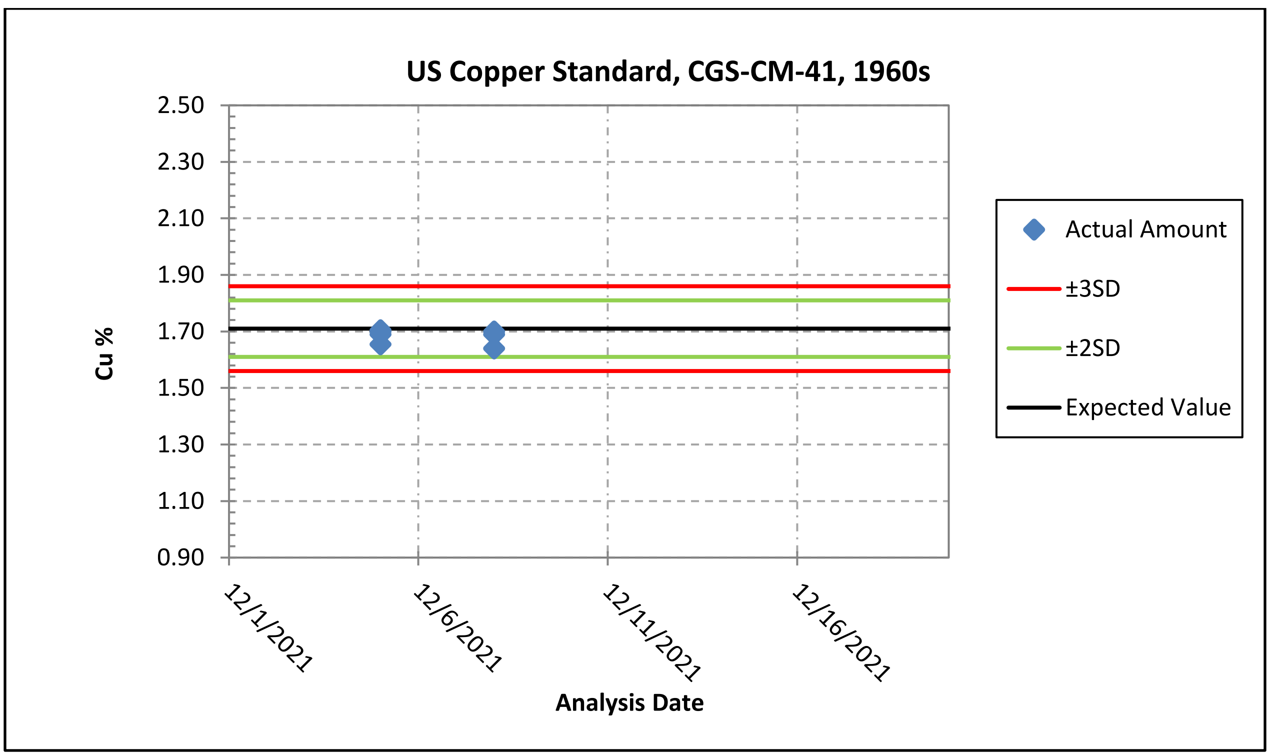
The Q-Q plot for copper effectively indicates that there is no scatter, with R2 values of 0.9952 for coarse duplicates (Figure 12-25).

Figure 12-22: CRMs for the 2021 CU Copper Re-assay Superior Historic Data



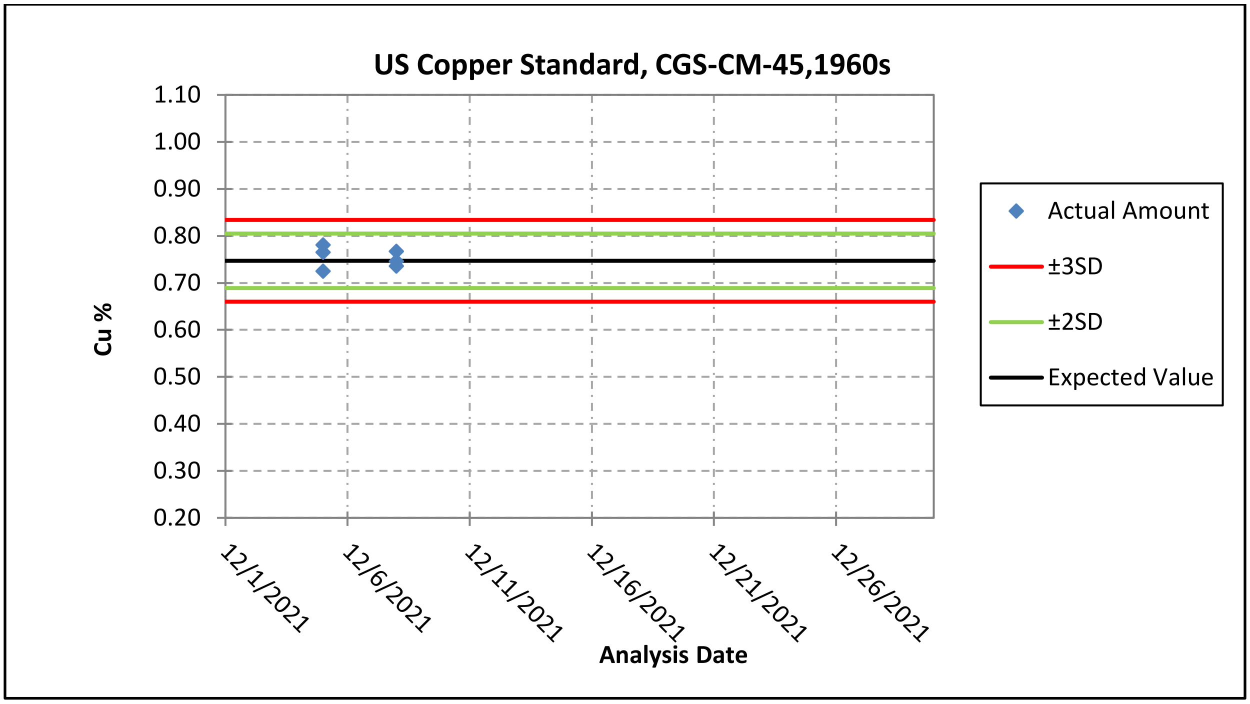
Source: GRE, 2024

Figure 12-23: CRMs for the 2021 CU Copper Re-assay Superior Historic Data



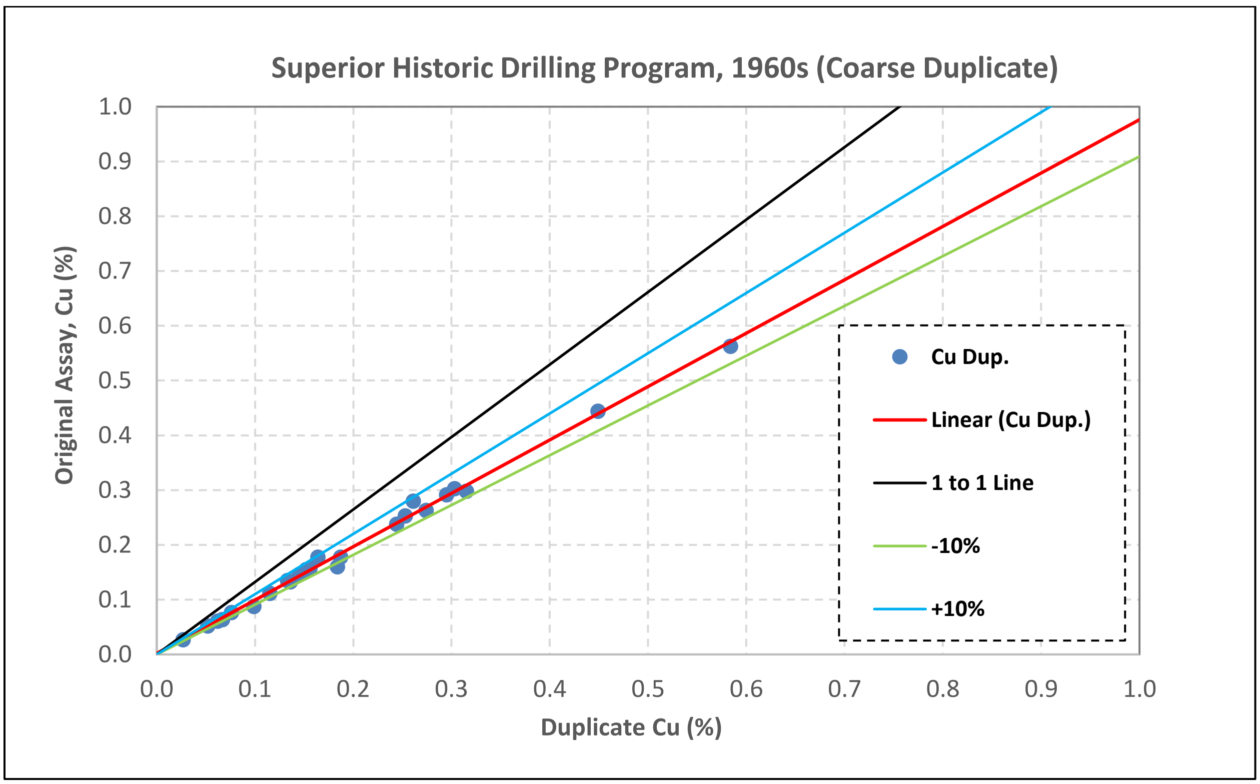
Source: GRE, 2024

Figure 12-24: CRMs for the 2021 CU Copper Re-assay Superior Historic Data



Source: GRE, 2024

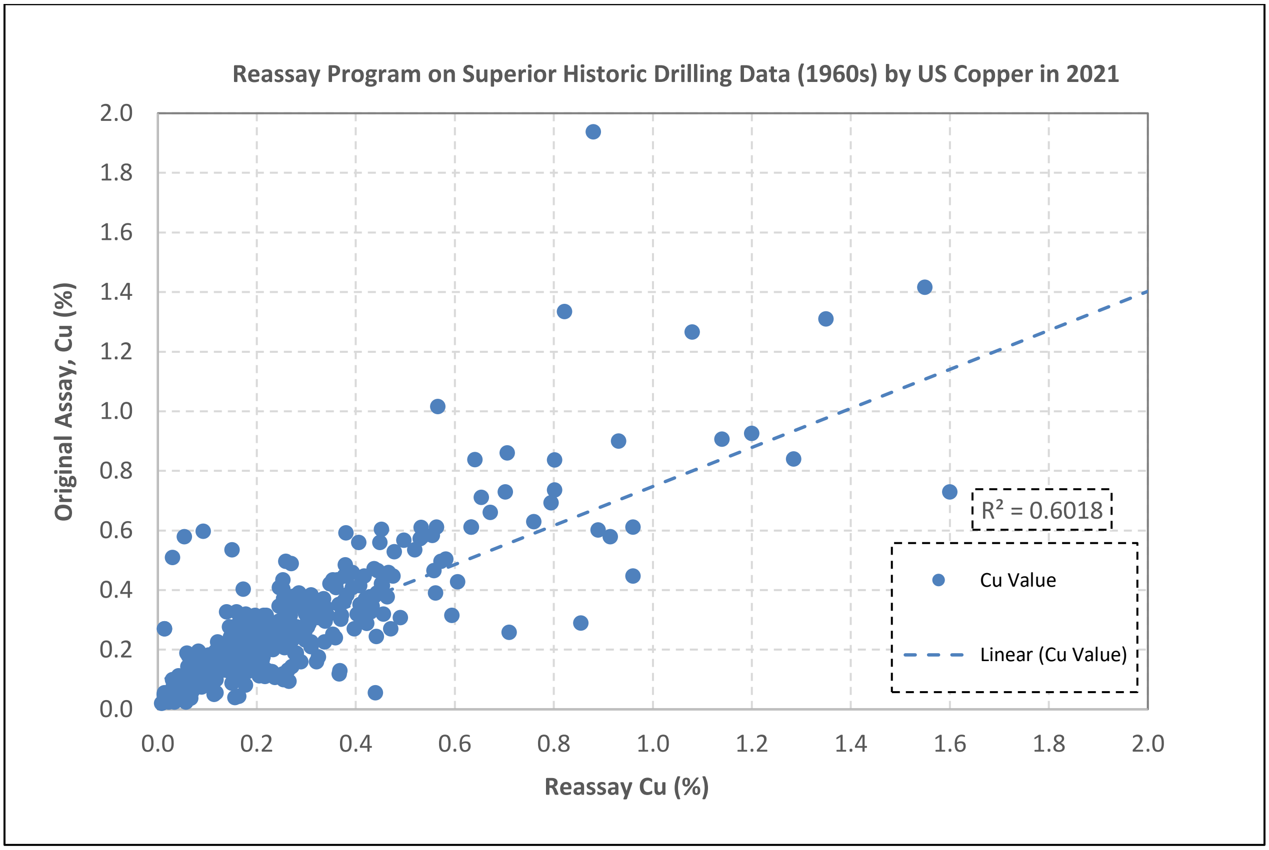
Figure 12-25: Coarse Duplicate Samples from for the 2021 CU Copper Re-assay Superior Historic Data



Source: GRE, 2024

In 2021, US Copper compiled 448 core samples out of 22 drill holes from the Superior historical data (1960s) for a re-assay program. Figure 12-26 is a scatterplot of the original sample versus the re-assay values for copper. The result from this program is the same as the two previous re-assay programs, conducted by Sheffield and Starfield in 2006 and 2009, respectively. This recent program confirmed that high-grade mineralization zones within the Superior deposit are mainly associated with the veinlets, with no homogenous distribution of mineralization.

Figure 12-26: Scatterplot of Original Cu Assay and Re-assay, Starfield in 2009



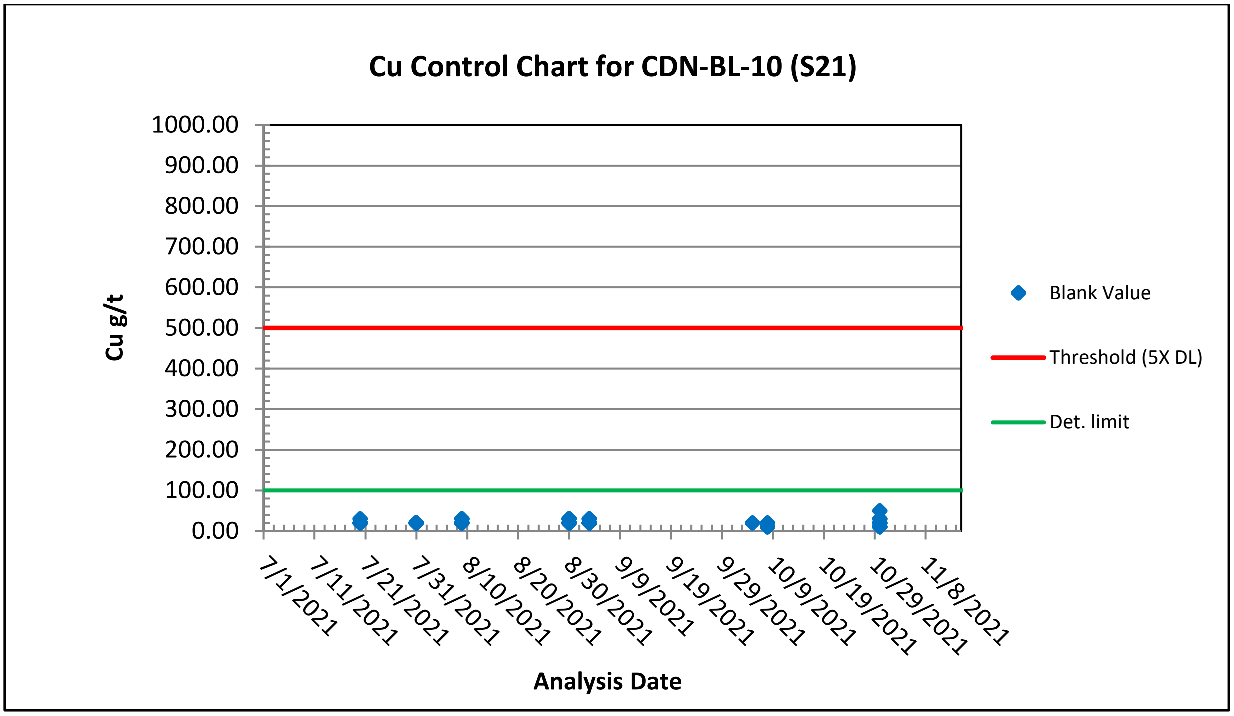
Source: GRE, 2024

12.2.4.2 Verification of the 2021 Drilling Program

In the 2021 drilling program, US Copper drilled seven core holes totaling 5,872 ft at Superior. For this program, 1,159 samples were collected and sent to ALS for analysis.

A total of 66 blank samples, which were purchased from CDN (CDN-BL-10), were inserted in the sample stream at a rate of six blank samples per 100 sample assays. The revaluation results show zero sample registered above the detection limit (Figure 12-27).

Figure 12-27: Blank Results for the 2021 Drilling Program

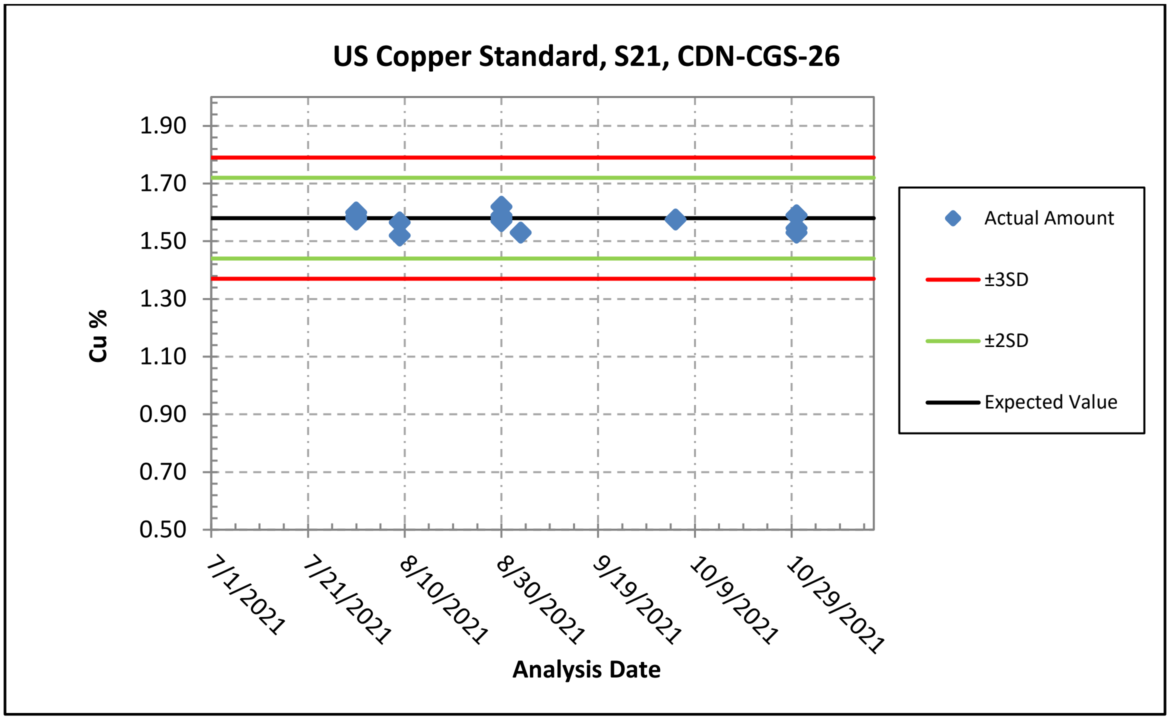


Source: GRE, 2024

US Copper used CRMs CDN-CGS-26, CDN-CGS-41, and CDN-CGS-45 for the 2021 drilling program. In total, CRMs for copper were inserted into the sample stream at a rate of six standards per 100 sample assays for all 1,159 core samples, which follows the industry standard.

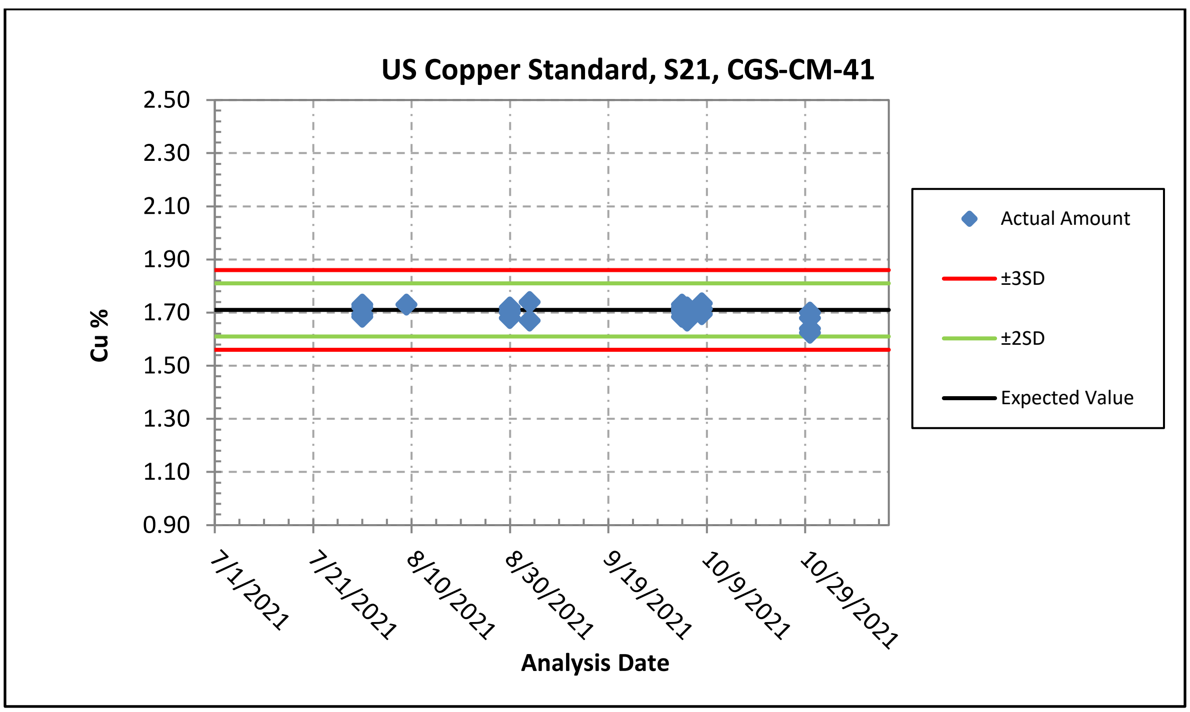
Analysis of CRMs charts for the high copper grades showed no obvious errors or bias (see Figure 12-28 through Figure 12-30).

Figure 12-28: CRM CDN-CGD-26 for the 2021 Drilling Program



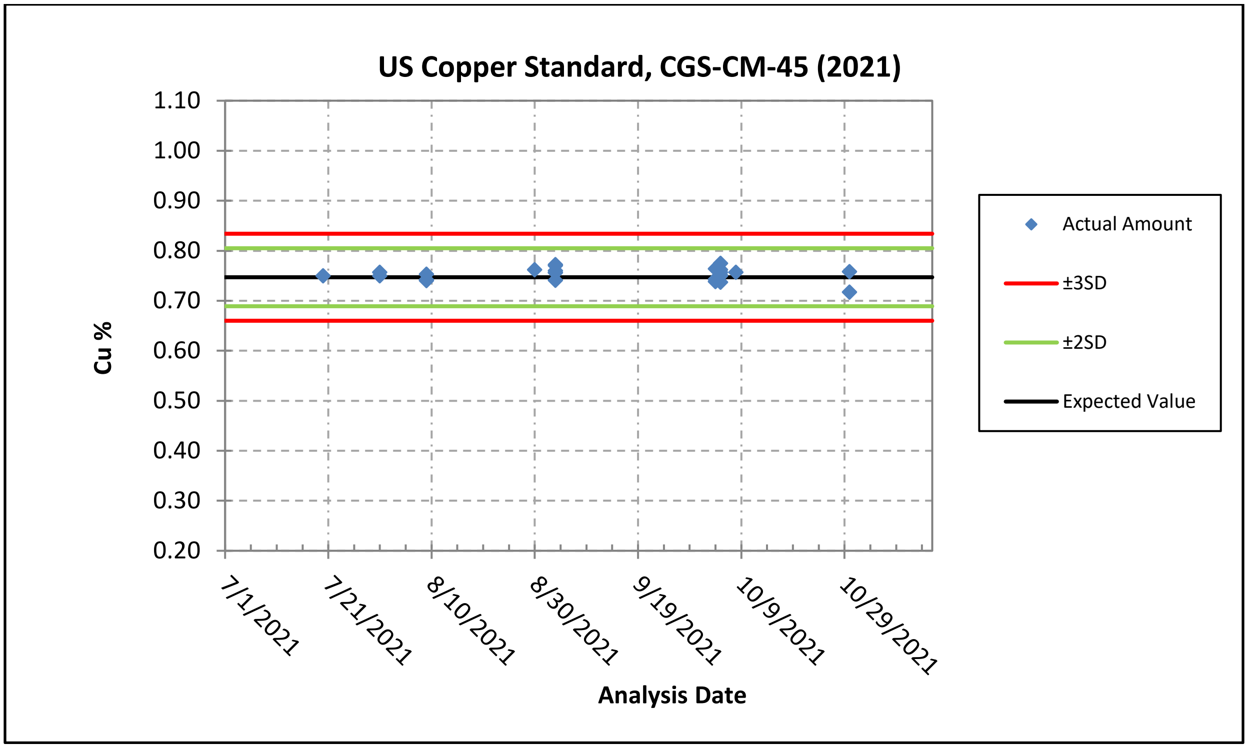
Source: GRE, 2024

Figure 12-29: CRM CDN-CGD-41 for the 2021 Drilling Program



Source: GRE, 2024

Figure 12-30: CRM CDN-CGD-45 for the 2021 Drilling Program

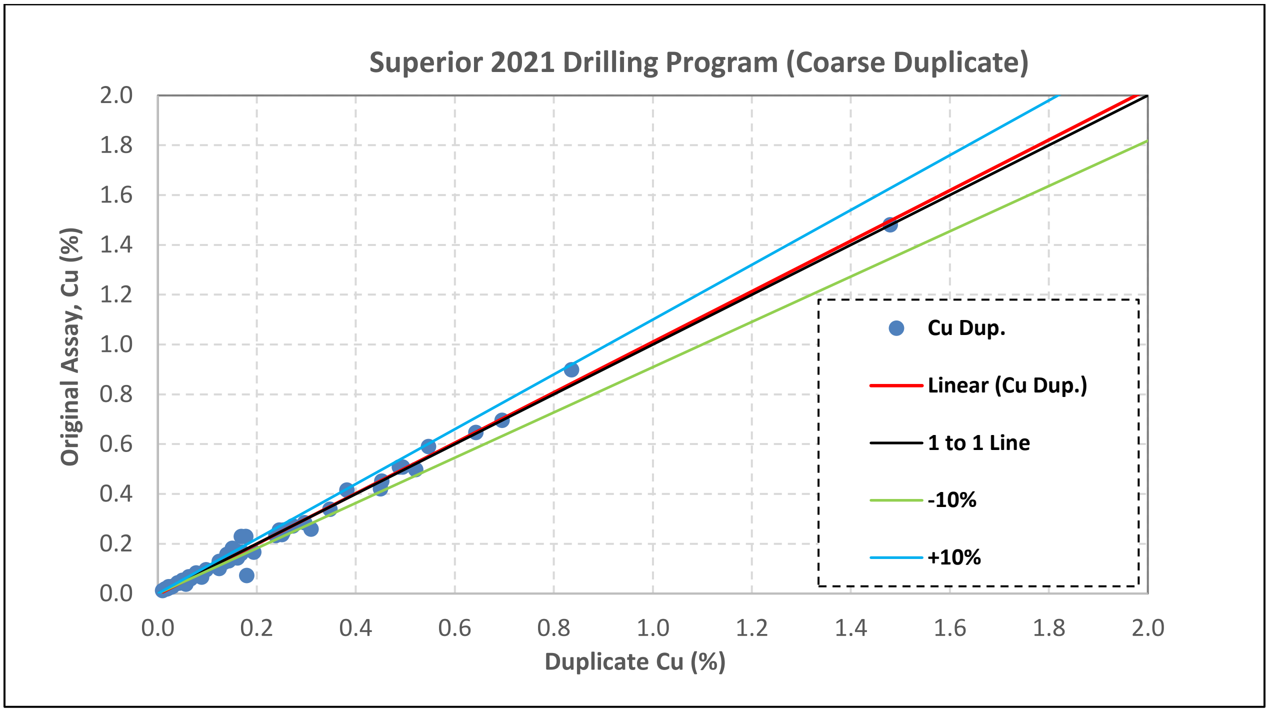


Source: GRE, 2024

US Copper considered 66 coarse duplicates for this program for all 1,159 core samples. Duplicate samples were prepared the same way as all assay samples and were assayed at ALS.

The Q-Q plot for copper effectively indicates that there is no scatter, with R2 values of 0.99 for coarse duplicates (Figure 12-31).

Figure 12-31: Coarse Duplicate Samples for the 2021 Drilling Program



Source: GRE, 2024

12.2.4.3 Verification of the 2023 Drilling Program

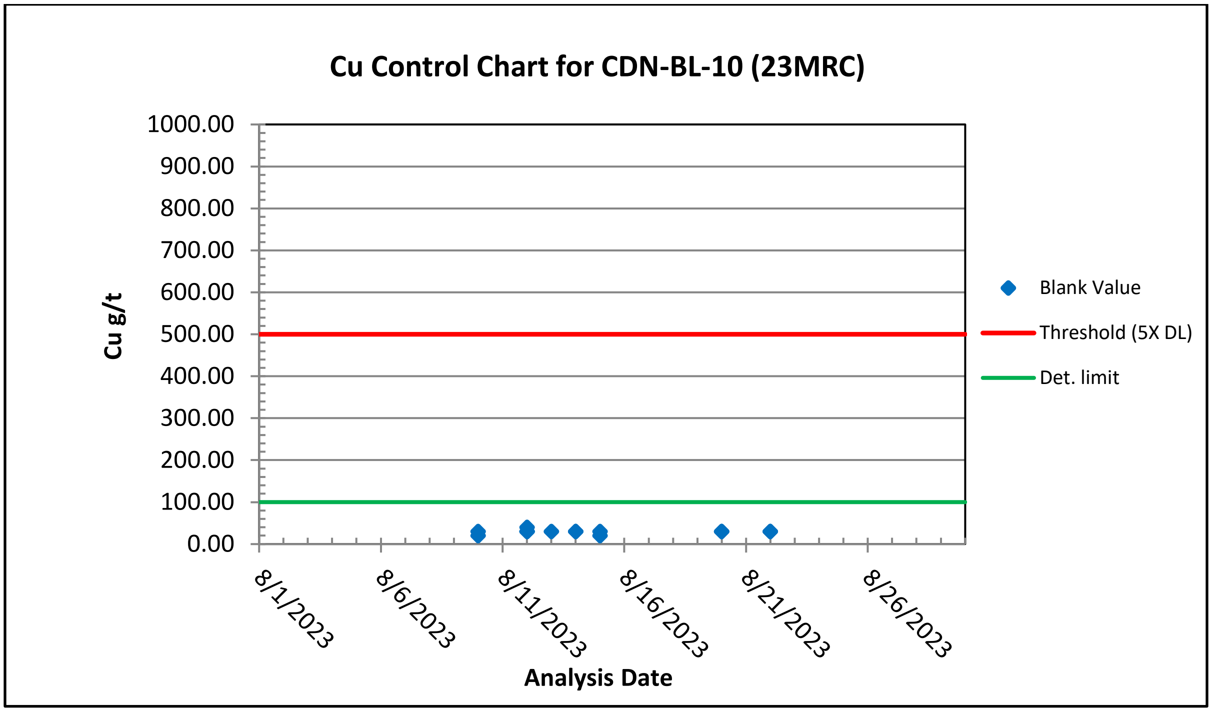
In the 2023 drilling program, US Copper drilled seven core holes totaling 5,872 ft at Superior. For this program, 1,159 samples were collected and sent to ALS for analysis.

For programs such as the 2021 program, only one type of blank sample (CDN-BL-10) was purchased from CDN and inserted into the sample streams.

For the 2023 Moonlight drilling program, a total of 15 blank samples were inserted in the sample stream at a rate of one blank sample per 100 samples assays for all 486 RC samples. For the 2023 Lamb's Ridge drilling program, only two blank samples were inserted in the sample stream at a rate of one blank sample per 100 samples assays for all 163 RC samples. For the 2023 Engels drilling program, a total of 13 blank samples were inserted in the sample stream at a rate of seven blank samples per 100 samples assays for all 199 RC samples.

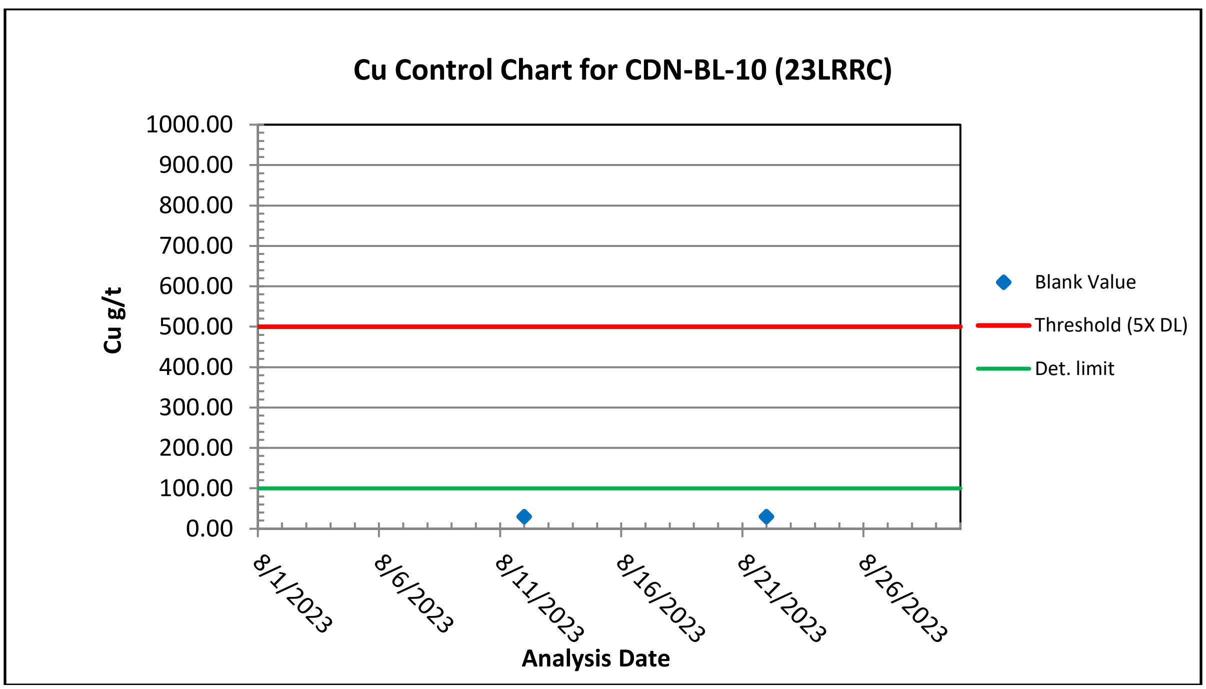
The revaluation results show zero sample registered above the detection limit (Figure 12-32 through Figure 12-34). Generally, except for the Engels program, the QA/QC blank sample insertion rates fall below the generally accepted industry standard, which is one blank sample for every 20 interval samples.

Figure 12-32: Blank Results for the 2023 Drilling Program, Moonlight



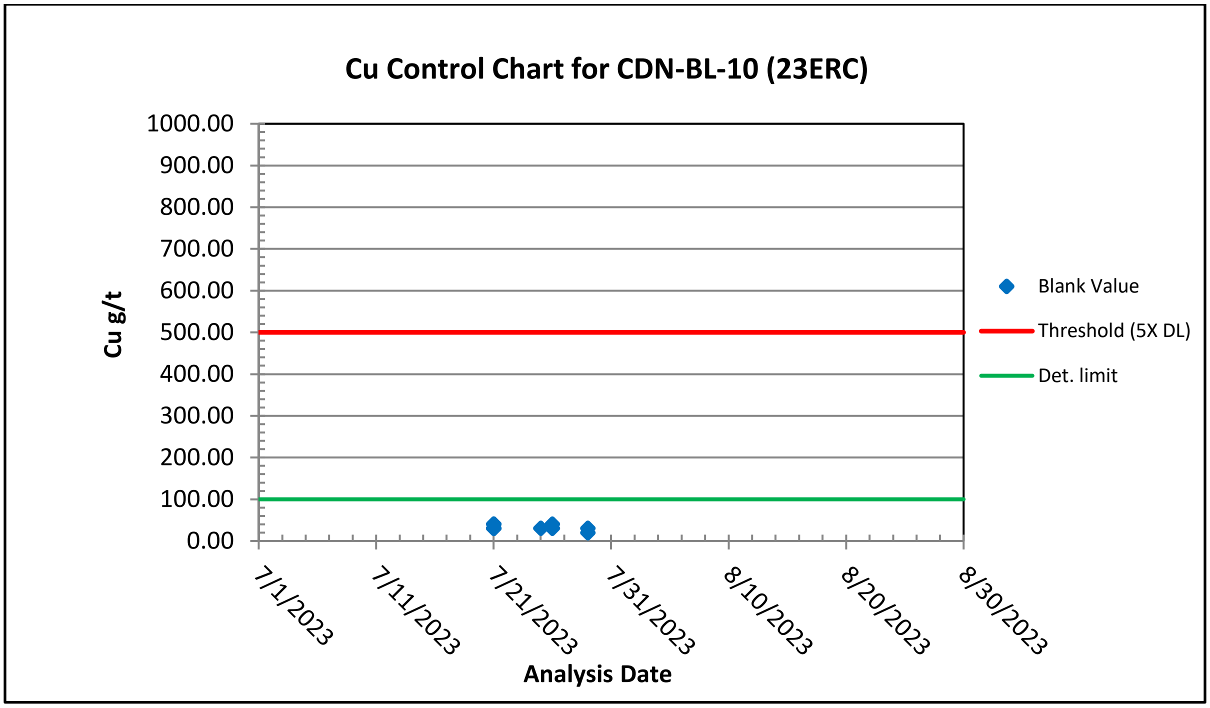
Source: GRE, 2024

Figure 12-33: Blank Results for the 2023 Drilling Program, Lambs Ridge



Source: GRE, 2024

Figure 12-34: Blank Results for the 2023 Drilling Program, Engels

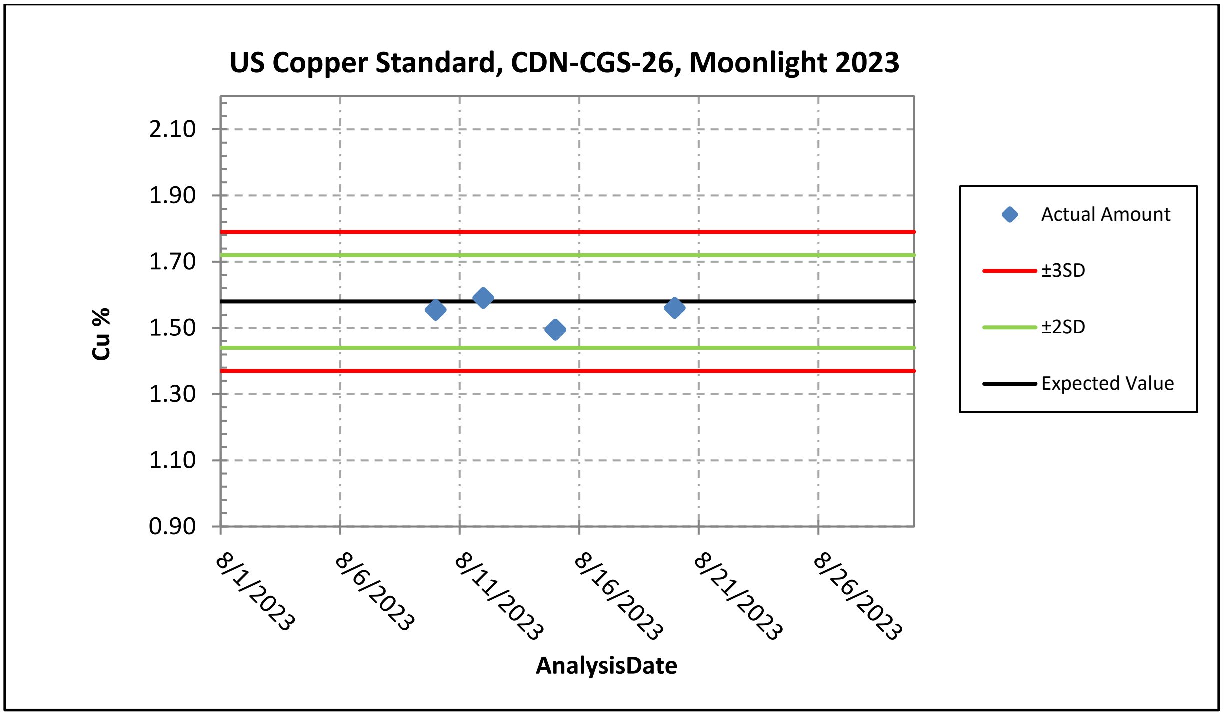


Source: GRE, 2024

US Copper used CRMs CDN-CGS-26, CDN-CGS-41, and CDN-CGS-45 for the 2023 drilling program. In total, CRMs for copper were inserted into the sample stream at a rate of one standard per 100 sample assays for all 486 core samples for Moonlight, at a rate of four standards per 100 sample assays for all 163 core samples for Lamb Ridge, and at a rate of nine standards per 100 sample assays for all 199 core samples for Engels, which, except for Engels, did not follow the industry standard.

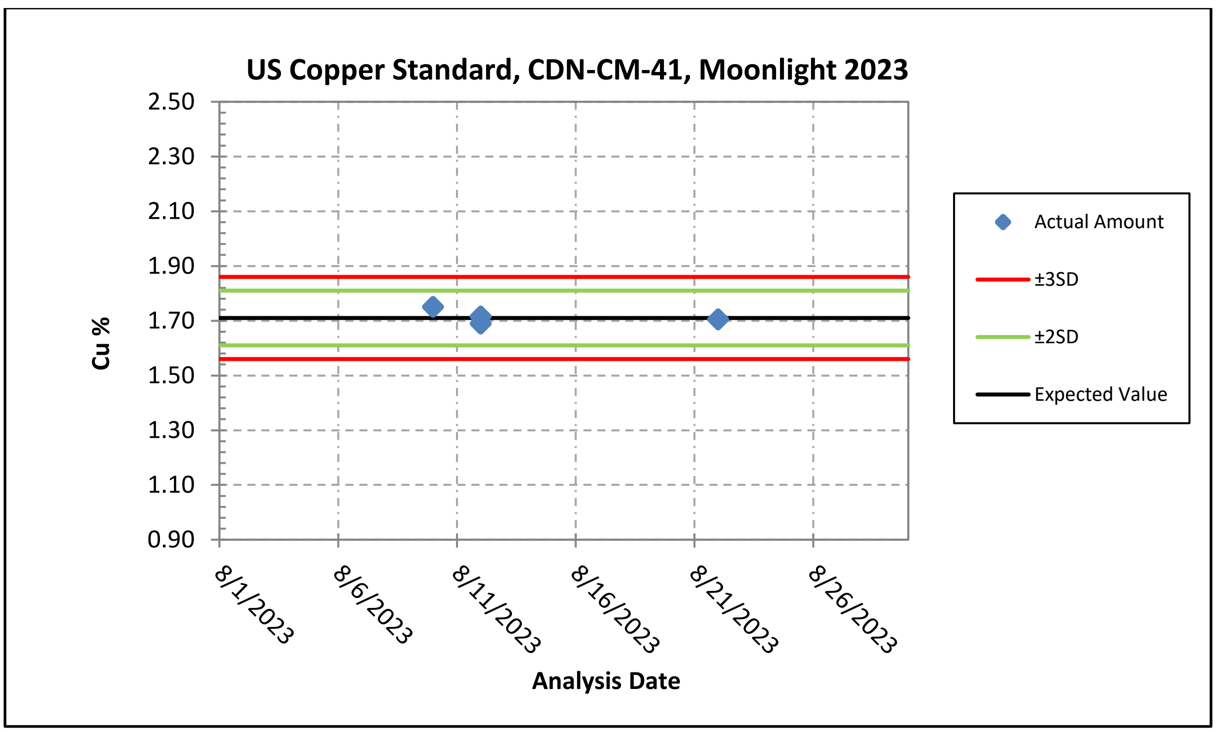
Analysis of CRMs charts for the high copper grades showed no obvious errors or bias (see Figure 12-35Figure 12-28 through Figure 12-43).

Figure 12-35: CRM CDN-CGD-26 for the 2023 Drilling Program, Moonlight



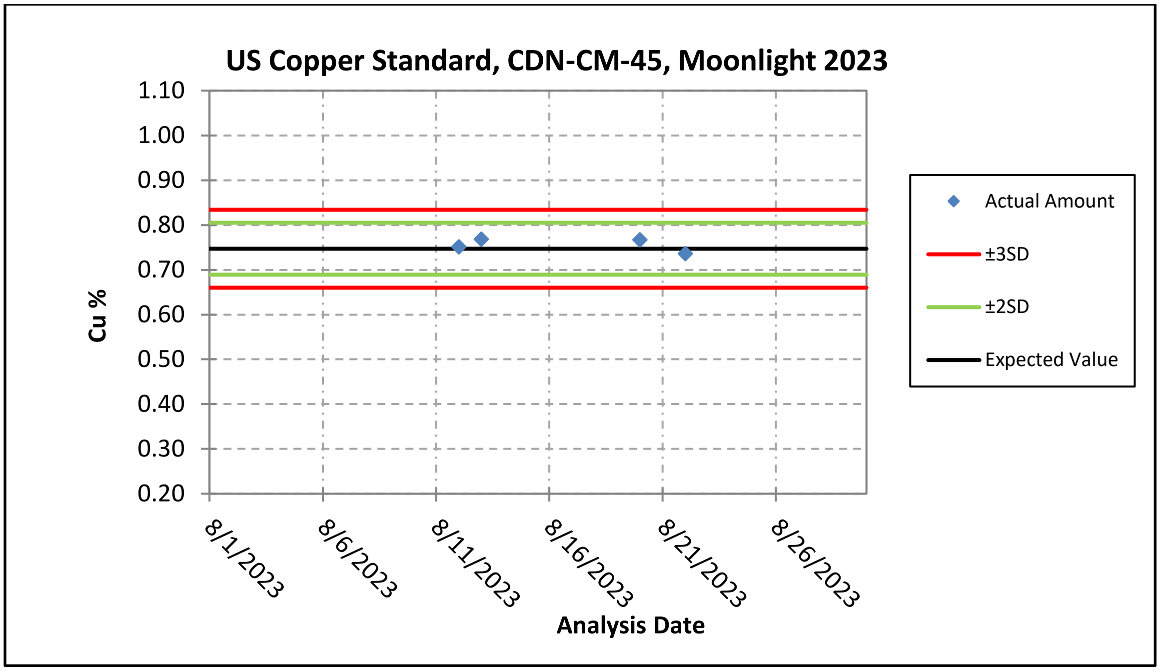
Source: GRE, 2024

Figure 12-36: CRM CDN-CGD-41 for the 2023 Drilling Program, Moonlight



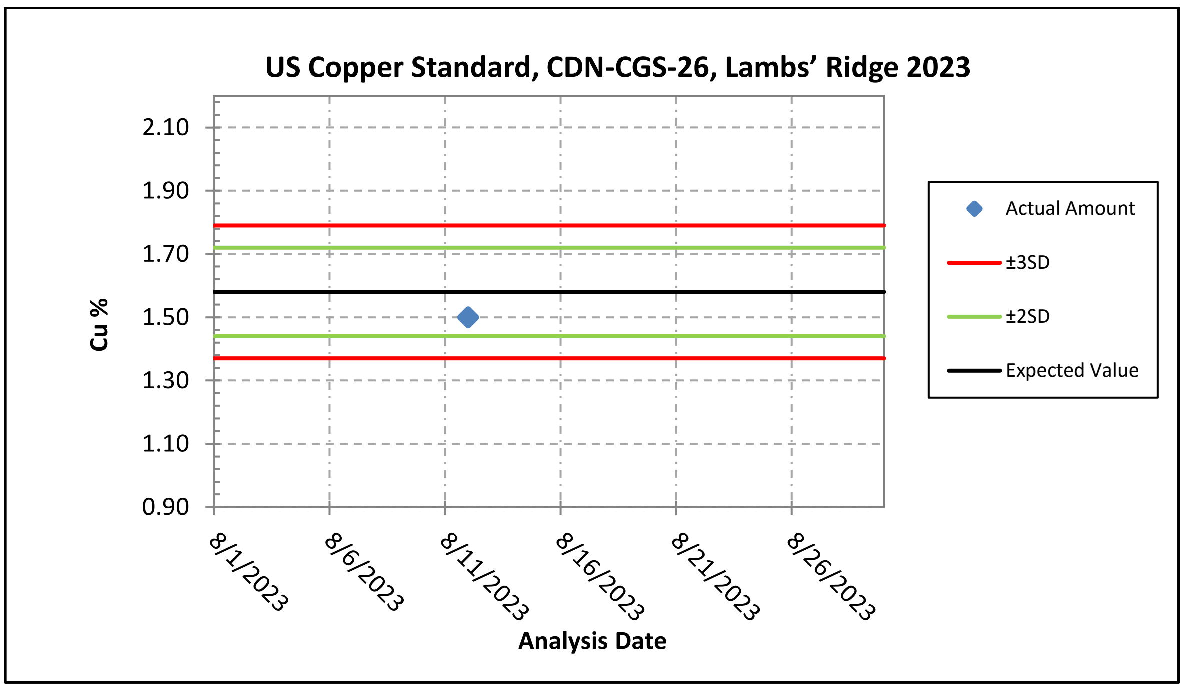
Source: GRE, 2024

Figure 12-37: CRM CDN-CGD-45 for the 2023 Drilling Program, Moonlight



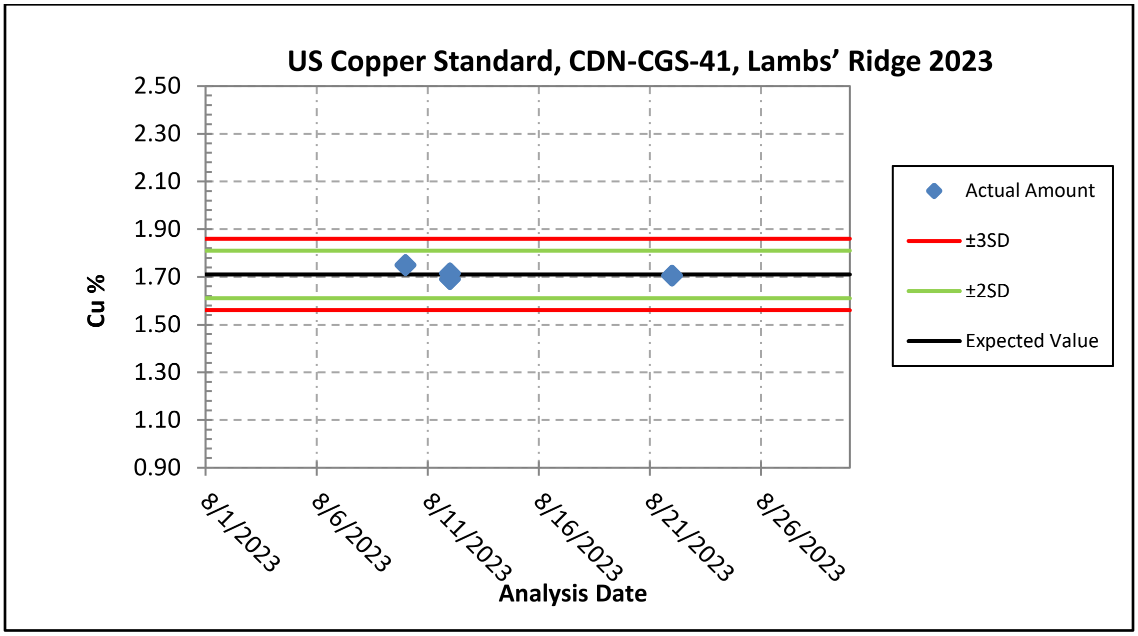
Source: GRE, 2024

Figure 12-38: CRM CDN-CGD-26 for the 2023 Drilling Program, Lambs Ridge



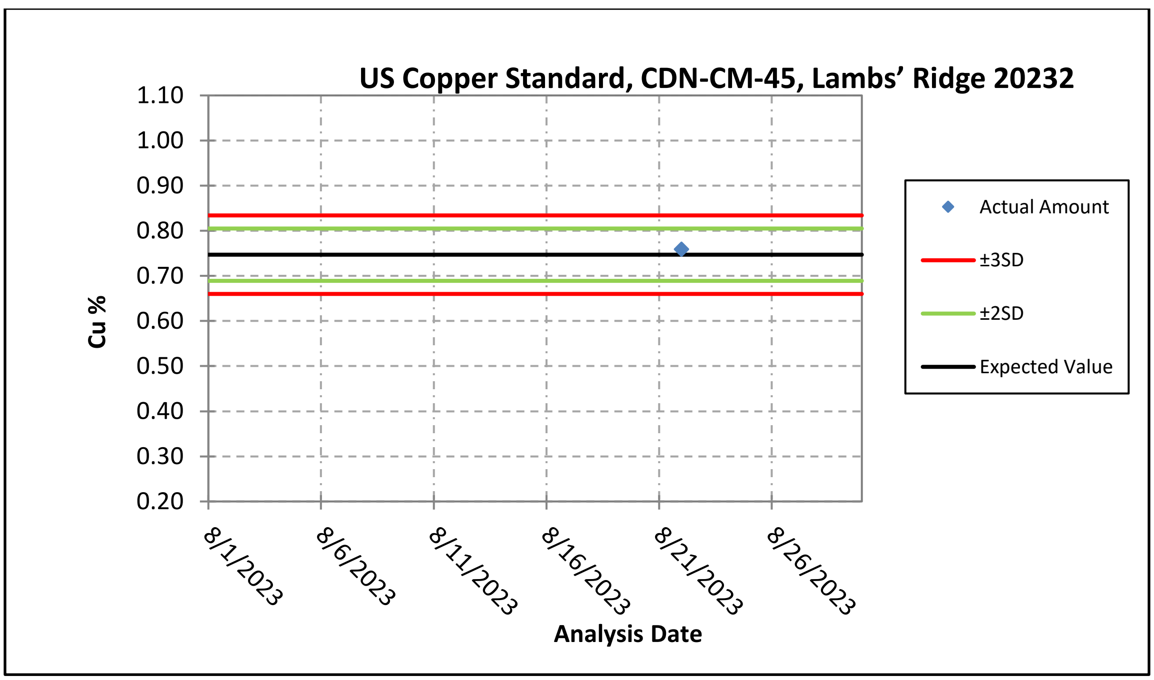
Source: GRE, 2024

Figure 12-39: CRM CDN-CGD-41 for the 2023 Drilling Program, Lambs Ridge



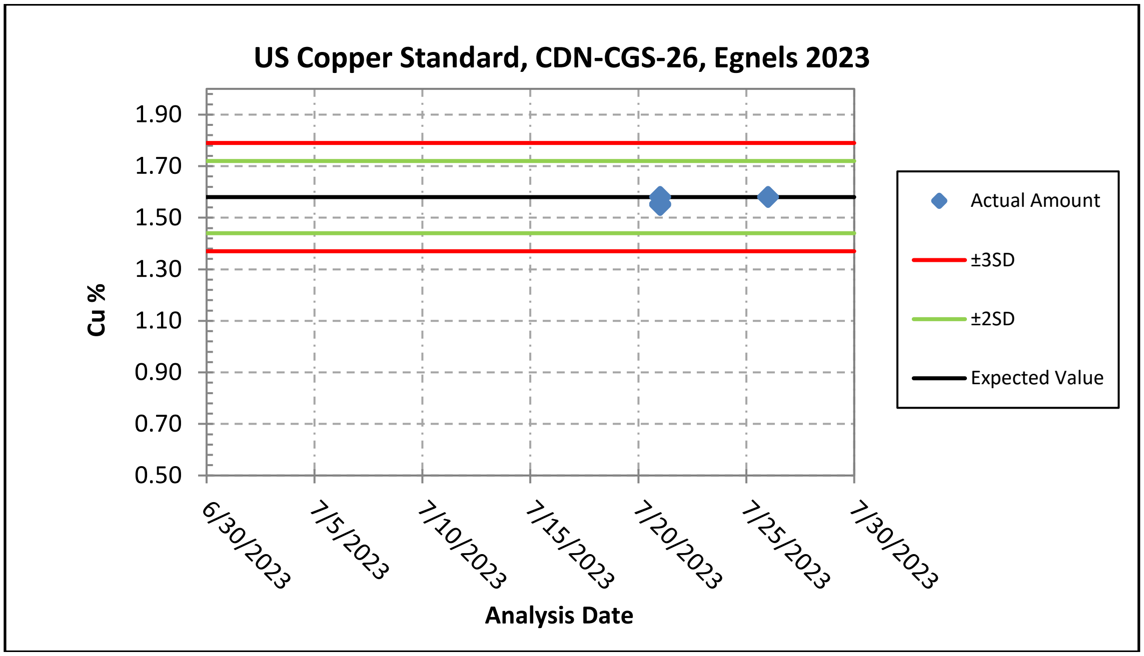
Source: GRE, 2024

Figure 12-40: CRM CDN-CGD-45 for the 2023 Drilling Program, Lambs Ridge



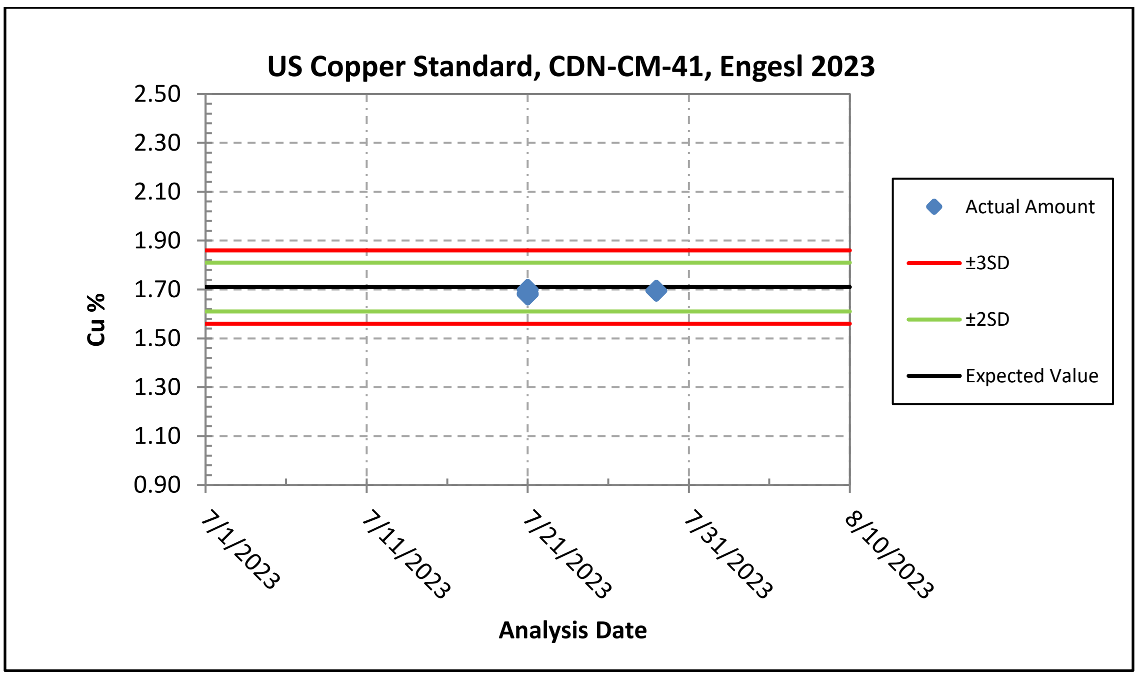
Source: GRE, 2024

Figure 12-41: CRM CDN-CGD-26 for the 2023 Drilling Program, Engels



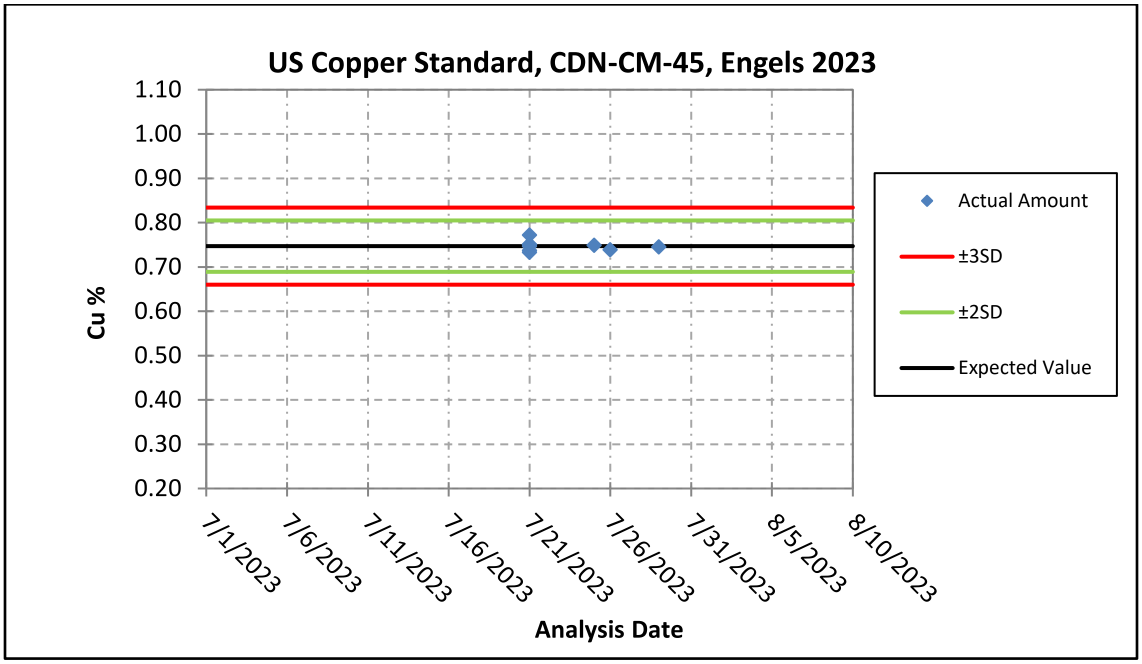
Source: GRE, 2024

Figure 12-42: CRM CDN-CGD-41 for the 2023 Drilling Program, Engels



Source: GRE, 2024

Figure 12-43: CRM CDN-CGD-45 for the 2023 Drilling Program, Engels

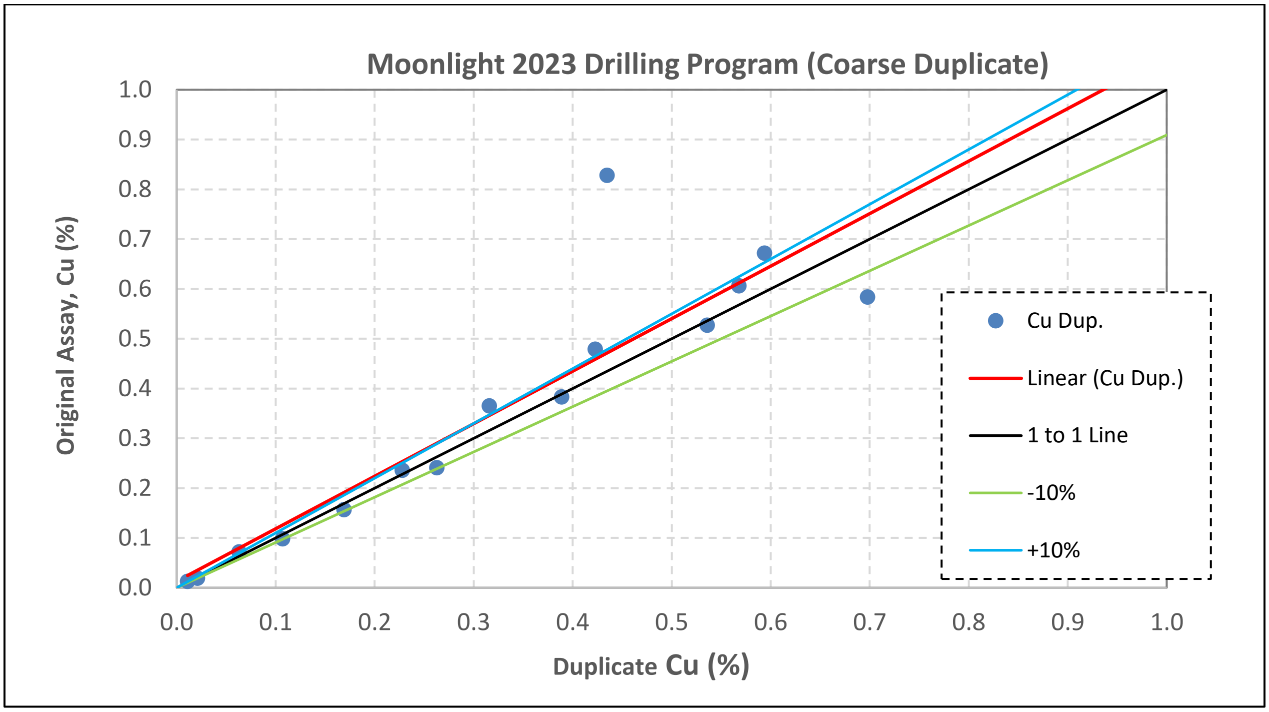


Source: GRE, 2024

For the 2023 drilling program, US Copper considered 15 coarse duplicates for all 486 RC samples for Moonlight and only two coarse duplicates for all 163 RC samples for Lamb Ridge. No duplicate was considered for Engles in the 2023 drilling program. Duplicate samples were prepared the same way as all assay samples and were assayed at ALS.

The Q-Q plot for copper effectively indicates that there is no scatter, with R2 values of 0.82 for Moonlight coarse duplicates (Figure 12-44). Having just two duplicate samples for Lamb Ridge cannot give the correct data for consideration.

Figure 12-44: Coarse Duplicate Samples for the 2021 Drilling Program



Source: GRE, 2024

12.3 Field Visit by GRE, Geological Data Verification and Check Assays

GRE's QP, Dr. Hamid Samari, conducted an on-site inspection at the project site from 7 to 8 August 2024, accompanied by US Copper's geologist, Mr. Justin Claiborne. The GRE's QP, Dr. Hamid Samari, conducted this field visit mainly to check exploration programs and to conduct field checks, including the validation and accuracy of collar coordinates, geological maps, and geological logging, and to take a few core and pulp samples for assay checking.

GRE used a handheld GPS, model Garmin 64 tons, to check the coordinates at each drill location being validated. Geographic coordinates for 22 of the existing drill hole collar locations (2006 to 2023) were recorded in the field using a hand-held GPS unit. The average variance between field collar coordinates and collar coordinates contained in the project database is roughly 3.3 meters, within the expected error margin. The average variance between field collar elevation and those contained in the project database is 10 feet (3.1 meters), which is within the expected margin of error. (Table 12-1, Photo 12-1, and Photo 12-2).

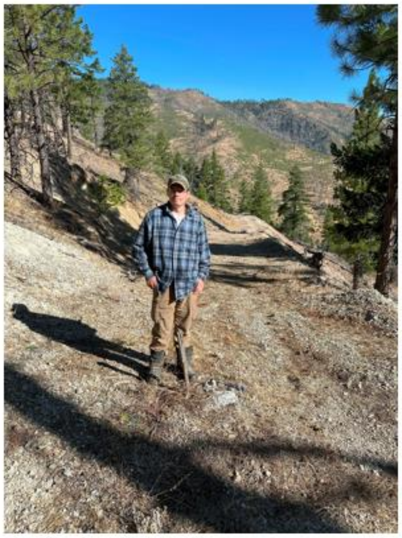
For hole 06-MN-12, the difference elevation between the elevation in the database and the elevation measured by GRE'QP was 39.3 feet (12 meters); for that reason, the elevation of this collar was adjusted using a topography map.

Table 12-1: Collar Coordinates Inspections

Hole ID	Deposit Area	From US Copper Database (UTM Zone 10 NAD83)			From Hand-held GPS by QP (UTM Zone 10 NAD83)			Distance Difference (m)	Elevation Difference (m)
		Easting	Northing	Elevation (ft)	Easting	Northing	Elevation (ft)		
S21-3	Superior	689875.40	4452498.44	1424.49	689873.00	4452499.00	1425.0	2.5	0.5
S21-6	Superior	689722.58	4452544.52	1413.92	689722.00	4452544.00	1415.0	0.8	1.1
S21-1	Superior	689889.91	4452598.91	1384.14	689787.00	4452599.00	1388.0	2.9	3.9
S21-7	Superior	689778.27	4452453.00	1465.94	689778.00	4452453.00	1465.0	0.3	0.9
23LRRC01	Lamb’s Ridge	690826.80	4454479.70	1637.72	690825.00	4454484.00	1632.0	4.7	5.7
23LRRC02	Lamb’s Ridge	690805.00	4454113.30	1524.24	690806.00	4454114.00	1531.0	1.2	6.8
E-07	Engels	692014.21	4455193.22	1648.15	692019.00	4455191.00	1651.0	54.3	2.8
23ERC11	Engels	692058.10	4455271.80	1648.88	692060.00	4455267.00	1653.0	5.2	4.1
23ERC06	Engels	692055.20	4455268.90	1648.88	692055.00	4455272.00	1654.0	3.1	5.1
ME10-05	Engels	692191.69	4455260.88	1668.03	692191.00	4455256.00	1670.0	4.9	2.0
23ERC13	Engels	692235.50	4455266.70	1676.40	692237.00	4455267.00	1675.0	1.5	1.4
23ERC10	Engels	692233.10	4455264.70	1675.36	692238.00	4455264.00	1675.0	4.9	0.4
23ERC14	Engels	692201.60	4455252.00	1674.57	692201.00	4455249.00	1670.0	3.1	4.6
07-E-08	Engels	692253.72	4455214.17	1680.88	692255.00	4455218.00	1688.0	4.0	7.1
23ERC01	Engels	692049.10	4455094.60	1611.78	692048.00	4455097.00	1619.0	2.6	7.2
23MRC16	Moonlight	687054.80	4455567.30	1682.65	687059.00	4455567.00	1682.5	4.2	0.2
23MRC27	Moonlight	687477.70	4455425.90	1731.42	687475.00	4455423.00	1731.9	4.0	0.5
06-MN-12	Moonlight	687325.96	4455416.66	1743.05	687331.00	4455417.00	1742.8	5.1	0.2
08-MN-20	Moonlight	687360.96	4455370.66	1750.47	687360.00	4455374.00	1751.1	3.3	0.6
23MRC26	Moonlight	687226.50	4455098.50	1736.32	687230.00	4455098.00	1744.0	3.5	7.7
23MRC19	Moonlight	687100.10	4455314.60	1691.46	687098.00	4455312.00	1695.0	3.3	3.5
23MRC20	Moonlight	687042.30	4455127.90	1697.80	687045.00	4455127.00	1700.0	2.8	2.2
Maximum Difference (m)								5.3	7.7
Minimum Difference (m)								0.3	0.2
Average Difference (m)								3.3	3.1

Source: GRE, 2024

Photo 12-1: Inspection of the Collar Coordinates



S21-3, E 689875
N 4452498, H 1424

S21-6, E 689722
N 4452544, H 1413

23 LRRC02, E 690804
N 4454113, H 1524

23 ERC01, E 692049
N 4455094, H 1611

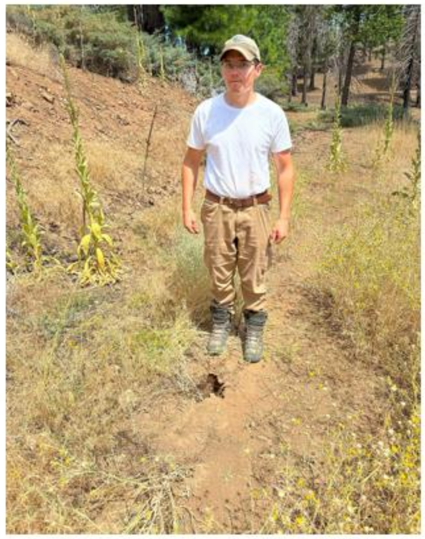
23ERC11, E 692058
N 4455271, H 1648

08-MN-20, E 687360
N 4455370, H 1751

Source: GRE, 2024

Photo 12-2: Inspection of the Collar Coordinates

ME10-05, E 692191
N 4455260 H 1668



08-E-40, E 692004
N 4455178, H 1651

06 MN 12, E 687325
N 4455416, H 1743

23MRC19, E 687100
N 4455314, H 1691

23MRC 20, E 687042
N 4455127, H 1697

23ERC13, E 692235
N 4455266, H 1676

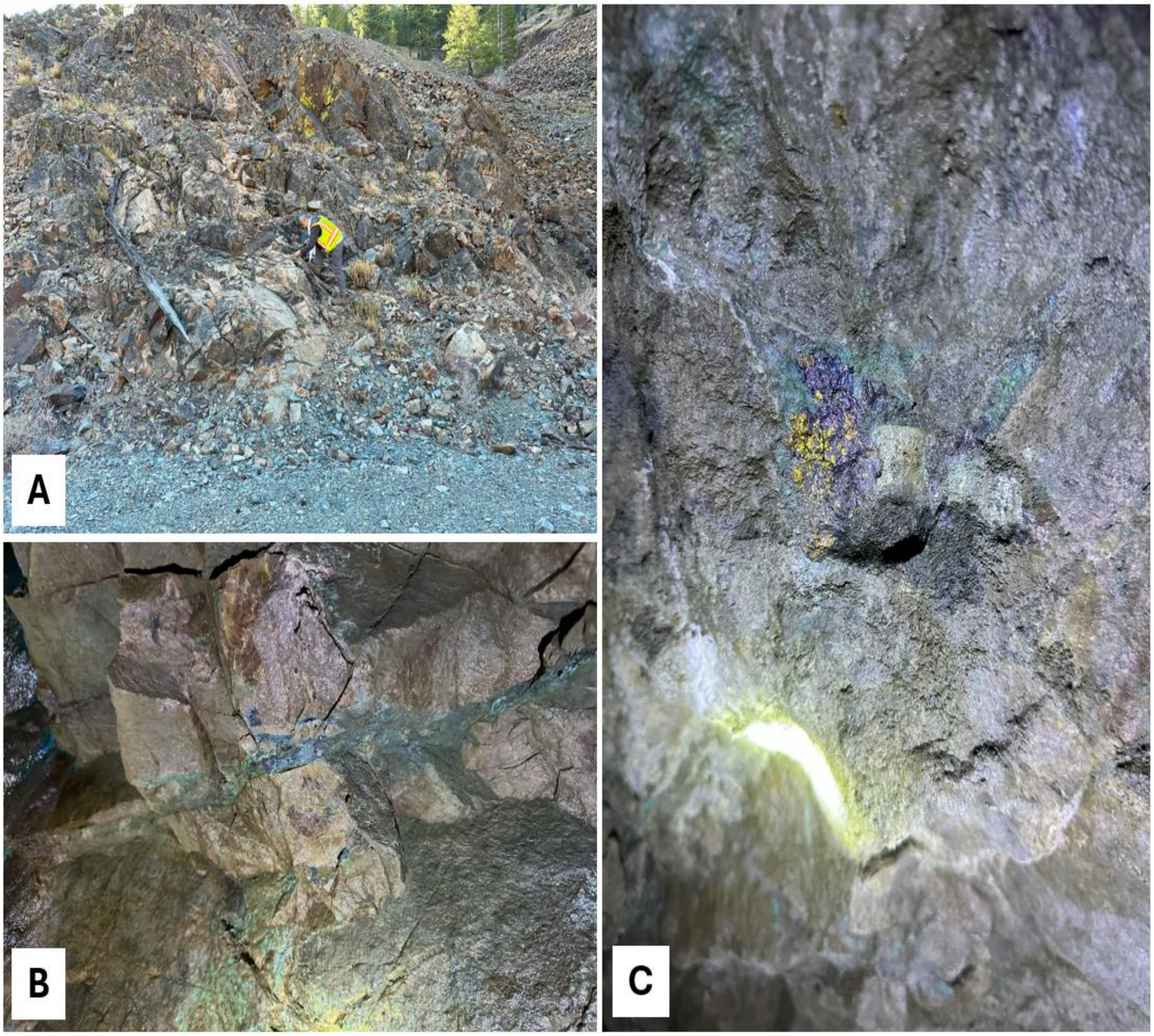
Source: GRE, 2024

12.3.1 Geological Data Verification and Interpretation

From 7 to 8 August 2024, the GRE'QP, Dr. Hamid Samari, spent one day at the project site checking some of the drill collars and the geological map prepared for the project area.

During the site visit, field visit observations generally confirmed the geologic map of the project area. The lithology of exposed bedrock, alteration types, and significant structural features is consistent with descriptions provided in previous project reports (US Copper, 2013 and 2018). Dr. Samari did not see any evidence in the field that might significantly alter or refute the current interpretation of the local geologic setting (Photo 12-3 through Photo 12-6).

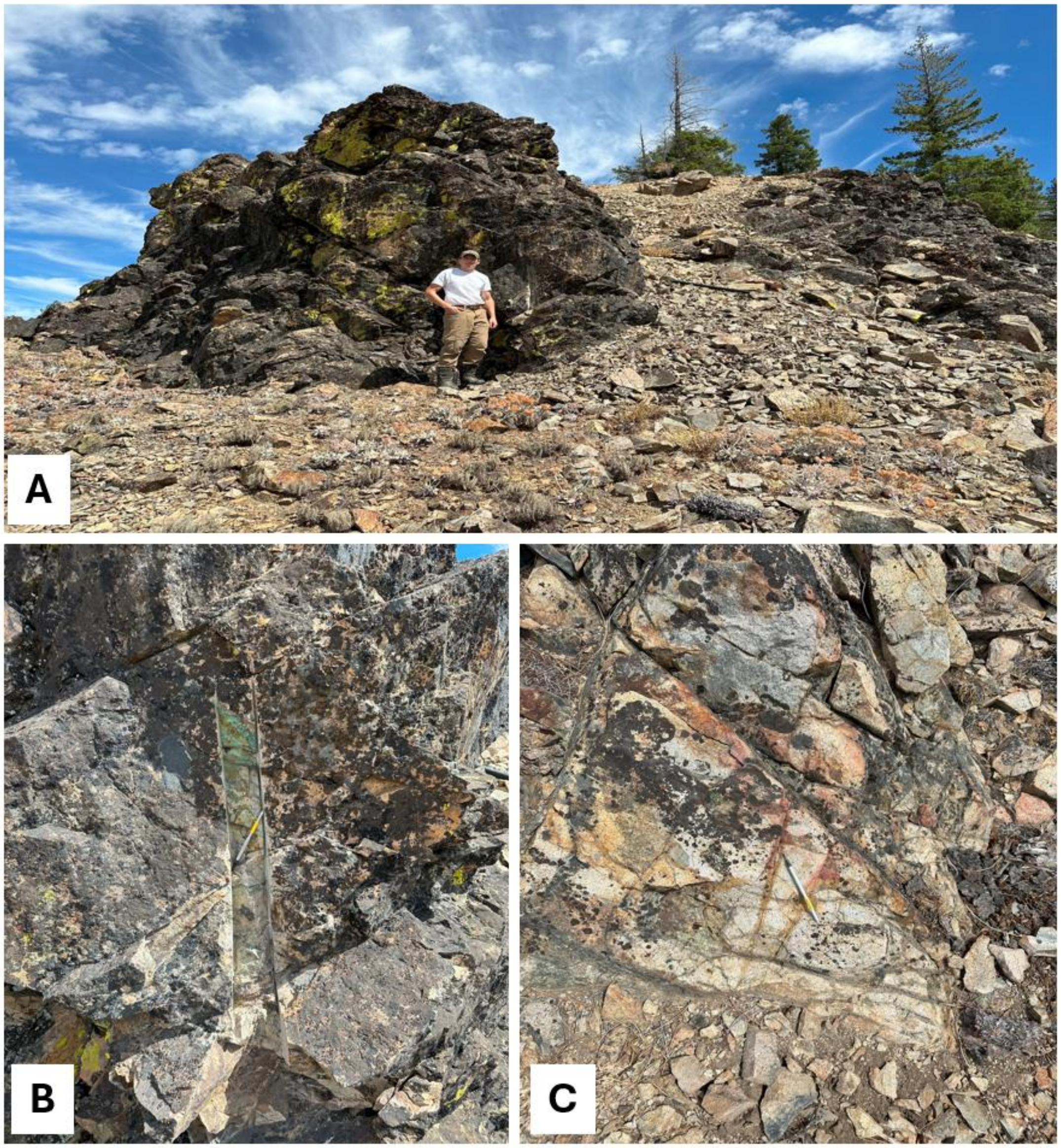
Photo 12-3: Quartz Monzonite at Superior (A), Underground Photos (B and C): A typical IOCG Deposit at Superior Consists of Fractures Filled with Chalcopyrite and Bornite, which are associated with Magnetite.



Source: GRE, 2024

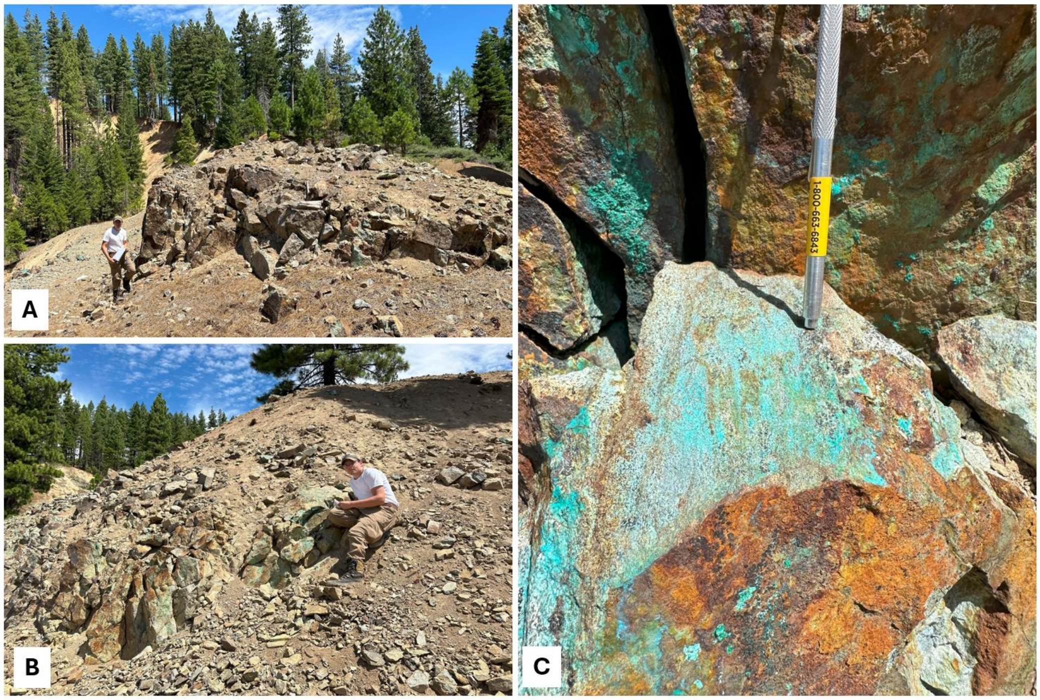
Copper Oxidation usually happens when copper is in contact with the atmosphere.

Photo 12-4: Quartz Monzonite at Lamb's Ridge (A), Copper Oxide (B), and Fractures Filled by Magnetite and Iron Oxide (C)



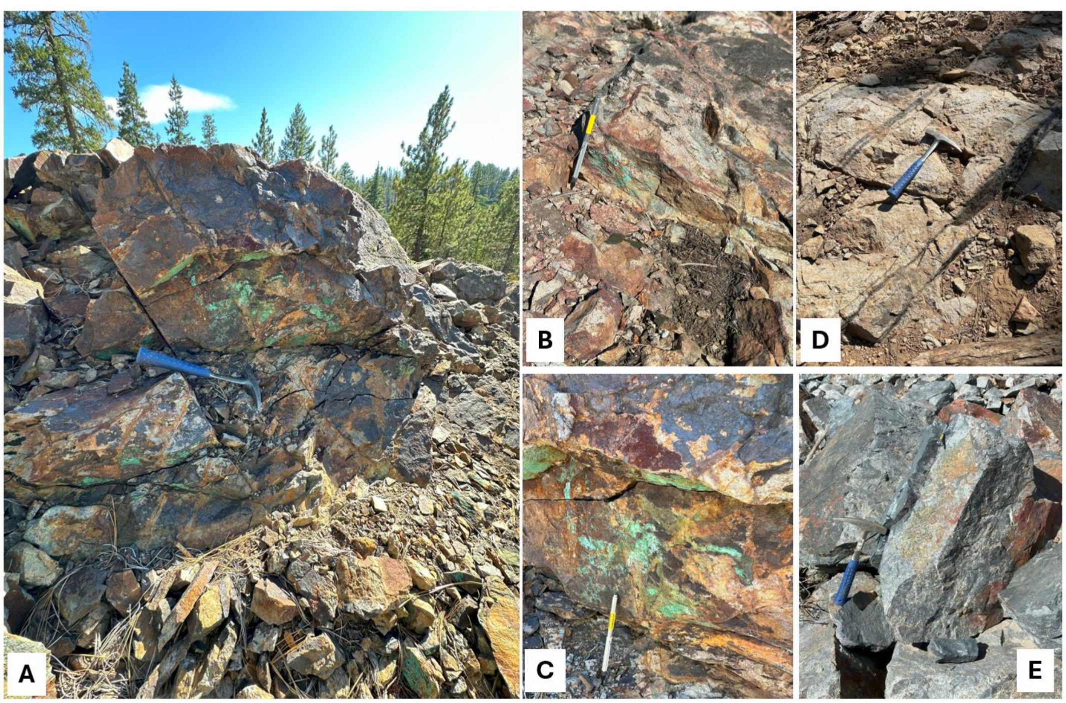
Source: GRE, 2024

Photo 12-5: Gabro at Engels with Copper Mineralization with the Northeast Trending (A and B), Close View from Copper Mineralization at Engels (C). Surface Copper Oxidation is Pervasive and Common at Engels



Source: GRE, 2024

Photo 12-6: Quartz Monzonite at Moonlight (A), Mineralization and Oxidation along the Fractures (B, C, & D), Specularite within the Metavolcanic Rock (E)

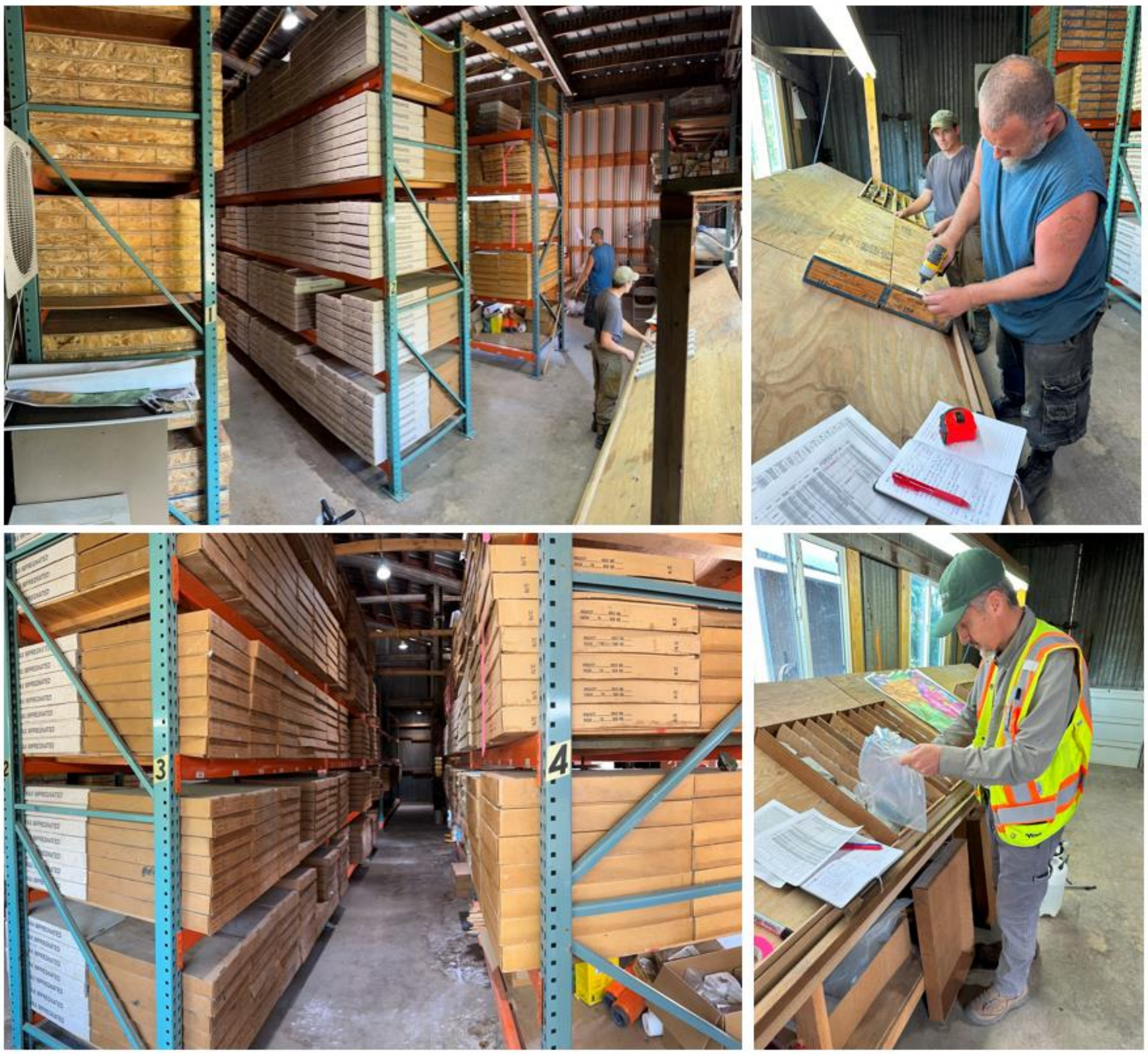


Source: GRE, 2024

12.3.2 Geological Logging Accuracy

During the site visit, GRE's QP Dr. Hamid Samari also spent one day of his site visit at US Copper's logging facility at Cresent Mills, approximately 21 km southwest of the Superior Deposit, where all core boxes and chip trays were visually inspected (Photo 12-7).

Photo 12-7: Core and Chip Trays Storage in US Copper's Facility at Cresent Mill



Source: GRE, 2024

GRE's QP, Dr. Hamid Samari, checked several core samples and chip trays from RC holes, which were drilled from 2005 to 2023, in US Copper's facility (Table 12-2). A total of 383.67 feet (116.94 meters) of sample intervals from pre-US copper drilling programs, including 368.67 feet (112.37 meters) of core samples and 15 feet (4.57 meters) of RC samples (chip trays), were checked visually using a geological hand lens. Also, a total of 296.2 feet (90.3 meters) of sample intervals from US Copper drilling programs, including 46.2 feet (14.1 meters) of core samples and 250 feet (76.2 meters) of RC samples (chip trays) from 2021 and 2023, were visually inspected. The inspection results show that the core and RC sample intervals accurately reflect the lithologies and alteration and sample descriptions recorded on the associated drill hole logs and within the project database (Photo 12-8 through Photo 12-20).

Table 12-2: Visual Inspection of Core and RC Sample Intervals

Area	No.	Hole No	From (feet)	To (feet)	Intervals (feet)	Certificate	Sample Designation	Lithology	Hole Type
Moonlight	1	05-MN-01	134.84	141.40	6.56	RE05114313	SHM-26	QM	Core
			141.40	147.97	6.56	RE05114313	SHM-27	QM	Core
			997.37	1003.93	6.56	RE06004758	SHM-269	QM	Core
			1003.93	1010.50	6.56	RE06004758	SHM-270	QM	Core
			1010.50	1017.06	6.56	RE06004758	SHM-271	QM	Core
	2	06-MN-12	216.54	223.10	6.56	RE06102150	SHM 1565	QM	Core
			223.10	229.66	6.56	RE06102150	SHM 1566	QM	Core
			229.66	236.22	6.56	RE06102150	SHM 1567	QM	Core
			236.22	242.78	6.56	RE06102150	SHM 1568	QM	Core
			242.78	249.34	6.56	RE06102150	SHM 1569	QM	Core
	3	08-MN-15	1085.00	1090.00	5.00	RE08155414	SHM-2136	QM	Core
			1090.00	1095.00	5.00	RE08155414	SHM-2137	QM	Core
			1095.00	1100.00	5.00	RE08155414	SHM-2138	QM	Core
	4	08-MN-20	30	35	5.00	RE08159710	SHM 2385	QM	Core
			35	40	5.00	RE08159710	SHM 2386	QM	Core
			40	45	5.00	RE08159710	SHM 2387	QM	Core
			50	55	5.00	RE08159710	SHM 2389	QM	Core
	6	07-MRC-09	85	90	5.00	RE08000022	07MRC9 85-90	QM	RC
			90	95	5.00	RE08000022	07MRC9 90-95	QM	RC
			95	100	5.00	RE08000022	07MRC9 95-100	QM	RC
	7	23MRC19	65	70	5.00	CI23204485	877249	QM	RC
			70	75	5.00	CI23204485	877250	QM	RC
			75	80	5.00	CI23204485	877251	QM	RC
	8	23MRC26	55	60	5.00	CI23204524	954965	QM	RC
			60	65	5.00	CI23204524	954966	QM	RC
			65	70	5.00	CI23204524	954968	QM	RC
	9	23MRC16	35	40	5.00	CI23204466	877184	QM	RC
			40	45	5.00	CI23204466	877185	QM	RC
			45	50	5.00	CI23204466	877187	QM	RC
	10	23MRC20	0	5	5.00	CI23204487	877256	QM	RC
			5	10	5.00	CI23204487	877257	QM	RC
			10	15	5.00	CI23204487	877258	QM	RC
			15	20	5.00	CI23204487	954759	QM	RC
			20	25	5.00	CI23204487	954760	QM	RC
			25	30	5.00	CI23204487	954762	QM	RC
			30	35	5.00	CI23204487	954763	QM	RC
			35	40	5.00	CI23204487	954764	QM	RC
			40	45	5.00	CI23204487	954765	QM	RC
			45	50	5.00	CI23204487	954767	QM	RC
	11	23MRC27	110	115	5.00	CI23204527	955006	QM	RC
			115	120	5.00	CI23204527	955007	QM	RC
			120	125	5.00	CI23204527	955008	QM	RC

Area	No.	Hole No	From (feet)	To (feet)	Intervals (feet)	Certificate	Sample Designation	Lithology	Hole Type
Superior	12	S21-1	240	245	5.00	RE21145938	360012	QM	Core
			245	248.5	3.50	RE21145938	360013	QM	Core
			248.5	253.5	5.00	RE21145938	360014	QM	Core
			253.5	258.9	5.40	RE21145938	360016	QM	Core
			258.9	263.5	4.60	RE21145938	360017	QM	Core
			263.5	268.1	4.60	RE21145938	360018	QM	Core
			270.4	278	7.60	RE21145938	360020	QM	Core
	13	S21-3	349.9	355	5.10	RE21169750	360366	QM	Core
			355	360	5.00	RE21169750	360367	QM	Core
	14	S21-6	195	200	5.00	EL21196777	659606	QM	Core
			200	205	5.00	EL21196777	659607	QM	Core
			205	210	5.00	EL21196777	659608	QM	Core
			210	215	5.00	EL21196777	659609	QM	Core
	15	S21-7	725	730.2	5.20	RE21268097	659971	QM	Core
			730.2	733	2.80	RE21268097	659972	QM	Core
			733	735.7	2.70	RE21268097	659973	QM	Core
			735.7	740	4.30	RE21268097	659974	QM	Core
			740	745	5.00	RE21268097	659975	QM	Core
			745	751.2	6.20	RE21268097	659976	QM	Core
Lamb’ s Ridge	16	23LRRC01	400	405	5.00	CI23204453	877089	QM	RC
			405	410	5.00	CI23204453	877090	QM	RC
	17	23LRRC02	50	55	5.00	CI23204460	877108	QM	RC
			55	60	5.00	CI23204460	877109	QM	RC
Engels	18	07-E-04	45.93	52.49	6.56	RE07085572	ESH-83	JTr	Core
			52.49	59.06	6.56	RE07085572	ESH-84	JTr	Core
			59.06	65.62	6.56	RE07085572	ESH-85	Gb	Core
			72.18	78.74	6.56	RE07085572	ESH-87	Gb	Core
	19	07-E-32	72.18	78.74	6.56	RE07147833	ESH 1235	Gb	Core
			78.74	85.30	6.56	RE07147833	ESH 1236	Gb	Core
			85.30	91.86	6.56	RE07147833	ESH 1237	Gb	Core
			91.86	98.43	6.56	RE07147833	ESH 1238	Gb	Core
			98.43	104.99	6.56	RE07147833	ESH 1239	Gb	Core
	20	08-E-40	154	159	5.00	RE08122483	ESH-1579	Gb	Core
			159	164	5.00	RE08122483	ESH-1581	JTr	Core
			164	169	5.00	RE08122483	ESH-1582	Gb	Core
			169	174	5.00	RE08122483	ESH-1583	Gb	Core
			174	179	5.00	RE08122483	ESH-1584	Gb	Core
	21	08-E-42	195	200	5.00	RE08164708	ESH 1814	Gb	Core
			200	205	5.00	RE08164708	ESH 1815	Gb	Core
			205	210	5.00	RE08164708	ESH 1816	Gb	Core
			210	215	5.00	RE08164708	ESH 1817	Gb	Core
			215	220	5.00	RE08164708	ESH 1818	Gb	Core
			220	225	5.00	RE08164708	ESH 1819	Gb	Core
			225	230	5.00	RE08164708	ESH 1821	Gb	Core
			230	235	5.00	RE08164708	ESH 1822	Gb	Core
			235	240	5.00	RE08164708	ESH 1823	Gb	Core

Area	No.	Hole No	From (feet)	To (feet)	Intervals (feet)	Certificate	Sample Designation	Lithology	Hole Type
	22	ME09-02	57	59	2.00	RE09130029	662523	Gb	Core
			59	63	4.30	RE09130029	662525	Gb	Core
			63	68	4.70	RE09130029	662526	Gb	Core
			68	70	2.00	RE09130029	662527	Gb	Core
			70	74.3	4.30	RE09130029	662528	Gb	Core
			79.3	81.3	2.00	RE09130029	662530	Gb	Core
			81.3	85.5	4.20	RE09130029	662531	Gb	Core
			85.5	89	3.50	RE09130029	662532	Gb	Core
	23	ME10-05	51	53	2.00	RE10077322	662656	JTr	Core
			53	58	5.00	RE10077322	662657	JTr	Core
			58	63	5.00	RE10077322	662658	JTr	Core
			63	68	5.00	RE10077322	662660	JTr	Core
			68	71	3.00	RE10077322	662661	JTr	Core
	24	23ERC01	160	180	20.00	CI23163304	660639-642-COMP	Gb	RC
	25	23ERC09	120	140	20.00	CI23184783	660985-989-COMP	Gb	RC
	26	23ERC11	320	340	20.00	CI23184915	661134-137-COMP	Gb	RC
			340	360	20.00	CI23184915	661138-141-COMP	Gb-Di	RC
	27	23ERC14	180	200	20.00	CI23177222	876878-882-COMP	Gb	RC
			200	220	20.00	CI23177222	876883-886-COMP	Gb	RC

Photo 12-8: Visual Inspection of Core Intervals from Holes S21-6 (Superior)

S21-6, 194.5-203 ft S21-6, 203-212.5 ft

Source: GRE, 2024

Photo 12-9: Visual Inspection of Core Intervals from Holes S21-7 (Superior)

S21-7, 723.3-732 ft

S21-7, 732-740 ft

S21-7, 740-748.3 ft

S21-7, 748.3-757 ft

Source: GRE, 2024

Photo 12-10: Visual Inspection of Core Intervals from Holes S21-1 (Superior)

S21-1, 241-249 ft

S21-1, 249-258.6 ft

S21-1, 258.6-268.0 ft

S21-1, 268.0-278 ft

Source: GRE, 2024

Photo 12-11: Visual Inspection of Core Intervals from Holes 08MN-15 (Moonlight)

08MN-15, 1084-1093 ft 08MN-15, 1093-1102 ft 08MN-15, 1102-1111 ft

Source: GRE, 2024

Photo 12-12: Visual Inspection of Core Intervals from Holes 08MN-20 (Moonlight)

08MN-20, 27-36 ft 08MN-20, 36-44.5 ft 08MN-20, 44.5-53.5 ft

Source: GRE, 2024

Photo 12-13: Visual Inspection of Core Intervals from Holes 08MN-12 (Moonlight)

06MN-12, 215.5-224 ft

06MN-12, 224-233 ft

06MN-12, 233-242 ft

06MN-12, 242-250 ft

Source: GRE, 2024

Photo 12-14: Visual Inspection of Core Intervals from Holes 05MN-1 (Moonlight)

05MN-1, 133-141 ft | 05MN-1, 141-150 ft | 05MN-1, 998-1008 ft | 05MN-1, 1008-1019.5 ft

Source: GRE, 2024

Photo 12-15: Visual Inspection of RC Intervals from Holes 23MRC20, 23MRC26, 23MRC27, 23MRC16, 07MRC9, and 23MRC19 (Moonlight)

23MRC20, 0-50 ft | 23MRC26, 0-100 ft | 23MRC27, 100-200 ft | 23MRC16, 100-200 ft | 07MRC9, 50-100 ft | 23MRC19, 50-100 ft

Source: GRE, 2024

Photo 12-16: Visual Inspection of Core Intervals from Hole 08 E 40 (Engels)

08 E 40, 157-170 ft
08 E 40, 170-179.5 ft

Source: GRE, 2024

Photo 12-17: Visual Inspection of Core Intervals from Hole 07 E 04 (Engels)

07 E 04, 47-55 ft
07 E 04, 55-63 ft
07 E 04, 63-71 ft

Source: GRE, 2024

Photo 12-18: Visual Inspection of Core Intervals from Hole ME09-02 (Engels)

ME09-02, 58-66 ft

ME09-02, 66-77 ft

ME09-02, 77-86 ft

ME09-02, 86-98 ft

Source: GRE, 2024

Photo 12-19: Visual Inspection of Core Intervals from Hole ME10-05 (Engels)

ME10-05, 52-59 ft ME10-05, 59-68 ft ME10-05, 68-77 ft

Source: GRE, 2024

Photo 12-20: Visual Inspection of RC Intervals from Holes 23ERC01, 23ERC09, 23ERC11, 23ERC14 (Engels) and 23LRRC01 and 23LRRC02 (Lambs Ridge)

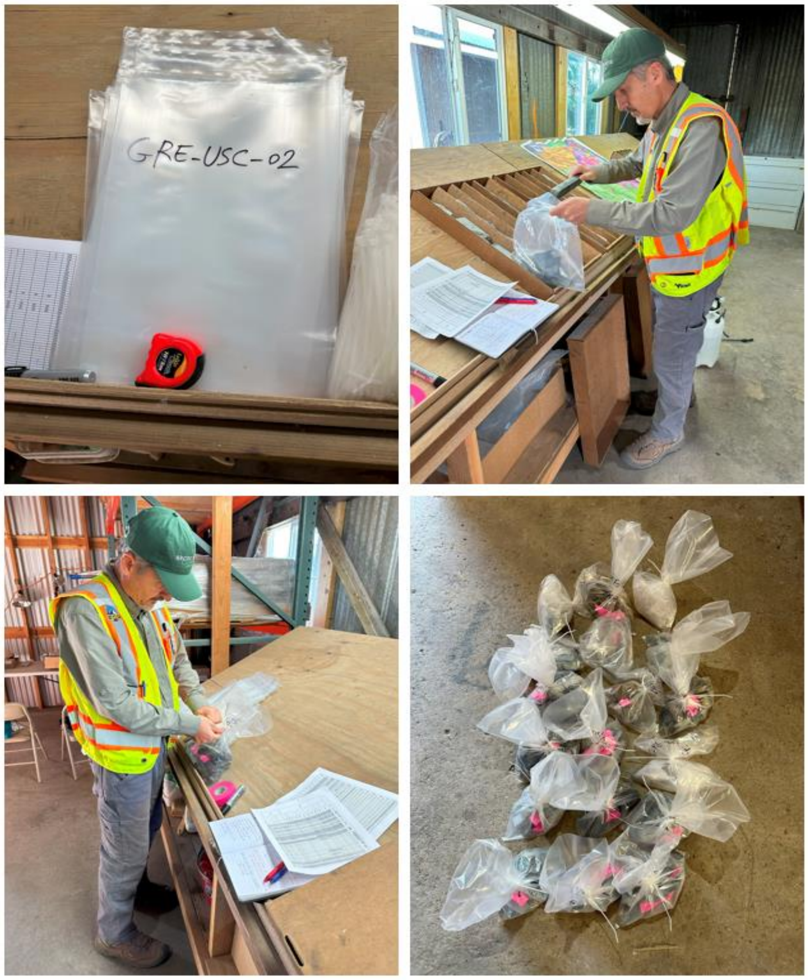
23ERC01, 100-200 ft 23ERC09, 100-200 ft 23ERC11, 300-400 ft 23ERC14, 100-200 ft 23ERC14, 200-300 ft 23LRRC01, 400-445 ft 23LRRC02, 0-100 ft

Source: GRE, 2024

12.3.3 Check Assay

At the time of GRE's QP site visit, Core and pulp reject samples from the 2005 to 2023 drilling campaigns were presented. Dr. Hamid Samari selected 13 core samples, four RC pulp samples together with one surface grab sample from the Lamb's Ridge deposit. All sample intervals selected by Dr. Samari for check assay were selected from 15 holes drilled from 2006 to 2023 drilling programs by taking the remaining pulp reject samples and ¼ core samples. All samples were labeled and put into a plastic bag by the GRE's QP Dr. Hamid Samari (Photo 12-21).

Photo 12-21: Sampling and Selected Core and Pulp Rejected Samples from the US Copper Logging Facility Cresent Mils



Source: GRE, 2024

A total of 18 check samples, were shipped by US Copper's geologist to Hazen Research Inc. (Hazen) in Golden, Colorado, USA, for 4-acid digestion and subjected to ICP for copper and for fire assay with AA finish for gold and silver (Table 12-3). Hazen received all check samples on August 12, 2024.

On August 23, 2024, GRE's QP Hamid Samari received Hazen's analytical report on the 18 selected samples for copper, gold, and silver (Table 12-3).

Table 12-3: Check Samples Submitted to Hazen Labs

No.	Deposit Area	Hole Number	From -ft	To-ft	US Copper Sample Designation	GRE Sample Number	Lithology	Type of check sample		Request Lab analysis by Hazen		
								1/4 core sample	Pulp sample	Cu (ppm) 4 Acid digestion, ICP	Au (ppm) Fire Assay, AA finish	Ag (ppm) Fire Assay, AA finish
1	Superior	S21-7	735.5	740	659974	GRE-USC-01	QM	✓		✓	✓	✓
2		S21-6	200	205	659607	GRE-USC-02	QM	✓		✓	✓	✓
3		S21-1	253.5	258.9	360016	GRE-USC-03	QM	✓		✓	✓	✓
4		S21-1	263.5	268.1	360018	GRE-USC-04	QM	✓		✓	✓	✓
5	Moonlight	08-MN-15	1090	1095	SHM-2137	GRE-USC-05	QM	✓		✓	✓	✓
6		06-MN-12	229.66	236.22	SHM-1567	GRE-USC-06	QM	✓		✓	✓	✓
7		23MRC16	40	45	877185	GRE-USC-07	QM		✓	✓	✓	✓
8	Engels	08-E-40	169	174	ESH-1583	GRE-USC-08	Gb	✓		✓	✓	✓
9		07-E-04	59.06	65.62	ESH-85	GRE-USC-09	Gb	✓		✓	✓	✓
10		ME09-02	59	63	662525	GRE-USC-10	Gb	✓		✓	✓	✓
11		08-E-42	200	205	ESH-1815	GRE-USC-11	Gb	✓		✓	✓	✓
12		08-E-42	225	300	ESH-1821	GRE-USC-12	Gb	✓		✓	✓	✓
13		07-E-32	91.86	98.43	ESH-1238	GRE-USC-13	Gb	✓		✓	✓	✓
14		23ERC11	340	360	661138-141	GRE-USC-14	Gb-Di		✓	✓	✓	✓
15		ME10-05	68	71	662661	GRE-USC-15	JTr	✓		✓	✓	✓
16	Lambs Ridge	23LRR C01	400	405	877089	GRE-USC-16	QM		✓	✓	✓	✓
17		23LRR C02	50	55	877108	GRE-USC-17	QM		✓	✓	✓	✓
18	Surface sample from Lamb's Ridge	N/A	N/A	N/A	N/A	GRE-LR-SS-01	QM	surface grab sample		✓	✓	✓

Source: GRE, 2024

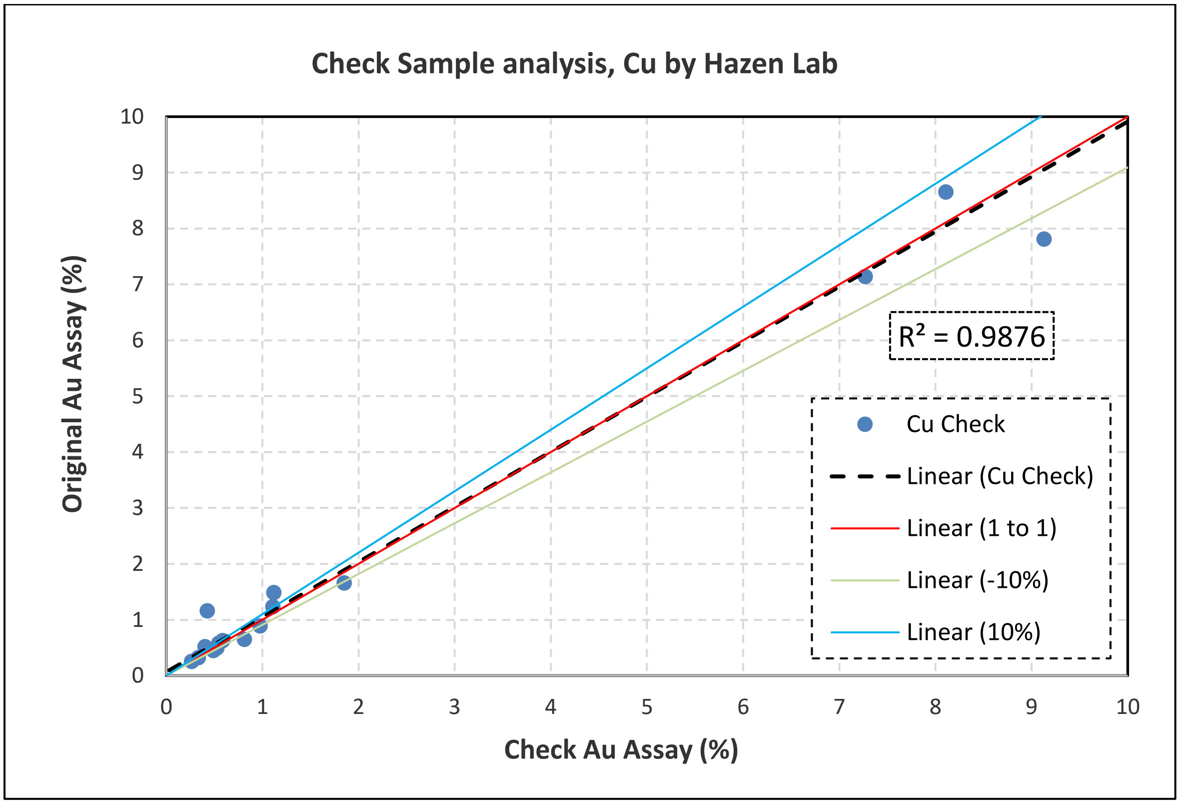
Table 12-4: Summary of Hazen Results with Original Assays

No.	Deposit Area	Hole Number	From -ft	To-ft	US Copper Sample Designation	GRE Sample Number	Original Assays			Hazen Results		
							Cu (%)	Au (ppm)	Ag (ppm)	Cu (%)	Au (ppm)	Ag (ppm)
1	Superior	S21-7	735.5	740	659974	GRE-USC-01	0.448	0.014	3.5	0.491	0.13	0.9
2		S21-6	200	205	659607	GRE-USC-02	1.235	0.041	7	1.11	0.09	4.6
3		S21-1	253.5	258.9	360016	GRE-USC-03	1.485	0.085	13	1.12	0.09	10.4
4		S21-1	263.5	268.1	360018	GRE-USC-04	0.582	0.031	6	0.549	0.06	6.1
5	Moonlight	08-MN-15	1090	1095	SHM-2137	GRE-USC-05	0.494	N/A	4.4	0.529	0.04	1.2
6		06-MN-12	229.66	236.22	SHM-1567	GRE-USC-06	1.16	0.028	31.5	0.429	0.03	3.8
7		23MRC16	40	45	877185	GRE-USC-07	0.518	0.007	1	0.405	0.1	<0.3
8	Engels	08-E-40	169	174	ESH-1583	GRE-USC-08	0.65	N/A	0.8	0.813	0.02	1.9
9		07-E-04	59.06	65.62	ESH-85	GRE-USC-09	7.81	0.09	88	9.13	0.24	93.2
10		ME09-02	59	63	662525	GRE-USC-10	8.65	0.3	85.8	8.11	0.14	66.8
11		08-E-42	200	205	ESH-1815	GRE-USC-11	0.626	N/A	4.9	0.586	0.07	1.7
12		08-E-42	225	230	ESH-1821	GRE-USC-12	7.14	N/A	54.1	7.27	0.53	52.3
13		07-E-32	91.86	98.43	ESH-1238	GRE-USC-13	1.655	0.3	8	1.85	0.04	7.6
14		23ERC11	340	360	661138-141	GRE-USC-14	0.888	N/A	8	0.977	0.02	6.2
15		ME10-05	68	71	662661	GRE-USC-15	13.05	0.085	142	12.6	0.06	45.3
16	Lambs Ridge	23LRR C01	400	405	877089	GRE-USC-16	0.321	0.013	1	0.335	0.01	0.3
17		23LRR C02	50	55	877108	GRE-USC-17	0.252	0.005	1	0.263	0.01	0.3
18	Surface sample from Lamb’s Ridge	N/A	N/A	N/A	N/A	GRE-LR-SS-01	N/A	N/A	N/A	0.364	0.25	4.9

Source: GRE, 2024

A comparison of the original versus check assay values for copper on the 17 check samples, including core and pulp samples, shows a good correlation between the results, with an R2 of 0.9876 for copper (Figure 12-45). Standard t-test statistical analysis was completed to determine whether there was a significant difference between the original and check assay population means. The t-test results showed no statistically significant difference between the means of the two trials (original versus check assay).

Figure 12-45: Hazen Check Assay Results (Cu)



Source: GRE, 2024

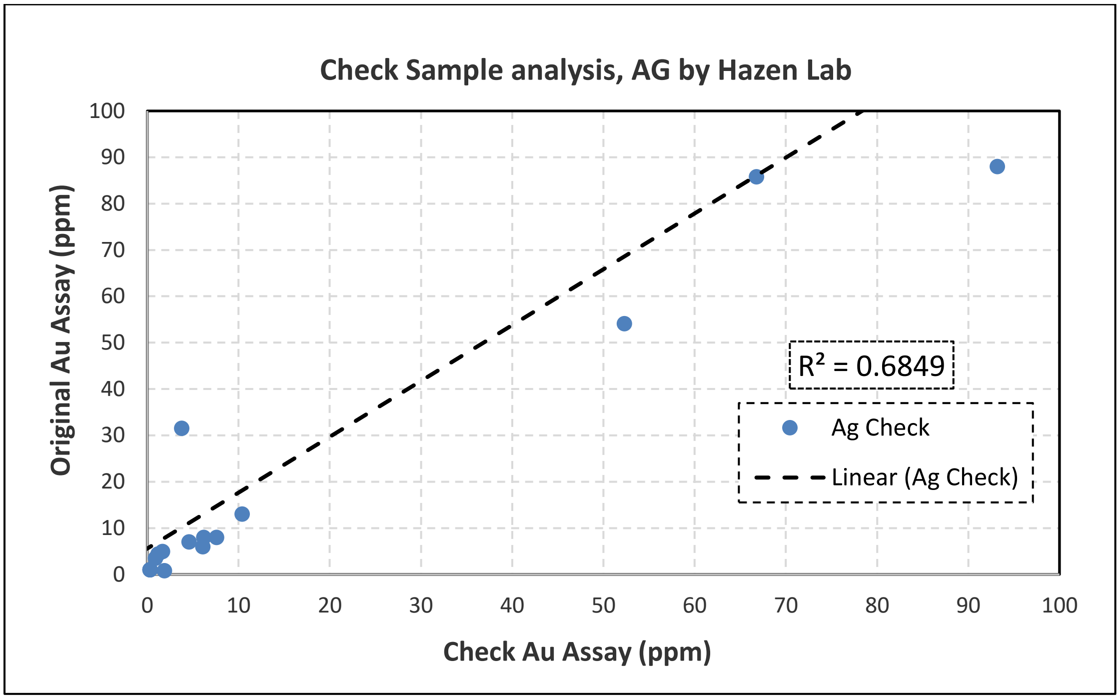
A comparison of the original versus check assay values for silver and gold on the 17 and 12 check samples, respectively, show acceptable correlations between the results for silver and a poor correlation between the results for gold, with an R2 of 0.6849 for Ag and 0.0349 for Au (Figure 12-46 and Figure 12-47).

The silver results show that three samples, SHM-1567, ESH-85, and ESH-1821, do not correlate well with their original assays. This is probably due to high-grade zones with high nugget effect and variability and the type of mineralization for those associated with veinlets. Duplicate quarter-core samples do not always provide the same grade of mineralization.

The poor correlation for gold is to be expected because most samples are low-grade, with less than 0.1 ppm Au. In this case, good correlation is seldom achieved.

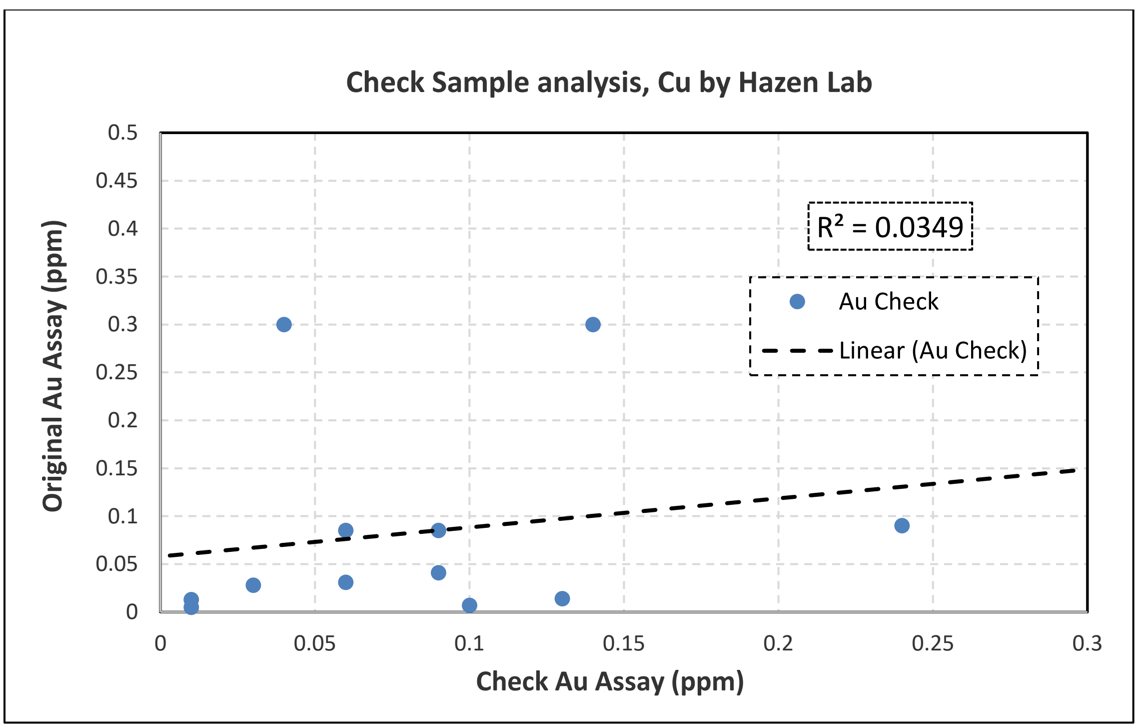
The assay result on the surface grab sample from Lamb's Ridge (GRE-LR-SS-01) confirms copper sulfide minerals such as chalcopyrite are associated with magnetite and tourmaline veinlets within Quartz monzonite.

Figure 12-46: Hazen Check Assay Results (Ag)



Source: GRE, 2024

Figure 12-47: Hazen Check Assay Results (Au)



Source: GRE, 2024

12.4 Database Audits

12.4.1 Pre-US Copper (1964-2016)

Dr. Samari, GRE's QP, completed a manual digital Project database audit. About 11% of original assay certificates for all drill holes were spot-checked with the database for accuracy and any clerical errors. The manual audit revealed no discrepancies between the hard-copy information and the digital database. As the Project advances and more data is collected, periodic database verification should be performed to maintain accuracy.

12.4.2 US Copper (2016-2023)

The database manual audit work by GRE's QP, Dr. Samari, which compared about 40% of original assay certificates with the database for the 2021 and 2023 drill campaigns, found no material errors. Dr. Samari recommends that US Copper establish a routine, internal mechanical audit procedure to check for overlaps, gaps, total drill hole length inconsistencies, non-numeric assay values, or any missing information in the database. After any significant update to the database, an internal mechanical audit should be conducted. The results of each audit, including any corrective actions taken, should be documented to provide a running log of the database validation.

12.5 Verification by Dr. Samari, Geological Data Adequacy QP

Based on the findings of GRE's QP Samari's check of the sampling practices, drill hole collars in the field, visual inspection of RC and core samples, and the results of manual database audit efforts for entire drilling campaigns, Dr. Samari considers the collar, lithology, and assay data contained in the Project database to be reasonably accurate and suitable for use in estimating mineral resources and mineral reserves.

12.6 Verification by Dr. Todd Harvey – Metallurgy QP

Dr. Todd Harvey, the Metallurgy QP, believes that the metallurgical testing was completed for the Moonlight-Superior project by a number of well-known commercial metallurgical laboratories. Dr. Harvey reviewed the sample selection and compositing used in the metallurgical test work and found that the selection of samples was representative for this type of deposit and geology. Dr. Harvey performed several mathematical tests to validate the metallurgical balances presented in the test work and they found the data presented in the metallurgical reports to be consistent with practices performed by reputable independent test laboratories. A complete discussion of the test work is provided in Section13.0. Though much of the work is historical in nature, the work appears to be professionally completed and is well documented, is supported by production data, and is suitable for estimation of heap leach and flotation copper recovery calculations in this PEA.

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

Mining and processing methods, costs, and infrastructure needs were verified by comparison to other similar sized open pit copper mines operating in the western USA and experience of the QPs, Ms. Lane and Dr. Harvey. Cost data used in the report was sourced from the most recent Infomine cost data report. All costs used in the analysis were verified and reviewed by Ms. Lane and were assessed to be current and appropriate for use. Finally, after the economic study was performed, the overall operating costs for different aspects of the operation (mining, process, and general & admin) were benchmarked against similar sized mines and recent technical reports to determine if they were similar; the results did benchmark well to other operations and economic studies.

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

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 EXECUTIVE SUMMARY

Test work has shown that the various deposits associated with this project have an oxide and transition cap that shows amenability to conventional acid heap leaching. This material is underlain by primary copper sulfides consisting of chalcopyrite with minor bornite, which show amenability to conventional sulfide flotation. More test work is required to properly identify the process design criteria, but initial indications show the potential for two process routes consisting of an acid heap leach and a conventional flotation concentrator.

13.2 CURRENT METALLURGICAL TEST WORK

In March of 2024 Kappes Cassiday & Associates was retained to conduct several copper speciation tests on potential heap leach materials (Kappes Cassiday & Associates, 2024).

The sequential copper tests were conducted as follows:

1. One gram of sample was combined with 40 milliliters (mL) of an acid solution and leached for two hours at ambient temperature. The acid solution was composed of 102.12 grams per liter (g/L) sulfuric acid and 21.95 g/L hydrated ferric sulfate. The slurry was loaded into a centrifuge tube and agitated on a shaker table.
2. After leaching, the slurry was centrifuged and the clear solution removed by decantation. The solution was assayed by flame atomic absorption spectrometry (FAAS) for Cu (reported as Acid Soluble Copper).
3. The solids in the centrifuge tube were washed with 25 mL of distilled water a total of three (3) times. With each cycle, the wash slurry was centrifuged and the clear solution decanted.
4. The solids in the centrifuge tube were combined with 40 mL of a sodium cyanide solution and leached for two hours at ambient temperature. The leach solution was composed of 5.0 g/L sodium cyanide and 2.5 g/L sodium hydroxide. The slurry was loaded into a centrifuge tube and agitated on a shaker table.
5. After leaching, the slurry was centrifuged and the clear solution removed by decantation. The solution was assayed by FAAS for Cu (reported as Cyanide Soluble Copper).
6. The solids in the centrifuge tube were washed with 25 mL of distilled water a total of three (3) times. With each cycle, the wash slurry was centrifuged and the clear solution decanted.
7. The solids in the centrifuge tube were transferred into a Teflon beaker and combined with 15 mL of hydrochloric acid. After gentle agitation, eight mL of hydrofluoric acid were added. The mixture was mixed well and placed on a hot plate. After the mixture was dried, six mL of hydrochloric acid were added along with 16 mL of nitric acid and eight mL of 17.5% hydrogen peroxide. The mixture was mixed, heated, and allowed to cool. The mixture was bulked with a 33% (v/v) hydrochloric acid solution and assayed by FAAS for copper (reported as 4-Acid Residue Copper).
8. The residue was discarded.

The results of the sequential copper analyses are presented are presented in Table 13-1 and are presented visually in Figure 13-1.

The results can be interpreted based on the dissolution of copper in each of the various leach stages as shown in Table 13-2. A significant portion of the samples examined as potential heap leach material show high acid and cyanide soluble copper. These materials should be amenable to conventional acid heap leaching with minimal biooxidation.

Acid bottle roll leach testing was conducted on a 5-kilogram portion of each composite sample. The bottle roll test procedure is outlined in the following:

1. A 5-kg portion of 100% passing 1.70 mL material was placed into a 20-liter bottle and slurried with 7.5 liters of tap water
2. The slurry was mixed thoroughly and the pH of the slurry checked
3. Sulfuric acid and hydrated ferric sulfate were added to the slurry to a target amount of 10 g/L sulfuric acid (H_2SO_4) and 5 g/L ferric iron [added as hydrated ferric sulfate].
4. Due to off-gassing during leach, a mixture of intermittent rolling and hand shaking mixed the slurry during the test. The slurries were mixed over an 8-hour period each day.
5. The slurry was checked at 2, 4, 8, 24, 48, 72, 96, 120, 144 and 168 hours for pH, oxidation-reduction potential (ORP), free acid, solution density, Cu, Total iron (Fe), and Ferrous Fe.
6. Additional sulfuric acid and hydrated ferric sulfate were added after each sample period, if required, to adjust the slurry to the target levels.
7. After completion of the leach period, the slurry was filtered. The filter cake was then washed with two liters of 1-g/L sulfuric acid solution followed by three liters of tap water. The wash filter cake was then dried.

From the dry tailings, duplicate portions were split out and individually ring and puck pulverized to 80% passing 0.075 millimeters. The pulverized portions were then assayed for total and sequential copper. The reject material was stored.

The copper extraction results of the bottle roll test are summarized in Figure 13-2.

In November 2024, GRE conducted several rougher flotation tests on material representing potential concentrator plant feed. The samples tested are shown in Table 13-4.

Table 13-1: Head Analyses – Sequential Copper

KCA Sample No.	Descriptions	Total Copper, mg/kg	Acid Soluble Copper, mg/kg	Acid Soluble Copper, %	Cyanide Soluble Copper, mg/kg	Cyanide Soluble Copper, %	Acid Soluble + CN Soluble, %	4-Acid Residue Copper, mg/kg	4-Acid Residue Copper, %	Calculated Total Copper, mg/kg	Total/ Calculated Copper
99528 A	23MRC24 24B Low Grade	2,550	1,196	47%	1,300	51%	98%	51	2%	2,547	1.00
99529 A	23MRC25 25B High Grade	9,720	1,204	12%	3,140	32%	45%	5,360	55%	9,704	1.00
99530 A	23MRC29 29 Mid Grade	3,740	2,256	61%	1,360	36%	97%	111	3%	3,727	1.00
99531 A	23ERC15 15C Low Grade	2,510	748	30%	246	10%	40%	1,508	60%	2,502	1.00
99532 A	23ERC08 8A Mid Grade	3,636	2,940	74%	486	12%	86%	572	14%	3,998	0.91
99533 A	23ERC01 1B High Grade	15,490	8,490	55%	6,240	40%	96%	686	4%	15,416	1.00

Figure 13-1: Sequential Copper Results

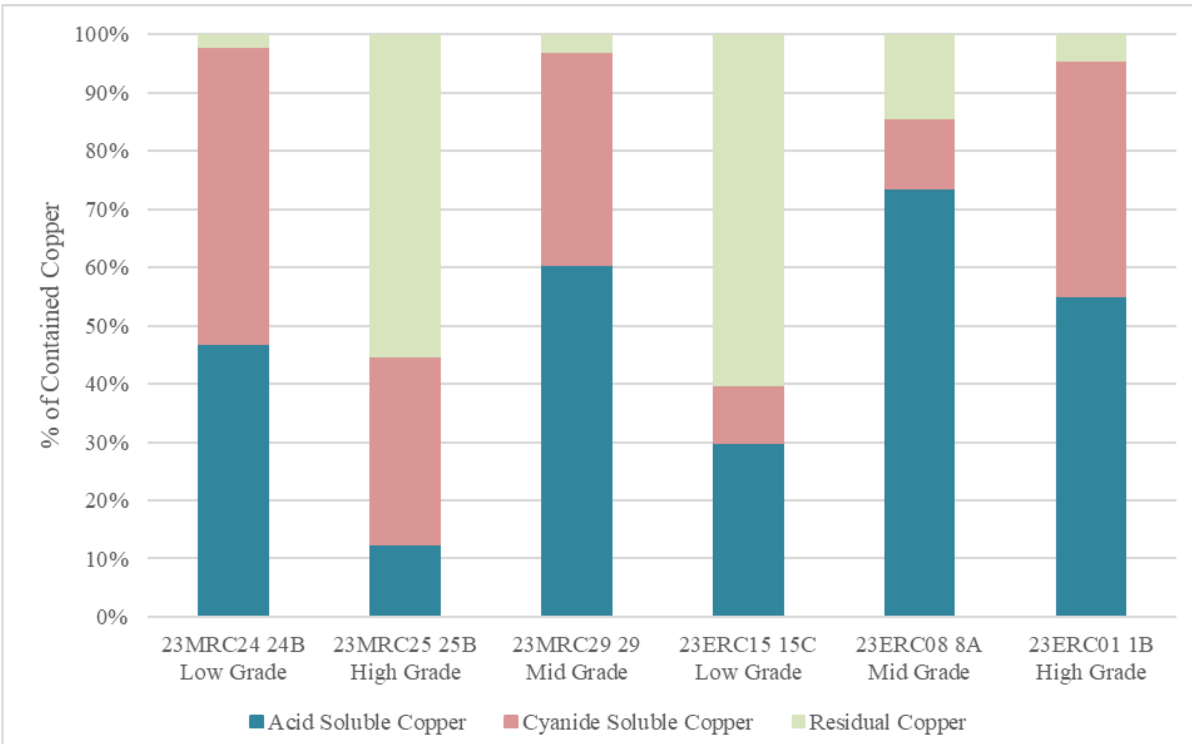


Table 13-2: Approximate Dissolution of Various Copper Species in Sulfuric Acid and Cyanide Solutions

Mineral Species	Composition	Approx Dissolution in Sulfuric Acid	Approx Dissolution in Cyanide
Oxides			
Atacamite	$Cu2Cl(OH)_3$	100	100
Azurite	$2CuCO_3Cu(OH)_2$	100	100
Cuprite	Cu2O	70	100
Chrysocolla	$CuSiO_32(H_2O)$	100	45
Malachite	$Cu_2CO_3(OH)_2$	100	100
Native Copper	Cu	5	100
Tenorite	CuO	100	100
Secondary Sulfides			
Chalcocite	Cu_2S	3	100
Covellite	CuS	5	100
Primary Sulfides			
Bornite	Cu_5FeS_4	2	100
Chalcopyrite	$CuFeS_2$	2	7

Table 13-3: Summary of Acid Bottle Roll Leach Test Work

KCA Sample No.	KCA Test No.	Description	Crush Size, mm	Assay Head (mg Cu/kg)	Calculated Head (mg Cu/kg)	Extracted (mg Cu/kg)	Avg. Tails (mg Cu/kg)	Cu Extracted (%)	Leach Time (hours)	Gross Cons. H_2SO_4 (kg/MT)	Net Cons. H_2SO_4 (kg/MT)
99528 A	99536 A	23MRC24 24B Low Grade	1.70	2,550	2,550	1,604	946	63%	168	49	47
99529 A	99536 B	23MRC25 25B High Grade	1.70	9,720	9,377	1,617	7,760	17%	168	69	66
99530 A	99536 C	23MRC29 29 Mid Grade	1.70	3,740	4,121	2,551	1,570	62%	168	55	51
99531 A	99537 A	23ERC15 15C Low Grade	1.70	2,510	2,546	779	1,767	31%	168	49	47
99532 A	99537 B	23ERC08 8A Mid Grade	1.70	3,636	3,649	2,680	969	73%	168	33	28
99533 A	99537 C	23ERC01 1B High Grade	1.70	15,490	15,665	9,415	6,250	60%	168	71	56

MT = metric tonne

Figure 13-2: Heads versus Bottle Roll Leach Test Results

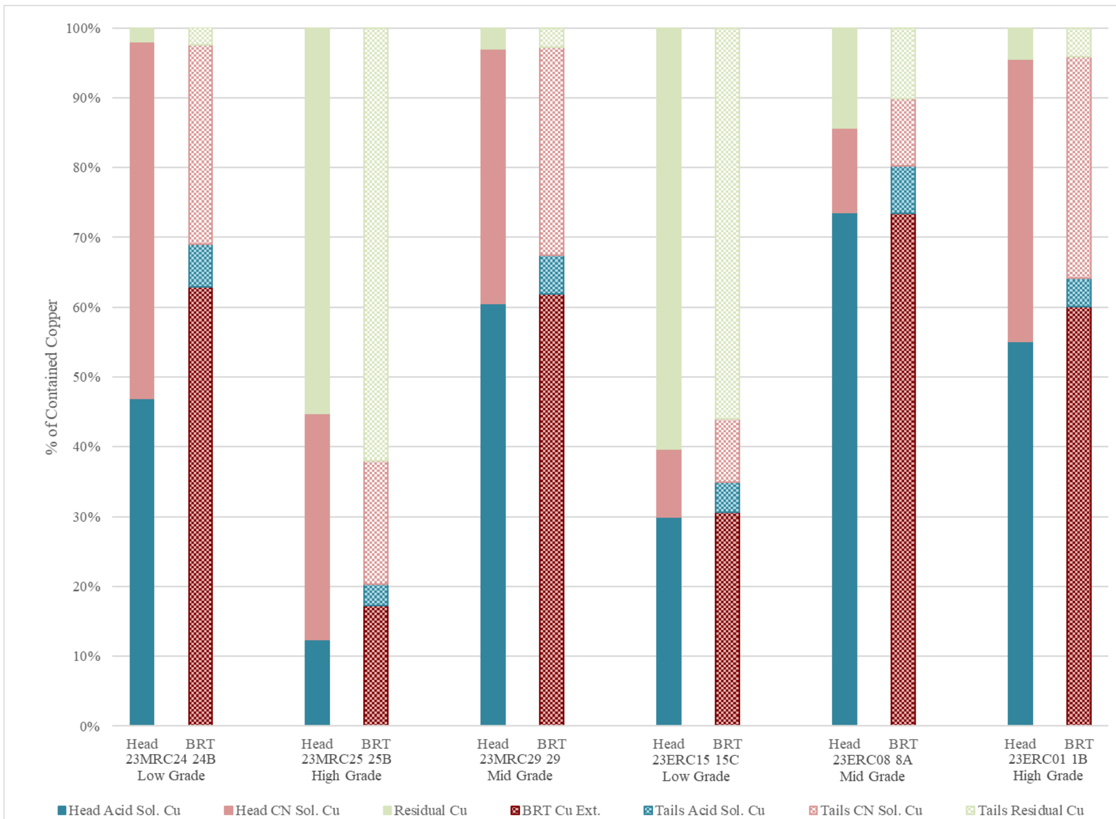


Table 13-4: GRE Flotation Test Samples

Deposit Area	Hole Number	From - ft	To-ft	US Copper Sample Designation	Lithology	Original Assays					Hazen Results		
						Fe%	S%	Cu (%)	Au (ppm)	Ag (ppm)	Cu (%)	Au (ppm)	Ag (ppm)
Superior	S21-7	735.5	740	659974	QM	N/A	N/A	0.448	0.014	3.5	0.491	0.13	0.9
Moonlight	08-MN-15	1090	1095	SHM-2137	QM	3.59	0.63	0.494	N/A	4.4	0.529	0.04	1.2

N/A = not applicable

The test variables were as follows:

- grinding size – P80 150 mesh
- conditioning time – 1 minutes
- float time – 14 minutes
- reagent type –potassium amyl xanthate (PAX), Di thiophosphate
- frother – Flottec 171
- pH – 10.0
- pulp bulk density – 30% solids by weight

The results of the flotation tests are shown in Table 13-5.

Table 13-5: GRE Flotation Test Results

Stream	Weight Dry	Assays (cum)				Distribution (cum)			
	% (cum)	Cu%	Fe%	Ag g/t	S= %	Cu%	Fe%	Ag (%)	S= %
Sample: GRE-USC-01 Deposit: Superior									
Feed (calc)		0.467%	9.751%	2.689	0.332%				
Conc 1 - 0.75 min	3.2%	8.86%	12.40%	37.9	7.03%	60.9%	4.1%	45.3%	68.0%
Conc 2 - 2 min	5.8%	6.33%	11.41%	29.8	4.73%	79.2%	6.8%	64.8%	83.3%
Conc 3 - 8 min	9.3%	4.43%	10.42%	22.8	3.20%	88.6%	10.0%	79.3%	90.1%
Conc 4 - 14 min	12.9%	3.31%	9.76%	17.4	2.37%	91.6%	12.9%	83.8%	92.1%
Tailings	100.0%								
Sample: GRE-USC-05 Deposit: Moonlight									
Feed (calc)		0.578%	4.542%	2.629	0.495%				
Conc 1 - 0.75 min	2.4%	14.60%	18.80%	42.3	13.40%	61.5%	10.1%	39.1%	65.9%
Conc 2 - 2 min	9.3%	5.30%	9.59%	19.6	4.73%	85.1%	19.6%	69.0%	88.7%
Conc 3 - 8 min	17.2%	3.19%	7.51%	13.1	2.74%	94.9%	28.5%	85.6%	95.4%
Conc 4 - 14 min	21.1%	2.62%	6.98%	11.0	2.25%	95.6%	32.4%	88.0%	96.0%
Tailings	100.0%								

Both the Superior and Moonlight samples showed good response to sulfide flotation at a moderate grind. Copper recovery to the concentrate was 91.6% and 95.6% for Superior and Moonlight, respectively. Silver recovery was 83.8% and 88.0% for Superior and Moonlight, respectively. These results are consistent with the previous flotation results presented. Further work is required to optimize the flotation parameters, but initial indications are that copper recoveries above 90% can be achieved.

13.3 REVIEW OF HISTORICAL METALLURGICAL TESTING

Much of the historical test work reported was sourced from the previous Tetra Tech report (TetraTech, 2018) but GRE does not have access to the original test reports. The results are being presented here for historical reference, but they do align with the more recent test work results.

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*. 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 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 kiloWatt-hours per short ton (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 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 Placer-Amex completed metallurgical testing on five bulk composite samples of cores from the Moonlight deposit. Composite core samples were sent to Kappes, Cassidy & Associates (KCA) 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-gram head splits from each of the five composites. Copper recoveries from the oxide samples after 72 hours of tests were variable across 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 pounds per short ton (lb/st).

Ten kilograms of finer than one-inch crushed composite was also leached with similar results. The South oxide composite returned a 92% extraction. The Central oxide composite and the North oxide composite returned 65% and 57% extraction, respectively. The sulfide composites returned a 27% copper extraction. Bottle roll leach tests are useful to provide an indication of the leachability of the material but are not necessarily an indication of potential heap leach extractions because of the fine size of the material employed during testing.

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 grams of material were carried out along with larger tests utilizing 500 grams 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. No copper extraction data appears to be available from this test work.

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 samples treated consumed up to 180 lbs of sulfuric acid per ton. As stated above, bottle roll leach tests are designed to define amenability to acid leaching, but extraction results and acid consumptions are not generally indicative of heap leach results.

In 2007, Sheffield drilled 15 RC 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 RC 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 RC drill samples using KCA. Two samples were from Moonlight RC holes 07MRC-03 from 20 to 25 feet and 07MRC-06 from 10 to 15 feet. Samples were coarse-crushed, with dilute sulfuric acid being added to form a slurry. The samples were bottle rolled for 144 hours. The 07MRC-3 sample returned 65% copper extraction. The 07MRC-06 sample returned an 81% copper extraction. Silver extraction was negligible in both samples, as expected.

Additional bottle roll and column leach tests were conducted on Moonlight deposit oxide mineralization samples in 2013. 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-inch, ¾-inch, and ½-inch. Closed column tests were cured for five days then leached for 30 days. The main results of the study concluded that the ½-inch size fraction had the best copper leach extractions of 89%. A sulfuric acid cure dosage of 15.5 lb/st (7.75 kilograms per metric tonne) gave the optimum cure dosage to obtain the highest copper extraction (SGS, 2013).

13.4 RECENT 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. The material provided by Crown Mining was identified as follows: 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. 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 13-6 to Table 13-8 show the results of the automated mineralogical analysis of each material.

Table 13-6: Moonlight Oxide Automated Mineralogical Analysis

Mineral	Chemistry	Percentage
Quartz	SiO2	32.94
Orthoclase	KAlSi3O8	24.70
Albite	NaAlSi3O8	18.56
Andalusite	Al2SiO5	15.19
Hematite	Fe2O3	5.81
Chlorite	(Fe,Mg,Al)6Si4O10(OH)8	2.07
Calcite	CaCO3	0.38
Dolomite	CaMg(CO3)2	0.09
Barite	BaSO4	0.07
Apatite	Ca5(PO4)3OH	0.07
Ilmenite	FeTiO3	0.05
Malachite	Cu2CO3(OH)2	0.04
Anorthite	CaAl2Si2O8	0.02
Chromite	FeCr2O4	0.01
Tetrahedrite	(Cu,Fe)12Sb4S13	<0.01
Bornite	Cu5FeS4	<0.01
Galena	PbS	<0.01
Titanite	CaTiSiO5	<0.01
Chalcopyrite	CuFeS2	<0.01

Table 13-7: Superior Sulfide Automated Mineralogical Analysis

Mineral	Chemistry	Percentage
Orthoclase	KalSi3O8	30.24
Quartz	SiO2	25.99
Albite	NaAlSi3O8	22.79
Chlorite	(Fe,Mg,Al)6Si4O10(OH)8	9.87
Hematite	Fe2O3	3.52
Andalusite	Al2SiO5	2.04
Chromite	FeCr2O4	1.92
Chalcopyrite	CuFeS2	1.52
Anorthite	CaAl2Si2O8	0.75
Calcite	CaCO3	0.44
Ilmenite	FeTiO3	0.40
Titanite	CaTiSiO5	0.28
Bornite	Cu5FeS4	0.10
Apatite	Ca5(PO4)3OH	0.14
Galena	PbS	<0.01
Malachite	Cu2CO3(OH)2	<0.01
Dolomite	CaMg(CO3)2	<0.01
Tetrahedrite	(Cu,Fe)12Sb4S13	<0.01
Barite	BaSO4	<0.01

Table 13-8: Moonlight Sulfide 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 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). 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 – P80 100 mesh, P70 100 mesh, P90 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 kilograms per short ton (kg/st), respectively
- frother – methyl isobutyl carbinol (MIBC) – as required

- pH – 10.0
- pulp bulk density – 40% solids by weight.

Table 13-9 identifies the rougher flotation test results, shown in decreasing order based on grind size.

The rougher concentrates were then cleaned using the following parameters:

- grinding size – P90 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 13-9: 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

Table 13-10 identifies the results of the cleaner flotation tests.

Table 13-10: Cleaner Flotation Test 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%

*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 reduce the particle size and improve liberation. As a next step, locked cycle flotation testing should be performed.

The metal grades in the Moonlight sample tested in 2017 are higher than the average deposit grade. This suggests that the samples tested may not be representative. Further tests on more representative samples should be conducted.

To advance this project a complete metallurgical testing program needs to be conducted. The test work data available for this current study is sufficient to provide a strong indication of the potential metallurgical results but the results are not definitive at this stage. A complete geometallurgical program should be undertaken to advance the project including:

- Mineralogical characterization of the various deposits
- Sample selecting for testing to include lithology, oxidation, grade and spatial variability
- Heap leach column testing
- Crushing and grinding testing
- Flotation testing include locked cycle tests

14 MINERAL RESOURCE ESTIMATE

This mineral resource estimate for the Moonlight-Superior Property was completed by Terre Lane SME QP, with GRE. Ms. Lane is a Qualified Person as defined by NI 43-101 and is independent of US Copper, the vendor, and the property. GRE estimated the mineral resource for the Project using an inverse distance squared interpolant. Geostatistics and mineral resource estimation were done with Leapfrog EDGE®. Model visualization was done with Leapfrog Geo® software, and the mineral resources were constrained with a Lerch-Grossman pit optimization. The metals of interest at the Project are copper, silver, and gold. The Mineral Resource estimate reported here was prepared in a manner consistent with the "CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines" adopted by CIM Council on November 29, 2019. The mineral resources are classified as Measured, Indicated, and Inferred in accordance with "CIM Definition Standards for Mineral Resources and Mineral Reserves," prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014. Classification of the resources reflects the relative confidence of the grade estimates. The effective date of the mineral resource estimate reported herein is December 16, 2024.

14.1 Drill Hole Database

The Moonlight-Superior mineral resource estimate is based on 552 drill holes (249,122.16 feet [75,932.43 meters]) and 28,419 associated assay values collected from 1960 to 2023. GRE was provided the drill hole database in Excel format by US Copper, which included collar locations, down hole survey data, assay data, and lithology data.

Collar elevations for 271 of the drill holes did not match topography (238 at Moonlight, 23 at Copper Mountain, one at Osmeyer Ridge, and nine at unidentified locations) and, after review of aerial imagery for the locations showed no noticeable development or terrain disturbance, were adjusted to match topography. Those locations and the elevation adjustments are presented in Table 14-1.

Table 14-1: Moonlight-Superior Project Drill Hole Collar Elevations Adjusted in Leapfrog

HoleNo	Area	Easting	Northing	Elevation	Revised Elevation
05-MN-01	Moonlight	6895634	1967063	5585	5557.789
05-MN-02	Moonlight	6895634	1967063	5585	5557.789
06-MN-03	Moonlight	6896145	1965841	5815	5801.6
06-MN-04	Moonlight	6896168	1965830	5815	5802.824
06-MN-05	Moonlight	6896158	1965811	5815	5804.898
06-MN-06	Moonlight	6896151	1965811	5815	5804.344
06-MN-07	Moonlight	6895627	1967073	5585	5556.429
06-MN-08	Moonlight	6895627	1967073	5585	5556.429
06-MN-09	Moonlight	6897217	1968696	5755	5719.585
06-MN-10	Moonlight	6897217	1968696	5755	5719.585
06-MN-11	Moonlight	6897217	1968696	5755	5719.585
06-MN-12	Moonlight	6897217	1968735	5760	5718.668
06-MN-13	Moonlight	6896181	1968199	5602	5545.583
06-MN-14	Moonlight	6896159	1968229	5575	5539.905
07-MRC-02	Moonlight	6897325	1968592	5735	5735
07-MRC-03	Moonlight	6897325	1968592	5735	5735

HoleNo	Area	Easting	Northing	Elevation	Revised Elevation
07-MRC-04	Moonlight	6897225	1968728	5719.786	5719.794
07-MRC-05	Moonlight	6897077	1968346	5656.892	5656.923
07-MRC-06	Moonlight	6895814	1965955	5774.427	5774.674
07-MRC-07	Moonlight	6895814	1965955	5774.427	5774.674
07-MRC-08	Moonlight	6895844	1966005	5770.6	5770.654
07-MRC-09	Moonlight	6895844	1966005	5770.6	5770.654
07-MRC-10	Moonlight	6895913	1965937	5783.426	5783.499
07-MRC-11	Moonlight	6896189	1966127	5761.544	5760.669
07-MRC-12	Moonlight	6896131	1966342	5745	5744.8
07-MRC-13	Moonlight	6896154	1966501	5736.392	5735.531
07-MRC-14	Moonlight	6896257	1966663	5727.167	5727.854
07-MRC-15	Moonlight	6896337	1966810	5720.638	5720.711
08-MN-15	Moonlight	6895733	1964802	5885	5843.189
08-MN-16	Moonlight	6895959	1965957	5789	5785.462
08-MN-17	Moonlight	6896171	1966431	5762	5749.433
08-MN-18	Moonlight	6896692	1967511	5696	5678.68
08-MN-19	Moonlight	6896999	1967996	5673	5663.24
08-MN-20	Moonlight	6897330	1968583	5743	5735
08-MN-21	Moonlight	6896218	1968245	5570	5545.827
C-02	N/A	6891764	1965936	5324.402	5323.978
C-05	N/A	6899563	1971376	5248.868	5248.905
C-07	N/A	6902310	1970070	5892.622	5892.508
C-08	N/A	6902407	1969076	6015.768	6015.577
C-09	N/A	6900161	1969972	5742.649	5742.613
CM-01	Copper Mountain	6898560	1965450	6305.59	6315.421
CM-02	Copper Mountain	6899327	1965485	6408.34	6405.667
CM-03	Copper Mountain	6900129	1965527	6221.85	6220.064
CM-04	Copper Mountain	6900933	1965564	6013.7	6014.032
CM-05	Copper Mountain	6898964	1965015	6388	6366.764
CM-06	Copper Mountain	6898946	1965090	6373.43	6376.881
CM-07	Copper Mountain	6899746	1965113	6288.94	6284.836
CM-09	Copper Mountain	6897761	1964625	6039.8	6031.006
CM-11	Copper Mountain	6899352	1964720	6299.63	6300.4
CM-12	Copper Mountain	6900153	1964741	6073.37	6081.632
CM-14	Copper Mountain	6898960	1964320	6256.42	6265.628
CM-15	Copper Mountain	6899773	1964313	6068.23	6070.712
CM-17	Copper Mountain	6899382	1963933	6227.73	6227.02
CM-18	Copper Mountain	6900527	1964371	5856.9	5863.054
CM-22	Copper Mountain	6901024	1966890	5933.82	5933.38
CM-23	Copper Mountain	6897776	1965448	6273.03	6268.754
CM-24	Copper Mountain	6899765	1964677	6170	6150.474
CM-25	Copper Mountain	6899115	1963577	6210	6208.207
CM-26	Copper Mountain	6899765	1963577	6030	6032.848
CM-27	Copper Mountain	6900365	1963677	5780	5757.347
CM-28	Copper Mountain	6897731	1963652	5990	5985.063
CM-29	Copper Mountain	6896737	1964479	5958	5951.173

HoleNo	Area	Easting	Northing	Elevation	Revised Elevation
DDH-04A	Osmeyer Ridge	6902365	1967877	5970	5936.553
DDH-04B	Copper Mountain	6900393	1964957	6043	6050.756
MN-1	Moonlight	6896515	1968377	5550	5545.393
MN-10	Moonlight	6896326	1966678	5739.41	5738.64
MN-100	Moonlight	6893981	1970885	5577.44	5577.447
MN-101	Moonlight	6895263	1970364	5583.68	5579.595
MN-102	Moonlight	6896430	1970311	5639.59	5633.662
MN-11	Moonlight	6895938	1965915	5783.03	5786.53
MN-12	Moonlight	6896407	1966313	5820.9	5821.38
MN-13	Moonlight	6896429	1965913	5830.59	5829.632
MN-14	Moonlight	6895500	1968230	5476.02	5472.691
MN-15	Moonlight	6895521	1967462	5510.99	5506.914
MN-16	Moonlight	6895529	1966690	5608.32	5610.633
MN-17	Moonlight	6895523	1965838	5741.68	5739.459
MN-18	Moonlight	6895565	1965041	5781.65	5768.273
MN-19	Moonlight	6894417	1967796	5472.14	5470.21
MN-2	Moonlight	6895283	1964570	5850.02	5834.134
MN-200	Moonlight	6895159	1965482	5636.71	5634.852
MN-201	Moonlight	6895561	1965461	5699.66	5699.35
MN-202	Moonlight	6896450	1965540	5893.51	5896.13
MN-203	Moonlight	6895364	1965694	5694.26	5692.698
MN-204	Moonlight	6895784	1965669	5759.23	5758.118
MN-205	Moonlight	6896249	1965716	5822.2	5827.543
MN-206	Moonlight	6895080	1965878	5617.65	5605.616
MN-207	Moonlight	6894978	1966110	5577.45	5576.107
MN-208	Moonlight	6895366	1966079	5651.64	5660.164
MN-209	Moonlight	6895755	1966074	5753.72	5752.759
MN-210	Moonlight	6896232	1966063	5773.46	5773.117
MN-211	Moonlight	6896626	1966119	5895.05	5895.101
MN-213	Moonlight	6895187	1966310	5610.65	5607.419
MN-214	Moonlight	6895578	1966283	5699.55	5700.15
MN-215	Moonlight	6895372	1966500	5632.71	5631.762
MN-216	Moonlight	6895829	1966451	5662.47	5667.722
MN-217	Moonlight	6896226	1966466	5757.42	5759.687
MN-218	Moonlight	6896582	1966495	5837.57	5843.917
MN-219	Moonlight	6897022	1966532	5928.65	5939.402
MN-22	Moonlight	6896272	1969096	5522.85	5518.543
MN-220	Moonlight	6895736	1966828	5599.85	5604.029
MN-221	Moonlight	6896134	1966899	5657.21	5666.579
MN-222	Moonlight	6896534	1966861	5748.78	5749.538
MN-223	Moonlight	6896954	1966941	5837.33	5856.991
MN-224	Moonlight	6897318	1966939	5954.83	5957.691
MN-225	Moonlight	6895754	1967225	5551.03	5550.013
MN-226	Moonlight	6896147	1967274	5592.17	5594.179
MN-227	Moonlight	6896501	1967272	5667.64	5669.252
MN-228	Moonlight	6896926	1967328	5769.85	5773.139

HoleNo	Area	Easting	Northing	Elevation	Revised Elevation
MN-229	Moonlight	6897314	1967335	5861.91	5866.02
MN-230	Moonlight	6895786	1967637	5527.42	5525.814
MN-231	Moonlight	6896163	1967649	5568.25	5570.557
MN-232	Moonlight	6896519	1967590	5635.15	5638.127
MN-233	Moonlight	6896898	1967705	5693.8	5694.058
MN-234	Moonlight	6897304	1967735	5766.39	5769.329
MN-235	Moonlight	6895745	1968064	5495.18	5494.516
MN-236	Moonlight	6896119	1968063	5537.33	5535.32
MN-237	Moonlight	6896501	1968071	5607.31	5606.617
MN-238	Moonlight	6896830	1968150	5598.14	5604.579
MN-239	Moonlight	6897213	1968194	5702.58	5697.347
MN-24	Moonlight	6896606	1969177	5559.23	5554.325
MN-240	Moonlight	6897389	1968440	5738.68	5735.636
MN-241	Moonlight	6895716	1968459	5477.18	5474.87
MN-242	Moonlight	6896108	1968481	5501.23	5498.846
MN-243	Moonlight	6896372	1968525	5548.5	5544.334
MN-244	Moonlight	6896809	1968592	5632.15	5631.376
MN-245	Moonlight	6897160	1968565	5704.48	5702.043
MN-246	Moonlight	6897537	1968664	5736.91	5733.083
MN-247	Moonlight	6895688	1968883	5462.52	5464.221
MN-248	Moonlight	6896079	1968934	5493.36	5490.43
MN-249	Moonlight	6896479	1968947	5544.15	5539.386
MN-25	Moonlight	6896565	1968741	5575.29	5572.348
MN-250	Moonlight	6896774	1969005	5590.26	5589.796
MN-251	Moonlight	6897154	1969033	5654.75	5657.066
MN-252	Moonlight	6897485	1969058	5651.38	5649.799
MN-253	Moonlight	6897354	1969224	5622.87	5620.621
MN-254	Moonlight	6895695	1969218	5499.28	5495.96
MN-255	Moonlight	6896090	1969254	5505.42	5500.682
MN-26	Moonlight	6896644	1967933	5625.5	5625.758
MN-27	Moonlight	6896683	1967531	5678.85	5675.878
MN-28	Moonlight	6896728	1967102	5756.59	5758.325
MN-29	Moonlight	6896777	1966731	5833.32	5832.98
MN-3	Moonlight	6896376	1964532	5959.64	5956.543
MN-30	Moonlight	6896853	1966287	5950.85	5955.798
MN-301	Moonlight	6895862	1965345	5779.43	5781.873
MN-302	Moonlight	6896248	1965361	5849.5	5846.872
MN-303	Moonlight	6897772	1968862	5657.96	5656.361
MN-304	Moonlight	6897973	1968670	5646	5643.291
MN-305	Moonlight	6897783	1968464	5695.07	5688.354
MN-306	Moonlight	6897593	1968243	5746.24	5743.65
MN-307	Moonlight	6897634	1967836	5792.05	5791.775
MN-308	Moonlight	6897827	1968049	5731.47	5730.341
MN-309	Moonlight	6897889	1967612	5824.79	5824.657
MN-31	Moonlight	6896862	1965988	5936.48	5912.036
MN-310	Moonlight	6897665	1969260	5635.53	5635.411

HoleNo	Area	Easting	Northing	Elevation	Revised Elevation
MN-311	Moonlight	6898169	1969281	5575.16	5573.228
MN-312	Moonlight	6898175	1968884	5600.45	5597.346
MN-313	Moonlight	6898197	1968482	5647.67	5647.459
MN-314	Moonlight	6898228	1968061	5729.9	5727.742
MN-315	Moonlight	6898250	1967638	5829.5	5830.481
MN-32	Moonlight	6897038	1969589	5583.94	5579.737
MN-321	Moonlight	6898560	1968900	5544.06	5540.403
MN-323	Moonlight	6898623	1968070	5713.19	5710.809
MN-325	Moonlight	6898671	1967275	5914.6	5913.333
MN-329	Moonlight	6895297	1969211	5480.11	5478.106
MN-33	Moonlight	6896936	1969187	5593.91	5592.344
MN-330	Moonlight	6895295	1968807	5449.69	5445
MN-331	Moonlight	6895302	1968428	5461.88	5458.388
MN-332	Moonlight	6895354	1968040	5473.93	5470
MN-333	Moonlight	6895321	1967605	5489.79	5489.324
MN-334	Moonlight	6895288	1967204	5549.86	5545.814
MN-335	Moonlight	6895320	1966820	5583.87	5583.919
MN-336	Moonlight	6896049	1965151	5844.63	5852.239
MN-337	Moonlight	6896449	1965167	5925.06	5932.197
MN-338	Moonlight	6896664	1965356	5960.64	5964.15
MN-339	Moonlight	6896634	1965743	5898.01	5888.768
MN-34	Moonlight	6896964	1968807	5628.22	5667.456
MN-340	Moonlight	6896865	1965553	6002.64	6010.077
MN-345	Moonlight	6897516	1966731	6021.7	6023.862
MN-346	Moonlight	6897503	1967533	5851.33	5853.218
MN-347	Moonlight	6897698	1967332	5911.88	5912.122
MN-348	Moonlight	6897723	1966953	6024.73	6024.094
MN-35	Moonlight	6897066	1967977	5675.32	5672.279
MN-353	Moonlight	6898045	1967867	5755.43	5759.346
MN-354	Moonlight	6898075	1967446	5872.25	5869.268
MN-359	Moonlight	6898441	1967851	5762.21	5761.746
MN-36	Moonlight	6897094	1967526	5757.15	5760.519
MN-37	Moonlight	6897118	1967130	5867.02	5871.054
MN-372	Moonlight	6895130	1967397	5517.34	5513.171
MN-374	Moonlight	6895158	1966616	5588.49	5587.448
MN-376	Moonlight	6898138	1969690	5533.5	5520.32
MN-377	Moonlight	6896251	1968271	5535.26	5541.395
MN-378	Moonlight	6896908	1968420	5623.15	5620.052
MN-38	Moonlight	6897108	1966749	5904.45	5905.244
MN-4	Moonlight	6896318	1968716	5530.28	5529.951
MN-40	Moonlight	6895849	1969440	5542.04	5539.648
MN-403	Moonlight	6900165	1967577	5800	5830.223
MN-41	Moonlight	6895885	1969078	5488.49	5480.616
MN-410	Moonlight	6897420	1970025	5576.91	5573.209
MN-411	Moonlight	6898091	1970483	5495.81	5494.908
MN-412	Moonlight	6898894	1970919	5400.83	5396.009

HoleNo	Area	Easting	Northing	Elevation	Revised Elevation
MN-413	Moonlight	6898914	1970127	5429.4	5428.357
MN-414	Moonlight	6898949	1969316	5486.18	5489.414
MN-416	Moonlight	6895837	1969849	5574.21	5573.928
MN-418	Moonlight	6894260	1969279	5437.86	5433.801
MN-42	Moonlight	6895900	1968656	5483.94	5479.485
MN-421	Moonlight	6898059	1971292	5362.65	5358.303
MN-424	Moonlight	6895531	1967030	5561.14	5559.234
MN-43	Moonlight	6895936	1967854	5526.54	5523.141
MN-44	Moonlight	6895967	1967457	5558.33	5556.578
MN-45	Moonlight	6895956	1967036	5610.91	5615.213
MN-46	Moonlight	6895946	1966646	5664.36	5665.346
MN-47	Moonlight	6896027	1966254	5711.9	5714.376
MN-48	Moonlight	6896054	1965555	5796.73	5791.188
MN-49	Moonlight	6897441	1969628	5600.62	5599.543
MN-5	Moonlight	6896918	1968225	5614.57	5615.391
MN-50	Moonlight	6897330	1968819	5714.07	5713.831
MN-500	Moonlight	6892756	1968496	5414.94	5409.112
MN-501	Moonlight	6893637	1968525	5412.07	5408.615
MN-502	Moonlight	6899610	1968705	5605.71	5602.977
MN-504	Moonlight	6893166	1968754	5426.33	5424.991
MN-505	Moonlight	6897337	1968587	5737.9	5735
MN-506	Moonlight	6897146	1968748	5709.53	5709.002
MN-507	Moonlight	6897002	1968610	5675.36	5673.43
MN-508	Moonlight	6897113	1968365	5671.83	5669.268
MN-509	Moonlight	6893107	1967833	5400.86	5393.403
MN-51	Moonlight	6897437	1968024	5760.7	5761.804
MN-512	Moonlight	6897163	1969395	5591.47	5590.075
MN-513	Moonlight	6894774	1965598	5648.48	5647.181
MN-514	Moonlight	6894658	1966039	5606.14	5598.18
MN-515	Moonlight	6894874	1966420	5549.96	5544.573
MN-516	Moonlight	6893557	1966387	5656.58	5656.689
MN-517	Moonlight	6894980	1967070	5541.73	5536.26
MN-518	Moonlight	6894919	1968127	5452.74	5450.94
MN-519	Moonlight	6896302	1969489	5546.13	5543.533
MN-519A	Moonlight	6896312	1969494	5546.16	5544.47
MN-52	Moonlight	6897509	1967139	5955.41	5958.998
MN-520	Moonlight	6898422	1969184	5561.08	5557.312
MN-521	Moonlight	6897309	1966279	6028.13	6030.158
MN-522	Moonlight	6892487	1968345	5423.64	5418.176
MN-523	Moonlight	6892533	1968696	5441.89	5437.387
MN-524	Moonlight	6893163	1968380	5424.12	5421.475
MN-525	Moonlight	6892738	1968133	5409.3	5406.456
MN-526	Moonlight	6898646	1969690	5498.48	5495.299
MN-527	Moonlight	6898254	1969924	5497.04	5494.865
MN-528	Moonlight	6899457	1969603	5542.24	5544.891
MN-529	Moonlight	6897764	1970103	5550.87	5547.105

HoleNo	Area	Easting	Northing	Elevation	Revised Elevation
MN-530	Moonlight	6898480	1970366	5491.14	5487.631
MN-532	Moonlight	6892215	1968627	5420	5439.264
MN-533	Moonlight	6892240	1968247	5420	5435
MN-6	Moonlight	6895898	1968268	5507.92	5505.328
MN-7	Moonlight	6896317	1967820	5569.34	5572.086
MN-8	Moonlight	6896312	1967374	5614.49	5614.731
MN-9	Moonlight	6896326	1967082	5658.64	5660.725
SH-001	N/A	6895291	1971001	5692.198	5692.235
SH-002	N/A	6896279	1971281	5609.622	5609.532
SH-003	N/A	6897098	1971098	5468.479	5468.501
SH-005	N/A	6891106	1968719	5456.976	5457.03
23MRC16	Moonlight	6896333	1969240	5520.5	5515.172
23MRC17	Moonlight	6895713	1968920	5471.5	5466.88
23MRC18	Moonlight	6896264	1968687	5529.6	5519.664
23MRC19	Moonlight	6896473	1968409	5549.4	5545.149
23MRC20	Moonlight	6896276	1967799	5570.2	5570.049
23MRC21	Moonlight	6896464	1968104	5602.7	5600.765
23MRC22	Moonlight	6895969	1967329	5573.7	5564.819
23MRC23	Moonlight	6896322	1967406	5621.9	5614.62
23MRC24	Moonlight	6897036	1969155	5627.8	5613.233
23MRC25	Moonlight	6896999	1968463	5650	5645.791
23MRC26	Moonlight	6896879	1967695	5696.6	5691.946
23MRC27	Moonlight	6897715	1968760	5680.5	5678.234
23MRC29	Moonlight	6897481	1967526	5849	5851.272
23MRC30	Moonlight	6896919	1967322	5785.4	5772.902

A number of drill holes at Engels did not match topography, but review of aerial imagery for the locations showed considerable terrain disturbance, indicating that the current ground elevation may differ from what it was at the time of drilling, so those drill hole collar elevations were not adjusted.

14.2 Estimation Areas

The project was divided into three areas: Northwest, Northeast, and South. The Northwest area includes Moonlight, Copper Mountain, and Gossan Ridge. The Northeast area includes Engels and Lamb's Ridge. The South area includes Superior.

Within each main area, grade shells representing mineralization equal to or exceeding 0.1% Cu were generated. Material inside the grade shell was tagged as "Inside Grade Shell Domain," while all other material was tagged as "Outside Grade Shell Domain."

14.3 Grade Capping

Log probability plots for Cu within each project area were evaluated to determine if grade capping was prudent. Locations of discontinuity on the log plot indicate the presence of outlier data and suggest that capping should be performed. The GRE QP determined that the following Cu grade capping should be applied:

- Engels – 17.783%
- Moonlight, Superior, Copper Mountain, Osmeyer Ridge – 2.818%

The grade capping resulted in 43 assay values being capped, four from Engels, 20 from Moonlight, and 19 from Superior.

No Ag or Au grade capping was performed.

14.4 Assay Statistics

Assay statistics (capped for Cu, uncapped for Ag and Au) are summarized in Table 14-2.

Table 14-2: Moonlight-Superior Project Assay Statistics

Area	Domain	Count	Mean	Std. Dev.	CV	Minimum	Maximum
Copper (capped)							
Northwest	Inside Grade Shell	8,152	0.256	0.266	1.037	0.002	2.818
	Outside Grade Shell	6,861	0.070	0.111	1.59	0.001	2.818
Northeast	Inside Grade Shell	2,054	0.80	1.56	1.96	0.007	17.783
	Outside Grade Shell	2,969	0.185	0.626	3.38	0.001	17.783
South	Inside Grade Shell	6,986	0.263	0.286	1.09	0.003	2.818
	Outside Grade Shell	3,956	0.089	0.111	1.24	0.002	2.35
Silver (uncapped)							
Northwest	Inside Grade Shell	2,138	3.61	4.43	1.23	0.02	35.48
	Outside Grade Shell	688	1.47	1.51	1.03	0.06	12
Northeast	Inside Grade Shell	1,401	10.39	20.79	2.00	0.2	223.87
	Outside Grade Shell	944	4.19	9.34	2.23	0.2	148
South	Inside Grade Shell	761	2.64	2.68	1.01	0.5	27.6
	Outside Grade Shell	370	1.68	1.62	0.96	0.5	14.2
Gold (uncapped)							
Northwest	Inside Grade Shell	846	0.034	0.16	4.54	0.005	1.778
	Outside Grade Shell	145	0.049	0.210	4.25	0.005	1.72
Northeast	Inside Grade Shell	1,002	0.08	0.21	2.58	0.003	2.82
	Outside Grade Shell	765	0.044	0.137	3.07	0.003	2.818
South	Inside Grade Shell	873	0.013	0.020	1.49	0.001	0.224
	Outside Grade Shell	568	0.008	0.018	2.08	0.001	0.224

Std. Dev = standard deviation

CV = coefficient of variation

14.5 Compositing

The average length of the drill hole samples is 8.7 feet (2.65 meters). The most frequent sample length is 10 feet (3 meters), with 67% of the samples at this length followed by 5-foot (1.5-meter) samples, which represent 13% of the sample set. Considering the assay interval length statistics, a down-hole composite length of 10 feet was selected. Composite statistics are summarized in Table 14-3.

Table 14-3: Moonlight-Superior Project Composite Statistics

Area	Domain	Count	Mean	Std. Dev.	CV	Minimum	Maximum
Copper							
Northwest	Inside Grade Shell	7,207	0.256	0.255	0.994	0.0025	2.818
	Outside Grade Shell	6,408	0.070	0.110	1.57	0.001	2.818
Northeast	Inside Grade Shell	1,561	0.80	1.38	1.73	0.008	16.22
	Outside Grade Shell	2,439	0.186	0.549	2.95	0.001	10.895
South	Inside Grade Shell	5,782	0.264	0.270	1.03	0.003	2.818
	Outside Grade Shell	3,471	0.089	0.10	1.19	0.002	1.938
Silver							
Northwest	Inside Grade Shell	1,304	3.57	3.81	1.07	0.096	35.48
	Outside Grade Shell	434	1.41	1.29	1.67	0.53	7.48
Northeast	Inside Grade Shell	994	10.17	17.67	1.73	0.2	202.71
	Outside Grade Shell	707	4.00	7.56	1.89	0.2	90.94
South	Inside Grade Shell	397	2.60	2.13	0.82	0.5	16.51
	Outside Grade Shell	208	1.62	1.27	0.78	0.5	9.56
Gold							
Northwest	Inside Grade Shell	546	0.033	0.13	3.91	0.005	1.32
	Outside Grade Shell	102	0.044	0.187	4.22	0.005	1.315
Northeast	Inside Grade Shell	617	0.08	.015	1.90	0.0037	1.59
	Outside Grade Shell	473	0.044	0.101	2.27	0.004	1.228
South	Inside Grade Shell	439	0.013	0.016	1.22	0.001	0.216
	Outside Grade Shell	305	0.008	0.015	1.79	0.001	0.216

14.6 Variography

Variography analysis of copper, silver, and gold grades was completed for both estimation domains within each project area. Variography describes how similar sample grades are as a function of distance and direction. This is performed by comparing the orientation and distance used in the estimation to the variability of other samples of similar relative direction and distance. The spherical variograms were constructed using a “Correlogram” method of organizing the variance pairs. The nugget was determined from the down-hole correlograms, and the total sill was determined from global correlograms. Correlogram parameters for each domain are presented in Table 14-4, and the copper correlograms are illustrated in Figure 14-1 through Figure 14-6.

Table 14-4: Moonlight-Superior Project Variogram Parameters

<table>
<tr><th>Area</th><th>Domain</th><th>Structure</th><th>Structure Type</th><th>Pitch</th><th>Dip</th><th>Dip Azimuth</th><th>Nugget</th><th>Sill</th><th>Major Range</th><th>Semi-Major Range</th><th>Minor Range</th></tr>
<tr><td colspan="12">Copper</td></tr>
<tr><td rowspan="4">Northwest</td><td rowspan="2">Inside Grade Shell</td><td>1</td><td>Spherical</td><td rowspan="2">43</td><td rowspan="2">15</td><td rowspan="2">290</td><td rowspan="2">0.37</td><td>0.32</td><td>270</td><td>84</td><td>31</td></tr>
<tr><td>2</td><td>Spherical</td><td>0.31</td><td>380</td><td>330</td><td>150</td></tr>
<tr><td rowspan="2">Outside Grade Shell</td><td>1</td><td>Spherical</td><td rowspan="2">167.5</td><td rowspan="2">16.5</td><td rowspan="2">308.9</td><td rowspan="2">0.40</td><td>0.51</td><td>260</td><td>150</td><td>30</td></tr>
<tr><td>2</td><td>Spherical</td><td>0.08</td><td>340</td><td>300</td><td>100</td></tr>
</table>

Area	Domain	Structure	Structure Type	Pitch	Dip	Dip Azimuth	Nugget	Sill	Major Range	Semi-Major Range	Minor Range
Northeast	Inside Grade Shell	1	Spherical	15	25	305	0.14	0.68	70	42	40
		2	Spherical					0.24	320	300	55
	Outside Grade Shell	1	Spherical	120	5	185	0.05	0.58	40	32	20
		2	Spherical					0.40	225	230	130
South	Inside Grade Shell	1	Spherical	165	30	5	0.24	0.24	100	120	45
		2	Spherical					0.30	500	375	150
	Outside Grade Shell	1	Spherical	75	320	30	0.22	0.67	90	165	50
		2	Spherical					0.11	600	500	60
Silver											
Northwest	Inside Grade Shell	1	Spherical	43	15	290	0.19	0.57	55	93	30
		2	Spherical					0.25	320	305	140
	Outside Grade Shell	1	Spherical	167.5	16.5	308.9	0.33	0.30	160	65	50.3
		2	Spherical					0.37	310	300	150
Northeast	Inside Grade Shell	1	Spherical	15	25	305	0.14	0.66	50	45	45
		2	Spherical					0.29	270	300	65
	Outside Grade Shell	1	Spherical	120	5	185	0	0.66	34	135	22
		2	Spherical					0.37	340	300	150
South	Inside Grade Shell	1	Spherical	165	30	5	0.15	0.66	33	105	47
		2	Spherical					0.20	305	305	50
	Outside Grade Shell	1	Spherical	75	30	320	0.05	0.55	70	285	40
		2	Spherical					0.41	590	510	110
Gold											
Northwest	Inside Grade Shell	1	Spherical	43	15	290	0	0.82	105	93	33
		2	Spherical					0.17	315	300	50
	Outside Grade Shell	1	Spherical	167.5	16.5	308.9	0	0.33	160	45	29
		2	Spherical					0.68	350	230	32
Northeast	Inside Grade Shell	1	Spherical	15	25	305	0.26	0.35	60	40	18
		2	Spherical					0.40	220	250	50
	Outside Grade Shell	1	Spherical	120	5	185	0.30	0.52	170	160	23.4
		2	Spherical					0.21	300	300	110
South	Inside Grade Shell	1	Spherical	165	30	5	0.33	0.50	42	105	37
		2	Spherical					0.18	300	310	50
	Outside Grade Shell	1	Spherical	75	320	30	0.51	0.33	145	270	40
		2	Spherical					0.18	800	510	90

Figure 14-1: Moonlight-Superior Project Copper Correlogram Northwest Area Inside Grade Shell

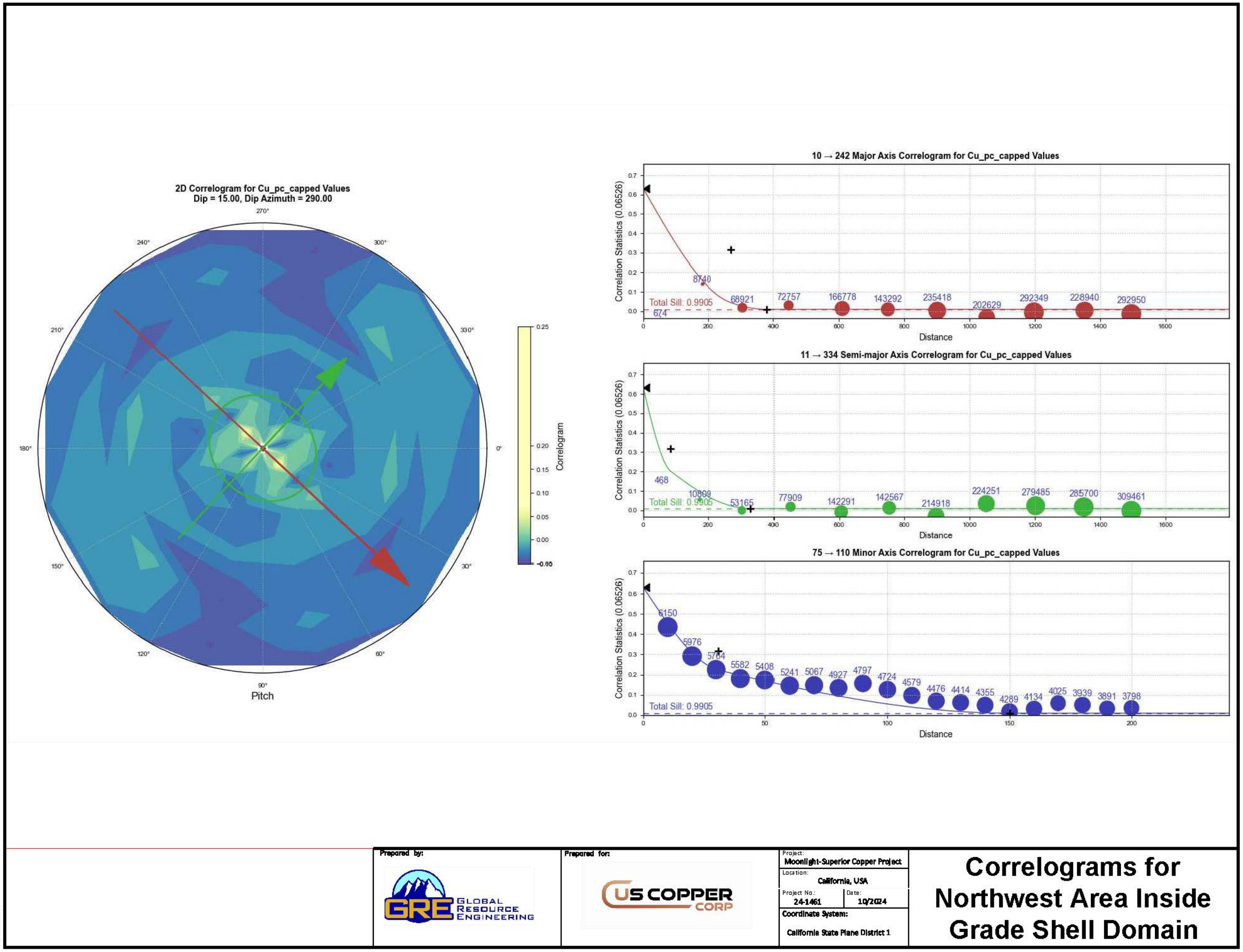


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Figure 14-2: Moonlight-Superior Project Copper Correlogram Northeast Area Inside Grade Shell

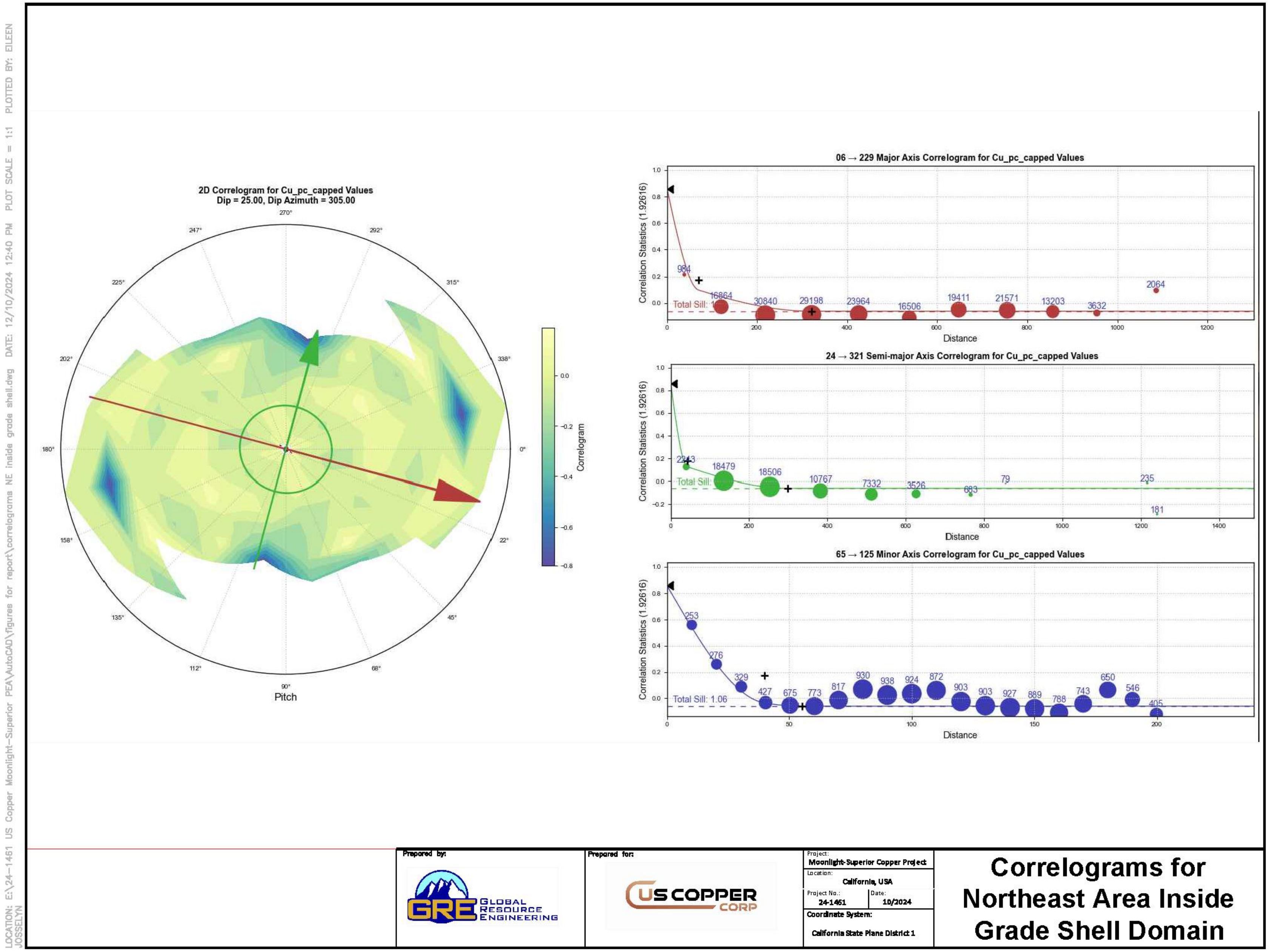

Figure 14-3: Moonlight-Superior Project Copper Correlogram South Area Inside Grade Shell

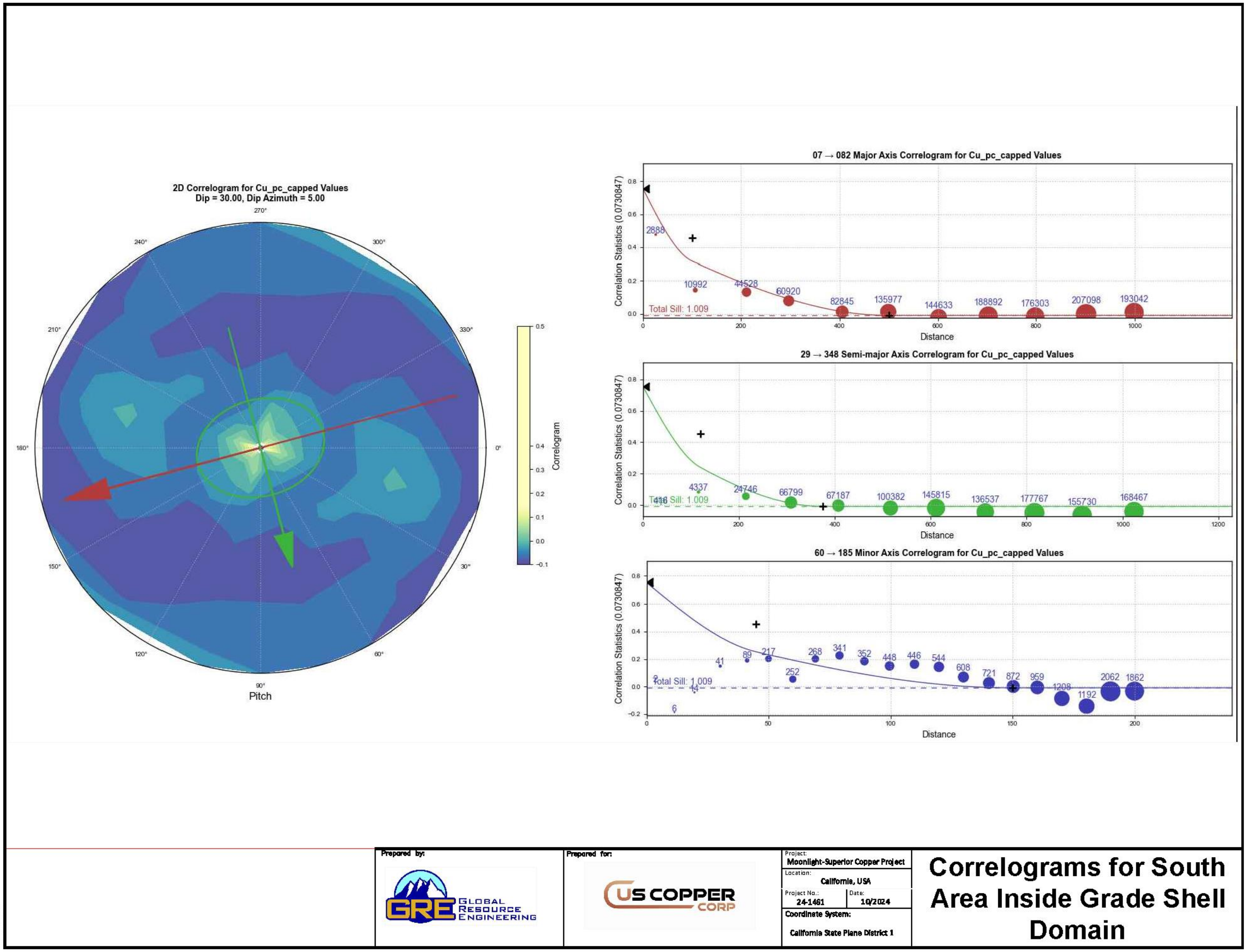

Figure 14-4: Moonlight-Superior Project Copper Correlogram Northwest Area Outside Grade Shell

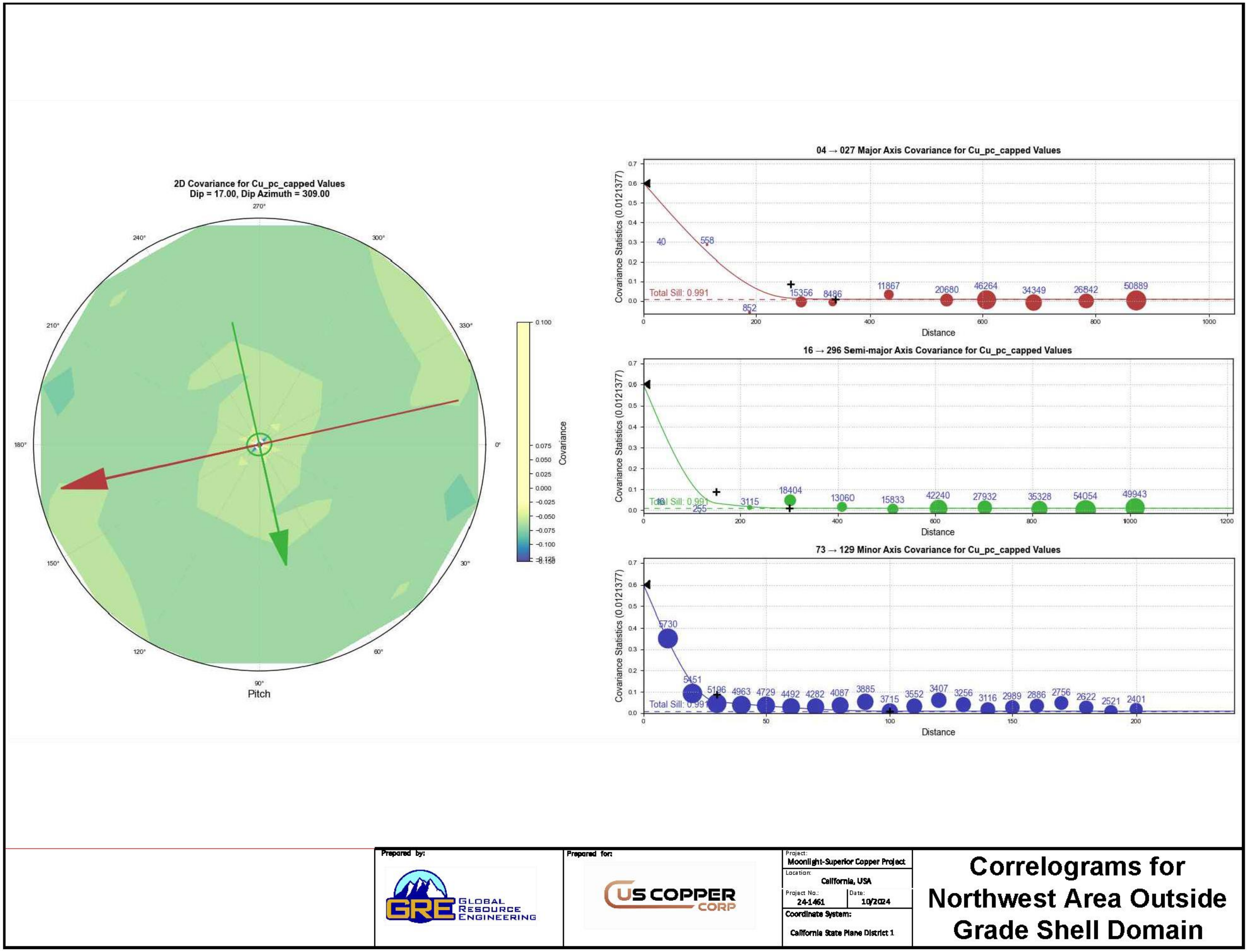

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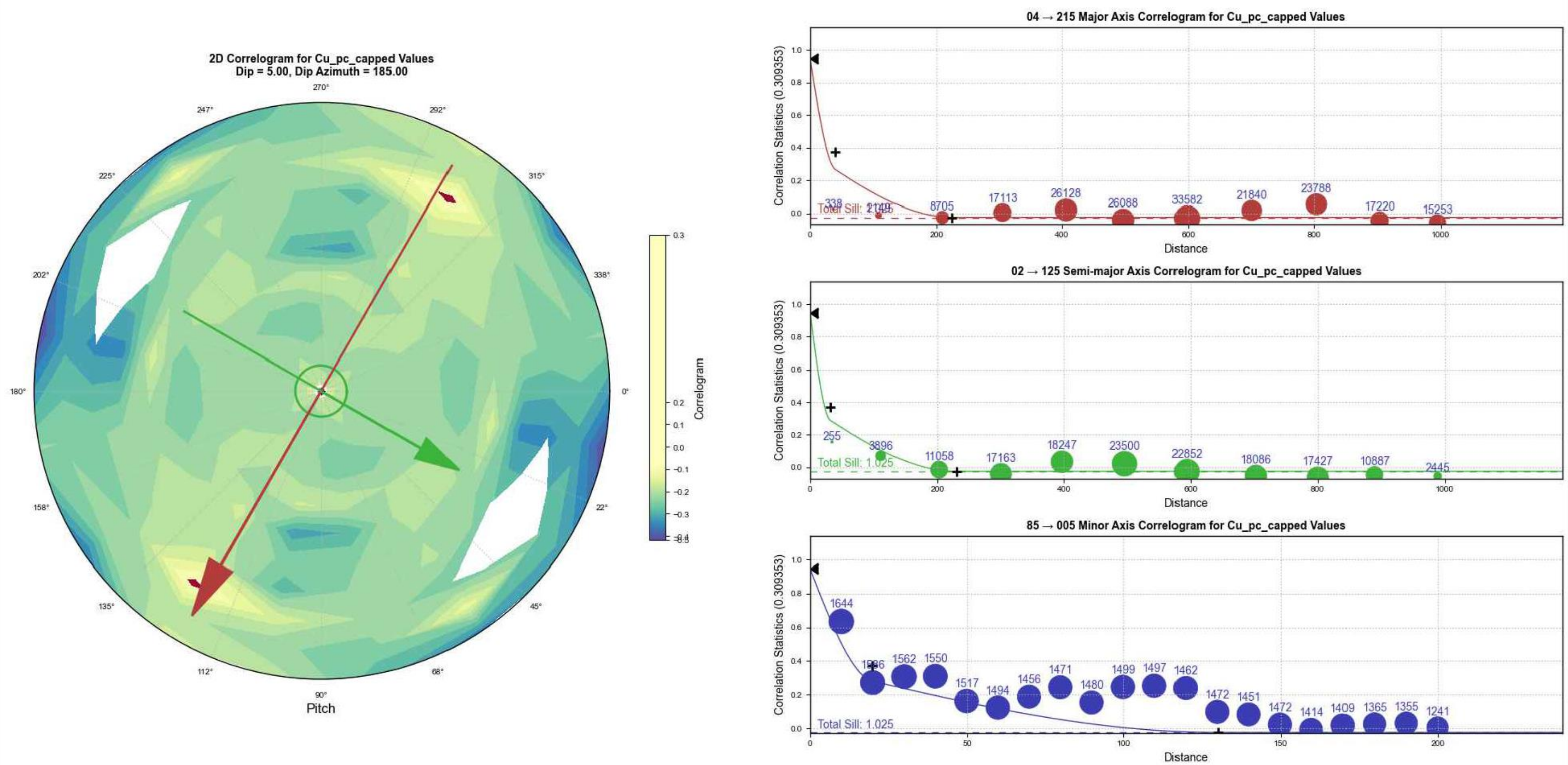
Figure 14-5: Moonlight-Superior Project Copper Correlogram Northeast Area Outside Grade Shell



Prepared by: GRE Global Resource Engineering

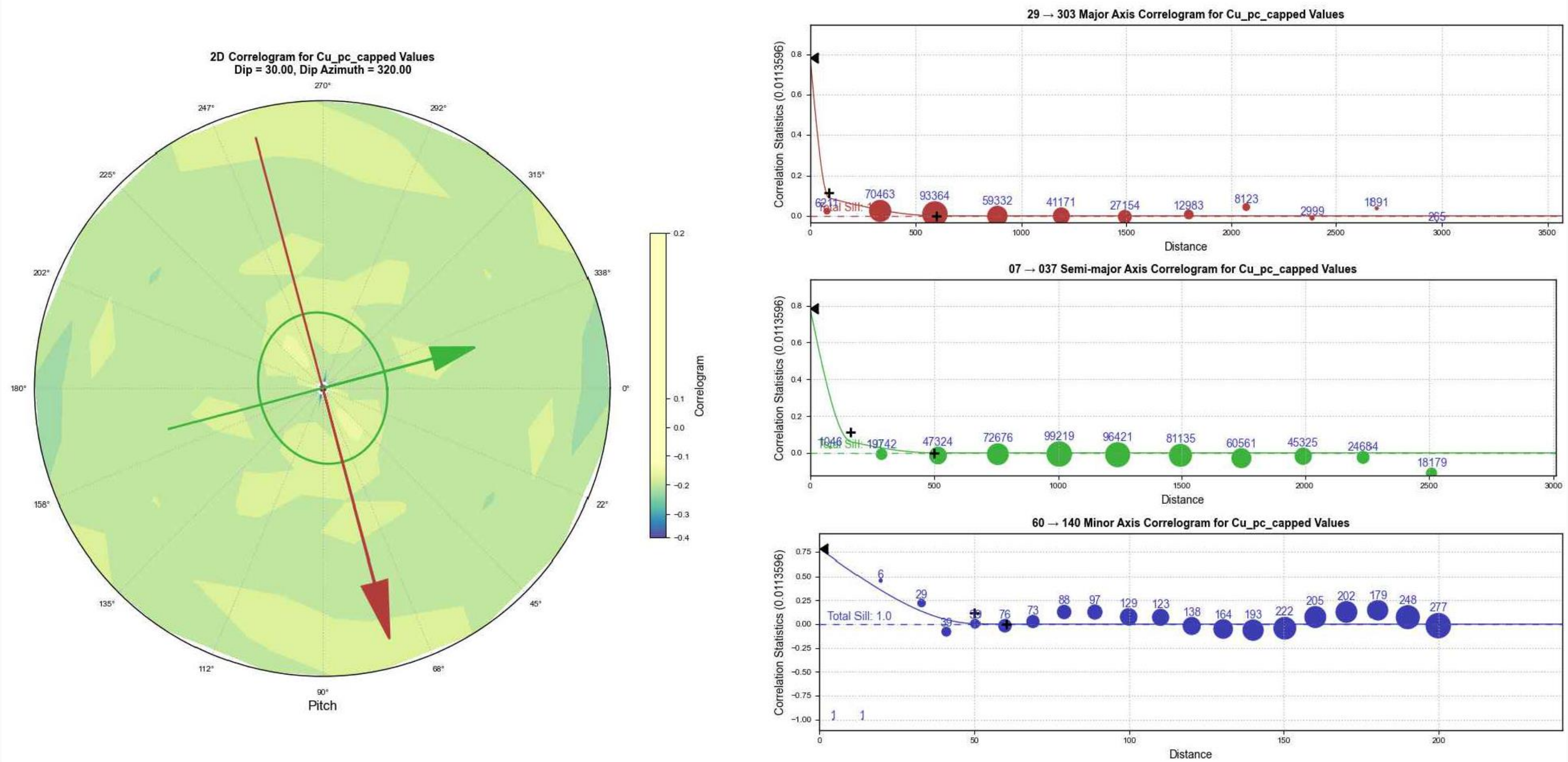
Prepared for: US Copper Corp

Project: Moonlight-Superior Copper Project	
Location: California, USA	
Project No.: 24-1461	Date: 10/2024
Coordinate System: California State Plane District 1	

Correlograms for Northeast Area Outside Grade Shell Domain

LOCATION: E:\24-1461 US Copper Moonlight-Superior PEA\AutoCAD\figures for report\correlograms NE outside grade shell.dwg DATE: 12/10/2024 12:41 PM PLOT SCALE = 1:1 PLOTTED BY: EILEEN JOSSELYN

Figure 14-6: Moonlight-Superior Project Copper Correlogram South Area Outside Grade Shell



Prepared by: GRE Global Resource Engineering

Prepared for: US Copper Corp

Project: Moonlight-Superior Copper Project

Location: California, USA

Project No.: 24-1461

Date: 10/2024

Coordinate System: California State Plane District 1

Correlograms for South Area Outside Grade Shell Domain

LOCATION: E:\24-1461 US Copper Moonlight-Superior PEA\AutoCAD\figures for report\correlograms S outside grade shell.dwg DATE: 12/10/2024 12:48 PM PLOT SCALE = 1:1 PLOTTED BY: EILEEN JOSSELYN

14.7 Oxide/Sulfide Boundary

Copper oxide and transitional domains were defined in the project by a combination of three copper grades in the assay table: Cupc (total copper %), Cu_AL_AA_H2SO4_pc (copper % by sulfuric acid solution), and Cu_CN_AA_CN_pc (copper % by cyanide solution). Two indicators were calculated: percent total acid soluble copper and percent total acid soluble plus cyanide soluble (pct_total_acid_solu and pct_acid_plus_cn). Indicator equations:

pct_total_acid_solu = min(1, [Cu_AL_AA_H2SO4_pc]/ [Cu_pc])

The result is the proportion of acid soluble copper in the total copper in each assay interval.

pct_acid_plus_cn= min(1, [Cu_CN_AA_CN_pc]/ [Cu_pc]) + pct_total_acid_solu

The result is the proportion of cyanide soluble copper and acid soluble copper in the total copper in each assay interval.

A volume was estimated for each of these indicators, Oxide and Transitional materials. Oxide was created in Leapfrog using an Indicator RBF Interpolant based on pct_total_acid_solu intervals greater than 0.5, or 50% of the copper assay that is acid soluble. Transitional volume was created in Leapfrog using an Indicator RBF Interpolant based on pct_acid_plus_cn intervals greater than 0.5, or 50% of the copper assay that is both acid soluble and cyanide soluble. The oxide volume took precedence over the transitional volume, and both were evaluated in the block model to determine which blocks are oxide and transitional.

14.8 Block Model

Three block models were created based on the definitions shown in Table 14-5, one for each project area. The block model origin coordinates are represented by the minimum easting "X", minimum northing "Y" and minimum "Z". The models were not rotated in any direction. Based on the anticipated mining methods, a block size of 40 feet × 40 feet × 40 feet was selected.

Table 14-5: Moonlight-Superior Project Block Model Definitions for Moonlight-Superior

Parameter	X	Y	Z
Northwest Area			
Origin	6,890,600	1,962,900	6,500
Block Size (feet)	40	40	40
No. Blocks	275	233	63
Boundary Size	11,000	9,320	2,520
Northeast Area			
Origin	6,907,040	1,963,000	6,600
Block Size (feet)	40	40	40
No. Blocks	220	198	65
Boundary Size	8,800	7,920	2,600
South Area			
Origin	6,901,500	1,952,260	6,800
Block Size (feet)	40	40	40
No. Blocks	208	275	85
Boundary Size	8,320	11,000	3,400

14.9 Estimation Methodology

Copper, silver, and gold grades were estimated using Inverse Distance squared (ID2). Domain estimation parameters are shown in Table 14-6. The estimate for each domain incorporated two passes, and all estimations required a minimum of five samples, a maximum of 12 samples, and a maximum of four samples per hole.

Table 14-6: Moonlight-Superior Project Estimation Search Parameters

Area	Domain	Estimate Pass	Ellipsoid Ranges (ft)		
			Maximum	Intermediate	Minimum
Copper					
Northwest	Inside Grade Shell	First Pass	380	300	150
		Second Pass	570	495	225
	Outside Grade Shell	First Pass	340	300	100
		Second Pass	680	600	150
Northeast	Inside Grade Shell	First Pass	320	300	55
		Second Pass	480	450	82.5
	Outside Grade Shell	First Pass	225	230	130
		Second Pass	450	460	195
South	Inside Grade Shell	First Pass	500	375	150
		Second Pass	750	562.5	225
	Outside Grade Shell	First Pass	600	500	60
		Second Pass	1200	1000	90
Silver					
Northwest	Inside Grade Shell	First Pass	320	305	140
		Second Pass	480	457.5	210
	Outside Grade Shell	First Pass	310	300	150
		Second Pass	620	600	225
Northeast	Inside Grade Shell	First Pass	270	300	65
		Second Pass	405	450	97.5
	Outside Grade Shell	First Pass	340	300	150
		Second Pass	680	600	225
South	Inside Grade Shell	First Pass	305	305	50
		Second Pass	457.5	457.5	75
	Outside Grade Shell	First Pass	590	510	110
		Second Pass	1,180	1,020	165
Gold					
Northwest	Inside Grade Shell	First Pass	315	300	50
		Second Pass	475.5	450	75
	Outside Grade Shell	First Pass	350	230	32
		Second Pass	700	460	67.5
Northeast	Inside Grade Shell	First Pass	220	250	50
		Second Pass	330	375	75
	Outside Grade Shell	First Pass	300	300	110
		Second Pass	600	600	165
South	Inside Grade Shell	First Pass	300	310	50
		Second Pass	450	465	75

Area	Domain	Estimate Pass	Ellipsoid Ranges (ft)		
			Maximum	Intermediate	Minimum
South	Outside Grade Shell	First Pass	800	510	90
		Second Pass	1,600	1,020	135

14.10 Validation

The GRE QP used several methods to validate the results of the estimation method. The combined evidence from these validation methods verifies the ID estimation model results.

14.10.1 Statistical Comparison

Nearest Neighbor (NN) and Ordinary Kriging (OK) models were run to serve as comparisons with the estimated results from the ID2 method. Descriptive statistics for the ID2 method along with those for the NN and OK estimates as well as drill hole composites are shown by domain in Table 14-7. The estimate means for the global population as well as the means for the estimation domains are similar, suggesting the ID2 estimate is not biased or overestimating the grades. The reduction in mean, CV, and maximum from composites to the ID2 estimate shows an appropriate amount of smoothing.

Table 14-7: Moonlight-Superior Project Model Comparison Descriptive Statistics by Domain

Area	Domain	Estimate	Count	Mean	Std. Dev.	CV	Minimum	Maximum
Copper								
Northwest	Inside Grade Shell	Composite	7,205	0.256	0.254	0.994	0.003	2.818
		NN	171,002	0.230	0.201	0.876	0.006	2.818
		OK	119,310	0.234	0.108	0.46	0.023	1.50
		ID2	119,310	0.237	0.120	0.51	0.009	2.01
	Outside Grade Shell	Composite	6,407	0.07	0.11	1.57	0.001	2.818
		NN	568,391	0.056	0.099	1.78	0.001	2.818
		OK	219,272	0.063	0.041	0.650	0.003	0.635
		ID2	219,272	0.061	0.043	0.71	0.001	0.854
Northeast	Inside Grade Shell	Composite	1,561	0.801	1.38	1.73	0.008	16.22
		NN	16,484	0.235	0.356	1.51	0.01	14.72
		OK	6,230	0.299	0.339	1.13	0.077	6.67
		ID2	6,230	0.311	0.372	1.20	0.077	6.74
	Outside Grade Shell	Composite	2,437	0.186	0.549	2.95	0.001	10.90
		NN	134,998	0.072	0.21	2.94	0.001	8.48
		OK	46,560	0.115	0.17	1.48	0.029	2.73
		ID2	46,560	0.106	0.15	1.42	0.004	4.27
South	Inside Grade Shell	Composite	5,782	0.264	0.270	1.03	0.003	2.818
		NN	81,591	0.217	0.183	0.842	0.003	2.818
		OK	60,567	0.234	0.111	0.474	0.049	2.23
		ID2	60,567	0.238	0.122	0.514	0.026	2.55
	Outside Grade Shell	Composite	3,471	0.089	0.107	1.19	0.002	1.938
		NN	332,596	0.076	0.082	1.07	0.002	1.938
		OK	103,438	0.096	0.045	0.470	0.003	0.705
		ID2	103,458	0.092	0.049	0.532	0.003	0.979

Area	Domain	Estimate	Count	Mean	Std. Dev.	CV	Minimum	Maximum
Silver								
Northwest	Inside Grade Shell	Composite	1,303	3.57	3.81	1.07	0.096	35.48
		NN	65,837	3.13	4.00	1.28	0.096	35.48
		OK	33,009	3.72	2.08	0.558	0.354	21.33
		ID2	33,009	3.63	2.01	0.556	0.365	22.64
	Outside Grade Shell	Composite	433	1.40	1.29	0.92	0.096	7.48
		NN	61,826	1.32	1.40	1.06	0.096	7.48
		OK	17,796	1.66	0.688	0.414	0.223	4.25
		ID2	17,796	1.63	0.755	0.463	0.245	5.28
Northeast	Inside Grade Shell	Composite	994	10.17	17.67	1.74	0.20	202.71
		NN	2,411	4.41	8.43	1.91	0.20	202.71
		OK	1,084	7.52	6.09	0.81	1.00	93.25
		ID2	1,084	7.83	6.76	0.86	0.71	94.00
	Outside Grade Shell	Composite	705	4.19	9.35	1.89	0.20	90.94
		NN	44,005	2.25	4.60	2.04	0.20	57.53
		OK	26,877	3.01	3.09	1.03	0.213	23.83
		ID2	26,877	2.84	2.66	0.94	0.203	36.21
South	Inside Grade Shell	Composite	397	2.60	2.13	0.82	0.50	16.51
		NN	18,138	2.46	1.92	0.781	0.5	16.51
		OK	9,258	2.61	1.12	0.428	0.821	8.72
		ID2	9,258	2.66	1.24	0.467	0.744	9.05
	Outside Grade Shell	Composite	208	1.62	1.27	0.781	0.50	9.56
		NN	44,012	1.74	1.50	0.861	0.5	9.56
		OK	29,539	1.66	0.68	0.41	0.68	6.33
		ID2	29,539	1.70	0.735	0.431	0.619	5.71
Gold								
Northwest	Inside Grade Shell	Composite	545	0.033	0.128	3.91	0.005	1.315
		NN	42,835	0.033	0.138	4.12	0.005	1.315
		OK	15,564	0.033	0.076	2.32	0.006	0.620
		ID2	15,564	0.035	0.097	2.74	0.005	1.015
	Outside Grade Shell	Composite	101	0.045	0.188	4.21	0.005	1.315
		NN	18,697	0.022	0.104	4.81	0.005	1.315
		OK	2,325	0.058	0.120	2.07	0.006	0.694
		ID2	2,325	0.061	0.151	2.49	0.006	1.073
Northeast	Inside Grade Shell	Composite	617	0.078	0.148	1.90	0.004	1.588
		NN	1,652	0.038	0.086	2.24	0.004	1.164
		OK	774	0.064	0.042	0.65	0.009	0.320
		ID2	774	0.066	0.048	0.728	0.008	0.413
	Outside Grade Shell	Composite	472	0.045	0.101	2.27	0.004	1.228
		NN	31,224	0.032	0.080	2.46	0.004	1.048
		OK	15,720	0.049	0.040	0.81	0.006	0.314
		ID2	15,720	0.041	0.032	0.78	0.006	0.554
South	Inside Grade Shell	Composite	439	0.013	0.016	1.23	0.001	0.216
		NN	18,348	0.013	0.014	1.12	0.001	0.216
		OK	9,427	0.014	0.008	0.551	0.002	0.078
		ID2	9,427	0.014	0.008	0.585	0.002	0.077

Area	Domain	Estimate	Count	Mean	Std. Dev.	CV	Minimum	Maximum
South	Outside Grade Shell	Composite	305	0.008	0.015	1.79	0.001	0.216
		NN	50,098	0.008	0.010	1.29	0.001	0.216
		OK	35,287	0.009	0.006	0.622	0.002	0.059
		ID2	35,287	0.008	0.005	0.633	0.001	0.080

The overall reduction of the maximum and standard deviation within the ID2 model represent an appropriate amount of smoothing to account for the point to block volume variance relationship while maintaining similar means. The reduction in mean from the composite to the estimates is the result of large volumes of low-grade material being estimated in low-grade domains with relatively fewer composites.

14.10.2 Swath Plots

Swath plots were generated to compare average estimated copper grade from the ID method to the NN and OK validation models. On a local scale, the NN model does not provide a reliable estimate of grade, but on a much larger scale, it represents an unbiased estimation of the grade distribution based on the total data set. Therefore, if the ID2 model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the distribution of grade from the NN.

Three sets of swath plots were generated for copper. Figure 14-7 shows the northwest area copper swath plot set, Figure 14-8 shows the northeast area copper swath plot set, and Figure 14-9 shows the south area copper swath plot set. Each set contains a swath plot along the X axis of the block model (upper left corner), the Y axis of the block model (upper right corner), and the Z axis of the block model (lower center).

Correlation between the grade models is generally good, though deviations occur. Areas where these deviations occur are the result of low sample density.

14.10.3 Section Inspection

Bench plans, cross-sections, and long sections comparing modeled grades to the 10-foot composites were evaluated. The example sections displaying estimated copper grades are shown in Figure 14-10 through Figure 14-21. The figures show good agreement between modeled grades and the composite grades. In addition, the modeled blocks display continuity of grades along strike and down dip.

Figure 14-7: Moonlight-Superior Project Northwest Area Block Model Swath Plots for Copper

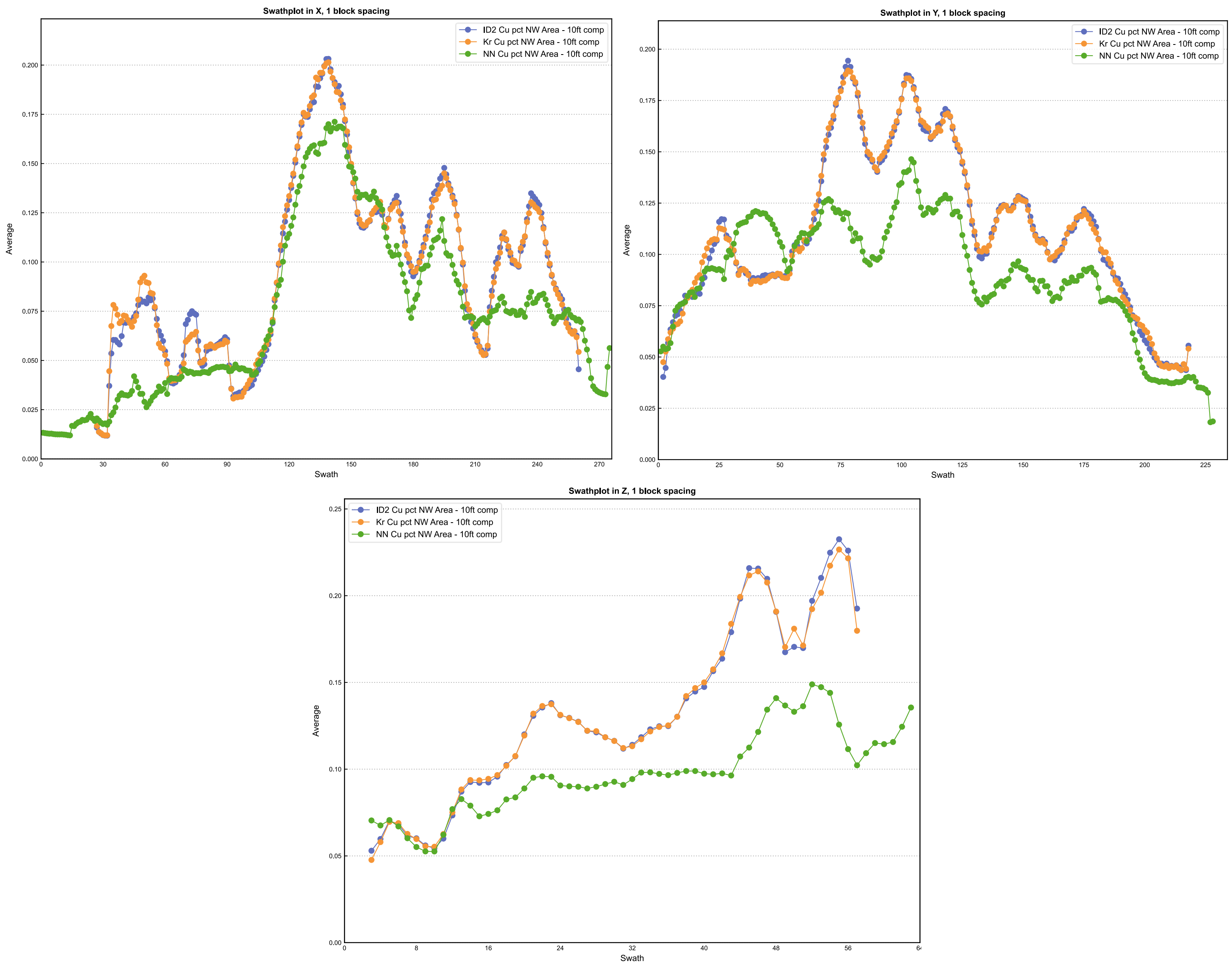


Figure 14-8: Moonlight-Superior Project Northeast Area Block Model Swath Plots for Copper

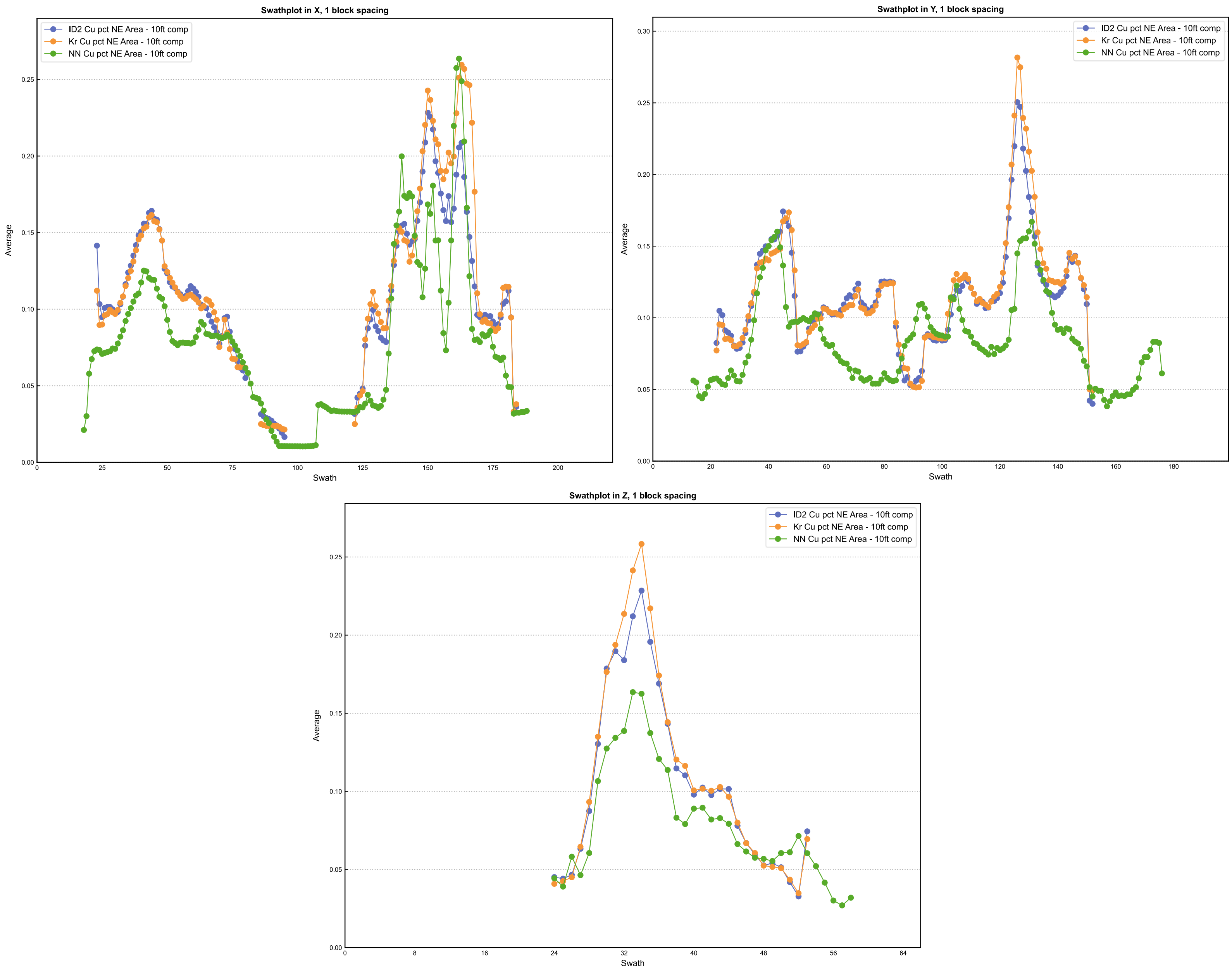


Figure 14-9: Moonlight-Superior Project South Area Block Model Swath Plots for Copper

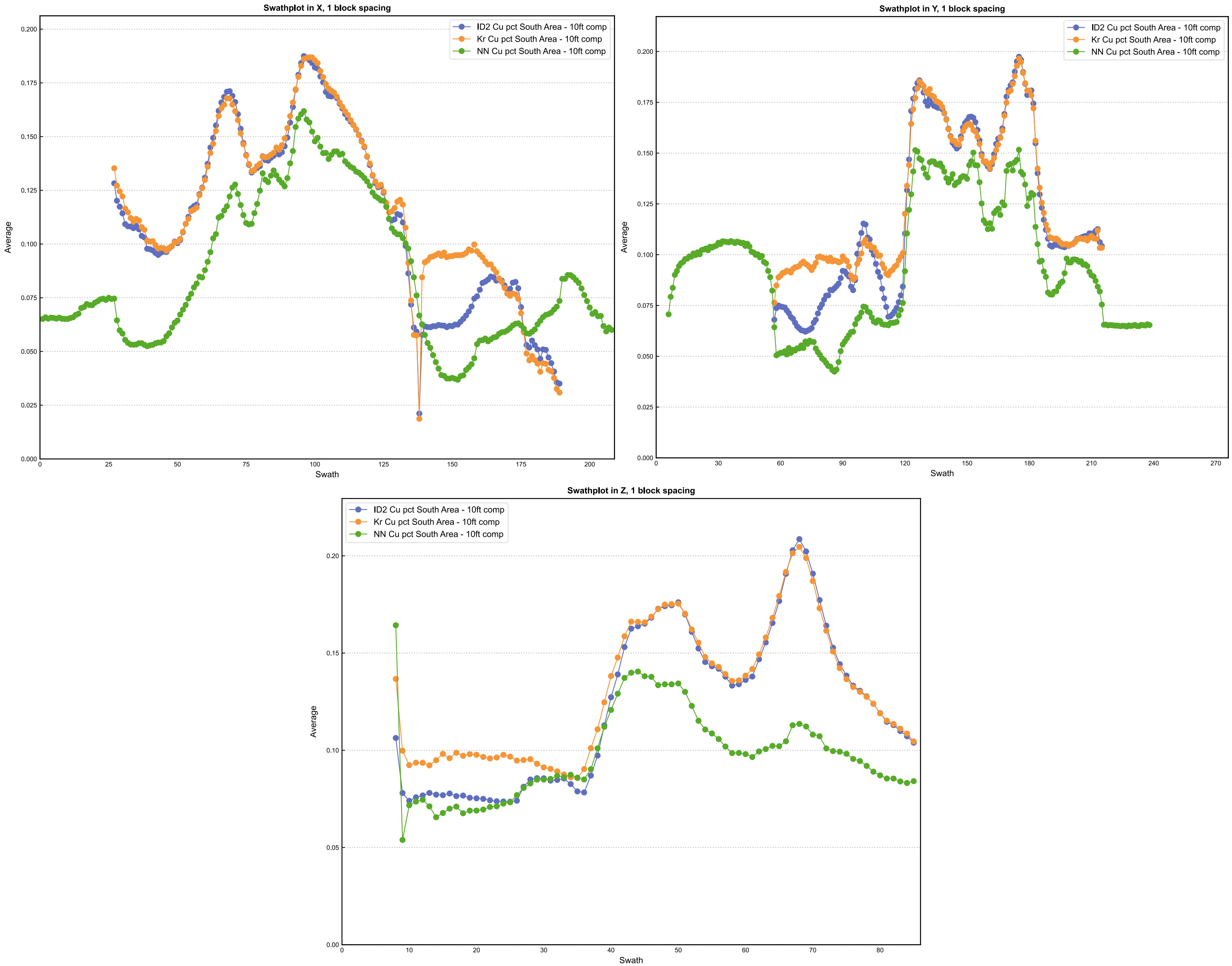


Figure 14-10: Moonlight-Superior Project Northwest Area Cross Section 1 of Estimated Copper Grades with Composites

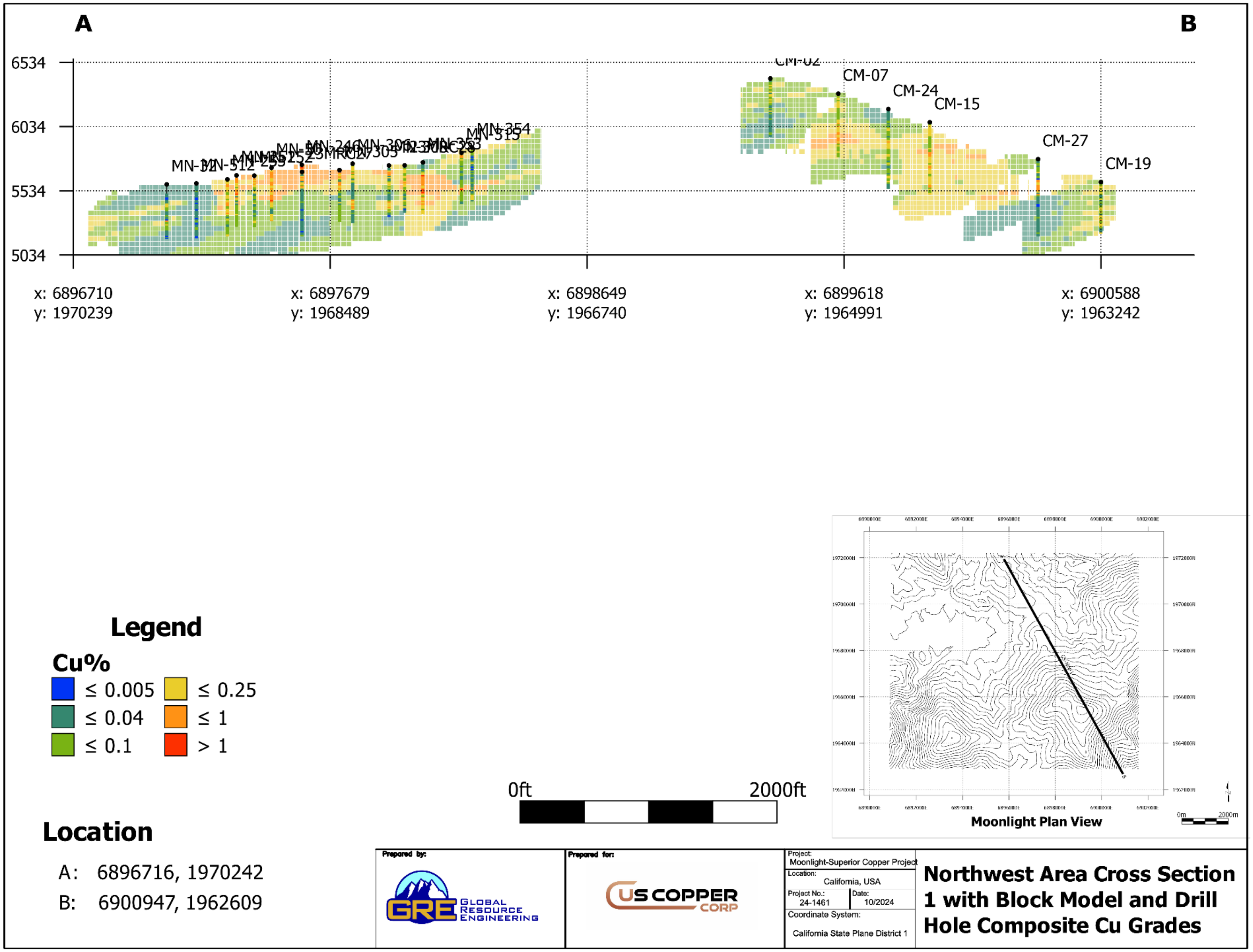


Figure 14-11: Moonlight-Superior Project Northwest Area Cross Section 2 of Estimated Copper Grades with Composites

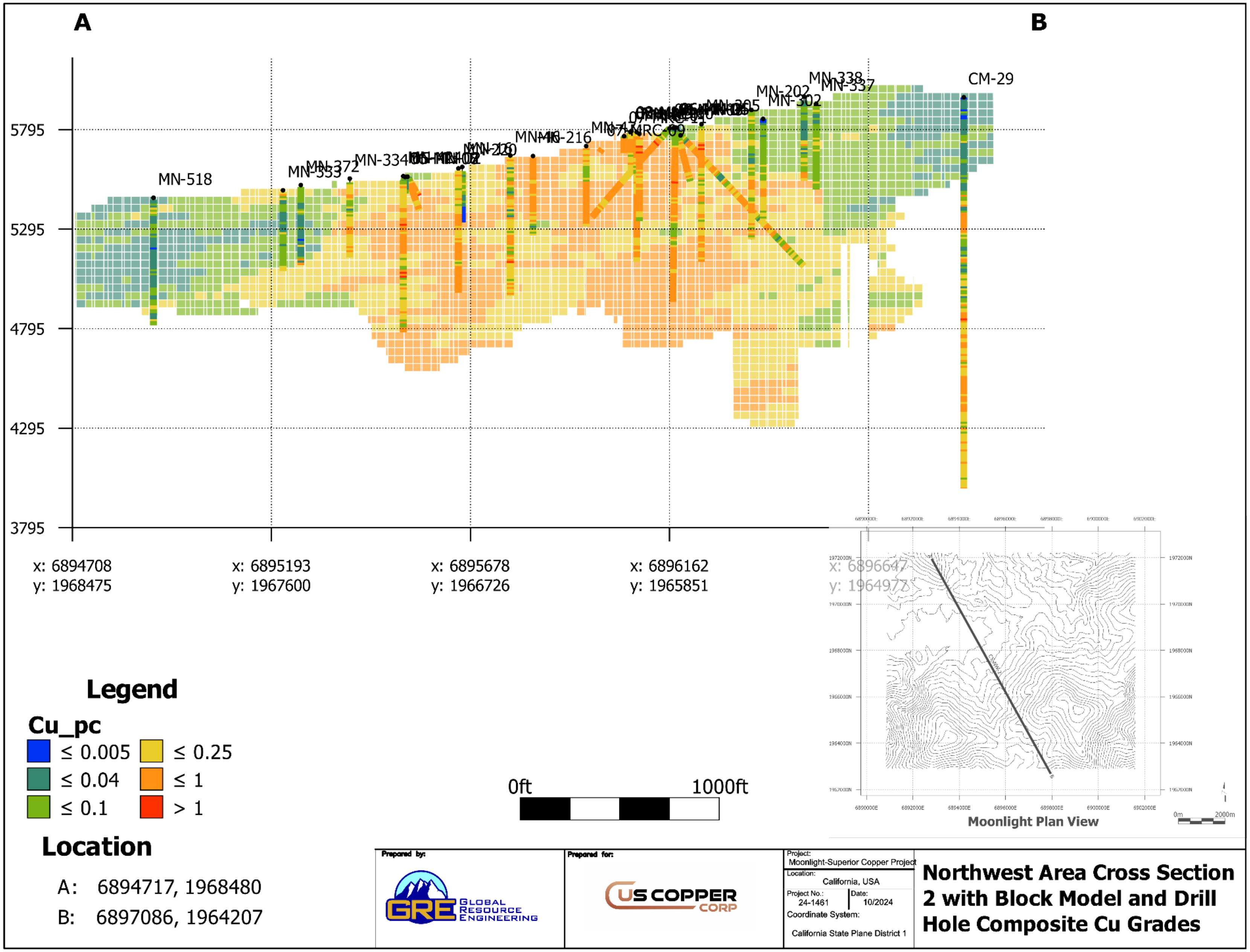


Figure 14-12: Moonlight-Superior Project Northwest Area Long Section 1 of Estimated Copper Grades with Composites

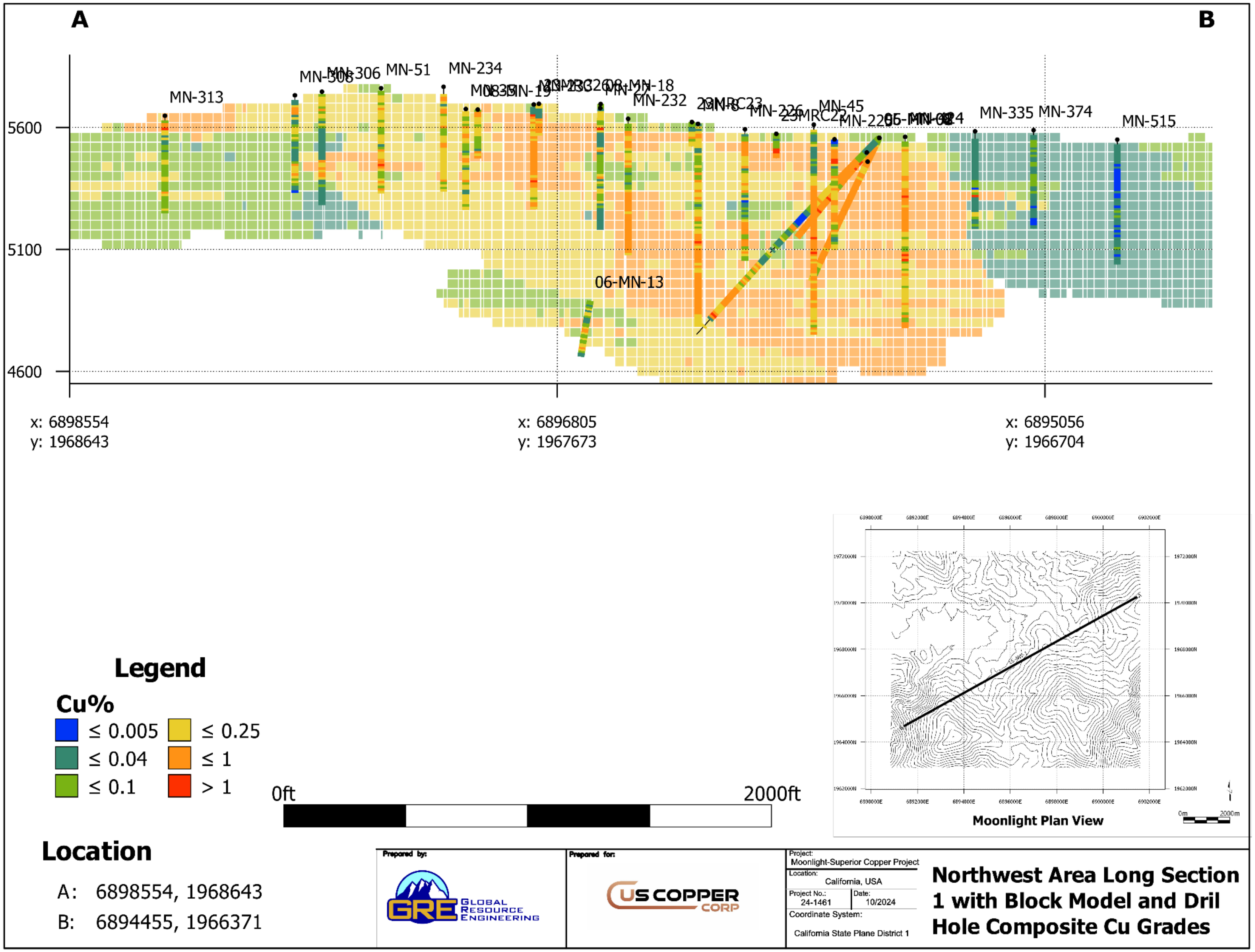


Figure 14-13: Moonlight-Superior Project Northwest Area Long Section 2 of Estimated Copper Grades with Composites

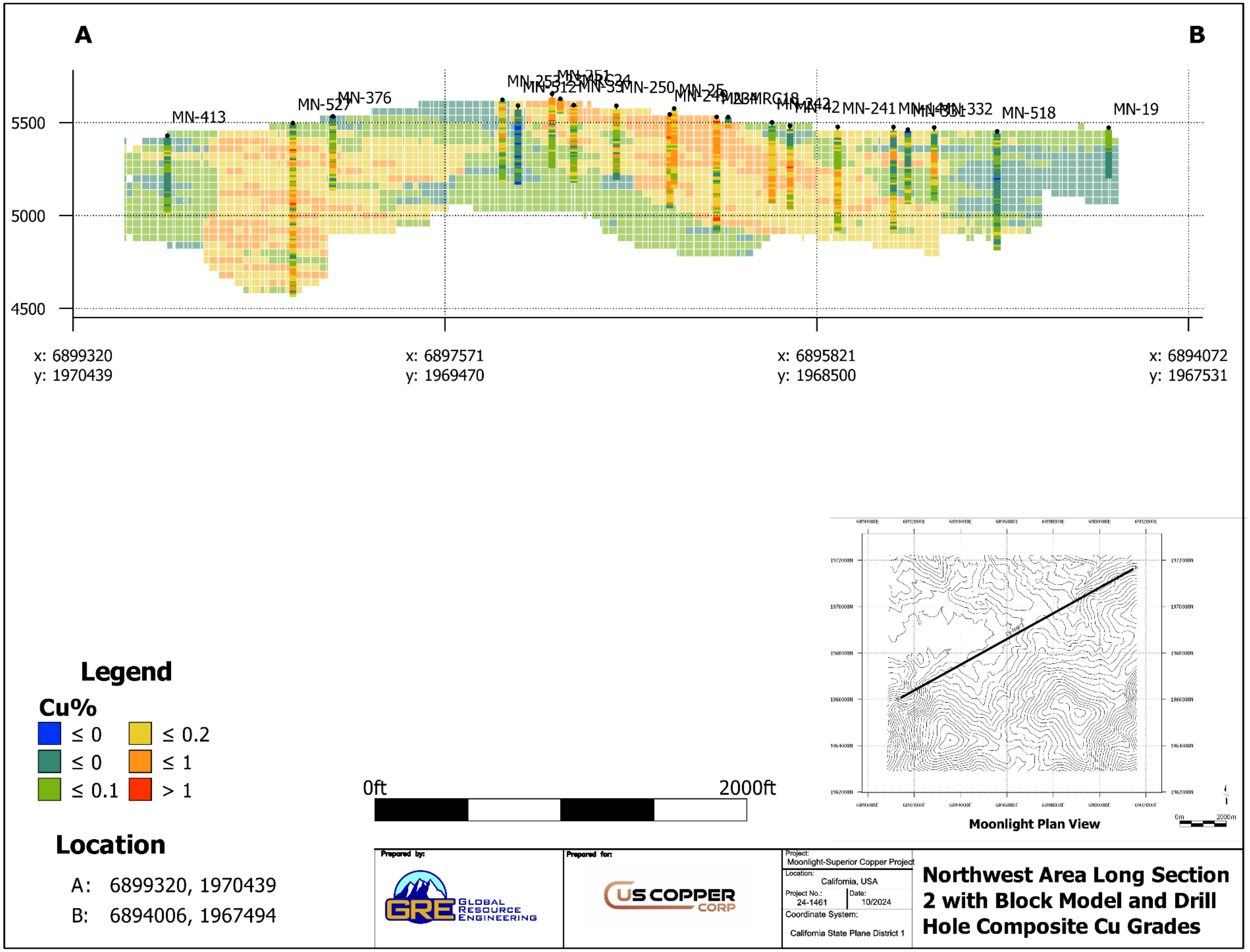


Figure 14-14: Moonlight-Superior Project Northeast Area Cross Section 1 of Estimated Copper Grades with Composites

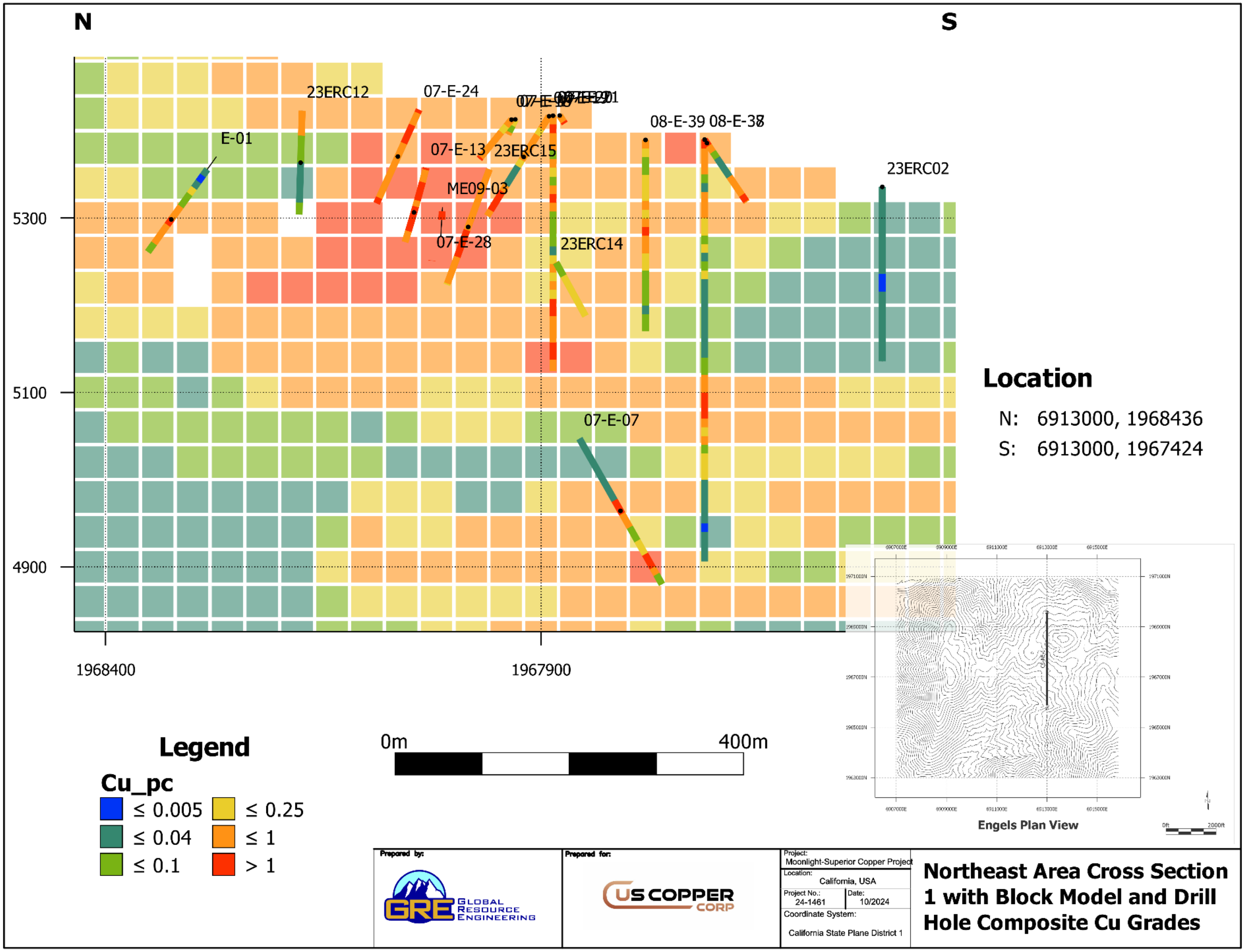


Figure 14-15: Moonlight-Superior Project Northeast Area Cross Section 2 of Estimated Copper Grades with Composites

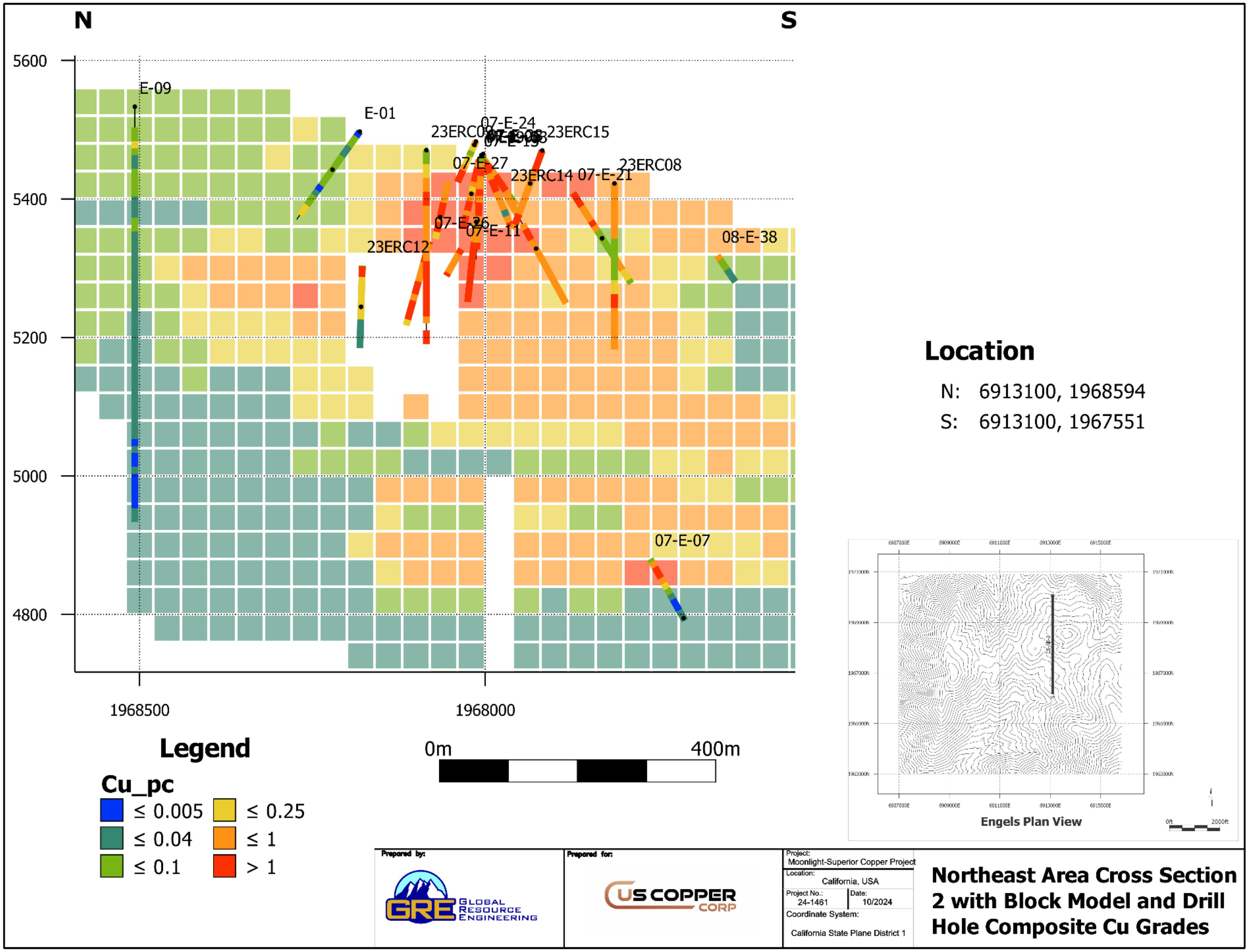


Figure 14-16: Moonlight-Superior Project Northeast Area Long Section 1 of Estimated Copper Grades with Composites

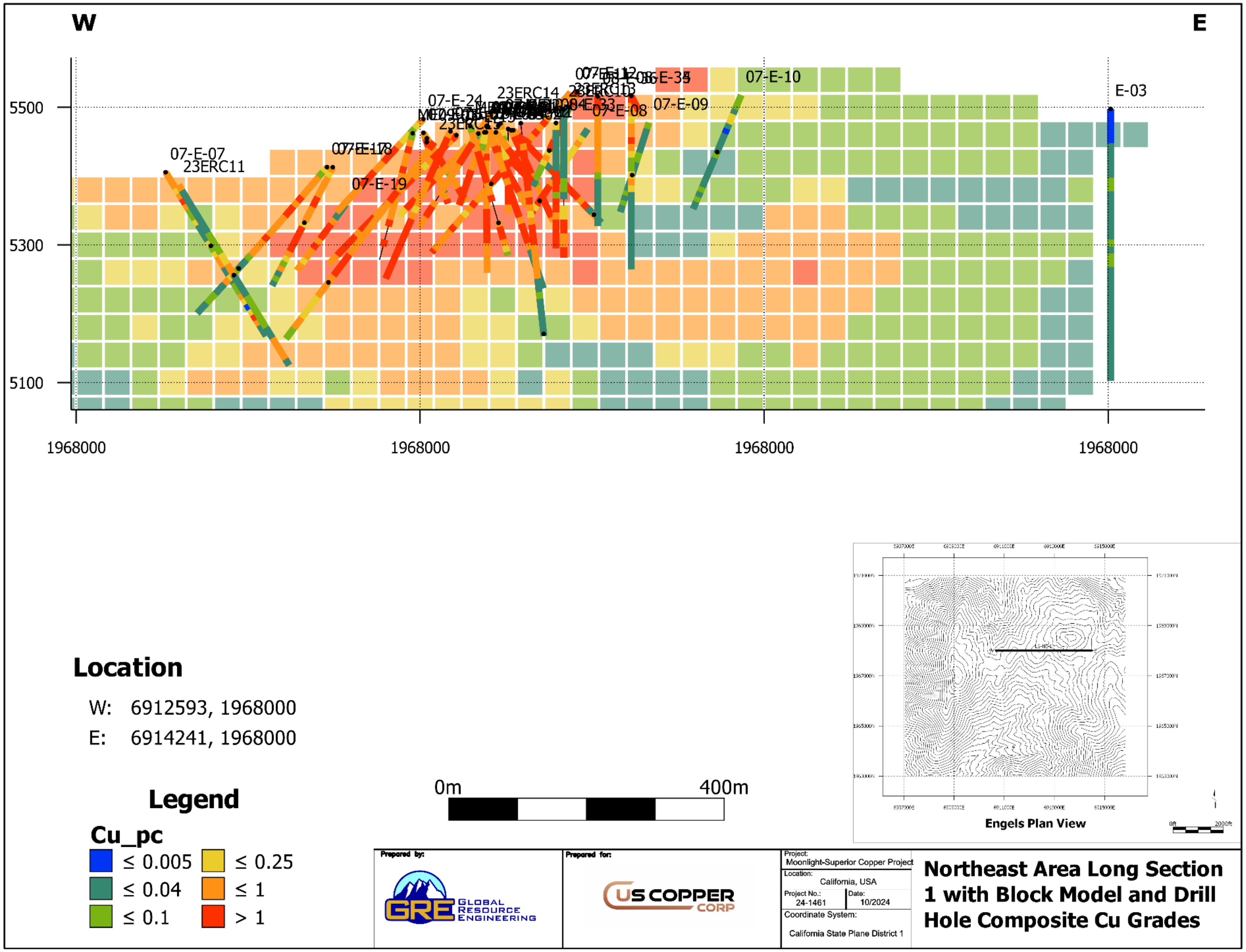


Figure 14-17: Moonlight-Superior Project Northeast Area Long Section 2 of Estimated Copper Grades with Composites

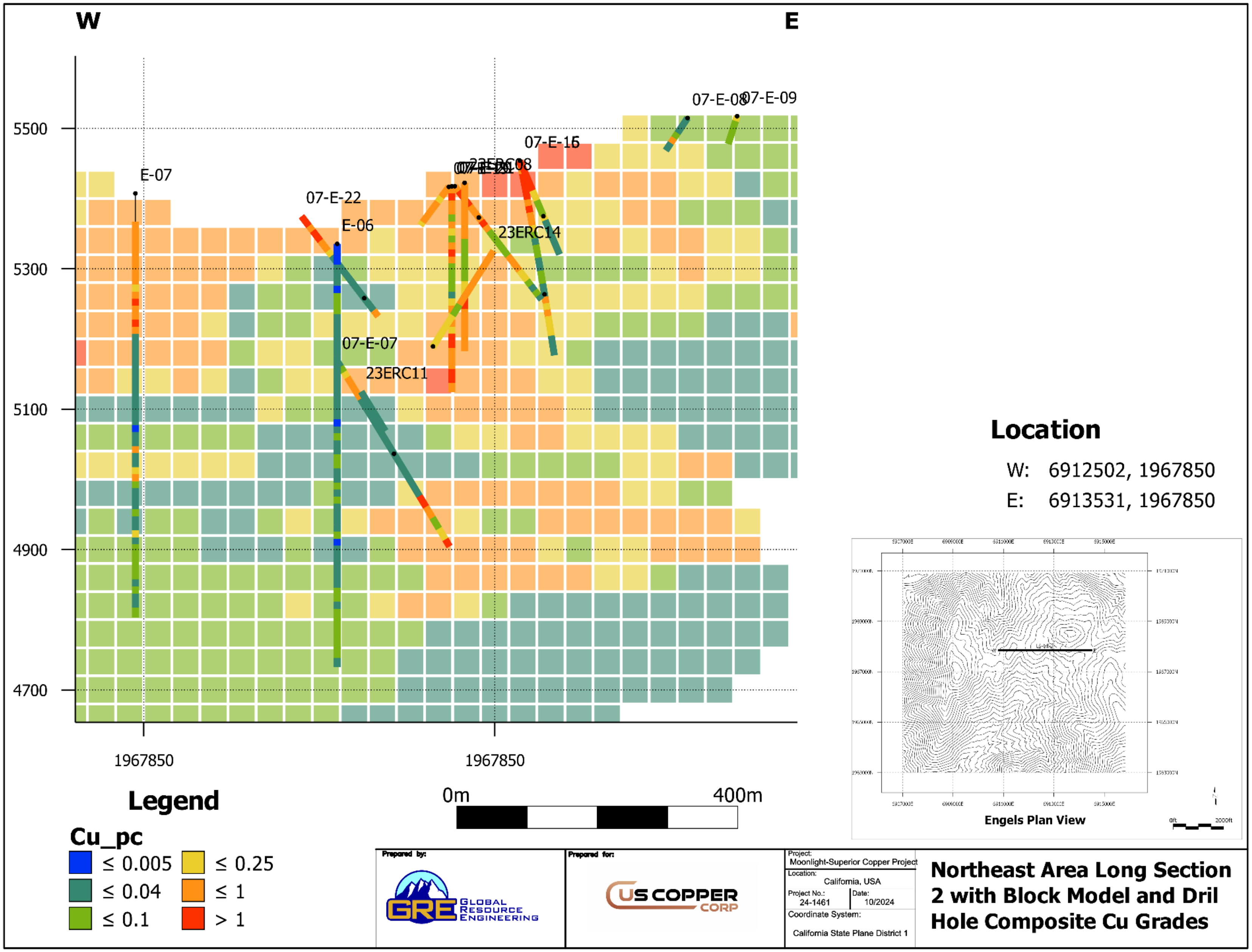


Figure 14-18: Moonlight-Superior Project South Area Cross Section 1 of Estimated Copper Grades with Composites

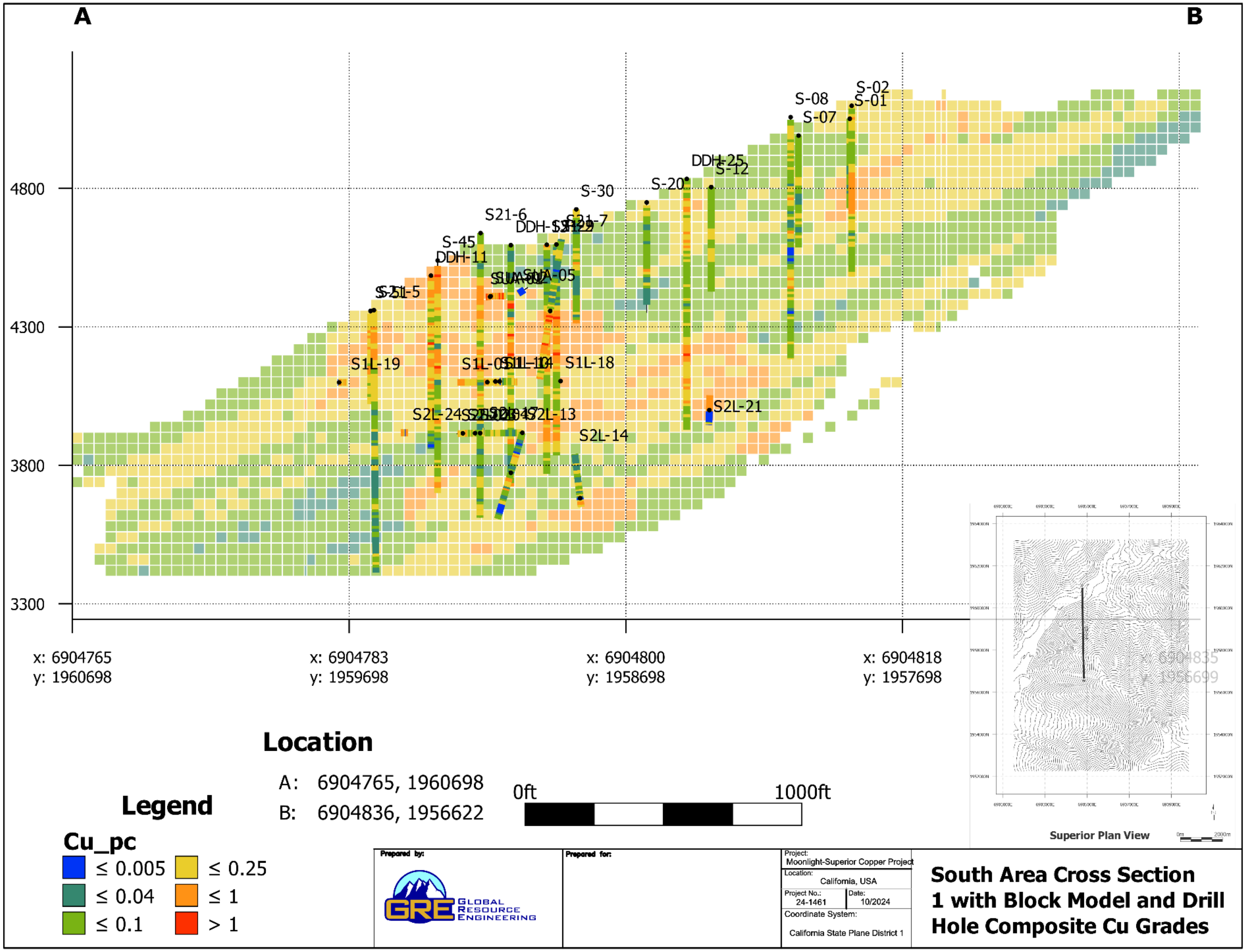


Figure 14-19: Moonlight-Superior Project South Area Cross Section 2 of Estimated Copper Grades with Composites

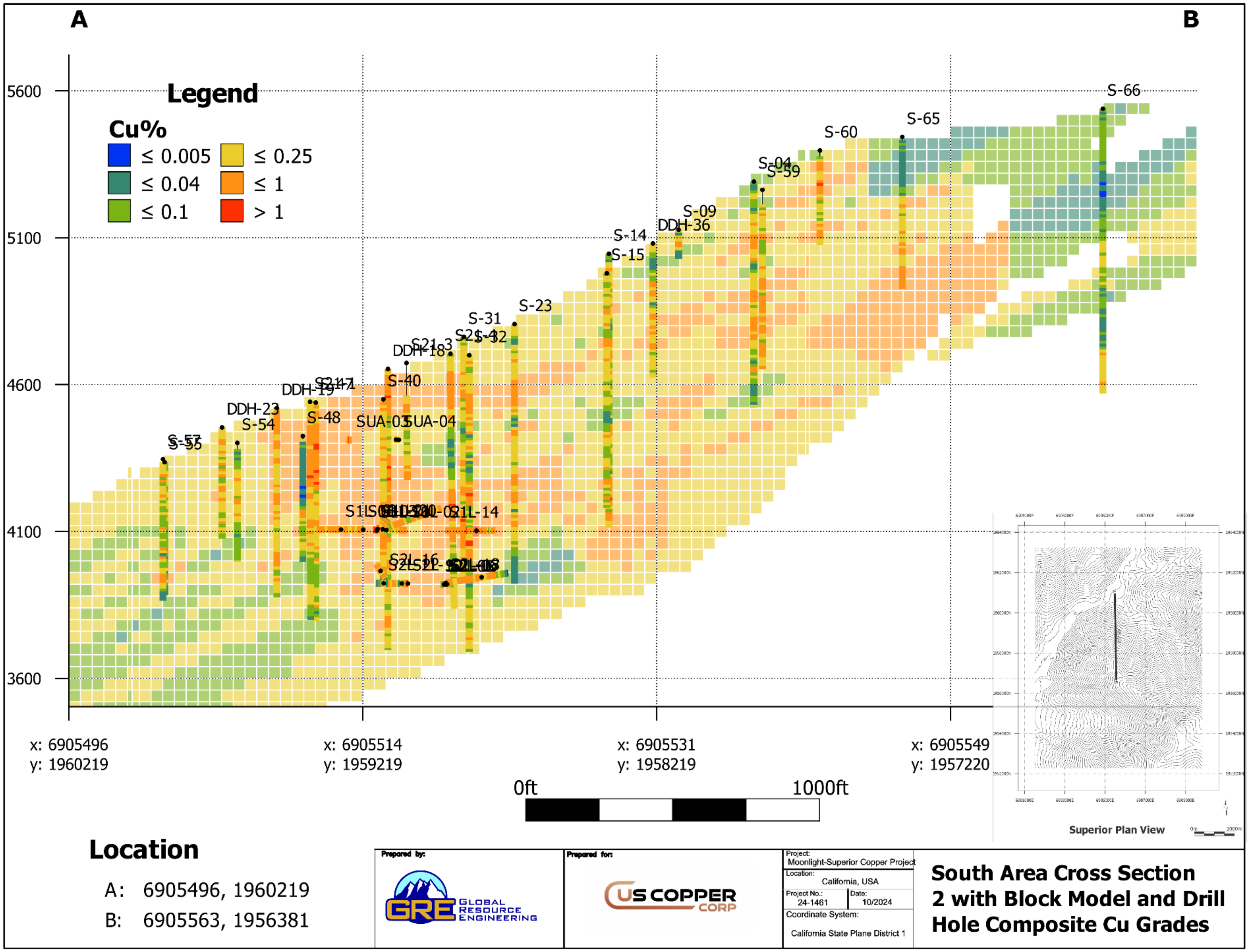


Figure 14-20: Moonlight-Superior Project South Area Long Section 1 of Estimated Copper Grades with Composites

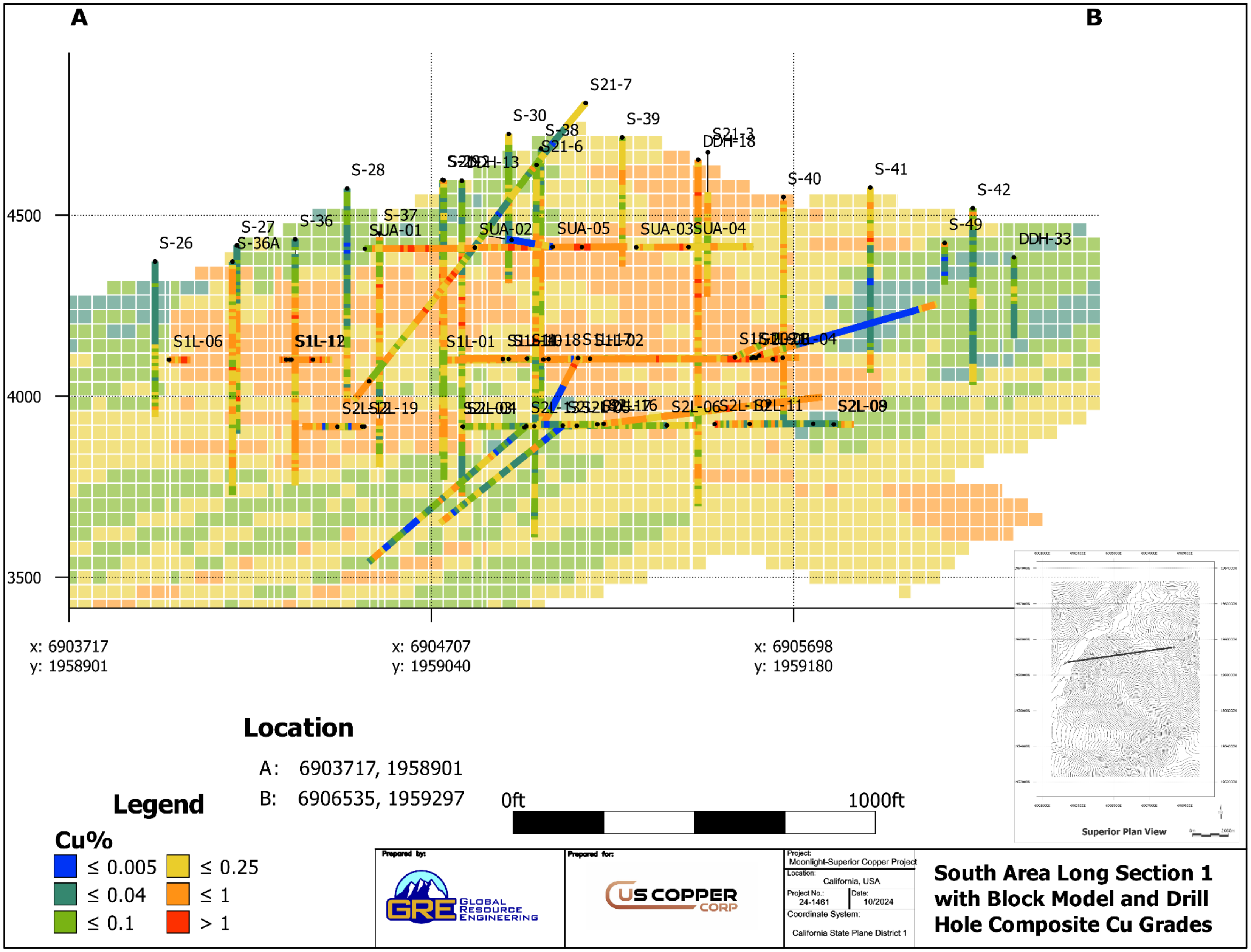
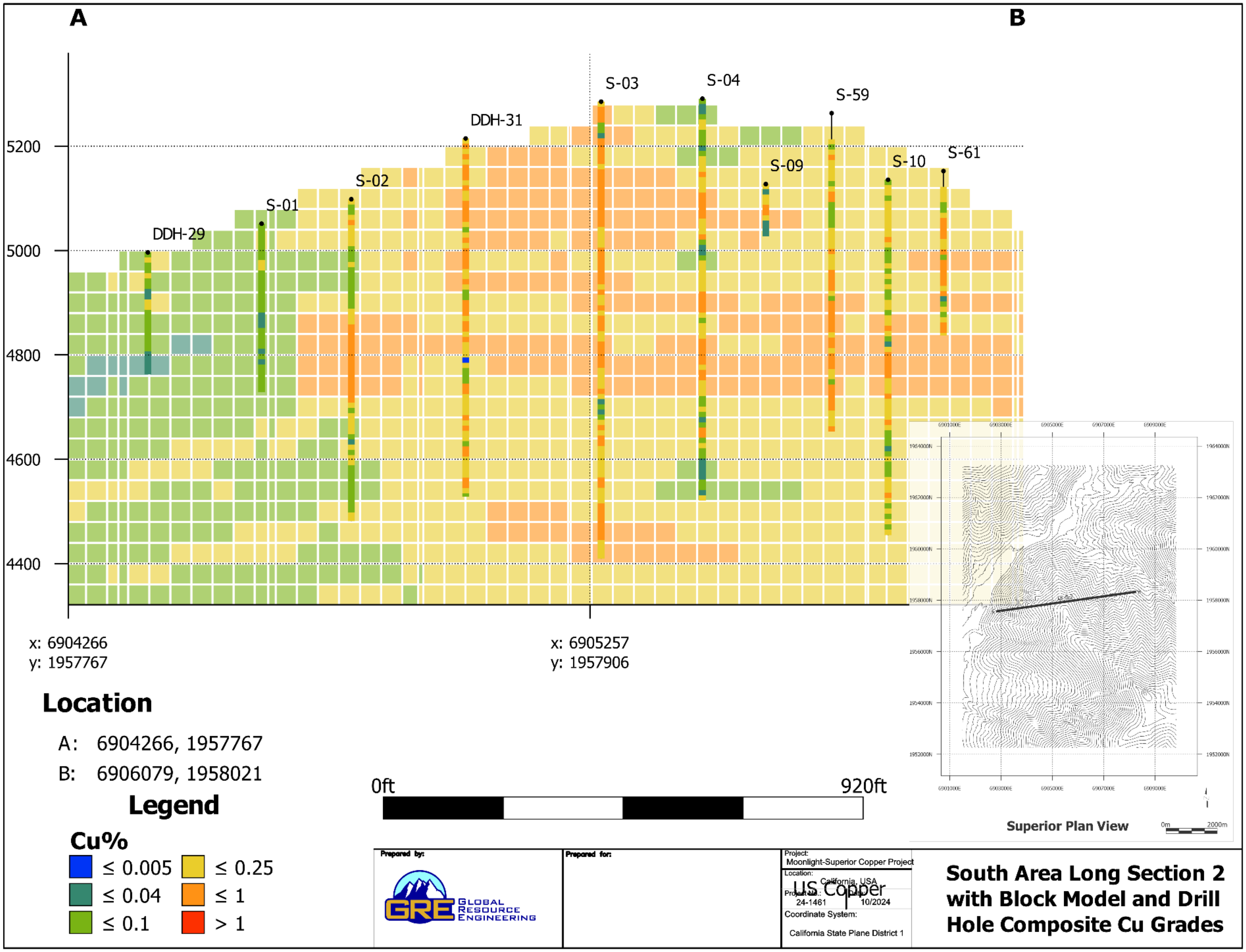


Figure 14-21: Moonlight-Superior Project South Area Long Section 2 of Estimated Copper Grades with Composites



14.11 Density

The database provided by US Copper contains 68 samples for bulk density, all of which were collected and tested by Sheffield from the MN-series core holes. The simple mean of the SGs was 2.67 tonnes/m^3 (equivalent to 0.0833 lb/ft^3), and all samples lay within two standard deviations of the mean. Fifteen samples above the oxide surface had the same mean as those below. Thus, a single value of 0.0833 lb/ft^3 was applied to all blocks in the models.

14.12 Mineral Resource Classification

14.12.1 Classifications

The mineral resources are classified as Indicated and Inferred in accordance with "CIM Definition Standards for Mineral Resources and Mineral Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014. Classification of the resources reflects the relative confidence of the grade estimates.

For all three block models, if a block was populated with grade in pass 1, either inside or outside of the 0.1% Cu grade shell, it was classified as Indicated. All other blocks with grade were classified as Inferred.

Representative illustrations of the block classifications are shown for each block model area in Figure 14-22 through Figure 14-24.

14.12.2 Existing Underground Workings

The Engels (Northeast Area) and Superior (South Area) deposits have existing underground workings. For each of these areas, blocks falling withing the existing workings were given Cu, Ag, and Au grades of 0, although tonnage was left in the model in the event any backfilling or collapse occurred.

14.12.3 Reasonable Prospects for Economic Extraction

The "reasonable prospects for economic extraction" requirement referred to in NI 43-101 was tested by designing a series of conceptual Lerchs-Grossman pit shells. The economic parameters used for this analysis are based on estimated operating costs at the project scaled to reflect production rates, expected processing costs, and estimated recoveries from metallurgical tests completed to date. Table 14-8 summarizes the cost and recovery parameters used in the analysis. Blocks classified as Indicated and Inferred were used to define the resource pit shell. The block model tons and estimated recovered copper are shown in Figure 14-8 at variable copper prices within corresponding pits, as a sensitivity analysis.

Figure 14-22: Moonlight-Superior Project Northwest Area Block Classifications

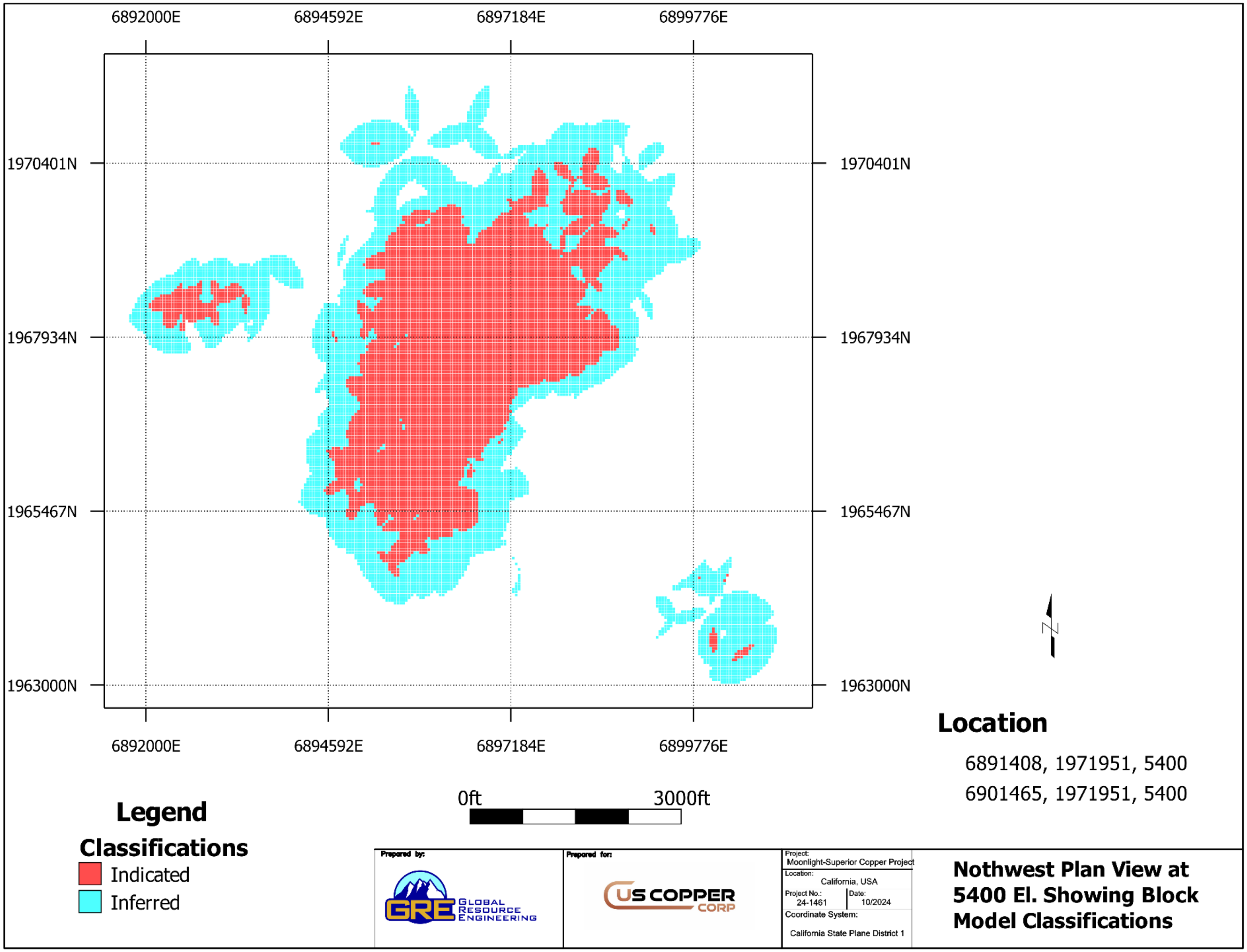


Figure 14-23: Moonlight-Superior Project Northeast Area Block Classifications

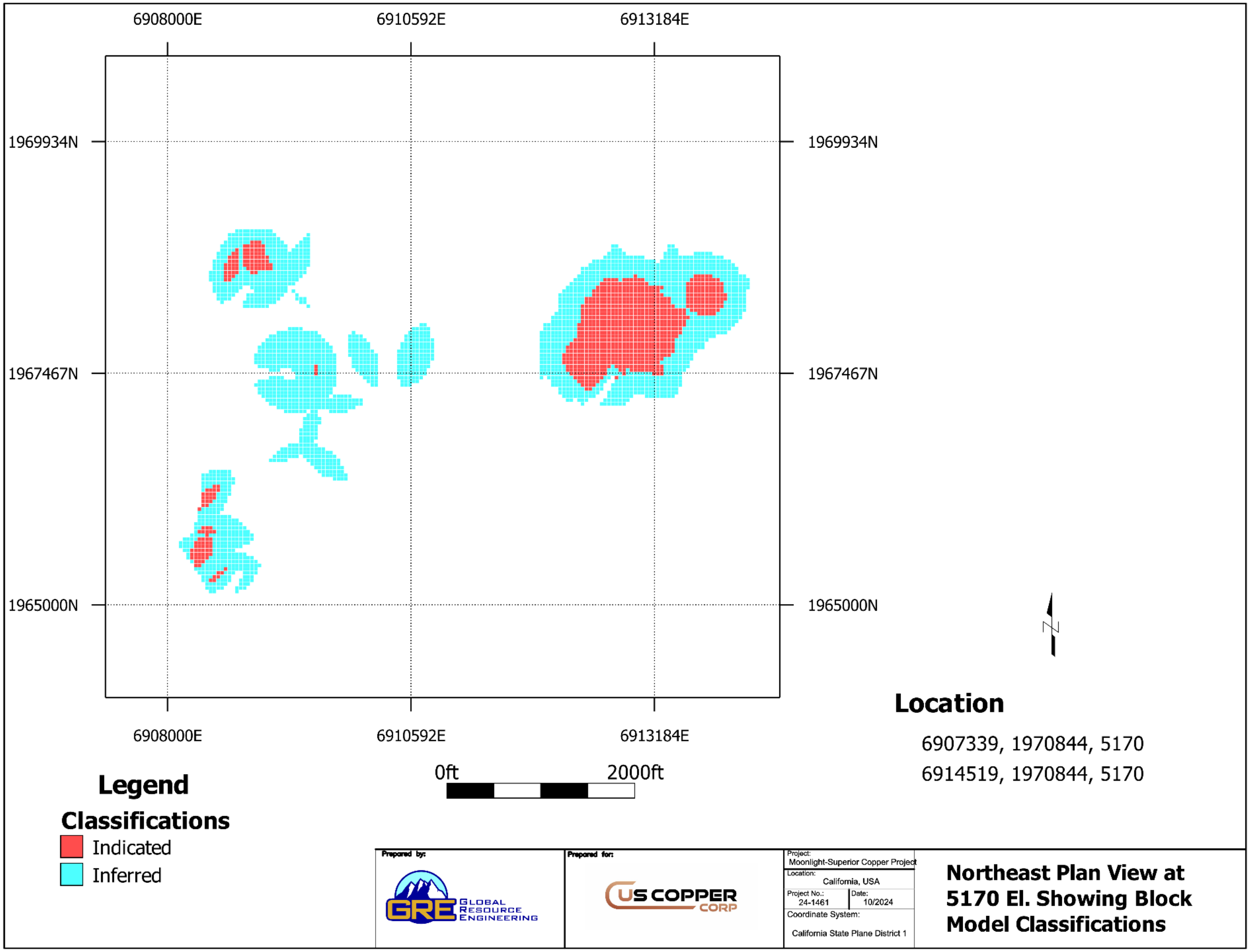


Figure 14-24: Moonlight-Superior Project South Area Block Classifications

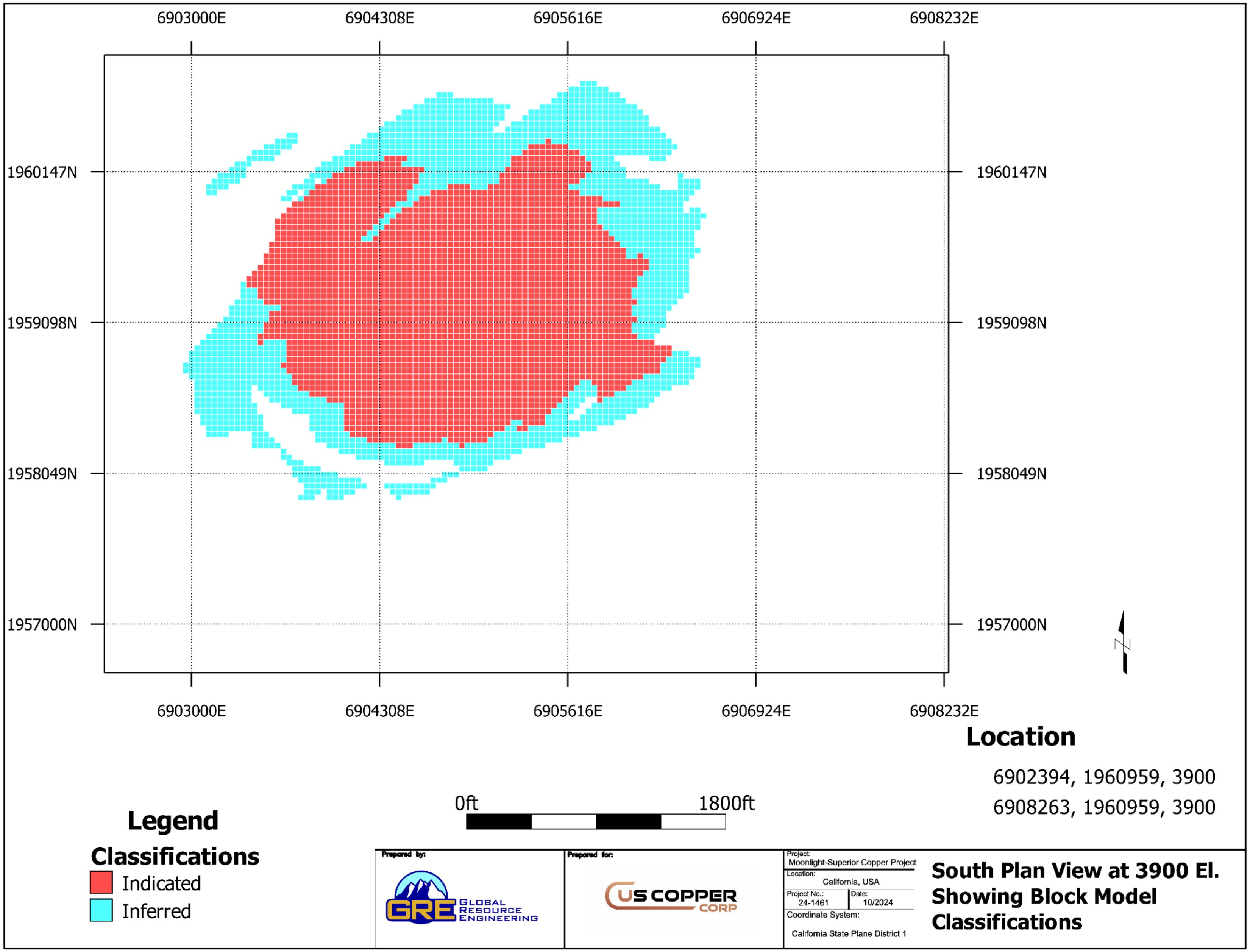


Table 14-8: Parameters used for Resource Pit Shell Generation

Item	Cost/Rate	Units
Sulfide Material		
Revenue, Smelting, and Refining		
Cu Price	$4.00	US$ per lb Cu
Ag Price	$23.50	US$/troy oz
Au Price	$1,900	US$/troy oz
Cu Recovery	89.7%	
Ag Recovery	70%	
Au Recovery	70%	
Cu Concentrate Deductions	1%	
Ag Concentrate Deductions	1	oz/ton
Au Concentrate Deductions	0.03	oz/ton
Ag Payable	90%	
Au Payable	98%	
Cu Refining Charges	0.08	US$/lb
Ag Refining Charges	0.30	US$/oz
Transport and Concentrate Loadout	40	US$/ton concentrate
Ocean Freight	25	US$/ton concentrate
Other Off-site Costs (Losses, Insurance, Sale Rep. Assay)	15	US$/ton concentrate
Cu Treatment Charges	80	US$/ton concentrate
Operating Cost Estimates		
Mining Cost	$2.35	US$ per total ton
Processing Cost	$9.80	US$ per process ton
General and Administrative (G&A)	$0.65	US$ per process ton
Process and Mining Recovery		
Mining Recovery	99.5%	
Dilution	2%	
Oxide and Transition Material		
Cu Price	$4.00	US$/ lb Cu
Cu Recovery – Oxide	75%	
Cu Recovery - Transition	60%	
Mining Cost	$2.35	US$ per total ton
Processing Cost	$6.85	US$ per process ton
G&A Cost	$0.65	US$ per process ton
Mining Recovery	99.5%	%
Dilution	2%	

For the sulfide material, Net Smelter Return (NSR) values were calculated within Leapfrog and populated into each block using the parameters specified in Table 14-8.

14.13 Mineral Resource Statement

Resources are reported within an optimized pit shell for each project area and meet the test of reasonable prospects for economic extraction. For sulfide material, a 10.45 NSR cutoff was chosen, and for oxide and transition material, a 0.16% Cu cutoff was chosen for reporting the mineral resource. The cutoff grades were calculated based on the parameters in Table 14-8.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources are that part of the mineral resource for which quantity and grade or quality are estimated based on limited geologic evidence and sampling, which is sufficient to imply but not verify grade or quality continuity. Inferred mineral resources may not be converted to mineral reserves. It is reasonably expected, though not guaranteed, that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.

Table 14-9 shows the Mineral Resource Statement for the Moonlight-Superior Project.

Table 14-9: Moonlight-Superior Project Mineral Resource Statement

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
Indicated										
Engels	Oxide	0.16	%	2.39	0.81	40,861	7.72	565,232	0.055	4,050
	Transition	0.16	%	7.52	0.50	79,941	4.75	1,093,948	0.042	10,194
	Sulfide	10.45	NSR/ton	8.32	0.46	76,750	5.83	1,415,487	0.056	13,585
Lambs Ridge	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	1.61	0.27	8,614	0.00	0	0.000	0
Moonlight	Oxide	0.16	%	1.35	0.36	10,244	3.77	154,364	0.128	5,460
	Transition	0.16	%	25.71	0.33	179,071	3.85	2,972,073	0.037	30,083
	Sulfide	10.45	NSR/ton	232.35	0.30	1,390,461	1.87	12,674,340	0.009	61,721
Copper Mountain	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	3.94	0.32	24,936	0.00	0	0.000	0
Superior	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	119.64	0.30	722,893	0.81	2,817,086	0.004	14,949
Total	Oxide	0.16	%	3.74	0.68	51,104	6.59	719,596	0.087	9,510
	Transition	0.16	%	33.23	0.39	259,012	4.20	4,066,021	0.042	40,277
	Sulfide	10.45	NSR/ton	365.86	0.30	2,223,654	1.58	16,906,913	0.008	90,255
Total				**402.83**	**0.31**	**2,533,771**	**1.85**	**21,692,531**	**0.012**	**140,042**
Inferred										
Engels	Oxide	0.16	%	0.15	1.18	3,740	11.91	55,046	0.010	48
	Transition	0.16	%	1.73	0.49	18,287	5.20	281,158	0.019	1,053
	Sulfide	10.45	NSR/ton	6.93	0.38	52,445	5.08	1,027,412	0.041	8,280
Lambs Ridge	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	3.46	0.30	20,954	0.00	0	0.000	0
Moonlight	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	30.82	0.28	175,635	0.09	81,857	0.000	35

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
Copper Mountain	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	3.90	0.27	21,320	0.00	0	0.000	0
Superior	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	17.60	0.29	101,817	0.01	2,681	0.000	23
Total	Oxide	0.16	%	0.15	1.25	3,740	12.64	55,046	0.011	48
	Transition	0.16	%	1.73	0.53	18,287	5.58	281,158	0.021	1,053
	Sulfide	10.45	NSR/ton	62.71	0.30	372,171	0.61	1,111,950	0.005	8,338
Total				**64.59**	**0.31**	**394,199**	**0.77**	**1,448,154**	**0.005**	**9,440**

Notes:

7. The effective date of the Mineral Resource is December 16, 2024.
8. The Qualified Person for the Mineral Resource Estimate is Terre Lane of GRE.
9. Mineral resources are reported at a 0.16% Cu cutoff for oxide and transition material and at a 10.45 NSR cutoff for sulfide material. The oxide and transition cutoff is calculated based on a long-term copper price of US$4.00/lb; assumed combined operating costs of US$7.50/ton (process and G&A); metallurgical recovery of 75% for copper. The sulfide cutoff is calculated as the breakeven NSR, which is equal to the combined process and G&A costs for the sulfide material.
10. Mineral resources are captured within an optimized pit shell and meet the test of reasonable prospects for economic extraction by open pit. The optimization used the same mining costs of US$2.35/ton mined and a 45º pit slope.
11. Rounding may result in apparent differences when summing tons, grade, and contained metal content.

Table 14-10 shows the sensitivity of the mineralization to cutoff grade.

Table 14-10: Mineral Resource Estimate by At Varying Cutoff Grades

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
Indicated										
Engels	Oxide	0.1	%	2.61	0.79	41,435	7.55	574,119	0.054	4,101
		0.12	%	2.54	0.81	41,285	7.72	571,729	0.055	4,050
		0.14	%	2.47	0.83	41,100	7.90	568,711	0.055	3,991
		0.16	**%**	**2.39**	**0.85**	**40,861**	**8.11**	**565,232**	**0.056**	**3,913**
		0.18	%	2.33	0.87	40,649	8.28	561,579	0.056	3,802
		0.2	%	2.26	0.89	40,407	8.46	557,835	0.057	3,731
		0.22	%	2.17	0.92	40,003	8.71	550,406	0.057	3,630
		0.24	%	2.11	0.94	39,734	8.89	546,314	0.058	3,567
		0.26	%	2.04	0.96	39,417	9.10	542,205	0.059	3,487
	Transition	0.1	%	8.54	0.48	82,617	4.66	1,160,137	0.042	10,511
		0.12	%	8.22	0.50	81,925	4.75	1,138,998	0.042	10,194
		0.14	%	7.84	0.52	80,923	4.88	1,115,045	0.043	9,866
		0.16	**%**	**7.52**	**0.53**	**79,941**	**4.99**	**1,093,948**	**0.043**	**9,529**
		0.18	%	7.24	0.55	79,007	5.11	1,078,045	0.044	9,252
		0.2	%	6.85	0.57	77,521	5.27	1,053,234	0.044	8,847
		0.22	%	6.42	0.59	75,725	5.44	1,018,992	0.045	8,384
		0.24	%	6.06	0.61	74,047	5.61	991,725	0.045	8,015
		0.26	%	5.67	0.64	72,109	5.82	962,489	0.046	7,612

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
	Sulfide	**10.45**	**NSR/ton**	**8.32**	**0.46**	**76,750**	**5.83**	**1,415,487**	**0.056**	**13,585**
		15	NSR/ton	6.20	0.56	69,204	6.58	1,189,815	0.062	11,122
		20	NSR/ton	4.62	0.66	61,116	7.44	1,003,334	0.069	9,273
		25	NSR/ton	3.48	0.77	53,403	8.45	858,899	0.077	7,814
		30	NSR/ton	2.72	0.86	47,056	9.41	746,574	0.085	6,779
		35	NSR/ton	2.13	0.97	41,280	10.26	638,217	0.094	5,839
		40	NSR/ton	1.72	1.07	36,612	10.95	548,470	0.098	4,907
		45	NSR/ton	1.42	1.15	32,851	11.55	479,814	0.101	4,189
		50	NSR/ton	1.14	1.26	28,701	12.50	415,993	0.109	3,621
Lambs Ridge	Oxide	0.1	%	0.00	0.00	0	0.00	0	0.000	0
		0.12	%	0.00	0.00	0	0.00	0	0.000	0
		0.14	%	0.00	0.00	0	0.00	0	0.000	0
		0.16	**%**	**0.00**	**0.00**	**0**	**0.00**	**0**	**0.000**	**0**
		0.18	%	0.00	0.00	0	0.00	0	0.000	0
		0.2	%	0.00	0.00	0	0.00	0	0.000	0
		0.22	%	0.00	0.00	0	0.00	0	0.000	0
		0.24	%	0.00	0.00	0	0.00	0	0.000	0
		0.26	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.1	%	0.00	0.00	0	0.00	0	0.000	0
		0.12	%	0.00	0.00	0	0.00	0	0.000	0
		0.14	%	0.00	0.00	0	0.00	0	0.000	0
		0.16	**%**	**0.00**	**0.00**	**0**	**0.00**	**0**	**0.000**	**0**
		0.18	%	0.00	0.00	0	0.00	0	0.000	0
		0.2	%	0.00	0.00	0	0.00	0	0.000	0
		0.22	%	0.00	0.00	0	0.00	0	0.000	0
		0.24	%	0.00	0.00	0	0.00	0	0.000	0
		0.26	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	**10.45**	**NSR/ton**	**1.61**	**0.27**	**8,614**	**0.00**	**0**	**0.000**	**0**
		15	NSR/ton	1.10	0.29	6,432	0.00	0	0.000	0
		20	NSR/ton	0.19	0.37	1,404	0.00	0	0.000	0
		25	NSR/ton	0.02	0.45	191	0.00	0	0.000	0
		30	NSR/ton	0.01	0.56	60	0.00	0	0.000	0
		35	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		40	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		45	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		50	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
Moonlight	Oxide	0.1	%	1.49	0.36	10,623	3.72	162,197	0.125	5,466
		0.12	%	1.47	0.36	10,565	3.77	161,261	0.128	5,460
		0.14	%	1.42	0.37	10,435	3.83	158,683	0.131	5,437
		0.16	**%**	**1.35**	**0.38**	**10,244**	**3.91**	**154,364**	**0.129**	**5,114**
		0.18	%	1.30	0.39	10,061	3.98	150,931	0.124	4,711
		0.2	%	1.24	0.40	9,816	4.07	146,893	0.121	4,359
		0.22	%	1.19	0.40	9,636	4.14	144,129	0.121	4,232
		0.24	%	1.10	0.42	9,195	4.30	137,854	0.114	3,660
		0.26	%	1.03	0.43	8,856	4.42	132,562	0.109	3,274

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
	Transition	0.1	%	28.66	0.33	187,005	3.80	3,177,807	0.037	30,910
		0.12	%	27.98	0.33	185,483	3.85	3,142,036	0.037	30,083
		0.14	%	26.96	0.34	182,805	3.90	3,065,684	0.036	28,261
		0.16	**%**	**25.71**	**0.35**	**179,071**	**3.96**	**2,972,073**	**0.035**	**26,164**
		0.18	%	24.38	0.36	174,531	4.03	2,868,073	0.035	25,227
		0.2	%	23.01	0.37	169,331	4.10	2,753,561	0.035	23,792
		0.22	%	21.41	0.38	162,616	4.17	2,606,060	0.036	22,503
		0.24	%	19.66	0.39	154,571	4.25	2,438,989	0.036	20,694
		0.26	%	17.94	0.41	145,962	4.34	2,269,550	0.037	19,293
Moonlight	Sulfide	**10.45**	**NSR/ton**	**232.35**	**0.30**	**1,390,461**	**1.87**	**12,674,340**	**0.009**	**61,721**
		15	NSR/ton	149.10	0.35	1,057,969	2.23	9,707,971	0.012	50,765
		20	NSR/ton	80.51	0.43	686,183	2.56	6,003,935	0.015	34,672
		25	NSR/ton	39.69	0.51	402,016	2.75	3,186,403	0.018	20,809
		30	NSR/ton	19.43	0.59	229,549	2.85	1,615,125	0.018	10,154
		35	NSR/ton	9.82	0.68	132,721	2.84	814,471	0.010	2,893
		40	NSR/ton	4.88	0.76	74,471	2.73	387,978	0.008	1,081
		45	NSR/ton	2.59	0.84	43,627	2.59	195,689	0.008	627
		50	NSR/ton	1.37	0.92	25,089	2.71	107,983	0.011	443
Copper Mountain	Oxide	0.1	%	0.00	0.00	0	0.00	0	0.000	0
		0.12	%	0.00	0.00	0	0.00	0	0.000	0
		0.14	%	0.00	0.00	0	0.00	0	0.000	0
		0.16	**%**	**0.00**	**0.00**	**0**	**0.00**	**0**	**0.000**	**0**
		0.18	%	0.00	0.00	0	0.00	0	0.000	0
		0.2	%	0.00	0.00	0	0.00	0	0.000	0
		0.22	%	0.00	0.00	0	0.00	0	0.000	0
		0.24	%	0.00	0.00	0	0.00	0	0.000	0
		0.26	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.1	%	0.00	0.00	0	0.00	0	0.000	0
		0.12	%	0.00	0.00	0	0.00	0	0.000	0
		0.14	%	0.00	0.00	0	0.00	0	0.000	0
		0.16	**%**	**0.00**	**0.00**	**0**	**0.00**	**0**	**0.000**	**0**
		0.18	%	0.00	0.00	0	0.00	0	0.000	0
		0.2	%	0.00	0.00	0	0.00	0	0.000	0
		0.22	%	0.00	0.00	0	0.00	0	0.000	0
		0.24	%	0.00	0.00	0	0.00	0	0.000	0
		0.26	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	**10.45**	**NSR/ton**	**3.94**	**0.32**	**24,936**	**0.00**	**0**	**0.000**	**0**
		15	NSR/ton	2.56	0.38	19,205	0.00	0	0.000	0
		20	NSR/ton	1.61	0.43	13,923	0.00	0	0.000	0
		25	NSR/ton	0.95	0.48	9,203	0.00	0	0.000	0
		30	NSR/ton	0.35	0.54	3,825	0.00	0	0.000	0
		35	NSR/ton	0.07	0.61	917	0.00	0	0.000	0
		40	NSR/ton	0.01	0.69	146	0.00	0	0.000	0
		45	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		50	NSR/ton	0.00	0.00	0	0.00	0	0.000	0

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
Superior	Sulfide	**10.45**	**NSR/ton**	**119.64**	**0.30**	**722,893**	**0.81**	**2,817,086**	**0.004**	**14,949**
		15	NSR/ton	106.40	0.25	532,387	0.64	1,986,205	0.003	10,445
		20	NSR/ton	51.47	0.32	334,349	0.83	1,249,990	0.004	6,401
		25	NSR/ton	28.35	0.38	214,354	0.88	730,686	0.004	3,699
		30	NSR/ton	13.81	0.45	124,236	0.92	372,400	0.005	1,915
		35	NSR/ton	7.34	0.53	78,489	0.90	193,244	0.004	947
		40	NSR/ton	4.03	0.63	51,021	0.91	107,043	0.004	494
		45	NSR/ton	2.22	0.78	34,645	0.78	50,709	0.004	231
		50	NSR/ton	1.33	0.94	25,034	0.61	23,576	0.003	108
Total	Oxide	0.1	%	4.10	0.63	52,058	6.15	736,317	0.080	9,567
		0.12	%	4.01	0.65	51,849	6.27	732,990	0.081	9,510
		0.14	%	3.89	0.66	51,535	6.41	727,394	0.083	9,428
		0.16	**%**	**3.74**	**0.68**	**51,104**	**6.59**	**719,596**	**0.083**	**9,027**
		0.18	%	3.63	0.70	50,710	6.74	712,510	0.080	8,513
		0.2	%	3.50	0.72	50,223	6.91	704,728	0.079	8,090
		0.22	%	3.36	0.74	49,639	7.09	694,535	0.080	7,862
		0.24	%	3.21	0.76	48,929	7.32	684,168	0.077	7,227
		0.26	%	3.07	0.79	48,273	7.53	674,767	0.075	6,761
	Transition	0.1	%	37.20	0.36	269,622	4.00	4,337,944	0.038	41,421
		0.12	%	36.21	0.37	267,407	4.05	4,281,033	0.038	40,277
		0.14	%	34.80	0.38	263,728	4.12	4,180,728	0.038	38,127
		0.16	**%**	**33.23**	**0.39**	**259,012**	**4.20**	**4,066,021**	**0.037**	**35,693**
		0.18	%	31.61	0.40	253,538	4.28	3,946,118	0.037	34,479
		0.2	%	29.86	0.41	246,852	4.37	3,806,795	0.037	32,638
		0.22	%	27.83	0.43	238,341	4.47	3,625,052	0.038	30,887
		0.24	%	25.72	0.44	228,618	4.57	3,430,714	0.038	28,710
		0.26	%	23.61	0.46	218,072	4.69	3,232,039	0.039	26,905
	Sulfide	**10.45**	**NSR/ton**	**365.86**	**0.30**	**2,223,654**	**1.58**	**16,906,913**	**0.008**	**90,255**
		15	NSR/ton	265.36	0.32	1,685,198	1.66	12,883,991	0.009	72,332
		20	NSR/ton	138.40	0.40	1,096,974	2.05	8,257,259	0.012	50,346
		25	NSR/ton	72.50	0.47	679,168	2.26	4,775,988	0.015	32,321
		30	NSR/ton	36.32	0.56	404,727	2.58	2,734,099	0.018	18,848
		35	NSR/ton	19.37	0.65	253,407	2.91	1,645,931	0.017	9,679
		40	NSR/ton	10.64	0.76	162,251	3.36	1,043,491	0.021	6,482
		45	NSR/ton	6.24	0.89	111,123	3.99	726,212	0.028	5,047
		50	NSR/ton	3.84	1.03	78,823	4.89	547,552	0.037	4,172
Inferred										
Engels	Oxide	0.1	%	0.17	1.11	3,797	11.26	56,058	0.011	56
		0.12	%	0.16	1.18	3,773	11.91	55,603	0.010	48
		0.14	%	0.16	1.18	3,773	11.91	55,603	0.010	48
		0.16	**%**	**0.15**	**1.25**	**3,740**	**12.64**	**55,046**	**0.009**	**39**
		0.18	%	0.14	1.29	3,721	13.01	54,641	0.008	35
		0.2	%	0.13	1.43	3,660	14.39	53,740	0.006	22
		0.22	%	0.13	1.43	3,660	14.39	53,740	0.006	22
		0.24	%	0.11	1.67	3,563	16.63	51,739	0.001	4

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
		0.26	%	0.11	1.67	3,563	16.63	51,739	0.001	4
Engels	Transition	0.1	%	1.99	0.48	18,991	5.10	296,580	0.019	1,101
		0.12	%	1.93	0.49	18,850	5.20	293,120	0.019	1,053
		0.14	%	1.81	0.51	18,544	5.42	286,428	0.018	970
		0.16	**%**	**1.73**	**0.53**	**18,287**	**5.58**	**281,158**	**0.018**	**904**
		0.18	%	1.66	0.54	18,051	5.74	277,551	0.018	870
		0.2	%	1.59	0.56	17,810	5.87	273,098	0.018	825
		0.22	%	1.56	0.57	17,654	5.96	270,839	0.018	805
		0.24	%	1.45	0.59	17,135	6.25	263,583	0.018	761
		0.26	%	1.35	0.62	16,682	6.50	256,775	0.019	732
	Sulfide	**10.45**	**NSR/ton**	**6.93**	**0.38**	**52,445**	**5.08**	**1,027,412**	**0.041**	**8,280**
		15	NSR/ton	5.23	0.44	46,246	5.77	880,158	0.045	6,851
		20	NSR/ton	3.71	0.51	38,220	6.49	703,184	0.054	5,832
		25	NSR/ton	2.70	0.58	31,435	7.30	575,814	0.059	4,689
		30	NSR/ton	1.90	0.65	24,816	8.08	448,550	0.066	3,681
		35	NSR/ton	1.26	0.73	18,566	8.44	311,021	0.075	2,756
		40	NSR/ton	0.79	0.83	13,173	9.26	213,204	0.083	1,916
		45	NSR/ton	0.52	0.93	9,591	10.65	160,694	0.094	1,416
		50	NSR/ton	0.38	0.99	7,625	11.89	133,174	0.106	1,183
Lambs Ridge	Oxide	0.1	%	0.00	0.00	0	0.00	0	0.000	0
		0.12	%	0.00	0.00	0	0.00	0	0.000	0
		0.14	%	0.00	0.00	0	0.00	0	0.000	0
		0.16	**%**	**0.00**	**0.00**	**0**	**0.00**	**0**	**0.000**	**0**
		0.18	%	0.00	0.00	0	0.00	0	0.000	0
		0.2	%	0.00	0.00	0	0.00	0	0.000	0
		0.22	%	0.00	0.00	0	0.00	0	0.000	0
		0.24	%	0.00	0.00	0	0.00	0	0.000	0
		0.26	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.1	%	0.00	0.00	0	0.00	0	0.000	0
		0.12	%	0.00	0.00	0	0.00	0	0.000	0
		0.14	%	0.00	0.00	0	0.00	0	0.000	0
		0.16	**%**	**0.00**	**0.00**	**0**	**0.00**	**0**	**0.000**	**0**
		0.18	%	0.00	0.00	0	0.00	0	0.000	0
		0.2	%	0.00	0.00	0	0.00	0	0.000	0
		0.22	%	0.00	0.00	0	0.00	0	0.000	0
		0.24	%	0.00	0.00	0	0.00	0	0.000	0
		0.26	%	0.00	0.00	0	0.00	0	0.000	0

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
Lambs Ridge	Sulfide	**10.45**	**NSR/ton**	**3.46**	**0.30**	**20,954**	**0.00**	**0**	**0.000**	**0**
		15	NSR/ton	2.67	0.33	17,529	0.00	0	0.000	0
		20	NSR/ton	1.16	0.40	9,292	0.00	0	0.000	0
		25	NSR/ton	0.46	0.45	4,182	0.00	0	0.000	0
		30	NSR/ton	0.07	0.50	752	0.00	0	0.000	0
		35	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		40	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		45	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		50	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
Moonlight	Oxide	0.1	%	0.00	0.00	0	0.00	0	0.000	0
		0.12	%	0.00	0.00	0	0.00	0	0.000	0
		0.14	%	0.00	0.00	0	0.00	0	0.000	0
		0.16	**%**	**0.00**	**0.00**	**0**	**0.00**	**0**	**0.000**	**0**
		0.18	%	0.00	0.00	0	0.00	0	0.000	0
		0.2	%	0.00	0.00	0	0.00	0	0.000	0
		0.22	%	0.00	0.00	0	0.00	0	0.000	0
		0.24	%	0.00	0.00	0	0.00	0	0.000	0
		0.26	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.1	%	0.00	0.00	0	0.00	0	0.000	0
		0.12	%	0.00	0.00	0	0.00	0	0.000	0
		0.14	%	0.00	0.00	0	0.00	0	0.000	0
		0.16	**%**	**0.00**	**0.00**	**0**	**0.00**	**0**	**0.000**	**0**
		0.18	%	0.00	0.00	0	0.00	0	0.000	0
		0.2	%	0.00	0.00	0	0.00	0	0.000	0
		0.22	%	0.00	0.00	0	0.00	0	0.000	0
		0.24	%	0.00	0.00	0	0.00	0	0.000	0
		0.26	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	**10.45**	**NSR/ton**	**30.82**	**0.28**	**175,635**	**0.09**	**81,857**	**0.000**	**35**
		15	NSR/ton	19.56	0.33	129,498	0.09	54,107	0.000	12
		20	NSR/ton	7.41	0.41	61,280	0.12	25,877	0.000	4
		25	NSR/ton	2.90	0.50	29,055	0.06	5,238	0.000	0
		30	NSR/ton	1.08	0.62	13,281	0.02	663	0.000	0
		35	NSR/ton	0.68	0.67	9,102	0.00	0	0.000	0
		40	NSR/ton	0.35	0.74	5,132	0.00	0	0.000	0
		45	NSR/ton	0.13	0.83	2,132	0.00	0	0.000	0
		50	NSR/ton	0.04	0.99	738	0.00	0	0.000	0
Copper Mountain	Oxide	0.1	%	0.00	0.00	0	0.00	0	0.000	0
		0.12	%	0.00	0.00	0	0.00	0	0.000	0
		0.14	%	0.00	0.00	0	0.00	0	0.000	0
		0.16	**%**	**0.00**	**0.00**	**0**	**0.00**	**0**	**0.000**	**0**
		0.18	%	0.00	0.00	0	0.00	0	0.000	0
		0.2	%	0.00	0.00	0	0.00	0	0.000	0
		0.22	%	0.00	0.00	0	0.00	0	0.000	0
		0.24	%	0.00	0.00	0	0.00	0	0.000	0
		0.26	%	0.00	0.00	0	0.00	0	0.000	0

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
Copper Mountain	Transition	0.1	%	0.00	0.00	0	0.00	0	0.000	0
		0.12	%	0.00	0.00	0	0.00	0	0.000	0
		0.14	%	0.00	0.00	0	0.00	0	0.000	0
		0.16	**%**	**0.00**	**0.00**	**0**	**0.00**	**0**	**0.000**	**0**
		0.18	%	0.00	0.00	0	0.00	0	0.000	0
		0.2	%	0.00	0.00	0	0.00	0	0.000	0
		0.22	%	0.00	0.00	0	0.00	0	0.000	0
		0.24	%	0.00	0.00	0	0.00	0	0.000	0
		0.26	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	**10.45**	**NSR/ton**	**3.90**	**0.27**	**21,320**	**0.00**	**0**	**0.000**	**0**
		15	NSR/ton	2.30	0.31	14,479	0.00	0	0.000	0
		20	NSR/ton	0.67	0.40	5,346	0.00	0	0.000	0
		25	NSR/ton	0.22	0.49	2,124	0.00	0	0.000	0
		30	NSR/ton	0.04	0.66	566	0.00	0	0.000	0
		35	NSR/ton	0.00	0.00	127	0.00	0	0.000	0
		40	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		45	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		50	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
Superior	Sulfide	**10.45**	**NSR/ton**	**17.60**	**0.29**	**101,817**	**0.01**	**2,681**	**0.000**	**23**
		15	NSR/ton	11.96	0.33	78,190	0.01	1,772	0.000	14
		20	NSR/ton	5.07	0.40	40,374	0.00	0	0.000	0
		25	NSR/ton	2.14	0.45	19,089	0.00	0	0.000	0
		30	NSR/ton	0.14	0.50	1,392	0.00	0	0.000	0
		35	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		40	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		45	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
		50	NSR/ton	0.00	0.00	0	0.00	0	0.000	0
Total	Oxide	0.1	%	0.17	1.11	3,797	11.26	56,058	0.011	56
		0.12	%	0.16	1.18	3,773	11.91	55,603	0.010	48
		0.14	%	0.16	1.18	3,773	11.91	55,603	0.010	48
		0.16	**%**	**0.15**	**1.25**	**3,740**	**12.64**	**55,046**	**0.009**	**39**
		0.18	%	0.14	1.29	3,721	13.01	54,641	0.008	35
		0.2	%	0.13	1.43	3,660	14.39	53,740	0.006	22
		0.22	%	0.13	1.43	3,660	14.39	53,740	0.006	22
		0.24	%	0.11	1.67	3,563	16.63	51,739	0.001	4
		0.26	%	0.11	1.67	3,563	16.63	51,739	0.001	4
	Transition	0.1	%	1.99	0.48	18,991	5.10	296,580	0.019	1,101
		0.12	%	1.93	0.49	18,850	5.20	293,120	0.019	1,053
		0.14	%	1.81	0.51	18,544	5.42	286,428	0.018	970
		0.16	**%**	**1.73**	**0.53**	**18,287**	**5.58**	**281,158**	**0.018**	**904**
		0.18	%	1.66	0.54	18,051	5.74	277,551	0.018	870
		0.2	%	1.59	0.56	17,810	5.87	273,098	0.018	825
		0.22	%	1.56	0.57	17,654	5.96	270,839	0.018	805
		0.24	%	1.45	0.59	17,135	6.25	263,583	0.018	761
		0.26	%	1.35	0.62	16,682	6.50	256,775	0.019	732

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
Total	Sulfide	**10.45**	**NSR/ton**	**62.71**	**0.30**	**372,171**	**0.61**	**1,111,950**	**0.005**	**8,338**
		15	NSR/ton	41.73	0.34	285,942	0.77	936,037	0.006	6,877
		20	NSR/ton	18.02	0.43	154,511	1.39	729,061	0.011	5,835
		25	NSR/ton	8.43	0.51	85,884	2.36	581,052	0.019	4,689
		30	NSR/ton	3.24	0.63	40,808	4.76	449,213	0.039	3,681
		35	NSR/ton	1.94	0.72	27,795	5.49	311,021	0.049	2,756
		40	NSR/ton	1.14	0.81	18,305	6.43	213,204	0.058	1,916
		45	NSR/ton	0.65	0.91	11,723	8.54	160,694	0.075	1,416
		50	NSR/ton	0.42	0.99	8,363	10.84	133,174	0.096	1,183

14.14 Factors that May Affect the Mineral Resource Estimate

The oxide/ sulfide boundary could change with the acquisition of additional data. The base of the oxide surface is not expected to change very much with the new drilling, but metallurgical recoveries could vary within the defined oxide zone.

GRE is unaware of any other known environmental, permitting, legal, title, taxation, socio-economic, marketing, political factors that may materially affect the mineral resource.

15 MINERAL RESERVES

There are no Mineral Reserves in this Technical Report.

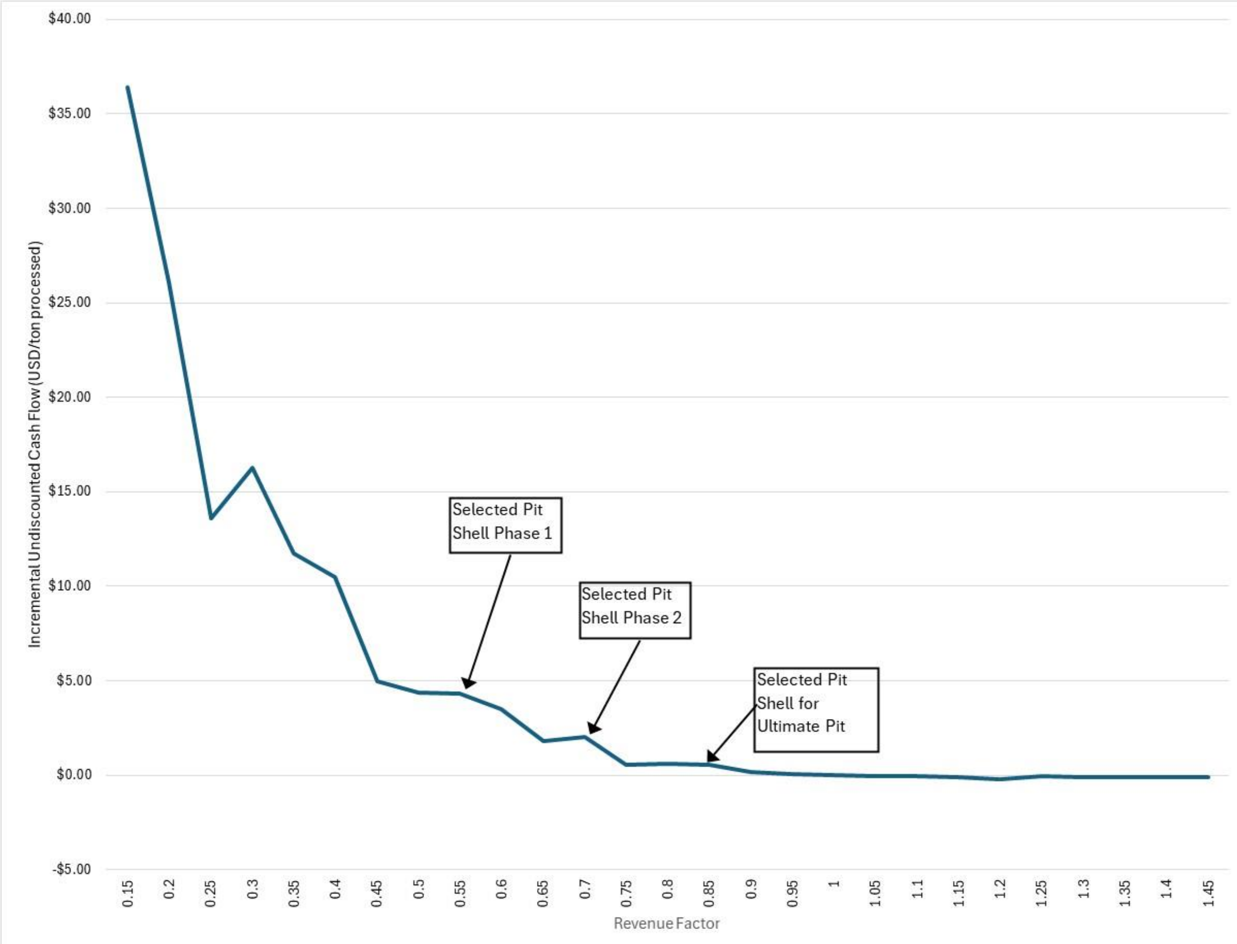
16 MINING METHODS

The Moonlight-Superior Copper Project will employ conventional open pit mining techniques using hydraulic shovels and rear dump rigid frame haul trucks. The mine plan is designed to deliver an average of 60,000 tons of sulfide material to the mill per day and 10,000 tons of oxide and transition material to the heap leach per day. The average daily waste production rate over the life of the mine would be 73,000 tons per day. Waste rock would be placed in waste rock storage facilities near each pit area.

16.1 Whittle Pit Shell Analysis

The Whittle pit shells created for the Resource Estimation, as described in Section 14, were analyzed to determine the optimal pit shell for economic extraction of the mineral resources contained in the block model. Ms. Lane of GRE examined the marginal impact on undiscounted cashflow for each Whittle pit shell series. This analysis examines the impact that each incremental increase in the pit shell has on the undiscounted cashflow divided by the number of tons that are processed. Ms. Lane of GRE selected a pit shell that gave a local spike in the marginal impact on undiscounted cashflow or that represented the largest pit shell before a significant increase in waste tonnage, as shown in Figure 16-1 through Figure 16-3. Ms. Lane also selected smaller Whittle pit shells with local spikes in the marginal impact on undiscounted cashflow for interim phases, as shown on Figure 16-1 through Figure 16-3.

Figure 16-1: Moonlight-Superior Project Open Pit Marginal Impact on Undiscounted Cashflow and Whittle Results - Northwest



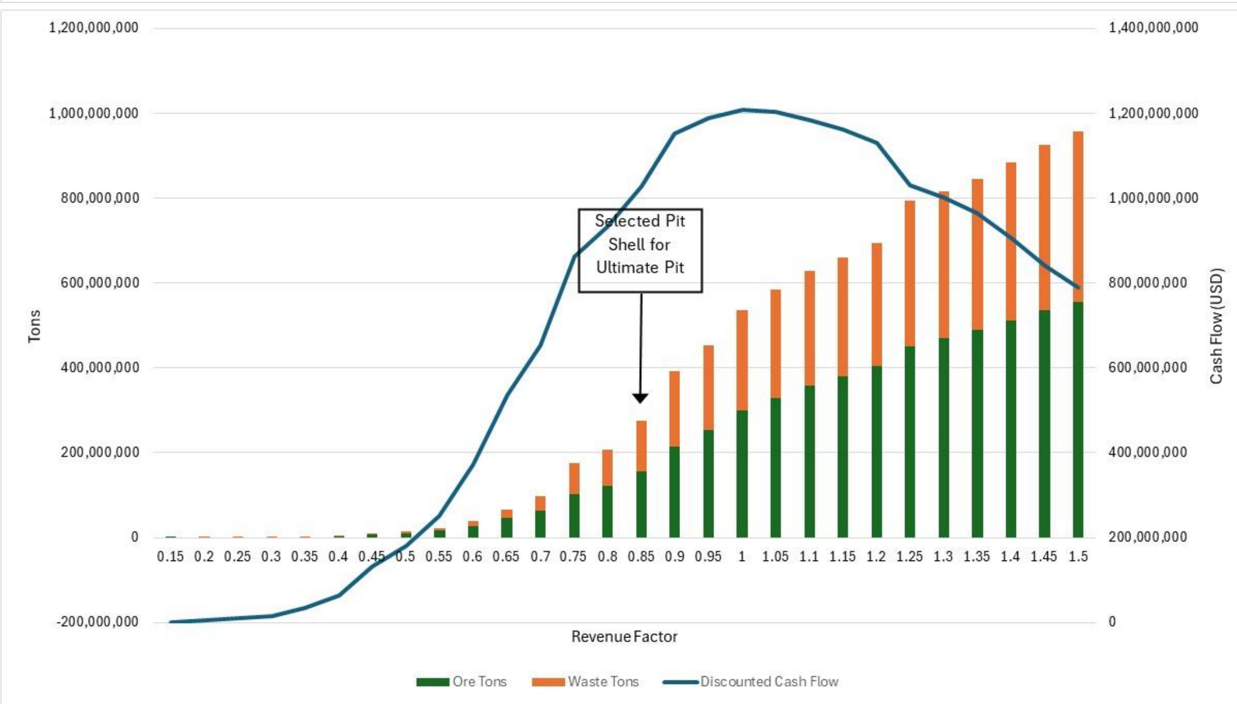
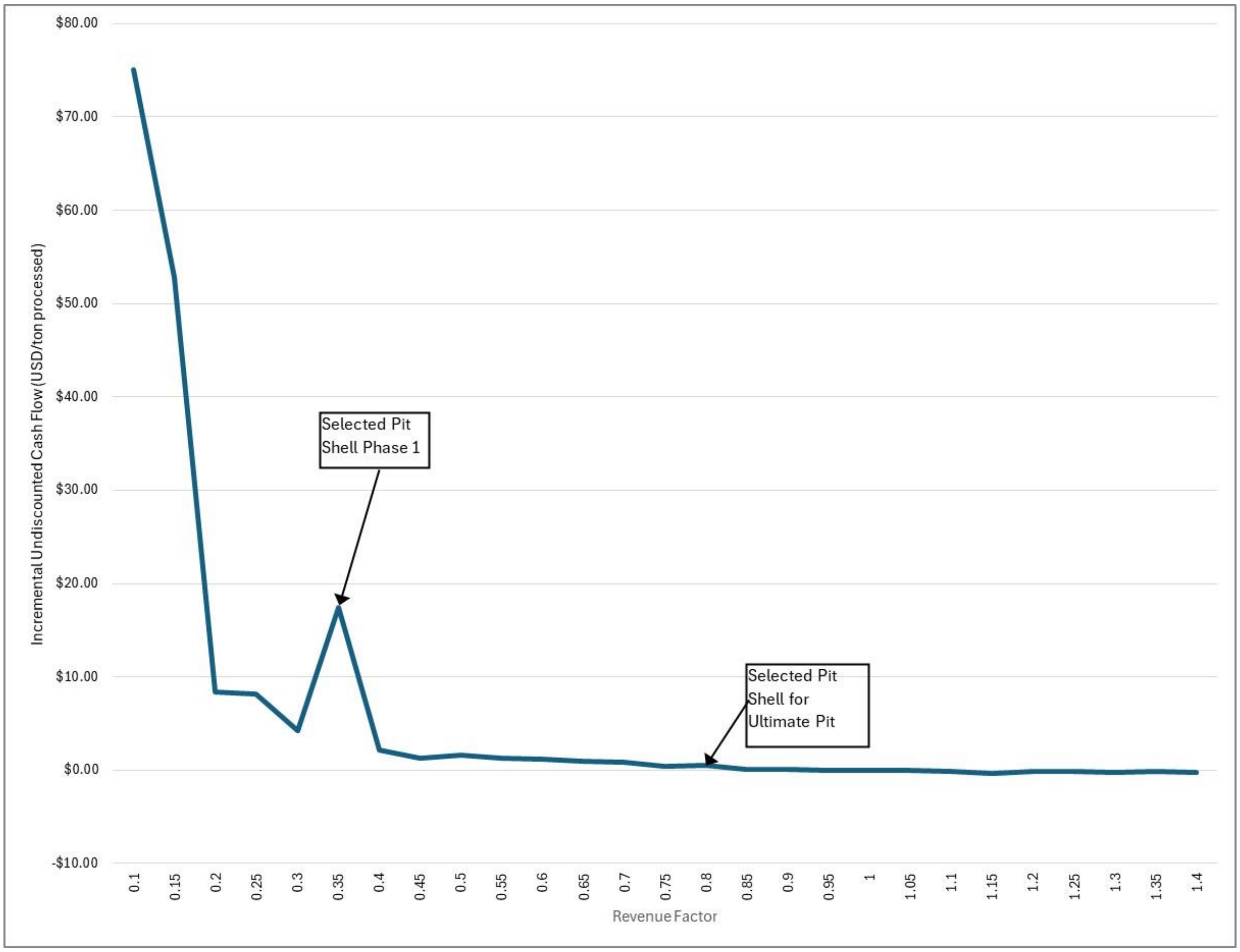


Figure 16-2: Moonlight-Superior Project Open Pit Marginal Impact on Undiscounted Cashflow Cashflow and Whittle Results – Northeast



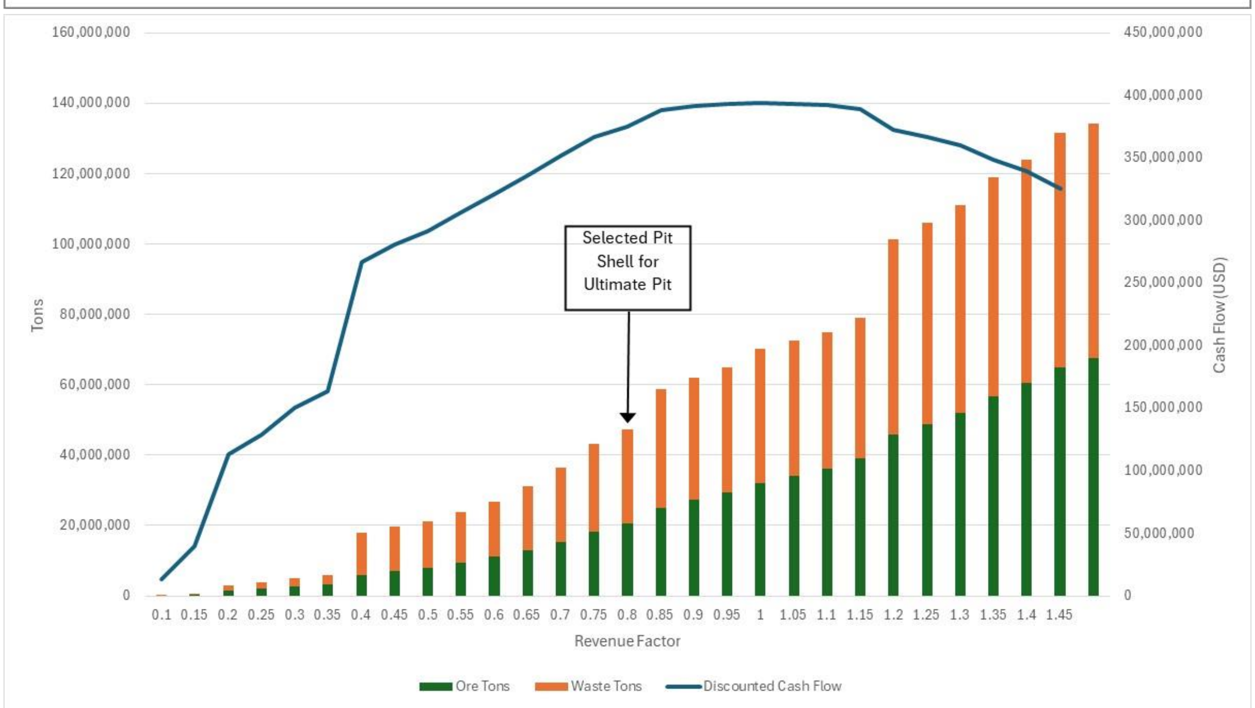
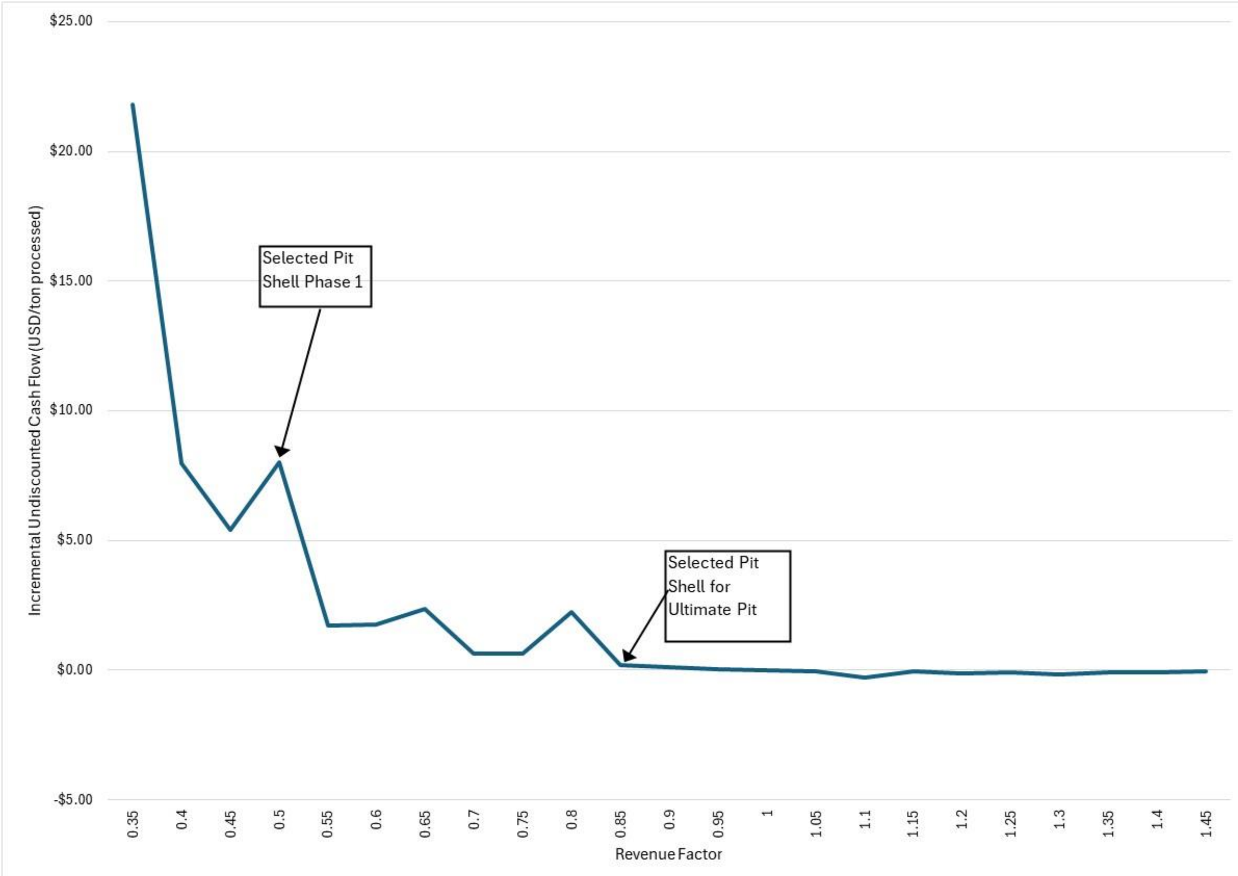
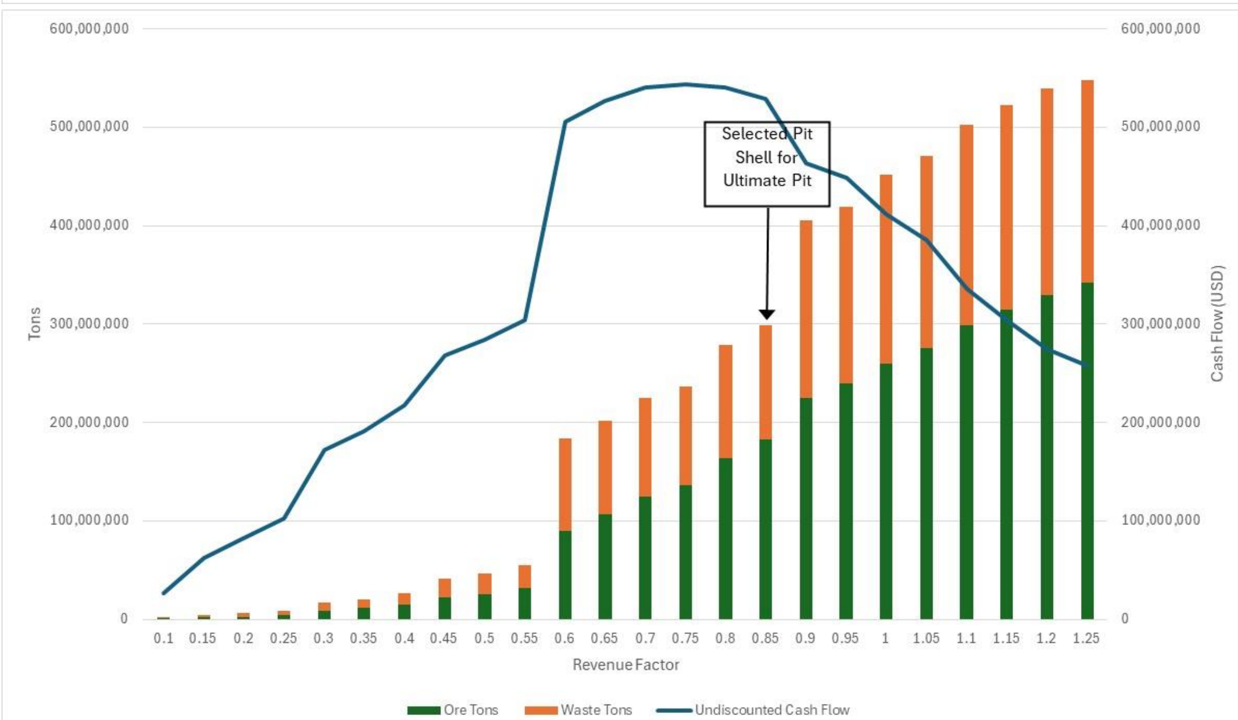


Figure 16-3: Moonlight-Superior Project Open Pit Marginal Impact on Undiscounted Cashflow and Whittle Results – South





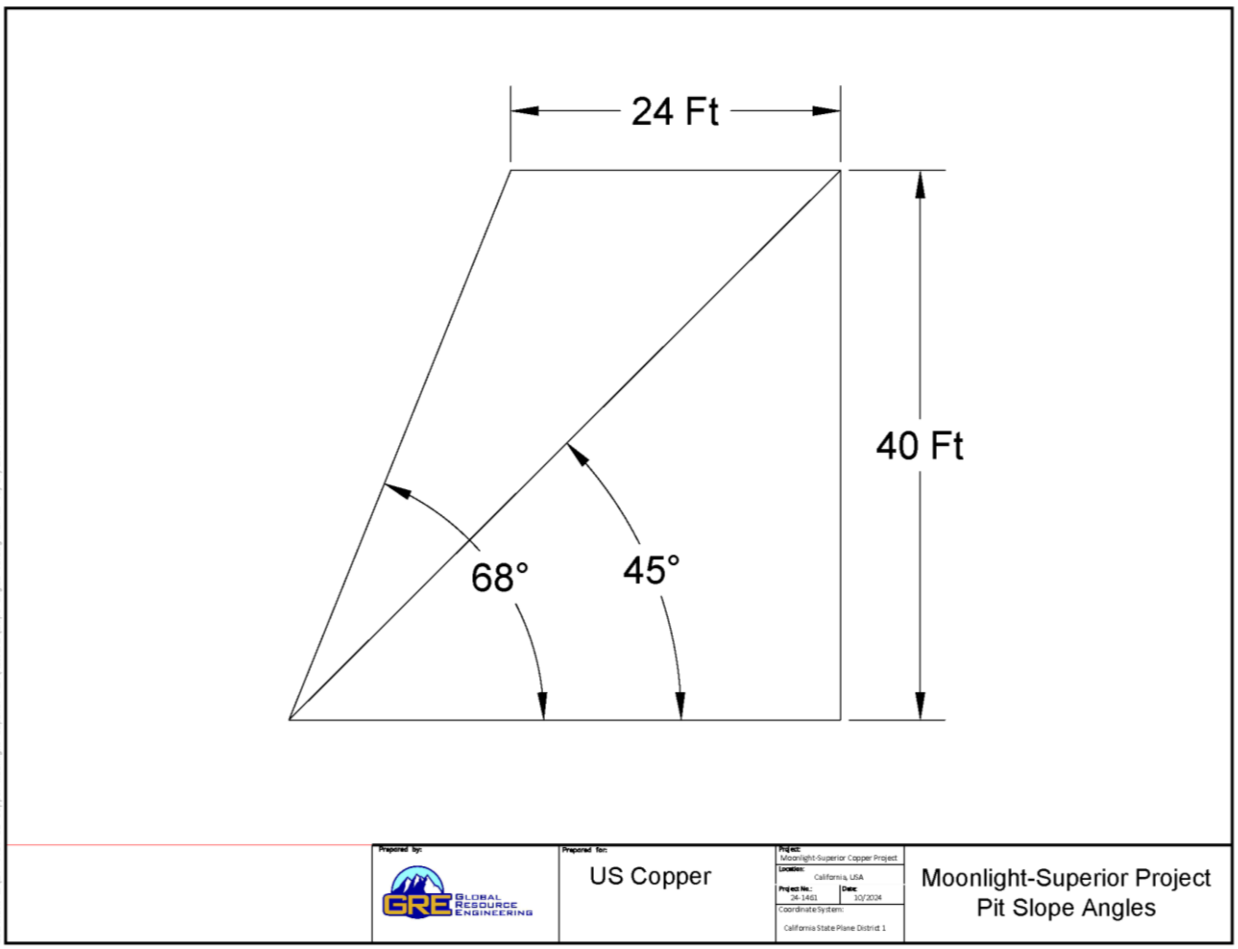
16.2 Pit Design

Ms. Lane of GRE used a single bench format consisting of 40-foot vertical benches with a horizontal 24-foot catch bench. The resulting open pit parameters are listed in Table 16-1 and illustrated in Figure 16-4.

Table 16-1: Moonlight-Superior Project Open Pit Geotechnical Design Parameters

Pit Design Parameters	Value (degrees)
Max Inter-ramp Angle Hard Rock	45
Max Bench Face Angle	68

Figure 16-4: Cross-Section of Typical Pit Slope



The Whittle pit shells selected for each phase of the mine design were imported into GEOVIA Surpac software and designed with pit slopes and benching as described above and with haul roads. Haul roads were designed with a minimum width of 112 feet and a maximum gradient of 10%. Haul ramps and roads have been designed to accommodate two-way traffic using 250-ton haul trucks, water diversion ditches, and safety berms. Minor sections were narrowed to a single lane of 70 feet.

The designed pits are illustrated in

Figure 16-5 through Figure 16-8.

Figure 16-5: Moonlight-Superior Project Moonlight and Copper Mountain Designed Pits

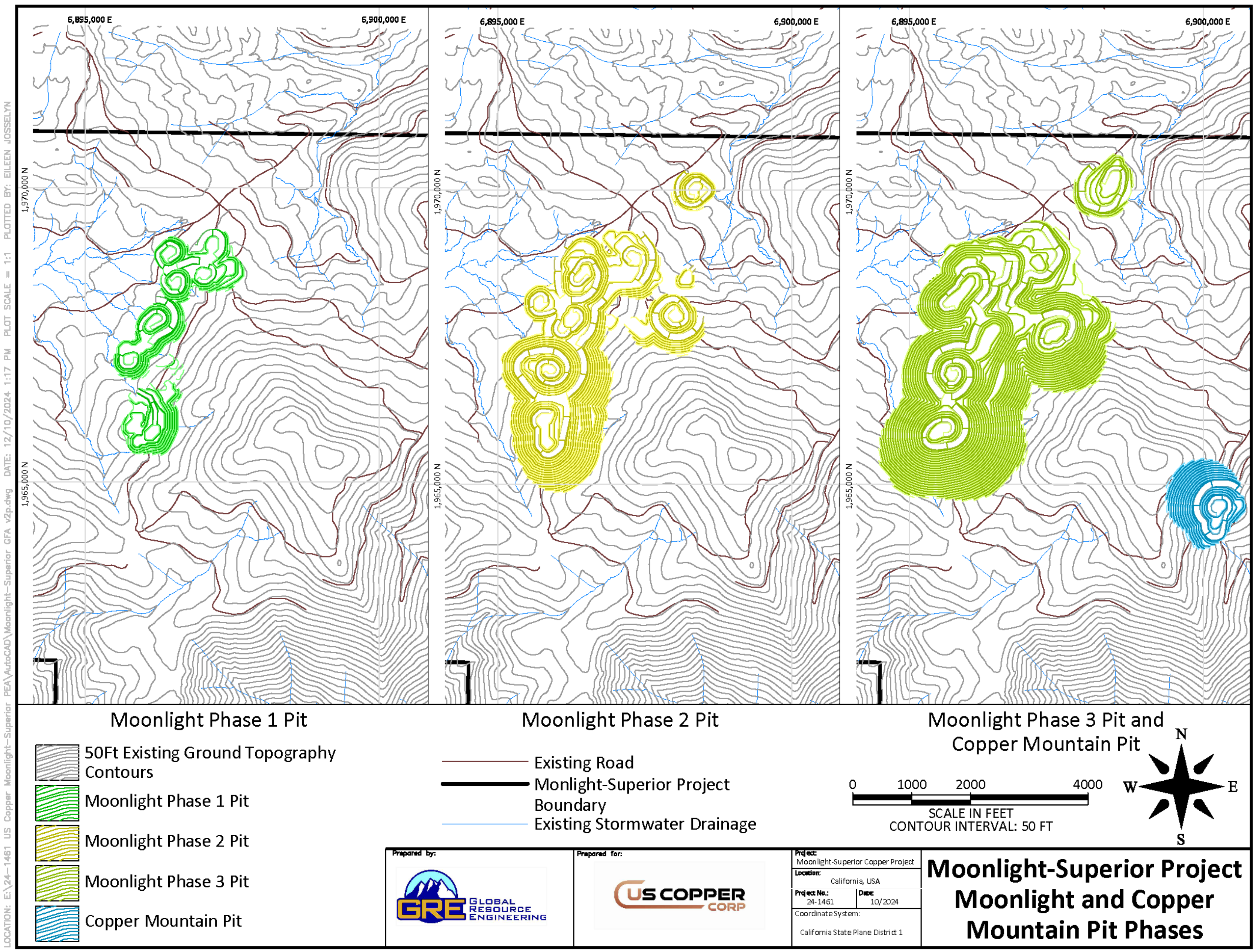


Figure 16-6: Moonlight-Superior Project Engels and Lamb's Ridge Designed Pits

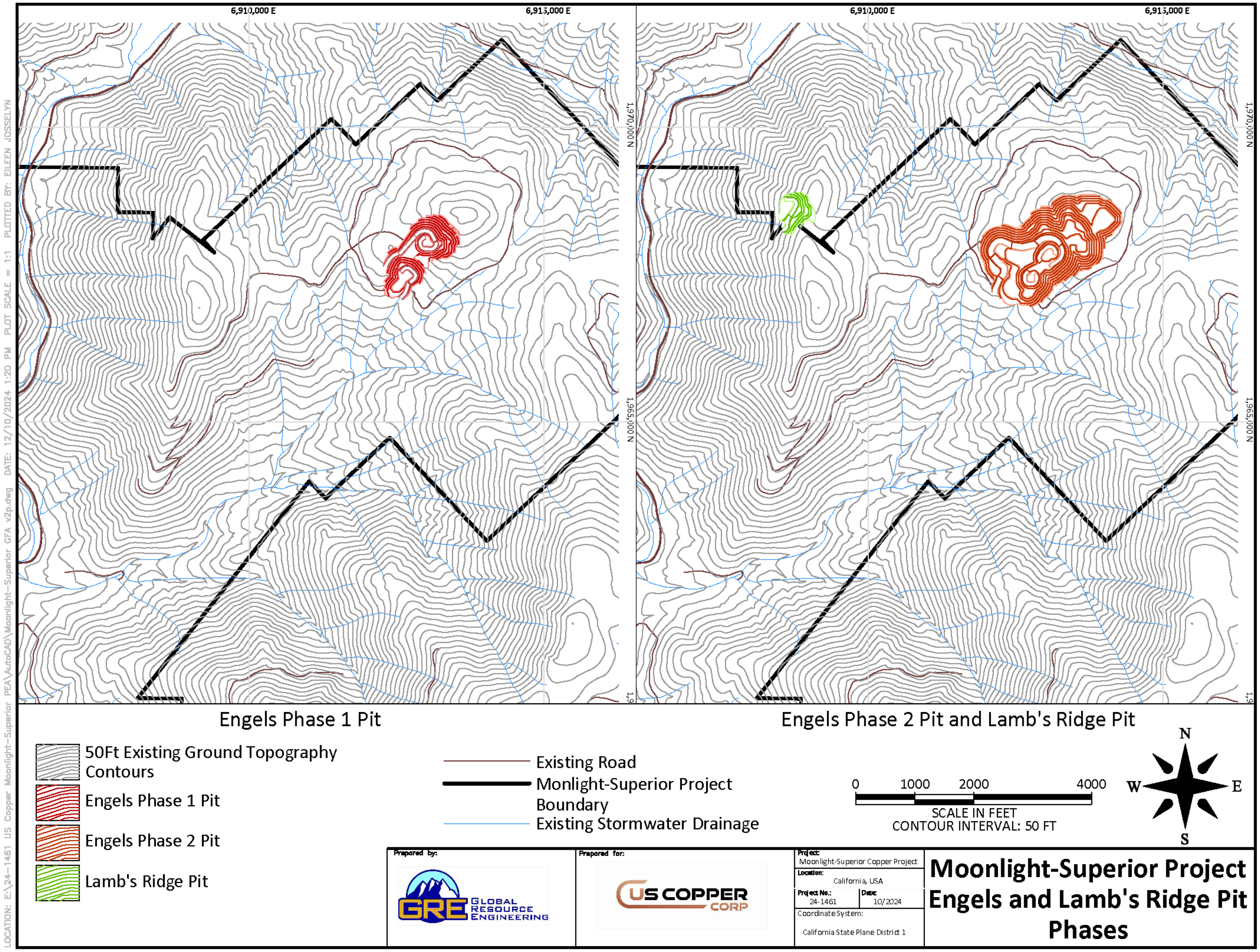


Figure 16-7: Moonlight-Superior Project Superior Designed Pits

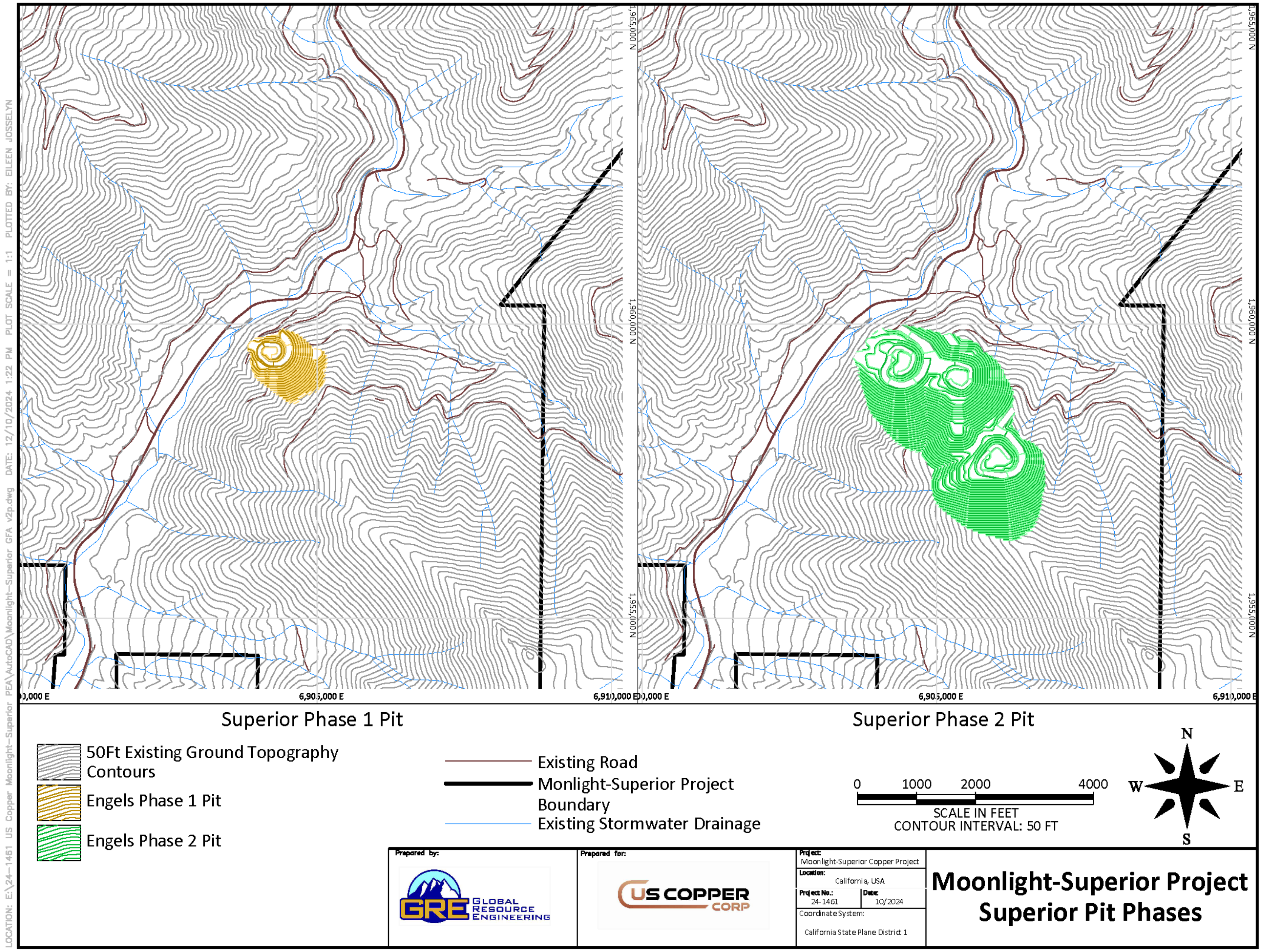
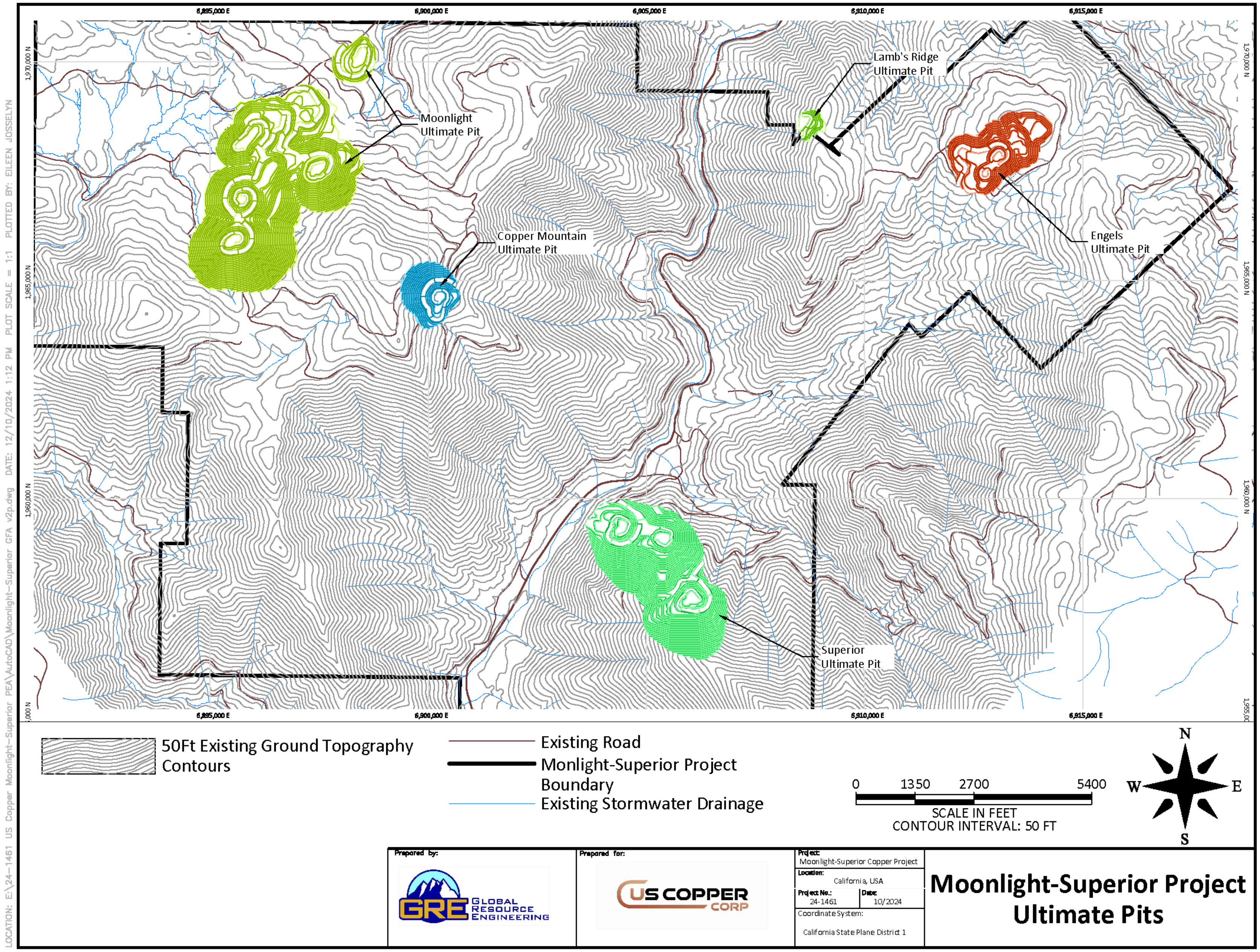


Figure 16-8: Moonlight-Superior Project Ultimate Designed Pits



16.3 Base Case

The resources for each ultimate pit and intermediate phase were reported out of Surpac by bench and pit phase. A summary of the reported resources for each pit phase are shown in Table 16-2. Cutoffs of 15 NSR for high-grade sulfide material, 10.45 NSR for low-grade sulfide material, and 0.28% Cu for oxide and transition material were used.

Table 16-2: Moonlight-Superior Copper Project Pit Phase Resources

Pit Phase	Mineralized Materal (million tons)	Waste (million tons)	Contained Cu ('000s lbs)	Contained Ag ('000s oz)	Contained Au ('000s oz)	Cu Grade (%)	Ag Grade (opt)	Au Grade (opt)	Stripping Ratio
Moonlight Phase 1	18.3	5.48	157,456	2,801	18	0.430	0.153	0.00096	0.30
Moonlight Phase 2	60.71	39.18	388,477	6,347	55	0.320	0.105	0.00091	0.65
Moonlight Phase 3	90.92	98.26	549,455	3,461	7	0.302	0.038	0.00008	1.08
Total Moonlight	**169.94**	**142.91**	**1,095,388**	**12,610**	**80**	**0.322**	**0.074**	**0.00047**	**0.84**
Engels Phase 1	5.06	2.61	78,420	1,129	7	0.776	0.223	0.00138	0.52
Engels Phase 2	13.35	29.06	136,178	2,291	19	0.510	0.172	0.00142	2.18
Total Engels	**18.41**	**31.66**	**214,599**	**3,421**	**26**	**0.583**	**0.186**	**0.00141**	**1.72**
Superior Phase 1	9.93	4.42	84,828	210	1	0.427	0.021	0.00009	0.45
Superior Phase 2	106.09	91.13	627,526	2,519	13	0.296	0.024	0.00013	0.86
Total Superior	**116.02**	**95.55**	**712,354**	**2,729**	**14**	**0.307**	**0.024**	**0.00012**	**0.82**
Copper Mountain	9.23	11.27	52,625	0	0	0.285	0.000	0.00000	1.22
Lamb's Ridge	1.02	0.65	7,419	0	0	0.364	0.000	0.00000	0.63
Total	**314.61**	**282.04**	**2,082,384**	**18,760**	**120**	**0.331**	**0.060**	**0.00038**	**0.90**

16.4 Mine Schedule

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

- High-Grade Sulfide Mining Production Rate (SMPR): 60,000 tons per day (tpd)
- Mine Operating Days per Week: 7
- Mine Operating Weeks per Year: 52
- Mine Operating Shifts per Day: 2
- Mine Operating Hours per Shift: 12

Pre-stripping of waste was included if waste occurred on a bench that had no corresponding processable material or if the tonnage of waste on a bench exceeded ten times the tonnage of processable material on that bench. The production rate for pre-strip benches was generally set to five times the sulfide material production rate, or 270,000 tpd, but varied to smooth out the mining fleet where possible.

The schedule included a gradual ramp-up to full production during the first production year as follows: 25% for the first quarter, 50% for the second quarter, 75% for the third quarter, and 100% for the remaining duration. The mining schedule is summarized in Table 16-3and illustrated in Figure 16-9.

Table 16-3: Moonlight-Superior Copper Project Base Case Mine Schedule Summary

Pit Phase	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Sulfide High Grade Mineralized Tons (millions)											
E1	0.92	-	-	-	-	-	-	-	-	-	**0.92**
LR1	1.00	-	-	-	-	-	-	-	-	-	**1.00**
M1	6.91	-	-	-	-	-	-	-	-	-	**6.91**
E2	2.27	4.75	-	-	-	-	-	-	-	-	**7.01**
M2	3.83	15.33	15.33	7.26	-	-	-	-	-	-	**41.75**
M3a	-	-	-	6.70	15.33	15.33	15.33	4.99	-	-	**57.68**
CM1	-	1.77	3.53	-	-	-	-	-	-	-	**5.30**
S1	-	-	3.04	4.77	-	-	-	-	-	-	**7.81**
S2	-	-	-	2.05	6.57	6.57	6.57	11.38	21.90	9.31	**64.36**
Oxide High Grade Mineralized Tons (millions)											
E1	1.48	-	-	-	-	-	-	-	-	-	**1.48**
LR1	-	-	-	-	-	-	-	-	-	-	**-**
M1	0.75	-	-	-	-	-	-	-	-	-	**0.75**
E2	0.05	0.44	-	-	-	-	-	-	-	-	**0.49**
M2	0.11	0.00	0.05	-	-	-	-	-	-	-	**0.17**
M3a	-	-	-	-	-	-	-	-	-	-	**-**
CM1	-	-	-	-	-	-	-	-	-	-	**-**
S1	-	-	-	-	-	-	-	-	-	-	**-**
S2	-	-	-	-	-	-	-	-	-	-	**-**
Transition High Grade Mineralized Tons (millions)											
E1	1.48	-	-	-	-	-	-	-	-	-	**1.48**
LR1	-	-	-	-	-	-	-	-	-	-	**-**
M1	9.16	-	-	-	-	-	-	-	-	-	**9.16**
E2	0.11	3.46	-	-	-	-	-	-	-	-	**3.57**
M2	1.72	2.46	0.86	-	-	-	-	-	-	-	**5.05**
M3a	-	-	-	1.37	0.39	-	-	-	-	-	**1.75**
CM1	-	-	-	-	-	-	-	-	-	-	**-**
S1	-	-	-	-	-	-	-	-	-	-	**-**
S2	-	-	-	-	-	-	-	-	-	-	**-**

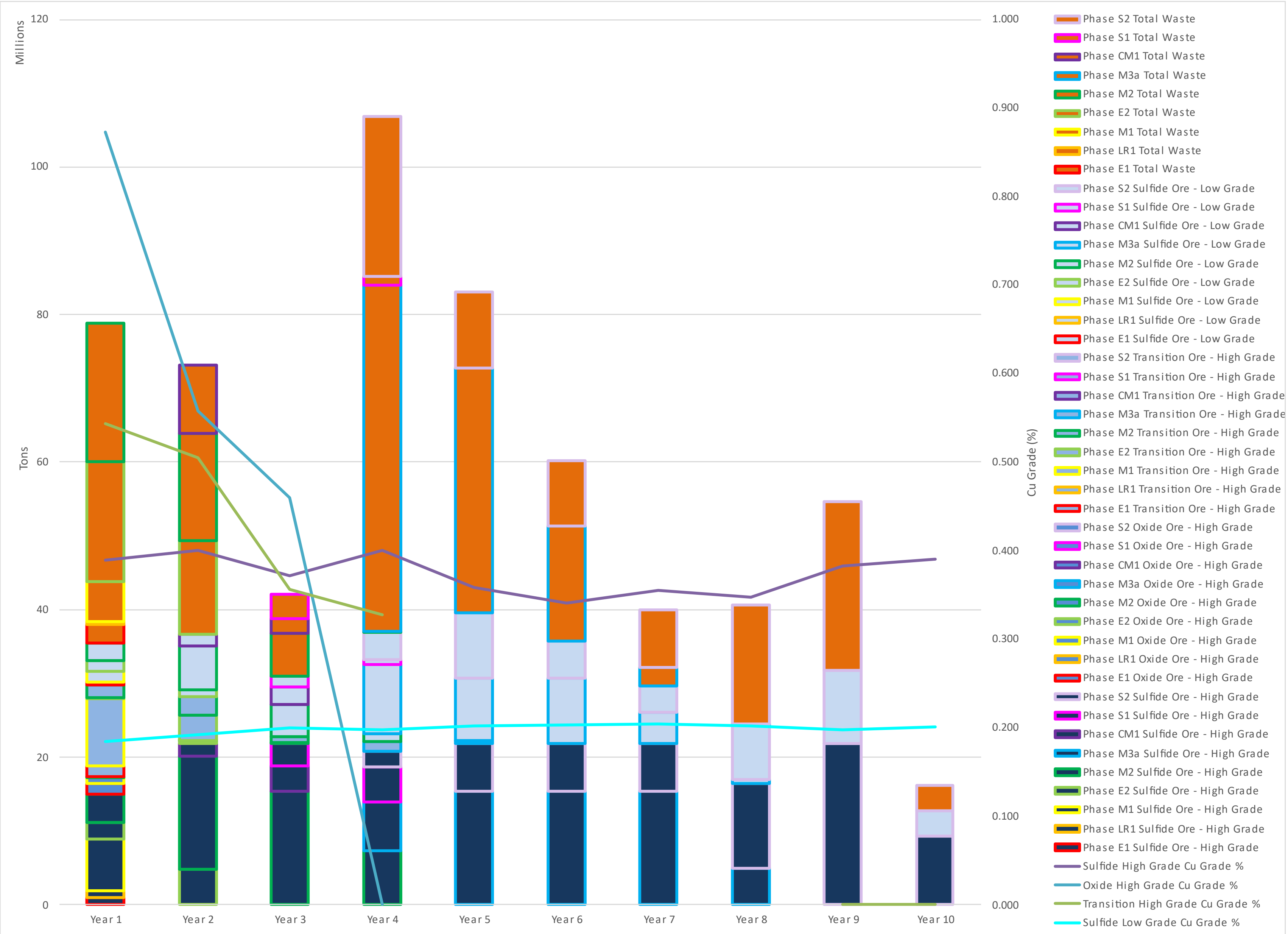
Pit Phase	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Sulfide Low Grade Mineralized Tons (millions)											
E1	0.42	-	-	-	-	-	-	-	-	-	**0.42**
LR1	0.01	-	-	-	-	-	-	-	-	-	**0.01**
M1	1.48	-	-	-	-	-	-	-	-	-	**1.48**
E2	1.36	0.92	-	-	-	-	-	-	-	-	**2.28**
M2	2.47	5.94	4.33	1.01	-	-	-	-	-	-	**13.75**
M3a	-	-	-	9.43	8.40	8.79	4.24	0.62	-	-	**31.49**
CM1	-	1.54	2.38	-	-	-	-	-	-	-	**3.93**
S1	-	-	1.46	0.66	-	-	-	-	-	-	**2.12**
S2	-	-	-	3.66	8.87	5.07	3.45	7.37	9.89	3.41	**41.73**
Total Mineralized Tons	**35.53**	**36.62**	**30.99**	**36.91**	**39.56**	**35.76**	**29.59**	**24.36**	**31.79**	**12.72**	**313.84**
Waste Tons (millions)											
E1	2.50	-	-	-	-	-	-	-	-	-	**2.50**
LR1	0.29	-	-	-	-	-	-	-	-	-	**0.29**
M1	5.45	-	-	-	-	-	-	-	-	-	**5.45**
E2	16.32	12.74	-	-	-	-	-	-	-	-	**29.06**
M2	18.73	14.55	5.74	0.15	-	-	-	-	-	-	**39.18**
M3a	-	-	-	46.94	33.14	15.58	2.51	0.08	-	-	**98.26**
CM1	-	9.27	2.00	-	-	-	-	-	-	-	**11.27**
S1	-	-	3.28	1.14	-	-	-	-	-	-	**4.42**
S2	-	-	-	21.77	10.29	8.79	7.81	16.14	22.91	3.41	**91.13**
Total Waste Tons	**43.29**	**36.55**	**11.02**	**70.01**	**43.43**	**24.37**	**10.32**	**16.22**	**22.91**	**3.41**	**281.55**
Sulfide High Grade Cu Lbs Contained (millions)											
E1	8.63	-	-	-	-	-	-	-	-	-	**8.63**
LR1	7.32	-	-	-	-	-	-	-	-	-	**7.32**
M1	57.57	-	-	-	-	-	-	-	-	-	**57.57**
E2	19.82	59.88	-	-	-	-	-	-	-	-	**79.70**
M2	22.78	101.69	117.84	56.02	-	-	-	-	-	-	**298.34**
M3a	-	-	-	43.33	114.84	104.23	110.45	39.37	-	-	**412.23**
CM1	-	13.41	22.85	-	-	-	-	-	-	-	**36.26**
S1	-	-	21.91	54.32	-	-	-	-	-	-	**76.23**
S2	-	-	-	12.56	41.96	44.94	45.28	74.29	167.54	72.63	**459.21**

Pit Phase	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Oxide High Grade Cu Lbs Contained											
E1	33.55	-	-	-	-	-	-	-	-	-	**33.55**
LR1	-	-	-	-	-	-	-	-	-	-	**-**
M1	6.98	-	-	-	-	-	-	-	-	-	**6.98**
E2	0.44	4.92	-	-	-	-	-	-	-	-	**5.36**
M2	0.74	0.03	0.49	-	-	-	-	-	-	-	**1.27**
M3a	-	-	-	-	-	-	-	-	-	-	**-**
CM1	-	-	-	-	-	-	-	-	-	-	**-**
S1	-	-	-	-	-	-	-	-	-	-	**-**
S2	-	-	-	-	-	-	-	-	-	-	**-**
Transition High Grade Cu Lbs Contained											
E1	34.83	-	-	-	-	-	-	-	-	-	**34.83**
LR1	-	-	-	-	-	-	-	-	-	-	**-**
M1	87.33	-	-	-	-	-	-	-	-	-	**87.33**
E2	1.17	41.75	-	-	-	-	-	-	-	-	**42.92**
M2	12.03	18.05	6.16	-	-	-	-	-	-	-	**36.24**
M3a	-	-	-	8.95	2.52	-	-	-	-	-	**11.46**
CM1	-	-	-	-	-	-	-	-	-	-	**-**
S1	-	-	-	-	-	-	-	-	-	-	**-**
S2	-	-	-	-	-	-	-	-	-	-	**-**
Sulfide Low Grade Cu Lbs Contained											
E1	1.41	-	-	-	-	-	-	-	-	-	**1.41**
LR1	0.05	-	-	-	-	-	-	-	-	-	**0.05**
M1	5.58	-	-	-	-	-	-	-	-	-	**5.58**
E2	4.94	3.27	-	-	-	-	-	-	-	-	**8.21**
M2	9.27	22.70	16.75	3.93	-	-	-	-	-	-	**52.64**
M3a	-	-	-	36.90	33.51	35.50	17.18	2.67	-	-	**125.76**
CM1	-	6.31	10.05	-	-	-	-	-	-	-	**16.36**
S1	-	-	5.85	2.74	-	-	-	-	-	-	**8.60**
S2	-	-	-	14.82	36.24	20.68	14.18	29.60	39.10	13.69	**168.32**
Total Cu Lbs Contained	**314.43**	**272.00**	**201.90**	**233.58**	**229.06**	**205.36**	**187.09**	**145.94**	**206.65**	**86.32**	**2,082.33**

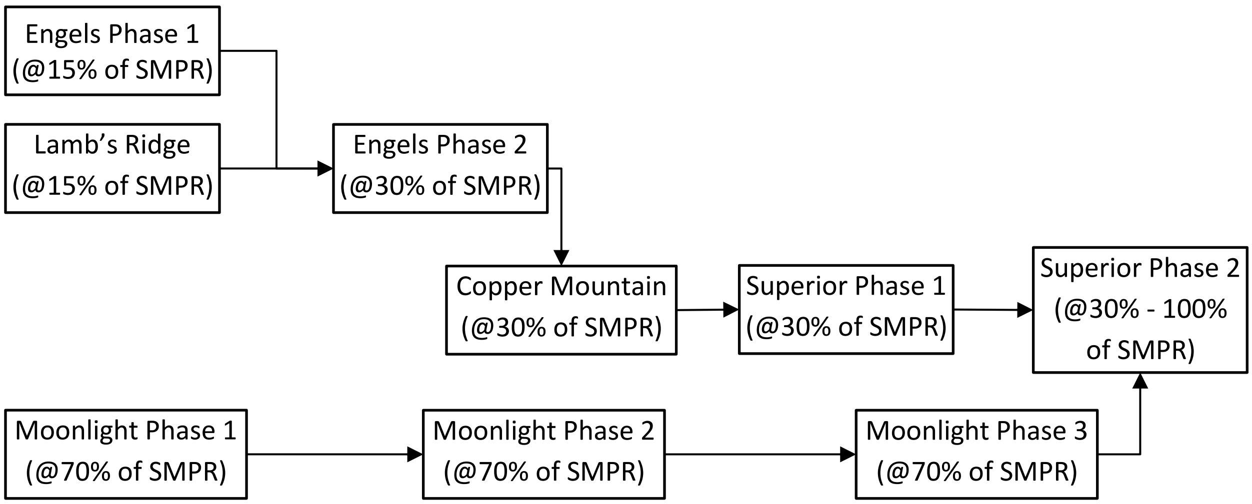
Pit Phase	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Sulfide High Grade Ag Oz Contained ('000s)											
E1	143.76	-	-	-	-	-	-	-	-	-	**143.76**
LR1	-	-	-	-	-	-	-	-	-	-	**-**
M1	1,103.79	-	-	-	-	-	-	-	-	-	**1,103.79**
E2	366.31	1,053.15	-	-	-	-	-	-	-	-	**1,419.46**
M2	400.62	1,888.94	1,505.24	807.16	-	-	-	-	-	-	**4,601.96**
M3a	-	-	-	216.57	582.36	819.33	599.07	93.04	-	-	**2,310.37**
CM1	-	-	-	-	-	-	-	-	-	-	**-**
S1	-	-	132.25	29.71	-	-	-	-	-	-	**161.96**
S2	-	-	-	-	-	0.56	2.96	114.70	1,228.22	397.26	**1,743.69**
Sulfide Low Grade Ag Oz Contained ('000s)											
E1	55.99	-	-	-	-	-	-	-	-	-	**55.99**
LR1	-	-	-	-	-	-	-	-	-	-	**-**
M1	165.40	-	-	-	-	-	-	-	-	-	**165.40**
E2	120.90	92.01	-	-	-	-	-	-	-	-	**212.91**
M2	150.49	564.62	350.63	106.22	-	-	-	-	-	-	**1,171.96**
M3a	-	-	-	254.42	313.96	311.59	157.81	15.84	-	-	**1,053.61**
CM1	-	-	-	-	-	-	-	-	-	-	**-**
S1	-	-	42.95	4.68	-	-	-	-	-	-	**47.63**
S2	-	-	-	-	-	0.37	3.68	127.37	475.41	168.95	**775.79**
Total Ag Oz Contained	**2,507.27**	**3,598.71**	**2,031.06**	**1,418.76**	**896.32**	**1,131.84**	**763.52**	**350.95**	**1,703.63**	**566.21**	**14,968.27**
Sulfide High Grade Au Oz Contained ('000s)											
E1	1.22	-	-	-	-	-	-	-	-	-	**1.22**
LR1	-	-	-	-	-	-	-	-	-	-	**-**
M1	8.88	-	-	-	-	-	-	-	-	-	**8.88**
E2	3.45	8.25	-	-	-	-	-	-	-	-	**11.70**
M2	13.99	17.89	2.97	2.00	-	-	-	-	-	-	**36.84**
M3a	-	-	-	0.77	0.67	0.71	0.94	0.23	-	-	**3.32**
CM1	-	-	-	-	-	-	-	-	-	-	**-**
S1	-	-	0.58	0.13	-	-	-	-	-	-	**0.72**
S2	-	-	-	-	-	0.00	0.03	0.56	6.51	2.02	**9.13**

Pit Phase	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Sulfide Low Grade Au Oz Contained ('000s)											
E1	0.60	-	-	-	-	-	-	-	-	-	**0.60**
LR1	-	-	-	-	-	-	-	-	-	-	**-**
M1	1.00	-	-	-	-	-	-	-	-	-	**1.00**
E2	1.30	0.86	-	-	-	-	-	-	-	-	**2.15**
M2	2.05	2.79	1.03	0.41	-	-	-	-	-	-	**6.29**
M3a	-	-	-	1.55	0.66	0.27	0.37	0.03	-	-	**2.88**
CM1	-	-	-	-	-	-	-	-	-	-	**-**
S1	-	-	0.18	0.02	-	-	-	-	-	-	**0.21**
S2	-	-	-	-	-	0.00	0.03	0.64	2.60	0.87	**4.14**
Total Au Oz Contained	**32.49**	**29.79**	**4.76**	**4.88**	**1.33**	**0.99**	**1.37**	**1.47**	**9.11**	**2.89**	**89.08**

Figure 16-9: Moonlight-Superior Copper Project Base Case Mine Schedule Summary



The schedule includes concurrent mining of pits and phases as follows:



16.5 Mine Operation and Layout

All facilities needed for the project, including administrative offices, warehouse, ammonium nitrate/fuel oil (ANFO) storage, equipment shop, fuel station, plant, leach pad, and waste storage, will need to be constructed. Conceptual layouts for the project were developed as illustrated in Figure 16-10.

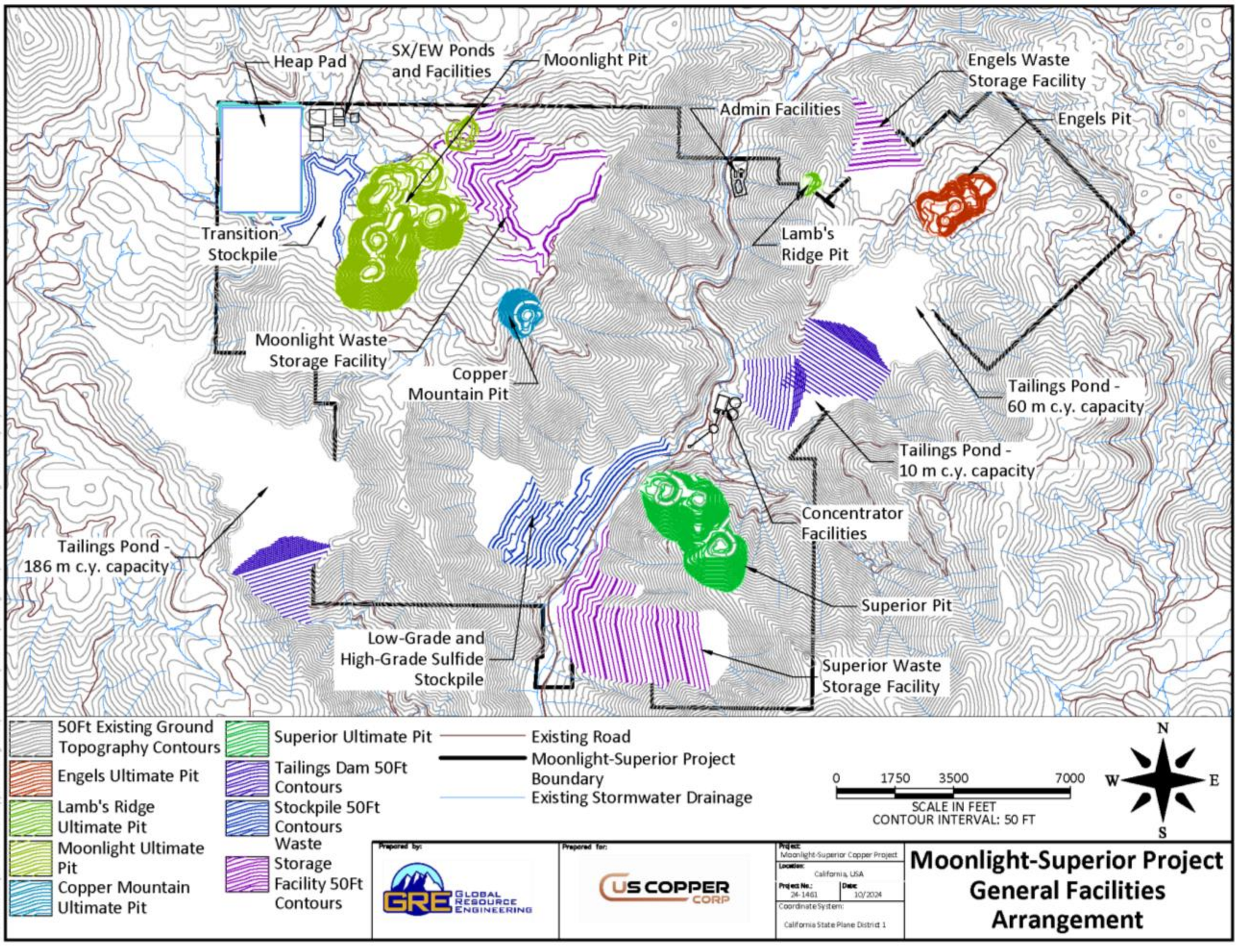
16.5.1 Drilling Blasting

The process material and waste rock material would be drilled and blasted using a rotary crawl driller and ANFO.

16.5.2 Loading and Hauling

For the pits and phases scheduled at 15% to 30% of the SMPR, the blasted rock would be loaded with 16-cubic yard (cy) capacity loaders into 105-ton capacity haul trucks. For the pits and phases scheduled at 70% to 100% of the SMPR, the blasted rock would be loaded with 29-cy capacity hydraulic shovels into 250-ton capacity haul trucks. Mineralized high-grade sulfide material would be hauled to the primary crusher, mineralized low-grade sulfide material would be hauled to a stockpile, mineralized oxide material would be hauled to the leach pad, mineralized transition material would be hauled to either the leach pad or a temporary stockpile, and waste material would be hauled to the waste storage facilities.

Figure 16-10: Moonlight-Superior Copper Project General Facility Arrangement



16.5.3 Waste Storage

Waste material would be stored in the waste storage facilities located near each of the pits. Approximately 151.5 million loose cy of waste would be mined and placed into the waste rock storage facilities. It is expected that construction of haul roads and administrative, mining, and plant facility areas would generate additional waste material that would be placed either in the waste rock storage facilities or in tailings dams. The waste storage facilities would be engineered to have overall final 3H:1V ultimate slopes.

16.5.4 Heap Leach

The heap leach facilities would be located northwest of the Moonlight pit area. The structure has been designed with a maximum capacity of 41 million tons. The heap leach facility would be constructed as detailed in Section 17.0 by stacking lifts to a maximum height of 328 feet (100 meters) and with overall final 3H:1V ultimate slopes.

17 RECOVERY METHODS

Based on the test work results, two process routes were developed for the Moonlight-Superior Copper Project: heap leach to treat the oxide and transitional materials and a conventional flotation concentrator to treat the sulfide materials.

Approximately 24.65 million short tons (22.36 million tonnes) of mixed oxide and transitional (10%/90%) material is available for treatment via acid heap leaching at a copper grade of 0.53% at a design production of 3.285 million short tons (2.98 million tonnes) per annum. Copper production is estimated at 259.9 million pounds (130,000 tons [118,000 tonnes]) over a 10-year leach period. Average annual copper production is 18.0 million lbs (9,000 tons [8,164 tonnes]). Peak production is estimated at 23.6 million lbs (11,800 tons [10,705 tonnes]) of copper.

Approximately 290.0 million tons (263.1 tonnes) of primary sulfide material is available for treatment in the concentrator with an average grade of 0.314% Cu. Annual production is 21.9 million tons (19.86 million tonnes) or 60,000 tons per day (54,348 tonnes per day). Copper production is estimated at 1.61 billion lbs (805,000 tons [730,000 tonnes]) with 13.2 million ounces of silver and 78,660 ounces of gold over a 14-year life. Payable silver and gold will be dependent on the smelter terms. Annual average copper production is estimated at 120.1 million lbs (60,050 tons [54,476 tonnes]). Peak production is estimated at 154.9 million lbs (77,450 tons [70,260 tonnes]) of copper in concentrate.

17.1 Concentrator Process Description

The concentrator has been designed to process a nominal 60,00 short tons per day (54,348 tonnes per day) of copper-silver mineralized material and is expected to produce a marketable copper concentrate ranging from 23 to 28% copper depending on the feed grade. Based on metallurgical testing and in-house expertise, the copper, silver, and gold recoveries have been estimated at 88 to 90%, 60 to 61% and 70 to 71%, respectively. The variation in recovery is a reflection of the feed grade to the plant. A complete MetSim model was developed for the process to allow for more accurate metallurgical forecasting.

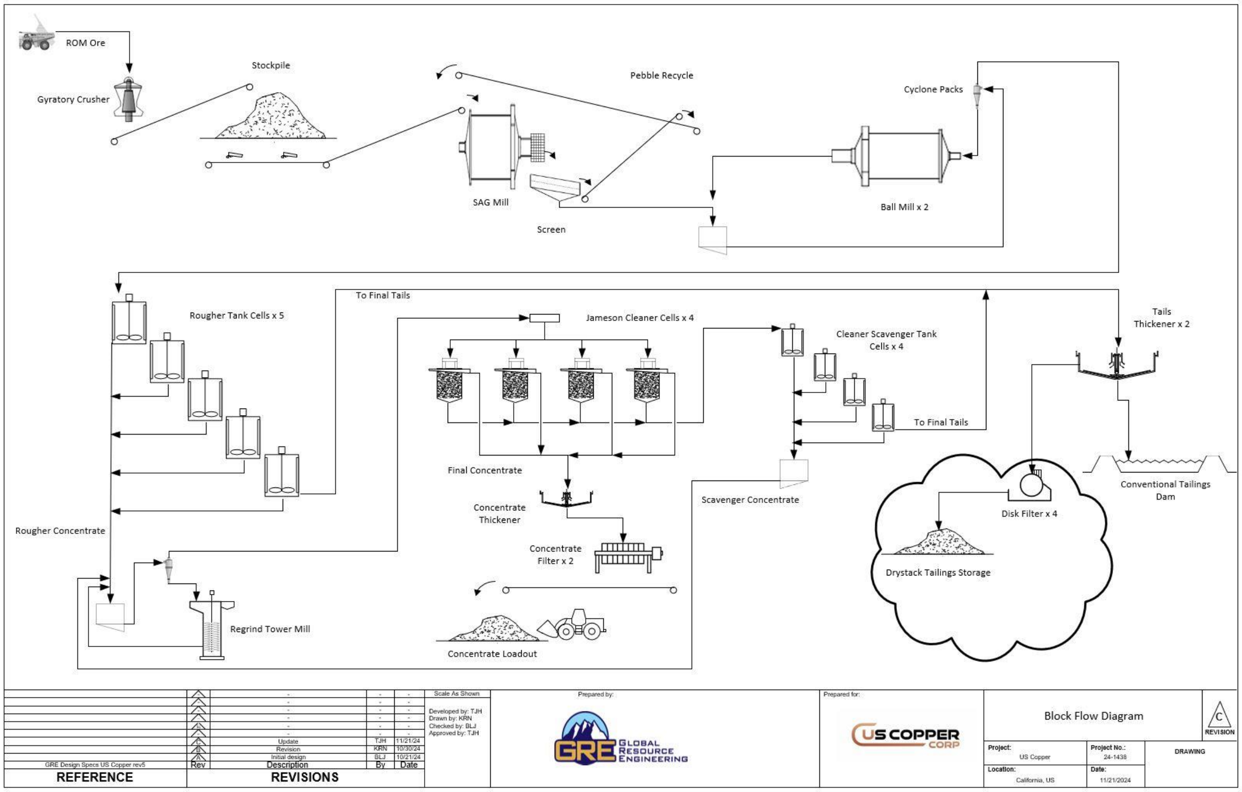
The concentrator design is based on a modern copper sulfide process typical of the industry. The capacity of the facility was based on the tradeoff between capital expenditure, operating costs, and mine life. The processing route has been designed to produce a saleable high-grade copper-silver concentrate. Gold has been included in the concentrate specifications, but it is unlikely to add any value at this stage due to smelter constraints on concentrate grade.

A simplified flowsheet of the concentrator is shown in 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 tailings management facility (TMF). Process water will be recycled from the tailings pond. Fresh water will only be used for gland service and reagent preparation.

Figure 17-1: Moonlight-Superior Concentrator Flowsheet



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

- run-of-mine (ROM) mineralized material receiving
- primary gyratory crusher
- crushed stockpile and reclaim
- semi-autogenous grinding (SAG) mill with screen for pebble recycle
- Ball mill with cyclones classification (SAG and Ball Mill discharge)
- Copper rougher flotation via large tank cells
- Copper concentrate regrinding via tower mill
- Copper cleaner flotation via Jameson cells
- Cleaner scavenger flotation via tank cells
- copper concentrate thickening, filtration, and dispatch
- tailings thickening and optional drystack or disposal to conventional tailings pond

The major criteria used in the design are outlined in Table 17-1.

Table 17-1: Concentrator Design Criteria

Area	Parameter	Units	Value
Crusher	Operating Time	days	330
	Hours per Day	hr	20
	Availability	%	75.3%
	Throughput		
	Design	tph	2,709
	Nominal	tph	2,258
Grinding	Operating Time	days	329
	Hours per Day	hr	24
	Availability	%	90.1%
	Throughput		
	Design	tph	2,265
	Nominal	tph	2,041
	SAG F80	mm	127,000
	SAG P80	mm	2,000
	SAG Pebble Recycle	%	31%
	Bond Work Index	kWh/t	18.1
	Ball Mill P80	um	150
	Regrind P80	um	45
Flotation	Rougher Retention Time	min	18.6
	Cleaner Retention Time	min	12.0
	Cleaner Scavenger Retention Time	min	16.0
Concentrate	Thickener	m2/tpd	0.575
	Plate and Frame Filter	l/hr/m2	750
Tailings	Thickener	m2/tpd	0.307
	Disk Filter	l/hr/m2	1,475

The design criteria are based on test work results and GRE'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.

The following describes the process plant sizing and equipment.

17.1.1 Primary Crushing

A conventional gyratory crusher facility has been designed to crush ROM 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 Mt/h).

The major equipment and facilities in this area include:

- dump pocket
- hydraulic rock breaker
- gyratory crusher – 1,370 mm x 1,910 mm (54 in x 75 in) 700 horsepower (hp)
- crusher discharge apron feeder
- crushed stockpile, 33,000 st live capacity

- reclaim apron feeders
- conveyor belts, metal detectors, self-cleaning magnets, and belt tear detectors
- belt scale
- sampling system
- dust collection system

The ROM material will be trucked from the open pit to the primary crusher by mine haul trucks. The ROM material will be reduced to 80% passing 127 mm using a gyratory crusher. A rock breaker will be installed to break any oversize rocks.

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 stockpile. The stockpile will have a live capacity of 33,000 st. The material will then be reclaimed from this stockpile by apron feeders at a nominal rate of 2,500 stph. The apron feeders will feed a 60-inch (1,520-mm) wide conveyor, which in turn will feed the SAG Mill. The conveyor belt will be equipped with a belt scale and self-cleaning magnet.

17.1.2 Grinding

The primary grinding circuit consists of a SAG mill and two ball mills. The SAG mill discharge reports to a vibrating screen with the oversize being conveyed back to the SAG mill feed end and the undersize reporting to a pump box. The ball mill discharge also reports to the pumps box, which feeds a cyclone pack. The cyclone overflow is directed to the rougher flotation circuit.

The major equipment associated with the grinding circuit is as follows:

- SAG Mill - 40.0-foot (12.2-meter) mill diameter inside shell x 24.9-foot (7.6-meter) mill length inside shell equipped with a ring motor of 26,000 hp.
- Vibrating screen – 10-foot x 20-foot (3.0-meter x 6.1-meter), single deck with a top deck screen size of 30 mm.
- Pebble return conveyors – 60 inches wide
- Ball Mills – two wet overflow ball mills 26-foot x 40.5-foot (7.9-meter x 12.3-meter) equipped with a ring motor of 18,000 hp
- Cyclone cluster – 12 operating and two standby of 33-inch diameter

Each ball mill will operate in closed-circuit with a common cyclone cluster. The product from each ball mill will be discharged into a common feed pump box, also receiving the SAG screen undersize. The P80 size target is 80% passing 150 µm, and the circulating load to the individual ball mill circuits will be 415%, with the cyclone underflow returning to the ball mill as feed material.

The SAG mill receives 2,500 tons (2,265 tonnes) per hour of new feed with approximately 31% recycle of pebble oversize (+30mm). The ball mills will operate at a speed of approximately 73% 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 to maintain the grinding efficiency. Lime is added to the ball mill discharge to adjust the pH to 10 for rougher flotation. A particle size analyzer is included for the cyclone overflow along with a density and mass flow analyzer.

17.1.3 Rougher Flotation

Ground material from the cyclone overflow reports to the rougher flotation circuit. The flotation circuit consists of five 600 cubic meter [m^3] tanks cells. The rougher tailings report to final tails and the rougher concentrate reports to the regrind circuit. Samplers are included for both concentrate and tailings.

An onstream XRF analyzer will take samples of each of the flotation streams and provide copper assays in roughly a 15-minute turnaround. Process control will be achieved by a Supervisory Control and Data Acquisition system.

17.1.4 Concentrate Regrind

The rougher concentrate of a nominal P80 of 150 µm reports to a cyclone feed box. The concentrate is passed to a cyclone cluster with a P80 target of 45 µm. The cyclone underflow reports to a tower mill that discharges into the cyclone feed box. The major equipment is as follows:

- Cyclone cluster of 12 operating and two spare x 12-inch cyclones
- Tower Mill – 4,000 hp
- Trash screen – 5-foot x 12-foot (1.5-meter x 3.7-meter), single deck at 1mm
- Stock tank for pH adjustment to 10.5 to 11 prior to cleaner flotation

17.1.5 Cleaner Flotation

The reground rougher flotation concentrate is split into four feed streams via a splitter box and distributed to four Jameson cells similar to E4500-12 or equivalent. The Jameson concentrate becomes the final flotation concentrate, and the tailings report to the cleaner scavenger circuit.

The cleaner scavenger circuit consist of four 100 m^3 tank cells in series. The cleaner scavenger concentrate reports back to the regrind circuit and the tails report to final tailings. Samplers are included for both concentrate and tailings.

17.1.6 Concentrate Filtration

A concentrate thickener (72-foot [22-meter] diameter) receives the final concentrate at approximately 15% solids. The concentrate is thickened to a target underflow density of 65% solids. A flocculation system is provided to add settling reagents to the thickener feed. The overflow reports back to the process water system. The underflow reports to a stock tank and a filter feed tank. A series of two plate and frame pressure filters in parallel filter the concentrate to a final moisture content of 8 to 10%.

The plate and frame filter is composed of 6.5 x 6.5-foot (2 x 2-meter) filter elements each with a chamber thickness of 1.6 inches (40 mm). Each filter has a total of 39 plates. The target filter cycle time is 15 minutes. The filter cake discharge is dropped onto a series of conveyors and transferred to the concentrate shed.

The concentrate shed is designed to hold approximately 26,500 tons (24,000 tonnes) of material. The loadout is conducted with a wheel loader into suitable haul trucks. The shed is also equipped with a tire wash system and a large truck scale. Concentrate sampling is conducted for each load.

17.1.7 Tailings

Rougher flotation tailings and cleaner scavenger flotation tailings report to a mix box on the tailings thickener (2 x 330-foot [100-meter] diameter). A flocculation system is provided to add settling reagents to the thickener feed. A sampler takes slurry samples for analysis. The tailings are diluted to 10 to 15% solids by an autodilution feature on the thickener. The overflow reports back to the process water system.

In the drystack tailings case the underflow reports to a stock tank and a filter feed tank. The underflow is split and fed to four rotating disk filters. The disk filters are 12.5 feet (3.8 meters) in diameter and consist of 12 disks per unit. The filter cake discharge is dropped onto a series of conveyors and transferred to the overland conveyor which feeds a series of grasshopper conveyors and a final tailings stacker.

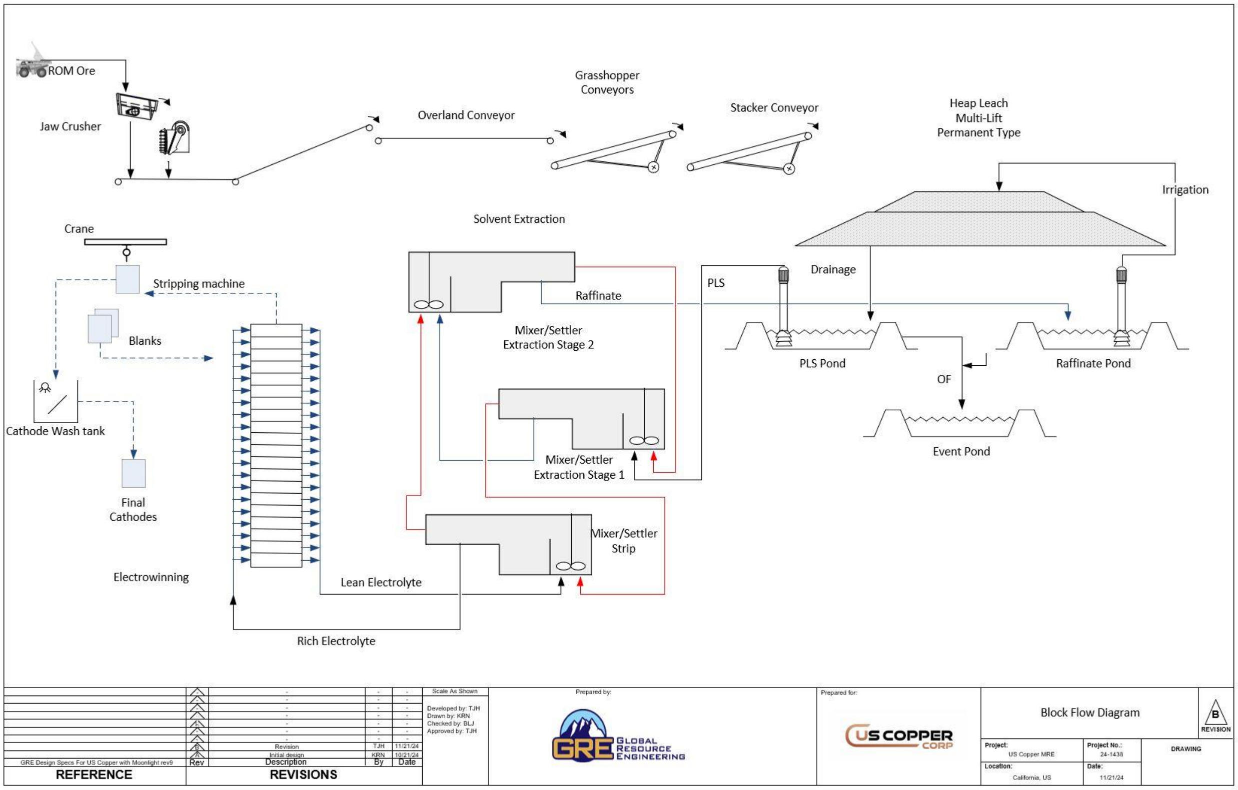
In the conventional tailings case, the thickened tailings are pumped to the tailings pond and spigoted to create a beach. The spigot locations are moved along the periphery of the tailings dam wall to create a continuous centerline dam.

17.2 Heap Leach Process Description

The heap is a permanent type with multiple lifts. Ore is crushed in a jaw crusher and conveyed to an agglomeration drum prior to conveyor stacking on the heap.

Ultimate copper extraction is estimated at 75% for the oxide material and 60% for the transition material. Leach extractions are based on a three-year cycle with 60%, 30% and 10% of the ultimate extraction occurring in year 1, 2, and 3 of the leach cycle, respectively. Figure 17-2 shows the simplified heap leach process flowsheet.

Figure 17-2: Moonlight-Superior Heap Leach Flowsheet



Copper is recovered via a conventional solvent extraction and electrowinning circuit (SX/EW) to produce Grade A cathodes.

17.2.1 Crushing

Ore is delivered via haul truck to the ROM stockpile or direct dumped into the ore bin ahead of the jaw crusher. A static grizzly limits the oversize feed to the crusher, and a hydraulic rock breaker is available to handle any oversize material.

The material is primary crushed to a target P80 of 75 mm (3 inches), and undersize material reports directly to the final product via a vibrating grizzly prior to the jaw crusher. The jaw crusher is sized at 48 x 60 inches (122 x 152 centimeters [cm]).

17.2.2 Heap Leach Pad and Ponds

The heap leach pad is a permanent type with a double lined high density polyethylene (HDPE) base including leak detection. The ore is stacked in 33-foot (10-meter) lifts on a pad that is 2,130 feet x 3,280 feet (650 meters x 1,000 meters). A total of eight lifts are required to hold the full design production, providing a total volume of 30.6 million cy (23.4 million m^3).

Three ponds have been designed to handle the solution flows. A raffinate pond receives raffinate from the solvent extraction circuit and is the primary irrigation solution. The raffinate pond is capable of storing 12 hours of solution with a working volume of 32,870 cy (25,130 m^3). The pregnant leach solution (PLS) pond receives the heap leach discharge solution and is capable of storing 72 hours of solution with a working volume of 183,100 cy (140,000 m^3). Solution from the PLS pond is pumped to the solvent extraction circuit for copper recovery. An event pond has been included that is capable of handling any pond overflows based on a 100-year rain event. The event pond has a working volume of 136,550 cy (104,400 m^3).

Irrigation is provided to the heap cells via emitter style pipes spaced at 23.6-inch (60-cm) intervals. Target irrigation is 0.003 gallons per minute per square foot (gpm/ft^2) (7.3 liters per hour per square meter [lph/m^2]). Maximum irrigation is designed at 0.005 gpm/ft^2 (12 lph/m^2). Irrigation to the heap leach is provided via the raffinate pumps located on a floating barge in the raffinate pond. Conventional booster pumps have been provided to ensure sufficient head is available to irrigate the maximum heap elevation. PLS solution is pumped to the solvent extraction circuit by similar barge pumps located in the PLS pond.

17.2.3 Solvent Extraction and Electrowinning

PLS solution from the heap leach is pumped to the solvent extraction circuit. The solvent extraction circuit consists of two parallel trains with two extraction stages and one strip stage. Conventional lixiviant and diluents are to be utilized such as LIX984N and ShellSol. The incoming PLS is mixed with the extractant mixture in a traditional mixer settler in a counter current fashion, the highest grade PLS is mixed with the second stage extractant first, the passed to the first stage. The loaded organic from the stage two extraction is passed to the strip stage where it is contacted with weak electrolyte from the electrowinning circuit. The strip stage and the electrowinning circuit are in closed circuit.

Strong or rich electrolyte from the strip circuit passes through a series of organic recovery systems primarily consisting of a series of dual media filters. The electrowinning circuit requires 2,012 cathodes configured in 45 cathodes per cell modules. The EW circuit is composed of two parallel trains containing 25 cells each.

An automated stripping machine has been provided to handle the cathode stripping, washing, and bundling.

Reagents storage for the heap leach (sulfuric acid) and the solvent extraction and electrowinning circuit has been provided and would typically be located near the SX/EW plant.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The project is located in Plumas County, California, approximately 10 miles northeast of Greenville, California, and approximately 100 miles northwest of Reno, Nevada. The Property is approximately 1.6 miles (2.5 km) from Diamond Mountain Road, a two-lane paved all weather highway. The Property is currently accessible through a network of existing forestry service roads.

18.2 Access Roads

The property can currently be accessed by an existing network of logging roads, mostly from the nearest paved highway (Diamond Mountain Road). The internal property road network will be constructed as the project is developed.

18.3 Buildings And Facilities

The major buildings at the plant site will include the concentrator building, administration building, truck shop complex, laboratory, primary crushing building, concentrate storage and concentrate loadout facility, substation, and warehouse. Additionally, the SX/EW buildings will house the heap leach staff, change room, and control room.

Figure 18-1 and Figure 18-2 illustrate the overall project site layout, and the general arrangement of the plant area.

Figure 18-1: Moonlight-Superior General Site Layout

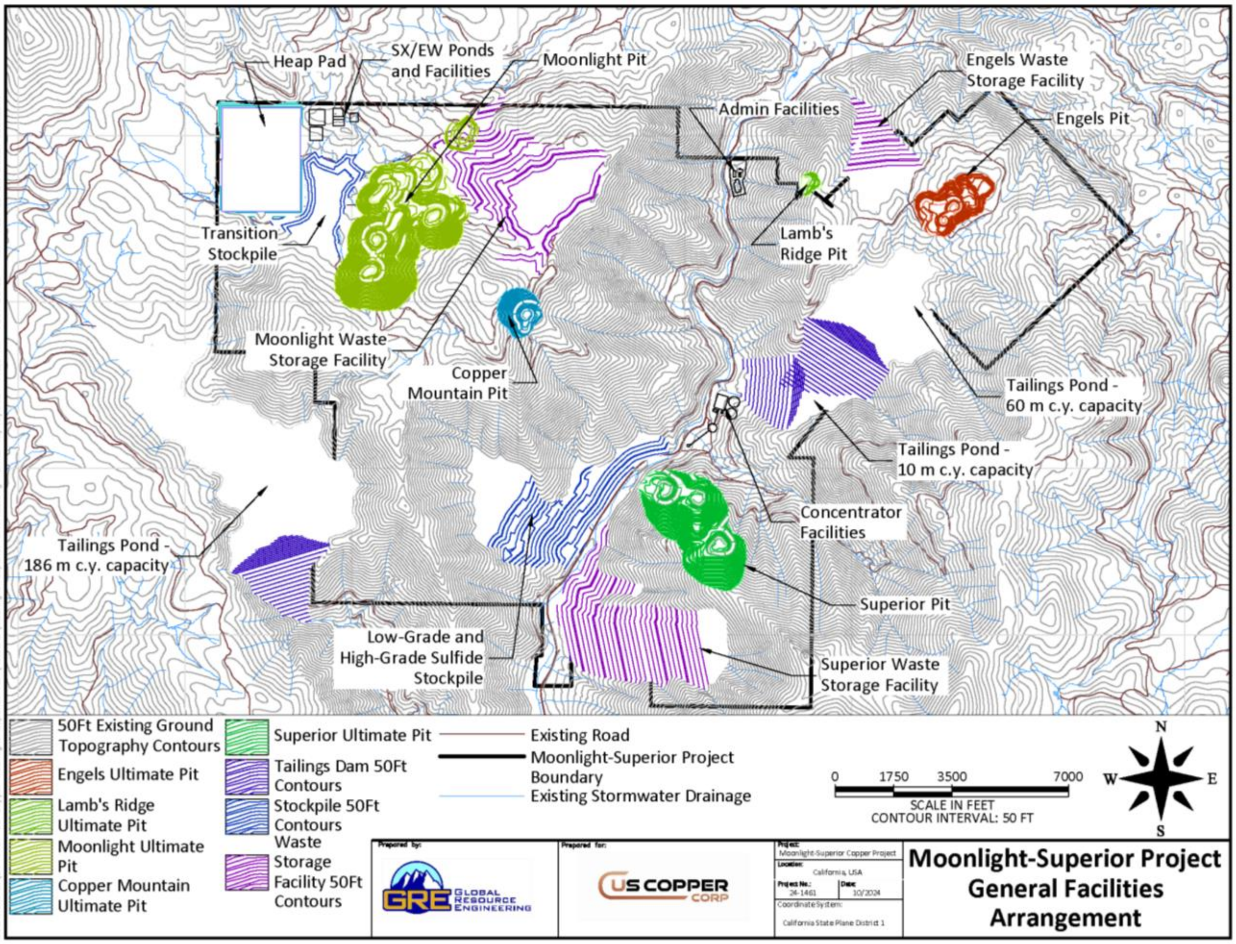
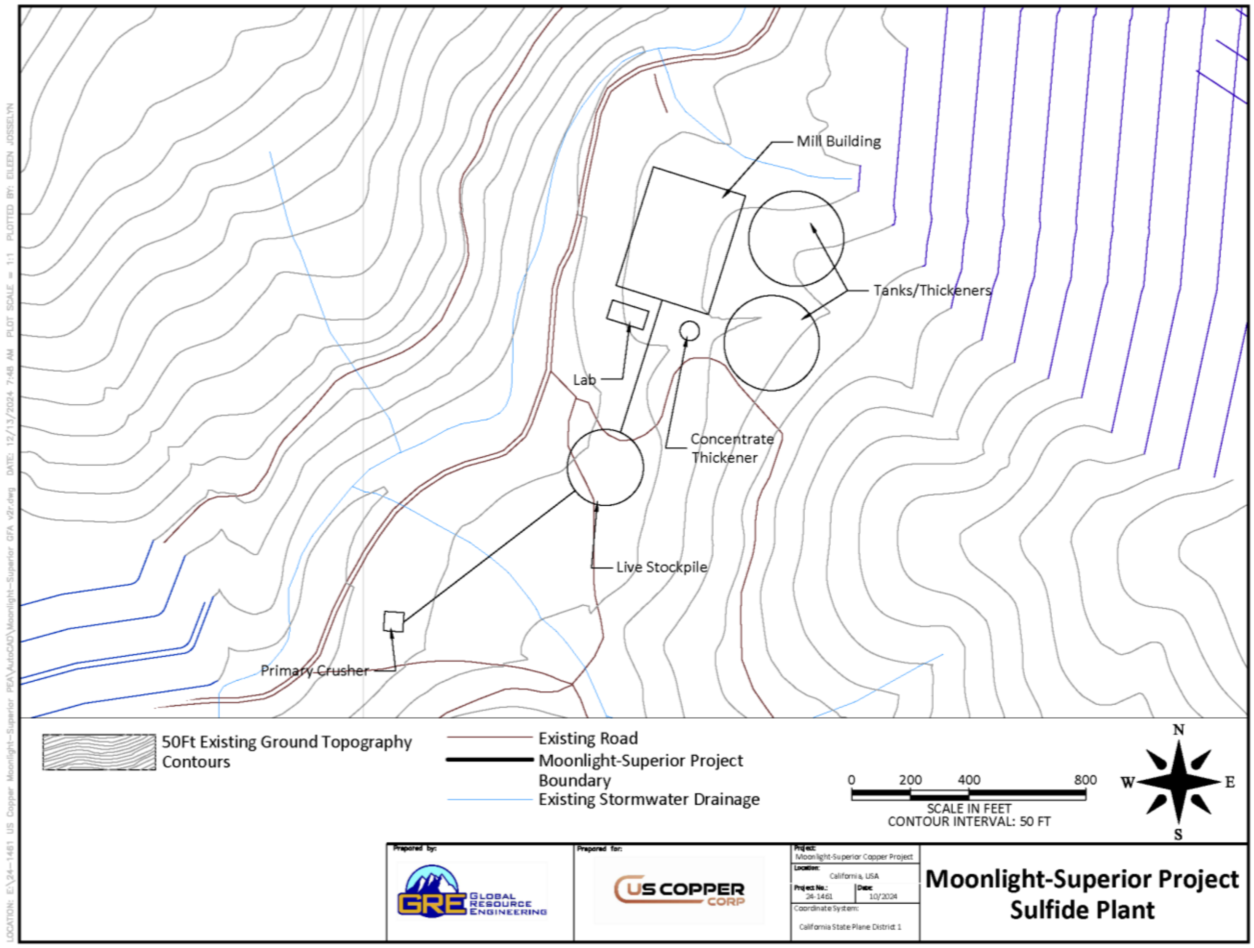


Figure 18-2: Moonlight-Superior Concentrator Area



18.4 Assay and Metallurgical Laboratory

A dual-purpose laboratory is provided for to include both assay and metallurgical testing facilities. The assay laboratory will function for grade control and process samples. The metallurgical laboratory will provide support to both the concentrator and the heap leach. The key equipment includes:

- Sample preparation including crushing, splitting, and pulverizing
- Screen analysis
- Microwave digestion
- ICP-AES
- X-ray fluorescence spectrometer (XRF)
- Sulfur and carbon analyzer
- Heap leach columns
- Grinding equipment
- Flotation cells
- Sample filtration
- Drying ovens

18.5 Tailings Storage Facility

The tailings storage facilities (TSF) are designed to accommodate over 297 million tons (270 million tonnes) of tailings, to be generated over the 16-year mine life. The design concentrator throughput is nominally 19.7 million tons (17.9 million tonnes) per annum. The tailings are expected to be non-acid generating due to the low sulfide and reasonable carbonate content.

Two tailings storage options have been developed for this study; conventional thickened tailings storage and drystack storage. Drystack tailings has the advantage of improved water recycle, enhanced stability and smaller footprint. The disadvantage is higher capital and operating costs.

Only approximate required areas and volumes have been developed at this stage of the project and further TSF design will be required.

18.6 Administration Building

The administration building will be a single-story steel structure with insulated steel roof and walls located near the main project entrance. The building will be supported on concrete spread footings with concrete grade walls along its perimeter. This facility will house the administrative, engineering, and geology staff.

A second building will be provided to house the mine dry, lockers, shower facilities, first aid, with emergency vehicles. The concentrator and heap leach change rooms will be located in the concentrator and heap leach buildings.

18.7 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.

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 warehouse and maintenance personnel. The facility is designed to service and maintain both the mining haul fleet and light vehicles.

18.8 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 ready line. The diesel fuel storage tank will have a capacity sufficient for approximately seven days of operation. Diesel storage will consist of above-ground tanks and a containment pad, complete with loading and dispensing equipment with card key access. Dedicated fuel and lubricant service trucks will transport diesel and lubricants to the mining equipment as required.

18.9 Water Supply

Due to the use of tailings (either conventional or dry stack) and heap leaching, the project could be a heavy consumer of water (with conventional tailings) or a moderate consumer of water (with dry tailings).

The selection between conventional and dry tailings must be conducted in the context of available water resources.

Two separate primary water supply systems have been included in the design including: fresh water and process water. Fresh water is sourced from a well field and process water is recycled from the concentrator solid/liquid separation systems.

Fresh Water Supply System

Fresh and potable water will be supplied to a fresh/fire water storage tank from wells. 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

Process water from the concentrate and tails thickeners will be recycled to the process water system along with any filtrate recovered. Process water will be the primary source of water for the concentrator and heap leach makeup.

The estimated fresh water usage is 85,000 gallons per hour (gph) (322 m^3/hr) including:

- 5,800 gph (22 m^3/hr) for potable usage
- 13,200 gph (50 m^3/hr) for mine usage
- 18,000 gph (68 m^3/hr) for heap leach makeup
- 48,000 gph (182 m^3/hr) for concentrator makeup

Water Management

The key facilities for the water management plan include:

- open pits
- concentrator water systems
- Tailings storage facility
- Surface water management
- fresh water supply

The water management strategy will use water within the project area to minimize fresh water demand. 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 in a separate water storage pond. The water supply sources for the project are as follows:

- precipitation runoff from the project areas
- water recycle from tailings and concentrate

- groundwater wells
- waste water treatment effluent for dust control usage

A preliminary water balance has been constructed (see above) but will require further review in the subsequent project stages.

Water rights and acquisition will be looked at further in the next phase of the project.

18.10 Power Supply And Distribution

The project is estimated to have connected load of approximately 50 megawatts (MW) for the concentrator and 11 MW for the heap leach facility.

Power is expected to be drawn from the existing network of transmission lines located in Westwood, CA, approximately 7.5 miles (12 km) 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 volts (V) at mill and mine for site-wide power distribution.

A single 4 MW, 4,160 V standby rated diesel generator set will be provided at the concentrator building to provide standby power for critical process loads.

19 MARKET STUDIES AND CONTRACTS

The Moonlight-Superior project would produce copper concentrate and copper cathode. A long-established, active, worldwide market exists for the buying and selling of copper. US Copper expects this to continue throughout the life of the Moonlight-Superior project. Further market studies are not deemed necessary to establish the existence of a market for the product.

The base case copper price used was $4.15/lb, which incorporates the 3-year trailing average of $4.06/lb and the one-year futures price of $4.30/lb. GRE has provided sensitivity analysis from -25% ($3.11/lb) to +25% ($5.19/lb). The price of copper has been rising, and the GRE QP believes the $4.15/lb base case price reflects the consensus market forecast for copper.

The base case silver price used was $27.40/oz, which incorporates the three-year trailing average of $24.19/oz and the one-year futures price of $32.26/oz.

The base case gold price used was $2,320/oz, which incorporates the three-year training average of $2,015/oz and the one-year futures price of $2,779/oz.

20 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 miles northwest of Reno, Nevada. The Project is approximately 3,205 acres in size consisting of a mix of patented mining claims, unpatented mining claims and fee lands in the Plumas National Forest (see Figure 4-1).

The site is in mountainous terrain with variable forest cover which has been subject to a recent forest fire. Elevations range from 3,600 feet above mean sea level in the lower Lights Creek valley to over 5,600 feet above mean sea level at the Engels Mine, with peaks in the 7,500-foot 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 miles 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.

Five areas of potential mining interest are included in the Project: Superior, Engels, Moonlight, Lamb's Ridge, and Copper Mountain (The Superior Mine, the larger 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 acres in size. A second tailings impoundment is located at the Lights Creek valley mouth and is approximately 100 acres in size. The Project does not include ownership of this area.

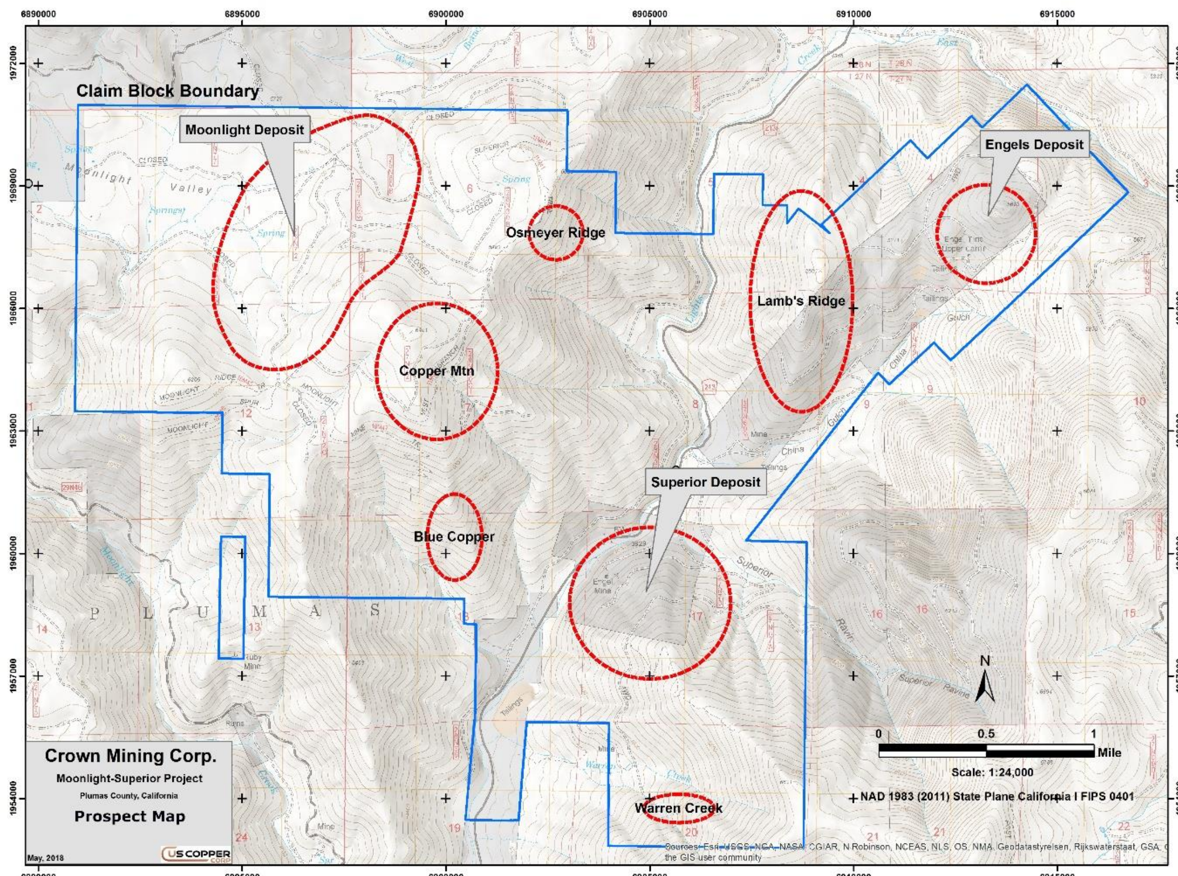
The Engles Mine is located on a ridge above a tributary to Lights Creek and possesses more options for mine development and waste management. Existing impacts include mine water discharge, waste rock and tailings impoundments of approximately one and five acres in size.

Figure 20-1). 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, Lamb's Ridge, and Copper Mountain are largely un-impacted; Moonlight and Copper Mountain are completely located on unpatented claims in the National Forest; Lamb's Ridge is mainly on patented land.

The Superior Mine, the larger 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 acres in size. A second tailings impoundment is located at the Lights Creek valley mouth and is approximately 100 acres 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 one and five acres in size.

Figure 20-1: Moonlight-Superior Project Areas of Potential Mining Interest



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.

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 related 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.

20.2.2 PERMITS REQUIRED

The major permits, approvals, and environmental reviews for a proposed project at Moonlight-Superior 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

20.2.2.1 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 five acres 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.

20.2.2.2 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-Superior. The combined EIR/EIS would not be required to meet the statutory requirements of just the EIR.

The granting of a Record of Decision (ROD) is the endpoint of the NEPA process and grants the mine permission to execute the project on federal lands (assuming other permits are likewise acquired). In the US, the NEPA process takes a minimum of five to seven years. Often, post-ROD legal challenges from nearby NGOs or stakeholders greatly lengthen the permitting timeline. For the PEA, GRE has assumed a best-case scenario of five to seven years.

20.2.2.3 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 waters are deemed significant, the USACE may decide to become co-lead with the land manager on the EIS.

20.2.2.4 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 with all available waste rock material to original contours. 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.

20.2.2.5 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 Preapplication 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.

20.2.2.6 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.

20.2.2.7 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. Modeling 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 miles 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 miles from the project. Regional haze and visibility issues will be the concern.

Class 1 Airshed designation requires additional air quality monitoring and modeling 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 miles (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 modeling would be required for Environmental Review and would require at least one year of weather and air quality monitoring.

20.2.2.8 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 modeling 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.

20.2.2.9 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.

20.2.2.10 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 79º F (26º 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-Superior 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).

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 analyzed 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 consistently 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*. This thesis 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 analyzed 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, while others were new to geochemical sampling (Table 20-3).

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 analyzed by Sheffield Resources Ltd. In 2007 (Orequest Consultants Ltd, 2007) are presented in Table 20-4.

Table 20-4: Sheffield Resources ABA

Sample	Source	Cu%	MPA tCaCO3/t ore	NNP tCaCO3/t ore	NP tCaCO3/t ore	NP:MPA
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 feet 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 in Table 20-5.

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

Analyte	Unit	Mn SUL	S SUL
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 indicate that acid generation is not a problem in the tens of years range.

20.5 Baseline Environmental Data Collection

The US Copper project must augment the environmental monitoring network to collect the necessary data to support the permitting effort described in Section 20.2 above. This involves environmental geochemistry, air quality, surface water quality, groundwater quality, noise monitoring, biodiversity monitoring and other data collection. A separate line-item for this investment is included in the economic model and recommendations (see Section 26.1.5).

20.6 SOCIAL OR COMMUNITY IMPACT

According to their website (http://www.countyofplumas.com/index.aspx?NID=190), Plumas County had a population of 19,486 in 2022, the last year for which there are data, but the county population has been in decline from a high of 20,824 in 2000. The median annual income in 2022 was $67,623, below the California median of $91,551.

Ninety-four percent of the population have received high school diplomas, 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. 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 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 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

The capital cost estimate has been prepared for the PEA under the assumption of processing of open pit mined sulfide material at a design rate of approximately 54,400 tonnes per day and oxide and transition material at a rate of 8,200 tonnes per day. Project costs were estimated using GRE in-house data, cost data from Infomine (2024), and experience of senior staff. The estimate assumes that the project will be operated by the owner with purchased equipment.

The initial capital costs are incurred in the years prior to production. GRE's QP expects there will be five to seven years of continued exploration, engineering, and permitting prior to a production decision.

Initial capital costs are defined as all capital costs until production starts. This includes labor and development costs in the pre-production years. Sustaining capital is defined as the capital costs incurred in the periods after production begins.

All capital costs cited in this Report are referenced in US dollars with an effective date of December 16, 2024.

Capital cost estimates were prepared based on current and expected long-term pricing assumptions and to a PEA level of ±35% level of accuracy.

The capital costs are summarized in Table 21-1.

21.1.1 Mine and Mobile Support Equipment

Mine equipment and mobile support equipment are assumed to be purchased. Mine and mobile support equipment costs occur throughout the life of the project and are shown in Table 21-2.

21.1.2 Tires

Costs for initial tire purchase is included for all major mobile equipment items requiring tires. Costs are incurred in year -1 and total $1.5 million.

21.1.3 Process Plant

The concentrator is assumed to require 18 to 24 months for construction with capital payments being made as equipment is ordered and delivered. The concentrator capital costs are shown in Table 21-3.

The heap leach pad is assumed to be constructed in two phases at years -2 and year 5. The SX/EW circuit will be built at full capacity over an anticipated period of 18 to 24 months. The heap leach and SX/EW capital costs are shown in Table 21-4.

21.1.4 Infrastructure

All buildings and associated infrastructure installed on a permanent or semi-permanent basis are considered infrastructure. They include material and installation costs. These costs are incurred in year -2 and -1. Each item's capital cost was estimated based on knowledge of nearby mine operations or senior engineers' experience. Table 21-5 shows total estimated costs for each infrastructure item.

Table 21-1: Moonlight-Superior Copper Project Capital Cost Summary ($millions)

Item	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	Total
Mine Equipment Capex	$0.00	$57.71	$75.00	$9.56	$0.03	$75.61	$0.07	$0.03	$0.03	$0.03	$0.03	$0.07	$0.00	$0.00	$0.00	$0.00	$0.00	**$218.17**
Process Capex	$281.64	$281.64	$0.00	$0.00	$0.00	$0.00	$48.27	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$611.54**
Infrastructure Capex	$61.00	$56.35	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$117.35**
G&A Capex	$0.00	$22.40	$7.50	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$50.00	**$101.72**
Working Capital	$0.00	$36.12	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$36.12**
Sustaining Capital	$0.00	$0.00	$0.00	$0.24	$0.00	$2.84	$4.21	$1.20	$0.00	$0.00	$0.78	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$9.27**
Contingency	$68.53	$90.84	$16.50	$2.29	$0.34	$16.03	$10.85	$0.58	$0.34	$0.34	$0.50	$0.35	$0.34	$0.34	$0.34	$0.34	$10.00	**$218.83**
Total Capital Costs	**$411.16**	**$545.05**	**$99.00**	**$13.76**	**$2.05**	**$96.15**	**$65.08**	**$3.49**	**$2.05**	**$2.05**	**$2.99**	**$2.10**	**$2.01**	**$2.01**	**$2.01**	**$2.01**	**$60.00**	**$1,313.00**

Table 21-2: Mine Equipment Capital Costs

Item	Quantity Required	Capital Costs ($millions)
Production Equipment		
Loader CAT 992K	4	$12.07
Hydraulic Shovel CAT 6040	5	$34.30
Haul Truck CAT 777G	15	$23.82
Haul Truck CAT 793F	28	$106.16
Bulldozer CAT D10T	5	$7.89
Blast Hole Drill Epiroc DM45	6	$7.20
Support Equipment		
Loader CAT 993K	1	$2.33
Loader CAT 992K	1	$3.02
Bulldozer CAT D9T	3	$3.79
Bulldozer CAT D6T	1	$0.71
Wheel Dozer CAT 834K	2	$2.65
Water Truck CAT 777G	2	$3.18
ANFO Truck	1	$0.30
Fuel/Lube Truck	2	$1.75
Mechanic Truck	2	$0.50
Grader CAT 16M3	3	$1.69
Compactor CAT CP-56B	1	$0.24
Blast Hole Drill Sandvik DR580	2	$2.40
Backhoe CAT 416	1	$0.14
Crane	1	$0.53
4x4 Pickup	10	$0.61
Dewatering Pump	3	$0.22
Generator	0	$0.00
Telehandler	0	$0.00
Light Plants	10	$0.31
Other Mining-Related Equipment		
Surveying Equipment	1	$0.13
Computers	20	$0.05
Operations Software	1	$0.11
Planning Software	1	$0.17
Geology Software	1	$0.11
Maintenance Software	1	$0.21
Dispatch System	1	$0.10
Plotter	1	$0.01
Radios	10	$0.00
Total Mining Equipment Capital Costs		**$216.67**

Table 21-3: Concentrator Capital Cost Summary – Conventional Tailings

Item	Cost	
	US$(millions)	%
Capital Costs		
Crushing - Stockpile	$11.12	8.1%

Item	Cost	
	US$(millions)	%
Grinding	$34.43	25.2%
Rougher Flotation	$5.41	4.0%
Regrinding	$10.42	7.6%
Cleaner Flotation	$7.45	5.4%
Concentrate Handling	$5.22	3.8%
Mill Building	$38.91	28.5%
Tailings - Conventional	$9.80	7.2%
Utilities	$10.78	7.9%
Reagents	$1.57	1.1%
Mobile Equipment	$1.62	1.2%
Total Equipment	**$136.72**	**100.0%**
Installation Labor	$57.46	22.0%
Concrete	$6.50	2.5%
Piping	$13.13	5.0%
Structural Steel	$7.77	3.0%
Instrumentation	$16.86	6.5%
Insulation	$1.97	0.8%
Electrical	$13.70	5.2%
Coatings and Sealants	$0.73	0.3%
Initial Tailings Dam	$26.77	10.2%
Spares and First Fill	$4.50	1.7%
Engineering/Management	$111.79	42.8%
Total Indirects	**$261.19**	**100.0%**
Total - Direct and Indirect	**$397.91**	
Taxes, Insurance and Freight		
Tax - refundable	$9.90	7.2%
Import Duties (50/50)	$4.10	3.0%
Insurance	$1.37	1.0%
Freight (50/50)	$4.79	3.5%
Total Taxes	**$20.15**	**14.7%**
Total Cost	**$418.07**	

Table 21-4: Heap Leach Capital Cost Summary

Item	Cost	
	US$(millions)	%
Capital Costs		
Crushing - Conveying	$5.20	6.8%
Leach Pad, Ponds, Sol'n Dist and Collection	$28.75	37.7%
SX/EW	$39.22	51.4%
Utilities	$1.94	2.5%
Mobile Equipment	$1.20	1.6%
Total Equipment	**$76.31**	**100.0%**
Installation Labor	$34.60	32.7%
Concrete	$4.66	4.4%
Piping	$6.57	6.2%

Item	Cost	
	US$(millions)	%
Structural Steel	$4.64	4.4%
Instrumentation	$2.78	2.6%
Insulation	$1.39	1.3%
Electrical	$5.61	5.3%
Coatings and Sealants	$0.49	0.5%
Spares and First Fill	$15.21	14.4%
Engineering/Management	$29.96	28.3%
Total Indirects	**$105.91**	**100.0%**
Total - Direct and Indirect	**$182.23**	
Taxes, Insurance and Freight		
Tax - refundable	$5.53	7.2%
Import Duties (50/50)	$2.29	3.0%
Insurance	$0.76	1.0%
Freight (50/50)	$2.67	3.5%
Total Taxes	**$11.25**	**14.7%**
Total Cost	**$193.47**	

Table 21-5: Moonlight-Superior Copper Project Infrastructure Capital Costs

Item	Capital Costs ($millions)
Pioneering/Clearing/Grubbing	$6.00
Haul Roads	$15.00
Office	$2.00
Warehouse	$2.00
Mine Shop	$10.00
Fuel Bay	$0.50
Wash Bay	$0.50
Cap Magazine and ANFO Storage Bin	$1.00
Camp	$0.00
Site General and Earthwork	$10.00
Surface Water Management	$5.00
Water Supply	$25.00
Back Up Gen Set	$0.35
Sub-station	$15.00
Power Line 33kV	$25.00
Total	**$117.35**

21.1.5 G&A Capital

General and administrative (G&A) capital costs include training, project management, owner's costs, bonding, closure, etc. The costs are incurred in Year -2 and Year -1 The total G&A capital costs are summarized in Table 21-6.

Table 21-6: Moonlight-Superior Copper Project G&A Capital Costs

Item	Capital Costs ($millions)
Startup Training	$1.75
Project Management	$5.00
Drilling and Met Testing	$4.25
Feasibility Study	$3.00
Construction Insurance	$1.40
Commissioning and Start-up	$2.50
Reclamation Bond	$29.32
Permitting	$3.50
Environmental Monitoring Network	$1.00
Closure	$50.00
Total	**$101.72**

21.1.6 Sustaining

Sustaining costs include equipment replacement costs, totaling $9.3 million, and are incurred in throughout the life of the project.

21.1.7 Working Capital

Working capital is the necessary cash on hand for the next period's operating cost. The estimated total is $36 million. This cost is recovered at the end of production.

21.1.8 Royalty Buyback

An existing 2% net smelter returns (NSR) royalty exists on the Moonlight-Superior resource exploitation concession. The royalty is assumed to be re-purchased for a cost of $3 million in accordance with the buyback provisions of the royalty.

21.1.9 Contingency

A 20% contingency was applied on all capital items, excluding the royalty buyback.

21.2 Operating Costs

The operating costs assume owner operation. Operating costs are summarized in Table 21-7.

Table 21-7: Moonlight-Superior Copper Project Operating Cost Summary

Item	Total Operating Cost ($millions)	Unit Operating Cost	Unit
Mining	$899	$1.51	$/ton mined
Processing - Sulfides	$1,520	$5.24	$/ton processed
Processing - Oxides and Transition	$215	$8.74	$/ton processed
Rehandle	$85	$0.75	$/ton processed
G&A	$108	$0.34	$/ton processed
Contingency	$283	$0.90	$/ton processed
Total	**$3,111**		

Operating cost estimates were prepared based on current and expected long-term pricing assumptions and to a PEA level of +/- 35% level of accuracy.

21.2.1 Labor

Hourly labor for the project is based on the number of people needed to operate and support equipment for each shift in a day plus additional crew to fill in for absences. Salaried labor in the project is based on job positions filled regardless of production changes or equipment units needed. Table 21-8 through Table 21-10 show the required labor, and Table 21-11 shows the estimated mining and G&A labor costs by year. Processing labor costs are built into the processing unit costs.

Table 21-8: Moonlight-Superior Copper Project Hourly Labor by Year

Position	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Drill Operator	0	20	28	28	20	32	28	24	20	20	24	12	**256**
Blaster	0	16	24	24	16	28	24	20	16	16	20	8	**212**
Blaster Helper	0	16	24	24	16	28	24	20	16	16	20	8	**212**
Haul Truck Driver	0	12	84	96	64	120	96	76	56	36	36	16	**692**
Loader/Shovel Operator	0	12	28	24	16	24	16	12	8	8	12	4	**164**
Dozer Operator	0	14	22	18	18	30	26	22	18	18	22	14	**222**
Loader Operator	0	4	4	4	4	4	4	4	4	4	4	4	**44**
General Equipment Operator	0	20	20	20	20	20	20	20	20	20	20	20	**220**
Water Truck Driver	0	8	8	8	8	8	8	8	8	8	8	8	**88**
Lube Truck Driver	0	8	8	8	8	8	8	8	8	8	8	8	**88**
Laborer	0	8	8	8	8	8	8	8	8	8	8	8	**88**
Heavy Duty Mechanic	0	0	23	29	15	0	0	0	0	0	0	0	**67**
Light Duty Mechanic	0	4	4	4	4	4	4	4	4	4	4	4	**44**
Tire Man	0	4	4	4	4	4	4	4	4	4	4	4	**44**
Total Hourly Mine Labor	**0**	**146**	**289**	**299**	**221**	**318**	**270**	**230**	**190**	**170**	**190**	**118**	**2,441**

Table 21-9: Moonlight-Superior Project Salaried Workers, Mine Management

Position	Number Each Year
Mine Ops/Technical Superintendent	2
Mine Shift Foreman	4
Clerk/Secretary	2
Open Pit Planning Engineer	2
Sampling/Geology Technician	2
Senior Engineer	2
Senior Geologist	2
Surveyor/Mine Tech helper	2
Surveyor/Mining Technician	2
Maintenance General Foreman	2
Maintenance Shift Foreman	4

Position	Number Each Year
Maintenance Superintendent	2
Grade Control Geologist/Modeler	2
Total	**30**

Table 21-10: Moonlight-Superior Project G&A Labor

Position	Number Each Year
General Manager	1
Purchasing Manager	1
Purchaser	2
Chief Accountant	1
Accounting Clerk	2
Human Resources/Relations Manager	1
Human Resources/Payroll Clerk	2
Security/Safety/Training Manager	1
Safety Officer	2
Environmental Supervisor	1
Environmental Technician	2
Logistics Administrator	1
IT Manager	1
Warehouseman	4
Accounts Payable Clerk	1
Receptionist/Secretary	1
Guard	4
Driver	1
Laborers/Janitorial	2
Total	**31**

Table 21-11: Moonlight-Superior Copper Project Labor Costs by Year ($ millions)

Item	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	Total
Mine Labor	$0.00	$20.14	$36.02	$37.22	$28.68	$38.72	$33.56	$29.29	$25.01	$22.80	$24.86	$7.38	$0.00	$0.00	$0.00	$0.00	**$303.67**
G&A Labor	$1.30	$2.93	$2.93	$2.93	$2.93	$2.93	$2.93	$2.93	$2.93	$2.93	$2.93	$2.93	$2.93	$2.93	$2.93	$2.93	**$42.28**
Total Labor Costs	**$1.30**	**$23.07**	**$38.95**	**$40.15**	**$31.61**	**$41.64**	**$36.49**	**$32.21**	**$27.94**	**$25.72**	**$27.78**	**$10.30**	**$2.93**	**$2.93**	**$2.93**	**$2.93**	**$345.94**

21.2.2 Mining Equipment and Consumables

Mining equipment includes production equipment and support equipment. Mining production equipment hours are calculated using the equipment productivity estimates and the number of tons required to be moved. It was assumed that all mining will be owner-operated.

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

Blasting costs were calculated assuming powder factors of 0.5 pounds (lbs) of ammonium nitrate fuel oil (ANFO)/ton of processable material and 0.4 lb/ton of waste rock and an ANFO unit rate of $0.34/lb. Caps and primers were included at a rate of one per blast hole each at a cost of $7.40 each. Ore control testing was included at a unit rate of $0.03/st, and a miscellaneous blasting cost of $500,000/year was included.

Table 21-12 summarizes the mining equipment costs by year, and Table 21-13 summarizes the blasting costs per year.

21.2.3 Concentrator Plant

The processing operating costs include labor, reagents and consumables, and power. Table 21-14 shows the estimated labor, consumable and power costs. The total unit rate for processing is $6.50 per tonne of material processed in the concentrator.

21.2.4 Heap Leach Plant

The processing operating costs include labor, reagents and consumables, and power. Table 21-14 shows the estimated labor, consumable and power costs. The total unit rate for processing is $9.62 per tonne of material processed in the heap leach. A summary of the process operating costs is provided in Table 21-15.

21.2.5 General and Administrative

General and administrative costs were estimated for two phases of the mine plan: open pit production operating and rinse and closure. The G&A costs include both salaried and hourly labor, supplies, office equipment, and anticipated regular expenses. Open pit production years have a G&A cost of $9.4 million per year and rinse and closure years have a G&A cost of 3.9 million per year.

21.2.6 Closure

Closure costs are estimated in one year at the end of production due and may include rinsing and neutralizing the leached material, and closure of waste stockpiles and TSFs, but does not, at this time, include backfilling of pits. The total estimated cost for site closure is $50 million.

21.2.7 Contingency

A 10% contingency was applied to operating costs.

Table 21-12: Moonlight-Superior Copper Project Mining Equipment Costs by Year ($millions)

Item	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	Total
Mine Production Equipment	$0.00	$0.16	$44.45	$42.23	$29.21	$74.81	$61.20	$47.01	$33.29	$25.09	$26.71	$10.03	$0.00	$0.00	$0.00	**$394.19**
Mine Support Equipment	$0.00	$7.51	$7.51	$7.51	$7.51	$7.51	$7.51	$7.51	$7.51	$7.51	$7.51	$3.19	$0.00	$0.00	$0.00	**$78.26**
Total	**$0.00**	**$7.66**	**$51.95**	**$49.74**	**$36.71**	**$82.32**	**$68.71**	**$54.52**	**$40.80**	**$32.60**	**$34.21**	**$13.23**	**$0.00**	**$0.00**	**$0.00**	**$472.45**

Table 21-13: Moonlight-Superior Copper Project Blasting Costs by Year ($millions)

Item	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	Total
Explosives	$0.00	$0.08	$13.40	$12.45	$7.52	$17.56	$14.04	$10.44	$7.15	$7.06	$9.47	$2.92	$0.00	$0.00	$0.00	**$102.11**
Caps	$0.00	$0.00	$0.41	$0.38	$0.23	$0.54	$0.43	$0.32	$0.22	$0.22	$0.29	$0.09	$0.00	$0.00	$0.00	**$3.15**
Primers	$0.00	$0.00	$0.36	$0.33	$0.20	$0.47	$0.38	$0.28	$0.19	$0.19	$0.25	$0.08	$0.00	$0.00	$0.00	**$2.74**
Ore Control Testing	$0.00	$0.00	$1.09	$1.10	$0.93	$1.11	$1.19	$1.07	$0.89	$0.73	$0.95	$0.38	$0.00	$0.00	$0.00	**$9.44**
Miscellaneous	$0.00	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.21	$0.00	$0.00	$0.00	**$5.21**
Total Blasting Costs	**$0.00**	**$0.58**	**$15.77**	**$14.77**	**$9.39**	**$20.18**	**$16.54**	**$12.62**	**$8.95**	**$8.70**	**$11.47**	**$3.68**	**$0.00**	**$0.00**	**$0.00**	**$122.65**

Table 21-14: Moonlight-Superior Copper Project Concentrator Operating Costs

Labor							
Area	**Position**	**Number**	**Base $/yr**	**Burden $/yr**	**Total $/yr**	**Total $/mt**	**Total $/lb Cu**
Metallurgical Staff							
	Superintendent	1	$175,000	$105,000	$280,000	$0.02	$0.003
	General Foreman	4	$110,000	$66,000	$704,000	$0.04	$0.007
	Maintenance Foreman	1	$110,000	$66,000	$176,000	$0.01	$0.002
	Shift Foreman	4	$75,000	$45,000	$480,000	$0.03	$0.005
	Chief Assay Chemist	1	$90,000	$54,000	$144,000	$0.01	$0.001
	Sr Metallurgist	2	$110,000	$66,000	$352,000	$0.02	$0.003
	Metallurgist	1	$75,000	$45,000	$120,000	$0.01	$0.001
	Process Technician	2	$65,000	$39,000	$208,000	$0.01	$0.002

Labor							
Area	Position	Number	Base $/yr	Burden $/yr	Total $/yr	Total $/mt	Total $/lb Cu
	Instrument Technician	2	$65,000	$39,000	$208,000	$0.01	$0.002
Subtotal		**18**			**$2,672,000**	**$0.15**	**$0.03**
Crusher							
	Operator	8	$75,000	$45,000	$960,000	$0.05	$0.009
	FEL Operator	4	$75,000	$45,000	$480,000	$0.03	$0.005
	Maintenance	3	$85,000	$51,000	$408,000	$0.02	$0.004
	Electrical	1	$85,000	$51,000	$136,000	$0.01	$0.001
Subtotal		**16**			**$1,984,000**	**$0.11**	**$0.02**
Mill							
	Grinding	8	$75,000	$45,000	$960,000	$0.05	$0.009
	Flotation	16	$75,000	$45,000	$1,920,000	$0.11	$0.019
	Regrind	4	$75,000	$45,000	$480,000	$0.03	$0.005
	Concentrate Loadout	8	$75,000	$45,000	$960,000	$0.05	$0.009
	Assay Laboratory	16	$75,000	$45,000	$1,920,000	$0.11	$0.019
	Samplers	4	$75,000	$45,000	$480,000	$0.03	$0.005
	Mechanic	6	$85,000	$51,000	$816,000	$0.05	$0.008
	Electrician	3	$85,000	$51,000	$408,000	$0.02	$0.004
Subtotal		**65**			**$7,944,000**	**$0.44**	**$0.08**
Tailings							
	Thickener	6	$75,000	$45,000	$720,000	$0.04	$0.007
	Filter	8	$75,000	$45,000	$960,000	$0.05	$0.009
	Stacking	8	$75,000	$45,000	$960,000	$0.05	$0.009
	Mechanic	4	$85,000	$51,000	$544,000	$0.03	$0.005
	Electrician	3	$85,000	$51,000	$408,000	$0.02	$0.004
Subtotal		**22**			**$3,592,000**	**$0.20**	**$0.04**
Total Labor		**121**			**$16,192,000**	**$0.91**	**$0.16**

Reagents and Consumables									
Area	Material	Unit Consumption	Units	Annual Consumption	Units	Cost Per Unit	Total $/yr	$/t	$/lb Cu
Crusher									
	Liners	3	set/yr	3.00	ea	$235,000	$705,000	$0.04	$0.007
	Conveyors	1	set/yr	1.00	ea	$15,000	$15,000	$0.00	$0.000
	Misc - belts, lube						$900,000	$0.05	$0.009
Subtotal							**$1,620,000**	**$0.09**	**$0.02**
Comminution									
	Liners/Balls	6.00	set/yr	6	ea	$3,600,000	$21,600,000	$1.21	$0.213
	Parts			2	ea	$970,000	$1,940,000	$0.11	$0.019
	Misc - belts, lube			1	ea	$840,000	$840,000	$0.05	$0.008
Subtotal							**$840,000**	**$1.36**	**$0.24**
Flotation									
	Collector	20.00	g/t	358	t	$4,590	$1,641,441	$0.09	$0.016
	Promotor	10.00	g/t	179	t	$4,590	$820,720	$0.05	$0.008
	Frother	20.00	g/t	358	t	$4,560	$1,630,712	$0.09	$0.016
	Lime	0.10	kg/t	1,807	t	$260	$469,890	$0.03	$0.005
Subtotal							**$4,562,763**	**$0.26**	**$0.05**
Regrind									
	Liners/Balls	3.00	set/yr	3	ea	$155,000	$465,000	$0.03	$0.005
	Parts			2	ea	$230,000	$460,000	$0.03	$0.005
	Misc - belts, lube			1	ea	$167,000	$167,000	$0.01	$0.002
Subtotal							**$1,092,000**	**$0.06**	**$0.01**
Thickening									
	Conc Thickener Floc	80.00	g/t	1,430	t	$4,080	$5,836,233	$0.33	$0.058
	Conc Thickener Maint						$595,000	$0.03	$0.006
	Tails Thickener Floc	50.00	g/t	894	t	$4,080	$3,647,646	$0.20	$0.036
	Tails Thickener Maint						$470,000	$0.03	$0.005
	Filter Maintenance						$350,000	$0.02	$0.003
Subtotal							**$10,898,879**	**$0.61**	**$0.11**
Consumables									
	Maintenance Items	3.0%	Equip Cost	$162	$M Eq Cost		$4,864,999	$0.27	$0.048
	Diesel	1,200	liters/hr	9,600,000	liter/yr	1.02	$9,792,000	$0.55	$0.097

Reagents and Consumables									
Area	Material	Unit Consumption	Units	Annual Consumption	Units	Cost Per Unit	Total $/yr	Total $/t	Total $/lb Cu
	Fresh Water	-	m^3/h	-	m^3	0	$-	$-	$-
	Lab Supplies						$500,000	$0.03	$0.005
	Misc Op Supplies	121	employ	$5,000.00	$/emp		$605,000	$0.03	$0.006
Subtotal							**$15,761,999**	**$0.88**	**$0.16**
Total Reagents and Consumables							**$34,775,641**	**$3.26**	**$0.58**

Power Consumption						
Area	Installed KW	Power kwh/yr	Cost $/kwh	Total $/yr	Total $/t	Total $/lb Cu
Crushing	1,330	7,571,432	$0.134	$1,010,786	$0.06	$0.010
Grinding	34,572	229,576,005	$0.134	$30,648,397	$1.71	$0.303
Flotation	4,257	23,863,060	$0.134	$3,185,719	$0.18	$0.031
Regrind	3,910	22,789,497	$0.134	$3,042,398	$0.17	$0.030
Thickening	3,077	15,750,561	$0.134	$2,102,700	$0.12	$0.021
Utilities	2,689	13,265,707	$0.134	$1,770,972	$0.10	$0.017
Reagents	141	333,152	$0.134	$44,476	$0.00	$0.000
Total	**49,974**	**313,149,414**		$41,805,447	**$2.34**	**$0.41**

Table 21-15: Moonlight-Superior Copper Project Heap Leach Operating Costs

Labor							
Area	Position	Number	Base $/yr	Burden $/yr	Total $/yr	Total $/t	Total $/lb
Metallurgical Staff							
	Plant Manager	1	$110,000	$66,000	$176,000	$0.06	$0.008
	General Foreman	1	$110,000	$66,000	$176,000	$0.06	$0.008
	Maintenance Superintendent	1	$110,000	$66,000	$176,000	$0.06	$0.008
	Shift Foreman	4	$75,000	$45,000	$480,000	$0.16	$0.023
	Sr Metallurgist	1	$110,000	$66,000	$176,000	$0.06	$0.008
	Instrument Technician	1	$65,000	$39,000	$104,000	$0.03	$0.005
Subtotal		**9**			**$1,288,000**	**$0.43**	**$0.06**

Labor							
			Base	Burden	Total		
Area	**Position**	**Number**	**$/yr**	**$/yr**	**$/yr**	**$/t**	**$/lb**
Crusher							
	Operator	8	$75,000	$45,000	$960,000	$0.32	$0.046
	FEL Operator	4	$75,000	$45,000	$480,000	$0.16	$0.023
	Maintenance	1	$75,000	$45,000	$120,000	$0.04	$0.006
	Electrical	1	$75,000	$45,000	$120,000	$0.04	$0.006
Subtotal		**14**			**$1,680,000**	**$0.56**	**$0.08**
Heap							
	Stacking	4	$75,000	$45,000	$480,000	$0.16	$0.023
	Irrigation Operator	4	$75,000	$45,000	$480,000	$0.16	$0.023
	Equipment Operator	4	$75,000	$45,000	$480,000	$0.16	$0.023
	Maintenance	2	$85,000	$51,000	$272,000	$0.09	$0.013
	Electrician	1	$85,000	$51,000	$136,000	$0.05	$0.007
Subtotal		**15**			**$1,848,000**	**$0.62**	**$0.09**
SX/EW							
	SX Operators	4	$75,000	$45,000	$480,000	$0.16	$0.023
	EW Operators	4	$75,000	$45,000	$480,000	$0.16	$0.023
	Cathode Striping	8	$75,000	$45,000	$960,000	$0.32	$0.046
	Maintenance	2	$85,000	$51,000	$272,000	$0.09	$0.013
	Electrician	1	$85,000	$51,000	$136,000	$0.05	$0.007
Subtotal		**19**			**$2,328,000**	**$0.78**	**$0.11**
Total		**57**			**$7,144,000**	**$2.40**	**$0.34**

Reagents and Consumables									
							Total		
Area	**Material**	**Unit Consumption**	**Units**	**Annual Consumption**	**Units**	**Cost Per Unit**	**$/yr**	**$/t**	**$/lb**
Crusher									
	Jaws	3	set/yr	3.00	ea	$135,000	$405,000	$0.14	$0.019
	Misc - belts, lube						$125,000	$0.04	$0.006
Subtotal							**$530,000**	**$0.18**	**$0.03**
Leach									
	Acid	6.00	kg/t	17,881	t	$200.00	$3,576,122	$1.20	$0.172

Reagents and Consumables									
Area	Material	Unit Consumption	Units	Annual Consumption	Units	Cost Per Unit	Total $/yr	$/t	$/lb
	Irrigation Cons	6.00	cells	10,000	m2/yr	$2.00	$120,000	$0.04	$0.006
Subtotal							**$3,696,122**	**$1.24**	**$0.18**
SX/EW									
	Organic	0.001	kg/m3	17	t	$12,128	$208,987	$0.07	$0.010
	Diluent	0.004	m3/m3	69	m3	$1,255	$86,504	$0.03	$0.004
	Cathode and Anode Reposition	2%	repo	2,012	num	$5,000	$201,243	$0.07	$0.010
	Guar	200.00	g/t	1.4	t	$5,000	$7,118	$0.00	$0.000
	Cobalt	100.00	mg/l	19.7	t	$35,000	$688,485	$0.23	$0.033
	Other Reagents						$250,000	$0.08	$0.012
	Natural Gas	5.00	mbtu/hr	43,800	mbtu	$2.85	$124,830	$0.04	$0.006
	Acid	3	kg/t	8,940	t	$200.00	$1,788,061	$0.60	$0.086
Subtotal							**$3,355,228**	**$1.13**	**$0.16**
Consumables									
	Maintenance Items	3.0%	Equip Cost	$76	Eq Cost (M)		$2,289,447	$0.77	$0.110
	Diesel	300	liters/hr	2,628,000	liter/yr	1.02	$2,680,560	$0.90	$0.129
	Buidling Heat	144,184	BTU/m2	1,211	mbtu	2.85	$3,452	$0.00	$0.000
	Fresh Water	0.20	m^3/t	596,020	m3/yr	0	$-	$-	$-
	Lab Supplies						$-	$-	$-
	Misc Op Supplies	57	employ	$5,000.00	$/emp		$285,000	$0.10	$0.014
Subtotal							**$5,258,459**	**$1.76**	**$0.253**
Total							**$12,839,809**	**$4.31**	**$0.62**

Power Consumption						
Area	Installed KW	KWhr/yr	Cost $/kwh	Total $/yr	$/t	$/lb
Crushing	3,051	16,035,223	$0.1335	$2,140,702	$0.72	$0.103
Leach	4,533	22,723,363	$0.1335	$3,033,569	$1.02	$0.146
SX/EW	3,481	24,877,267	$0.1335	$3,321,115	$1.11	$0.160
Utilities	224	1,489,971	$0.1335	$198,911	$0.07	$0.010
Total	**11,065**	**63,635,853**		**$8,694,298**	**$2.92**	**$0.418**

22 ECONOMIC ANALYSIS

Readers are advised that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under National Instrument 43-101. This 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 under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this preliminary economic assessment will be realized.

22.1 Model Cases

A multi scenario analysis method was used to analyze the economic performance of the project by varying the cutoff grade, plant and heap leach locations, and method of procuring mobile production and support equipment.

Ms. Lane of GRE evaluated the following options:

- Sulfide high-grade cutoffs of 15, 16, 17, 18, 19, and 20 NSR
- Oxide + transition material cutoff grades of 0.12%, 0.16%, 0.20%, 0.24%, and 0.28 % copper

After analyzing the economic results of all cases considered, Ms. Lane of GRE selected the 15 NSR high-grade sulfide cutoff and 0.28% copper oxide+transition cutoff as the base case as it results in the best overall economics.

22.2 Economic Analysis

22.2.1 Assumptions

Ms. Lane of GRE performed an economic analysis of the project by building an economic model based on the following assumptions:

- Copper price of $4.15/lb, based on using a weighted average of the 3-year trailing average copper price and the 1-year futures price, calculated as: 60% x 3-year trailing average price of $4.06/lb + 40% x 1-year futures price of $4.30/lb
- Silver price of $27.40/oz, based on using a weighted average of the 3-year trailing average silver price and the 1-year futures price, calculated as: 60% x 3-year trailing average price of $24.19/oz + 40% x 1-year futures price of $32.26/oz
- Gold price of $2,320/oz, based on using a weighted average of the 3-year trailing average gold price and the 1-year futures price, calculated as: 60% x 3-year trailing average price of $2,015/oz + 40% x 1-year futures price of $2,779/oz
- Sulfide material mineral recoveries of: 90.2% for copper, 80.4% for silver, and 71.0% for gold
- Heap leach mineral recoveries of: 75% for oxide material copper and 60% for transition material copper
- Leach recovery delay as follows: 60% recovered during the first year on the heap, 30% recovered in the second year on the heap, and 10% recovered during the third year on the heap

- Copper 100% payable
- $.036/lb Cu from the heap leach cathode premium
- $160/ton transportation and off-site charges
- $3 million cost up front to purchase back royalties
- All costs input to the model are in US dollars.
- Sales and use taxes are not included in the model
- Taxes, depreciation, amortization, and deductions as described below.

22.2.2 Taxes

Note: Ms. Lane is not an expert in US taxes and relied on information provided by US Copper and obtained from on-line searches of US tax codes to generate a tax model for the project. The calculations are based on the tax regime as of the date of this 2024 PEA. The tax calculations should be considered approximations because actual tax estimates involve complex calculations that can be accurately determined only during operations.

22.3 Results

Ms. Lane of GRE considered the following key economic parameters to determine the best scenario: Net Present Value (NPV), Internal Rate of Return (IRR), payback period, mine life, and initial capital cost. Table 22-1 summarizes the results of the economic model.

Table 22-2 presents the key economic results for the project.

Table 22-1: Moonlight-Superior Copper Project Economic Model Summary

Item	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	Total
Mine Production																		
Total Mineralized Material (million tons)	0.00	0.01	36.29	36.62	30.99	36.91	39.56	35.76	29.59	24.36	31.79	12.72	0.00	0.00	0.00	0.00	0.00	**314.61**
Total Sulfide Material (million tons)	0.00	0.01	20.68	30.25	30.08	35.54	39.18	35.76	29.59	24.36	31.79	12.72	0.00	0.00	0.00	0.00	0.00	**289.96**
Total Oxide Material (million tons)	0.00	0.00	2.39	0.44	0.05	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**2.89**
Total Transition Material (million tons)	0.00	0.00	13.22	5.92	0.86	1.37	0.39	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**21.77**
Prestrip Waste (million tons)	0.00	0.50	11.13	5.71	0.58	11.43	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**29.34**
In Bench Waste (million tons)	0.00	0.00	32.16	30.84	10.44	58.58	43.43	24.37	10.32	16.22	22.91	3.41	0.00	0.00	0.00	0.00	0.00	**252.70**
Total Waste (million tons)	0.00	0.50	43.29	36.55	11.02	70.01	43.43	24.37	10.32	16.22	22.91	3.41	0.00	0.00	0.00	0.00	0.00	**282.04**
Stripping Ratio		93.01	1.19	1.00	0.36	1.90	1.10	0.68	0.35	0.67	0.72	0.27						**0.90**
Total Tons (million tons)	0.00	0.50	79.58	73.17	42.02	106.92	82.99	60.14	39.92	40.58	54.70	16.13	0.00	0.00	0.00	0.00	0.00	**596.66**
Sulfide Plant Production																		
High Grade Tons Processed (million tons)	0.00	0.00	13.69	21.90	21.90	21.90	21.90	21.90	21.90	16.45	21.90	9.31	0.00	0.00	0.00	0.00	0.00	**192.75**
Low Grade Tons Processed (million tons)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	5.45	0.00	12.59	21.90	21.90	21.90	13.47	0.00	**97.22**
Concentrate Produced (million tons)	0.00	0.00	0.17	0.28	0.28	0.28	0.28	0.28	0.28	0.25	0.28	0.21	0.17	0.17	0.17	0.10	0.00	**3.18**
Recovered Metals																		
Sulfide Cu - High Grade (million lbs)	0.00	0.00	96.00	158.24	146.67	157.77	141.43	134.56	140.47	103.08	151.12	65.51	0.00	0.00	0.00	0.00	0.00	**1,294.85**
Sulfide Cu - Low Grade (million lbs)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	19.58	0.00	45.19	78.62	78.62	78.62	48.37	0.00	**349.00**
Total Cu (million lbs)	0.00	0.00	96.00	158.24	146.67	157.77	141.43	134.56	140.47	122.67	151.12	110.70	78.62	78.62	78.62	48.37	0.00	**1,643.85**
Sulfide Ag - High Grade ('000s oz)	0.00	0.00	1,484.51	2,371.64	1,316.54	967.30	468.22	659.19	484.03	175.62	987.49	319.40	0.00	0.00	0.00	0.00	0.00	**9,233.93**
Sulfide Ag - Low Grade ('000s oz)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	148.32	0.00	363.83	632.98	632.98	632.98	389.47	0.00	**2,800.56**
Total Ag ('000s oz)	0.00	0.00	1,484.51	2,371.64	1,316.54	967.30	468.22	659.19	484.03	323.94	987.49	683.23	632.98	632.98	632.98	389.47	0.00	**12,034.49**
Sulfide Au - High Grade ('000s oz)	0.00	0.00	17.92	18.63	2.52	3.51	0.47	0.51	0.69	0.67	4.62	1.43	0.00	0.00	0.00	0.00	0.00	**50.98**
Sulfide Au - Low Grade ('000s oz)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.64	0.00	1.60	2.77	2.77	2.77	1.71	0.00	**12.26**
Total Au ('000s oz)	0.00	0.00	17.92	18.63	2.52	3.51	0.47	0.51	0.69	1.31	4.62	3.03	2.77	2.77	2.77	1.71	0.00	**63.25**
Heap Leach Production																		
Tons Processed (million tons)	0.00	0.00	3.29	3.29	3.28	3.29	3.28	3.29	3.29	1.66	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**24.65**
Recovered Metals																		
Oxide Cu (million lbs)	0.00	0.00	18.77	11.61	4.46	0.48	0.04	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**35.36**
Transition Cu (million lbs)	0.00	0.00	3.30	11.98	16.63	17.45	18.51	19.56	20.03	14.13	5.05	1.01	0.00	0.00	0.00	0.00	0.00	**127.67**
Total Cu (million lbs)	0.00	0.00	22.07	23.59	21.09	17.93	18.55	19.56	20.03	14.13	5.05	1.01	0.00	0.00	0.00	0.00	0.00	**163.03**
Revenue																		
Cu Revenue (millions $)	$0.00	$0.00	$490.00	$754.57	$696.21	$729.17	$663.90	$639.60	$666.11	$567.72	$648.14	$463.61	$326.26	$326.26	$326.26	$200.75	$0.00	**$7,498.57**
Ag Revenue (millions $)	$0.00	$0.00	$40.68	$64.98	$36.07	$26.50	$12.83	$18.06	$13.26	$8.88	$27.06	$18.72	$17.34	$17.34	$17.34	$10.67	$0.00	**$329.75**
Au Revenue (millions $)	$0.00	$0.00	$41.58	$43.23	$5.84	$8.14	$1.10	$1.18	$1.60	$1.56	$10.73	$3.32	$0.00	$0.00	$0.00	$0.00	$0.00	**$118.28**
Cathode Premium (millions $)	$0.00	$0.00	($0.79)	($0.85)	($0.76)	($0.65)	($0.67)	($0.70)	($0.72)	($0.51)	($0.18)	($0.04)	$0.00	$0.00	$0.00	$0.00	$0.00	**($5.87)**
Cu Refining/Selling Cost (millions $)	$0.00	$0.00	($7.68)	($12.66)	($11.73)	($12.62)	($11.31)	($10.76)	($11.24)	($9.81)	($12.09)	($8.86)	($6.29)	($6.29)	($6.29)	($3.87)	$0.00	**($131.51)**
Ag & Au Refining/Selling Cost (millions $)	$0.00	$0.00	($0.45)	($0.72)	($0.40)	($0.29)	($0.14)	($0.20)	($0.15)	($0.10)	($0.30)	($0.21)	($0.19)	($0.19)	($0.19)	($0.12)	$0.00	**($3.63)**
Transportation Charges (millions $)	$0.00	$0.00	($29.36)	($46.04)	($45.84)	($45.59)	($45.63)	($45.72)	($45.75)	($41.01)	($44.55)	($34.36)	($26.98)	($26.98)	($26.98)	($16.60)	$0.00	**($521.39)**
Royalty (millions $)	$0.00	($3.00)	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**($3.00)**
Net Revenue (millions $)	$0.00	($3.00)	$533.97	$802.53	$679.40	$704.68	$620.07	$601.45	$623.11	$528.20	$628.80	$445.90	$316.58	$316.58	$316.58	$194.79	$0.00	**$7,309.65**
Operating Costs																		

Item	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	Total
Mine Opex (millions $)	$0.00	($7.66)	($51.95)	($49.74)	($36.71)	($82.32)	($68.71)	($54.52)	($40.80)	($32.60)	($34.21)	($13.23)	$0.00	$0.00	$0.00	$0.00	$0.00	**($472.45)**
Mine Labor (millions $)	$0.00	($20.14)	($36.02)	($37.22)	($28.68)	($38.72)	($33.56)	($29.29)	($25.01)	($22.80)	($24.86)	($7.38)	$0.00	$0.00	$0.00	$0.00	$0.00	**($303.67)**
Blasting (millions $)	$0.00	($0.58)	($15.77)	($14.77)	($9.39)	($20.18)	($16.54)	($12.62)	($8.95)	($8.70)	($11.47)	($3.68)	$0.00	$0.00	$0.00	$0.00	$0.00	**($122.65)**
Process Opex (millions $)	$0.00	$0.00	($100.47)	($143.57)	($145.31)	($145.81)	($145.70)	($146.00)	($146.00)	($134.69)	($114.83)	($124.27)	($131.26)	($131.26)	($131.26)	($80.76)	$0.00	**($1,821.19)**
G&A Opex (millions $)	($3.25)	($7.00)	($7.00)	($7.00)	($7.00)	($7.00)	($7.00)	($7.00)	($7.00)	($7.00)	($7.00)	($7.00)	($7.00)	($7.00)	($7.00)	($7.00)	$0.00	**($108.32)**
Contingency (millions $)	($0.32)	($3.54)	($21.12)	($25.23)	($22.71)	($29.40)	($27.15)	($24.94)	($22.78)	($20.58)	($19.24)	($15.56)	($13.83)	($13.83)	($13.83)	($8.78)	$0.00	**($282.83)**
Total Operating Costs (millions $)	**($3.57)**	**($38.93)**	**($232.34)**	**($277.53)**	**($249.80)**	**($323.43)**	**($298.67)**	**($274.37)**	**($250.54)**	**($226.36)**	**($211.62)**	**($171.12)**	**($152.09)**	**($152.09)**	**($152.09)**	**($96.54)**	**$0.00**	**($3,111.10)**
Taxes																		
Income																		
Federal Tax (millions $)	$0.00	$0.00	($26.92)	($60.36)	($39.97)	($32.24)	($25.11)	($26.01)	($33.34)	($25.05)	($40.73)	($19.46)	($14.20)	($14.44)	($14.42)	($5.76)	$0.00	**($378.03)**
State Tax (millions $)	$0.00	$0.00	($11.28)	($25.30)	($16.75)	($13.51)	($10.52)	($10.90)	($13.97)	($10.50)	($17.07)	($8.16)	($5.95)	($6.05)	($6.04)	($2.41)	$0.00	**($158.41)**
Property																		
California Property Tax (millions $)	($4.00)	($8.07)	($8.22)	($7.53)	($6.60)	($6.53)	($6.16)	($5.19)	($4.43)	($3.68)	($2.94)	($2.31)	($1.70)	($1.08)	($0.46)	$0.00	($0.30)	**($69.20)**
Capital Costs																		
Mine Equipment Costs (millions $)	$0.00	($57.71)	($75.00)	($9.56)	($0.03)	($75.61)	($0.07)	($0.03)	($0.03)	($0.03)	($0.03)	($0.07)	$0.00	$0.00	$0.00	$0.00	$0.00	**($218.17)**
Process Capital Costs (millions $)	($281.64)	($281.64)	$0.00	$0.00	$0.00	$0.00	($48.27)	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**($611.54)**
Infrastructure Capital Costs (millions $)	($61.00)	($56.35)	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**($117.35)**
G&A Capital Costs (millions $)	$0.00	($22.40)	($7.50)	($1.68)	($1.68)	($1.68)	($1.68)	($1.68)	($1.68)	($1.68)	($1.68)	($1.68)	($1.68)	($1.68)	($1.68)	($1.68)	($50.00)	**($101.72)**
Sustaining Capital (millions $)	$0.00	$0.00	$0.00	($0.24)	$0.00	($2.84)	($4.21)	($1.20)	$0.00	$0.00	($0.78)	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**($9.27)**
Working Capital (millions $)	$0.00	($36.12)	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**($36.12)**
Contingency (millions $)	($68.53)	($90.84)	($16.50)	($2.29)	($0.34)	($16.03)	($10.85)	($0.58)	($0.34)	($0.34)	($0.50)	($0.35)	($0.34)	($0.34)	($0.34)	($0.34)	($10.00)	**($218.83)**
Total Capital Costs (millions $)	**($411.16)**	**($545.05)**	**($99.00)**	**($13.76)**	**($2.05)**	**($96.15)**	**($65.08)**	**($3.49)**	**($2.05)**	**($2.05)**	**($2.99)**	**($2.10)**	**($2.01)**	**($2.01)**	**($2.01)**	**($2.01)**	**($60.00)**	**($1,313.00)**
Net Cashflow pre-tax (millions $)	($414.73)	($586.98)	$202.63	$511.23	$427.55	$285.10	$256.32	$323.59	$370.52	$299.79	$414.19	$272.68	$162.48	$162.48	$162.48	$96.23	($60.00)	**$2,885.55**
Net Cashflow after tax (millions $)	($418.73)	($595.05)	$156.21	$418.04	$364.24	$232.82	$214.52	$281.50	$318.78	$260.56	$353.45	$242.74	$140.63	$140.90	$141.55	$88.06	($60.30)	**$2,279.91**

Table 22-2: Moonlight-Superior Copper Project Key Economic Results

Economic Measure	Value
After Tax NPV @ 7% (millions)	$1,075
IRR	23%
Initial Capital (millions)	$956
Payback Period (year)	5.3
All-in Sustaining Cost ($/lb Cu Produced)	$2.51

22.4 Sensitivity Analyses

Ms. Lane of GRE evaluated the after-tax NPV@7% and IRR sensitivity to changes in capital costs, operating costs, copper price, and copper grade. For this analysis, Ms. Lane of GRE used a base case copper price of $4.15/lb. The results are shown in indicate that the after-tax NPV@7% and IRR are most sensitive to copper price and copper grade and moderately sensitive to operating cost and capital costs (Table 22-3 for NPV@7% and Table 22-4 for IRR). Additionally, the NPVs and IRRs at specific copper prices are shown in Table 22-5.

Table 22-3: Moonlight-Superior Copper Project Post-Tax NPV@7% Sensitivities

Variable	% of Base Case		
	75%	100%	125%
Capital Cost	$1,329	$1,075	$816
Operating Cost	$1,447	$1,075	$671
Copper Price	$91	$1,075	$2,014
Copper Grade	$114	$1,075	$1,990

Figure 22-1: Moonlight-Superior project Post-Tax NPV@7% Sensitivities

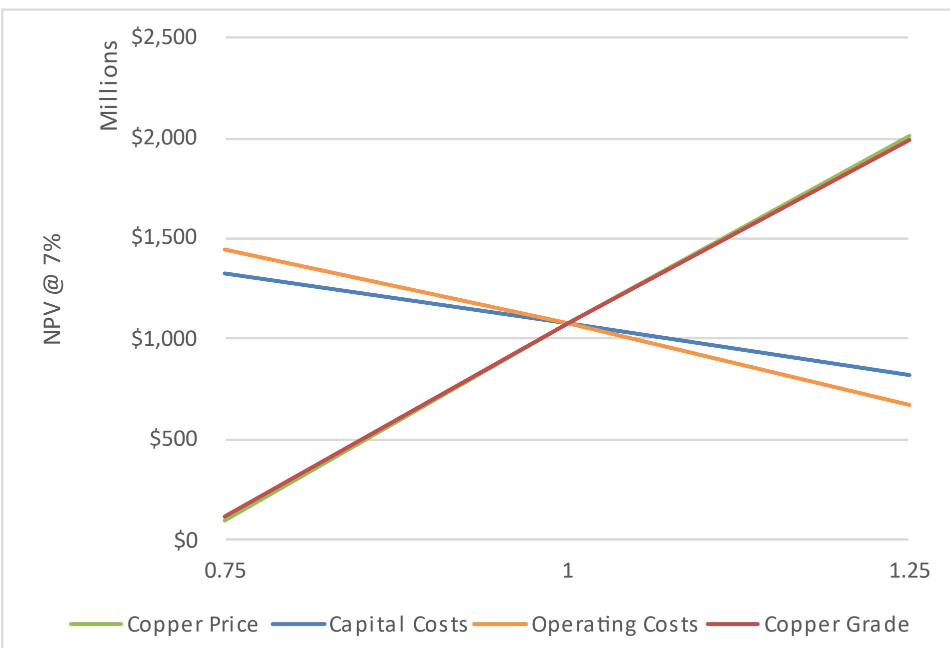


Table 22-4: Moonlight-Superior Copper Project Post-Tax IRR Sensitivities

Variable	% of Base Case		
	75%	100%	125%
Capital Cost	30%	23%	18%
Operating Cost	28%	23%	17%
Copper Price	8%	23%	34%
Copper Grade	9%	23%	34%

Figure 22-2: Moonlight-Superior Copper Project Post-Tax IRR Sensitivities

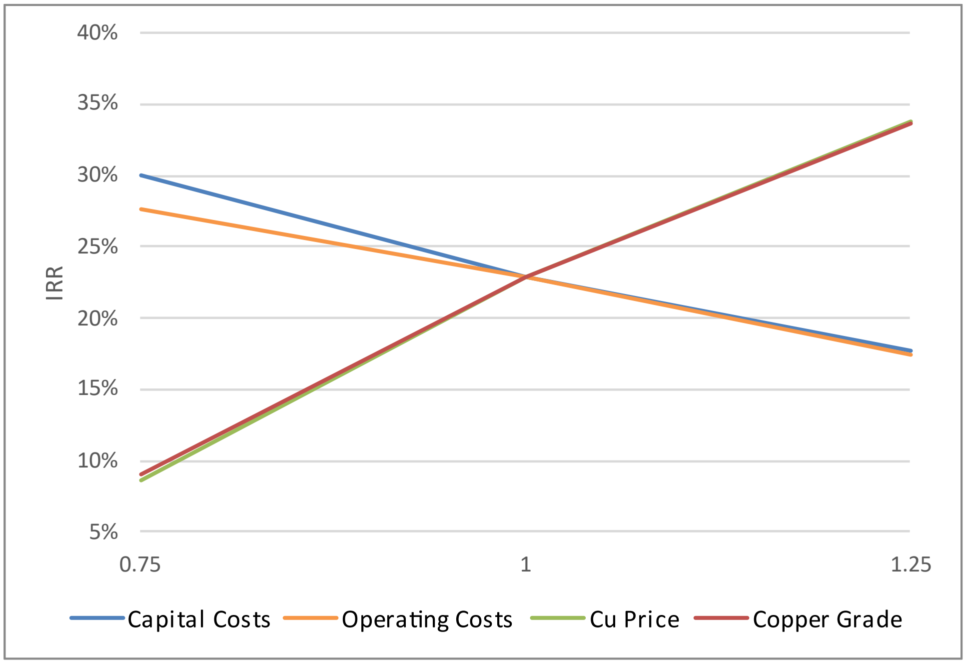


Table 22-5: Moonlight-Superior Copper Project NPV@7% and IRR at Specific Copper Prices

Parameter	Copper Price			
	$4.00	$4.50	$5.00	$5.50
NPV@7%	$935	$1,394	$1,847	$2,291
IRR	21%	27%	32%	37%

22.5 Conclusions of Economic Model

The project economics shown in the PEA are favorable, providing positive NPV values at varying copper prices, copper grades, capital costs, and operating costs. 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 under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this preliminary economic assessment will be realized.

23 ADJACENT PROPERTIES

GRE knows of no immediately adjacent properties which might materially affect the interpretation or evaluation of the mineralization, exploration targets, or economics of the Moonlight-Superior Project. Codelco's El Teniente Mine is located 21.8 miles (35 km) west.

The Walker Mine is located at the southern end of the Plumas Copper Belt approximately 12 miles (19 km) southeast of the Moonlight deposit. Numerous small mines and copper showings exist between the Walker Mine and the US Copper Mining land package. The Walker Mine is reported to have produced about 168 million lbs of copper, 180,000 oz of gold, and 3.6 million oz of silver from 5.3 million tons of ore between 1916 and 1941. Assuming 80% recovery, the feed grade would have been 1.98% copper, 0.85 opt silver, and 0.041 opt 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 OTHER RELEVANT DATA AND INFORMATION

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

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology and Mineralization

The Lights Creek Stock refers to the quartz monzonite, which is the ore host at the Moonlight, Superior, and Lamb's Ridge deposits. Most of the mineralization in the Lights Creek District appears to be related to the tourmaline-rich Lights Creek Stock or related dikes. While the Engels deposit lies just outside the stock, in the surrounding gabbroic-phase intrusive and metavolcanics, narrow dikes of granitic composition with abundant tourmaline have been noted.

The mineralization at Superior is hosted in the Lights Creek Quartz Monzonite and minor generally flat-lying diabase dikes. Both disseminated and associated copper mineralization with magnetite and tourmaline veinlets are seen at Superior.

The geology and mineralization at Lamb's Ridge appears to be most similar to Superior and was characterized by Placer-Amex geologists as a porphyry system. The wide-spaced (328- to 656-foot [100- to 200-meter]) drilling indicates disseminated copper mineralization similar to that found at Superior; however, no occurrences of the high-grade breccia-veins mined at Superior has been encountered in the drill holes.

The Engels deposit lies outside the Lights Creek Stock, immediately adjacent to its eastern margin in an area represented by both gabbroic-phase intrusive and roof-pendant metavolcanics. Engels is a structurally-controlled tabular shear-zone hosted deposit striking north-east and dipping steeply to the southeast. Mineralized widths range from 16 feet (5 meters) to over 65 feet (20 meters). The historically mined total strike length for the main ore shoot ranges from 328 feet (100 meters) to 820 feet (250 meters), while a narrower ore shoot to the south along strike was mined at lengths from 66 feet (20 meters) up to 197 feet (60 meters). The vertical extent mined is approximately 1,900 feet (580 meters). Mineralization in the Engels Mine area occurs in a 1,280-foot (390-meter) by 656-foot (200-meter) pipe like zone. Mineralization is associated with brecciated zones that exhibit features of both an intrusion breccia and a hydrothermal breccia.

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 LCS 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.

Considering all data, an IOCG deposit type has been recognized at Superior-Moonlight Project area.

Understanding Moonlight-Superior deposit setting, lithologies, mineralization, and the geological, structural, and structural controls on mineralization is sufficient to support the estimation of Mineral Resource and Mineral Reserves.

25.2 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

The exploration programs completed at the Project to date could reveal important aspects of geology, mineralization, and the style of deposit on the Property.

There are considerable drill holes at Superior (156 holes) and at Moonlight (261 holes). Drill holes at Engels are located at the vicinity of the historical mined area, and at Lamb's Ridge are very widely spaced with intervals of between 328 feet (100 meters) and 656 feet (200 meters), and are relatively shallow.

The historical drilling and sample collection methods and the recent drilling and sampling conducted by US Copper at the Project are acceptable for Mineral Resource and Mineral Reserve estimation.

The sample preparation, analysis, and security practices used by Century at the Project are acceptable and meet industry-standard practices and are sufficient to support Mineral Resource and Mineral Reserve estimation.

Sampling, sample preparation, assay analysis and QA/QC protocols are demonstrated to be consistent with current NI 43-101 standards only for the 2005, 2006, and 2008 drilling done at Engels and for the 2007, 2008, 2009, and 2010 at Moonlight. The QA/QC submission rates meet industry-accepted standards for only the 2005 and 2009 drilling programs at Moonlight and Engels, respectively. For all historic drilling the documentation was not available in the information supplied and the QA/QC protocols described are limited to extensive re-analysis of pulps and more limited re-analysis of split core.

US Copper initiated a dynamic QA/QC program for the Project and used it in all sample collection and analysis streams in 2021 and 2023. The QA/QC protocol became more comprehensive and detailed with progressive years. The QA/QC submission rates meet industry-accepted standards for the 2021 and 2023 drilling programs at Superior and Engels, respectively, and did not detect any material sample biases in the data reviewed that support the Mineral Resource and Mineral Reserve estimations.

Data verification concluded that the data collected from the Project adequately supports the geological interpretations and constituted a database of sufficient quality to support the use of the data in Mineral Resource and Mineral Reserve estimation.

The Author also recommends that less expensive, but important issues be addressed as well including:

- obtain high resolution topography and link to past drill collar coordinates
- conduct a more focused and organized SG test program using an independent laboratory
- re-submit the Engels pulps for sequential copper analysis to permit accurate assessment of the potential for heap leach SX-EW treatment.

25.3 Mineral Resource

The mineral resource estimate for the Moonlight-Superior Property was completed by Terre Lane (GRE), Society for Mining, Metallurgy and Exploration (SME)-Registered Member (RM). Ms. Lane is a Qualified Person as defined by NI 43-101 and is independent of US Copper. Ms. Lane estimated the mineral resource for the Project using an inverse distance squared interpolant. Geostatistics and mineral resource estimation were done with Leapfrog EDGE®. Model visualization was done with Leapfrog Geo® software, and the mineral resources were constrained with a Lerch-Grossman pit optimization. The metals of interest at the Project are copper, silver, and gold. The Mineral Resource estimate reported here was prepared in a manner consistent with the "CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines" adopted by CIM Council on November 29, 2019. The mineral resources are classified as Measured, Indicated, and Inferred in accordance with "CIM Definition Standards for Mineral Resources and Mineral Reserves," prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014. Classification of the resources reflects the relative confidence of the grade estimates. The effective date of the mineral resource estimate reported herein is December 16, 2024.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources are that part of the mineral resource for which quantity and grade or quality are estimated based on limited geologic evidence and sampling, which is sufficient to imply but not verify grade or quality continuity. Inferred mineral resources may not be converted to mineral reserves. It is reasonably expected, though not guaranteed, that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.

The Engels (Northeast Area) and Superior (South Area) deposits have existing underground workings. For each of these areas, blocks falling withing the existing workings were given Cu, Ag, and Au grades of 0, although tonnage was left in the model in the event any backfilling or collapse occurred.

Resources are reported within an optimized pit shell for each project area and meet the test of reasonable prospects for economic extraction. For sulfide material, a 10.45 NSR cutoff was chosen, and for oxide and transition material, a 0.16% Cu cutoff was chosen for reporting the mineral resource. The cutoff grades were calculated based on the parameters in Table 14-8.

Table 25-1 shows the Mineral Resource Estimate for the Project.

Table 25-1: Moonlight-Superior Project Mineral Resource Statement

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
Indicated										
Engels	Oxide	0.16	%	2.39	0.81	40,861	7.72	565,232	0.055	4,050
	Transition	0.16	%	7.52	0.50	79,941	4.75	1,093,948	0.042	10,194
	Sulfide	10.45	NSR/ton	8.32	0.46	76,750	5.83	1,415,487	0.056	13,585
Lambs Ridge	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	1.61	0.27	8,614	0.00	0	0.000	0
Moonlight	Oxide	0.16	%	1.35	0.36	10,244	3.77	154,364	0.128	5,460
	Transition	0.16	%	25.71	0.33	179,071	3.85	2,972,073	0.037	30,083
	Sulfide	10.45	NSR/ton	232.35	0.30	1,390,461	1.87	12,674,340	0.009	61,721
Copper Mountain	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	3.94	0.32	24,936	0.00	0	0.000	0
Superior	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	119.64	0.30	722,893	0.81	2,817,086	0.004	14,949
Total	Oxide	0.16	%	3.74	0.68	51,104	6.59	719,596	0.087	9,510
	Transition	0.16	%	33.23	0.39	259,012	4.20	4,066,021	0.042	40,277
	Sulfide	10.45	NSR/ton	365.86	0.30	2,223,654	1.58	16,906,913	0.008	90,255
Total				**402.83**	**0.31**	**2,533,771**	**1.85**	**21,692,531**	**0.012**	**140,042**

Deposit	Material	Cutoff Grade	Units	Mass (million tons)	Cu Grade (%)	Cu Content ('000 lb)	Ag Grade (ppm)	Ag Content (troy oz.)	Au Grade (ppm)	Au Content (troy oz.)
Inferred										
Engels	Oxide	0.16	%	0.15	1.18	3,740	11.91	55,046	0.010	48
	Transition	0.16	%	1.73	0.49	18,287	5.20	281,158	0.019	1,053
	Sulfide	10.45	NSR/ton	6.93	0.38	52,445	5.08	1,027,412	0.041	8,280
Lambs Ridge	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	3.46	0.30	20,954	0.00	0	0.000	0
Moonlight	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	30.82	0.28	175,635	0.09	81,857	0.000	35
Copper Mountain	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	3.90	0.27	21,320	0.00	0	0.000	0
Superior	Oxide	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Transition	0.16	%	0.00	0.00	0	0.00	0	0.000	0
	Sulfide	10.45	NSR/ton	17.60	0.29	101,817	0.01	2,681	0.000	23
Total	Oxide	0.16	%	0.15	1.25	3,740	12.64	55,046	0.011	48
	Transition	0.16	%	1.73	0.53	18,287	5.58	281,158	0.021	1,053
	Sulfide	10.45	NSR/ton	62.71	0.30	372,171	0.61	1,111,950	0.005	8,338
Total				**64.59**	**0.31**	**394,199**	**0.77**	**1,448,154**	**0.005**	**9,440**

Notes:

12. The effective date of the Mineral Resource is December 16, 2024.
13. The Qualified Person for the Mineral Resource Estimate is Terre Lane of GRE.
14. Mineral resources are reported at a 0.16% Cu cutoff for oxide and transition material and at a 10.45 NSR cutoff for sulfide material. The oxide and transition cutoff is calculated based on a long-term copper price of US$4.00/lb; assumed combined operating costs of US$7.50/ton (process and G&A); metallurgical recovery of 75% for copper. The sulfide cutoff is calculated as the breakeven NSR, which is equal to the combined process and G&A costs for the sulfide material.
15. Mineral resources are captured within an optimized pit shell and meet the test of reasonable prospects for economic extraction by open pit. The optimization used the same mining costs of US$2.35/ton mined and a 45º pit slope.
16. Rounding may result in apparent differences when summing tons, grade, and contained metal content.

25.4 Costs

The capital cost estimate has been prepared for the PEA under the assumption of mill processing of sulfide mineralized material at a design rate of 60,000 tpd, and heap leaching of oxide and transition material at a design rate of 10,000 tpd. Project costs were estimated using cost data from Infomine (2024) and experience of senior staff. The estimate assumes that the project will be operated by the owner with purchased equipment.

The capital costs are summarized in Table 25-2.

Table 25-2: Moonlight-Superior Copper Project Capital Cost Summary

Item	Total ($millions)
Mine Equipment	$218.17
Process	$611.54
Infrastructure	$117.35
G&A	$101.72
Working	$36.12
Sustaining	$9.27
Contingency	$218.83
Total	**$1,313.00**

Operating costs are summarized in Table 25-3.

Table 25-3: Moonlight-Superior Copper Project Operating Cost Summary

Item	Total Operating Cost ($millions)	Unit Operating Cost	Unit
Mining	$899	$1.51	$/ton mined
Processing – Sulfides	$1,520	$5.24	$/ton processed
Processing – Oxides and Transition	$215	$8.74	$/ton processed
Rehandle	$85	$0.75	$/ton processed
G&A	$108	$0.34	$/ton processed
Contingency	$283	$0.90	$/ton processed
Total	**$3,111**		

25.5 Economics

The key economic results for the base case are summarized in Table 25-4.

Table 25-4: Moonlight-Superior Copper Project Key Economic Results

Economic Measure	Value
After Tax NPV @ 7% (millions)	$1,075
IRR	23%
Initial Capital (millions)	$956
Payback Period (year)	5.3
All-in Sustaining Cost ($/lb Cu Produced)	$2.51

The project economics shown in the PEA are favorable, providing positive NPV values at varying copper prices, capital costs, and operating costs. 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 under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this preliminary economic assessment will be realized.

26 RECOMMENDATIONS AND OPPORTUNITIES

26.1 Recommendations

The QPs recommend the following items and budget (inclusive of contingency) to advance the Moonlight-Superior Copper project towards production (Table 26-1).

Table 26-1: Moonlight-Superior Copper Project Estimated Costs to Complete the Phase 1 Work Program

Exploration Cost Area	Total
Exploration Drilling	$5,000,000
Metallurgical Testing	$400,000
Permitting	$500,000
Total	**$5,900,000**

26.1.1 Drilling

Although drilling campaigns could reveal four main deposits (Superior, Lamb's Ridge, Engels, and Moonlight) across the project area, these deposits are open at almost all directions and at depth, thus they can be extended by more drilling along the structures and at depth. A few potential target areas, including Warren Creek, Blue Copper, Copper Mountain, and Osmeyer Prospect, also were explored through the previous surface sampling and limited drillings. These potential targets are capable of more exploration, including geological mapping, more surface sampling, geophysical land survey, and drilling.

Except existing deposits and potential targets, there are significant remaining new exploration targets within the Moonlight-Superior project area. Exploration targets include eleven areas (see section 9.3.2 of this technical report) with copper anomalies through the surface sampling and associated with northeast and northwest trending faults. These areas need to be explored in detail by geological mapping, surface sampling, followed by geophysical land survey, and then they need to be tested by drilling.

Since mineralization in the property is controlled by the structures, any future drilling should be designed to investigate the possible existence of other high-grade structurally-controlled orebodies to the northeast, northwest, and at depth.

26.1.2 Sampling and QA/QC

Sample collection, preparation, analysis, and security and in-house QA/QC program for the 2021 and 2023 RC and core drilling programs are in line with industry-standard methods for copper, gold, and silver deposits and should be retained for future drill campaigns. Standards, blanks, and duplicates including one standard, one duplicate, and one blank sample should be inserted every 20 interval samples, as is common within industry standards for the future drilling programs.

Geological and structural data collected during future drilling programs should continue to be used to increase geological understanding of the spatial correlation between mineralization and geological structures and inform the resource modeling process.

26.1.3 Metallurgical Testing

A comprehensive metallurgical test program is recommended to properly evaluate both the sulfide and oxide materials. Sulfide testing should include SAG mill and ball mill testing, flotation, thickening and filtration tests. For the heap leach the testing should include bottle roll leach tests in conjunction with column leach tests. Additionally, these tests should include both conventional acid leaching and bioleaching.

The variables that should be examined include grade, resource spatial distribution, and mineralogy.

26.1.4 Phase 2 Program

A Phase 2 program would be contingent upon positive results from the Phase 1 program, and the scope of the Phase 2 program are conditioned on the results of the Phase 1 program. For the purposes of conceptual level planning, it is assumed that a Phase 2 program would consist of a nominal $25M program that would include an expanded infill drill program to upgrade resources to reserves and engineering and economics studies that would result in a Pre-feasibility Study. Drilling would be at least planned at an appropriate spacing so that new mineralization could be largely included as reserves. Infill drilling over the area of the current resource estimate would decrease the hole spacing within the porphyry to an average of 300 feet and within the skarn to roughly 150 feet.

The QPs recommend further engineering evaluation of different projects sizes and the optimization of mine plans.

The QPs recommend the evaluation and incorporation of existing and/or future technologies to improve sustainability and reduce environmental impacts of the Project

26.1.5 Environmental Recommendations

Baseline studies are recommended to support the preparation of permitting documents. Baseline studies should include surface water, groundwater, noise, air quality, fauna and flora, archeology, human component, paleontology and landscape.

Development of other preliminary engineering studies that will support early preparation of an EIS are recommended. The following studies should be conducted to support infrastructure designs:

- Seismic study
- Hydrology and hydrogeology
- Geomorphology and geological risk
- Geotechnical studies
- Condemnation drilling

GRE recommends additional evaluation of the potential for PAG, ML, and groundwater mobilization of contaminants.

26.1.6 General

The QPs recommend further engineering evaluation of different project sizes and the optimization of mine plans.

The QPs recommend the evaluation and incorporation of existing and/or future technologies to improve sustainability and reduce environmental impacts of the Project.

26.2 Opportunities

The QPs believe there are opportunities to improve sustainability using technologies such as electric mining equipment, regenerative conveyor systems, and alternative leaching technologies.

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CERTIFICATE OF QUALIFIED PERSON

I, Hamid Samari, PhD, of 17301 W Colfax Ave, Suite 400, Golden, Colorado, 80401, the co-author of the report entitled "Preliminary Economic Assessment NI 43-101 Technical Report, Moonlight-Superior Copper Project, Plumas County, CA, USA" with an effective date of December 16, 2024, and an issue date of January 6, 2025 (the "Technical Report"), DO HEREBY CERTIFY THAT:

1. I am a MMSA Qualified Professional in Geology, #01519QP.
2. I hold a degree of PhD of Science (2000) in geology (Tectonics-structural geology) from Tehran Azad University (Sciences & Research Branch).
3. I have practiced my profession since 1994, holding positions ranging from expert geologist to senior geologist and project manager in geology and mining exploration.
4. I have practiced in the areas of geology, mining, and civil industry for over 25 years. I have worked for Azad University, Mahallat branch, as an assistant professor and head of the geology department for 19 years, for Tamavan consulting engineers as a senior geologist for 12 years, and for Global Resource Engineering for nearly eight years. I have worked on geologic reports and resource statements for gold, silver, copper, and lithium deposits in Iran, the United States, and Latin America. This includes porphyry and vein-type copper projects in Iran, epithermal gold and silver deposits in Peru, gold deposits in Nevada and Utah, Lithium in Nevada, and mixed precious metals deposits elsewhere in the Western Hemisphere. I have also worked on several gold and silver epithermal deposits in Nevada and Utah.
5. I have been involved with many studies, including scoping studies, pre-feasibility studies, and feasibility studies.
6. I have read the definition of "Qualified Person" in National Instrument 43-101 and certify that, by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101), and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
7. I visited the Project site from 7 to 8 August 2024 for two days.
8. I am responsible for Sections 11.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 4.1, 4.2, 4.3, 5, 6, 7, 8, 9, 10, 11, 12.1, 12.2, 12.3, 12.4, 12.5, 25.1, 25.2, 26.1.1, 26.1.2, and 27 of the Technical Report.
9. I am independent of US Copper Corp. as described in section 1.5 by National Instrument 43-101.
10. I have been involved with the Moonlight-Superior Project while preparing the 2024 NI 43-101 Technical Report.
11. I have read National Instrument 43-101 and Form 43-101F1. The geological section has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
12. As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Hamid Samari, PhD

"Hamid Samari"

Principal Geologist

Global Resource Engineering, Ltd.

Golden, Colorado

Date of Signing: January 6, 2025

CERTIFICATE OF QUALIFIED PERSON

I, Terre A. Lane, of 17301 W Colfax Ave, Suite 400, Golden, Colorado, 80401, the co-author of the report entitled "Preliminary Economic Assessment NI 43-101 Technical Report, Moonlight-Superior Copper Project, Plumas County, CA, USA" with an effective date of December 16, 2024, and an issue date of January 6, 2025 (the "Technical Report"), DO HEREBY CERTIFY THAT:

1. I am a MMSA Qualified Professional in Ore Reserves and Mining, #01407QP, and I am a Registered Member of the Society for Mining, Metallurgy, and Exploration
2. I hold a degree of Bachelor of Science (1982) in Mining Engineering from Michigan Technological University.
3. I have practiced my profession since 1982 in capacities from mining engineer to senior management positions for engineering, mine development, exploration, and mining companies. My relevant experience for the purpose of this PEA is as the resource estimator, mine planner, and economic modeler with 25 or more years of experience in each area.
4. I have taken classes in geology, structural geology, mineralogy, Mineral Resource estimation in university, and have taken several short courses in geostatistics subsequently.
5. I have worked in geology, managed geologic teams, created lithological and structural models, and I have been involved in or conducted the estimation of resources for several hundred projects at locations in North America, Central America, South America, Africa, Australian/New Zealand, India, China, Russia and Europe using nearly all estimation techniques.
6. I have estimated resources and created mine plans for several copper/molybdenum porphyry/skarn deposits including Vizcachitas, Henderson, Climax, Mount Emmons, North Ore Shoot (Bingham Canyon), and others and have overseen the resource estimate and mine design of many other similar porphyry deposits.
7. I have created or overseen the development of mine plans for several hundred open pit and underground projects and operating mines.
8. I have been involved in or managed several hundred studies including scoping studies, prefeasibility studies, and feasibility studies.
9. I have been involved with the mine development, construction, startup, and operation of several mines.
10. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
11. I most recently visited the Moonlight-Superior property on February 10, 2022 for one day and have reviewed previous geological data, geochemical results, metallurgical and technical reports on the subject property. I have previously visited the site a number of times in the last half of the 1990s while I was employed as VP Engineering for General Minerals Corp.
12. I am responsible for Sections 1.1, 1.10, 1.11, 1.13, 1.14, 1.15, 2, 3, 12.7, 14, 15, 16, 18, 19, 20, 21, 22, 23, 24, 25.3, 25.4, 25.5, 26.1.4, 26.1.6, and 26.2 of the Technical Report.
13. I am independent of US Copper as described in section 1.5 by National Instrument 43-101.

14. I previously worked on the Moonlight-Superior Project for South American Silver.
15. I have read National Instrument 43-101 and Form 43-101F1. The Technical Report has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
16. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Terre A. Lane

"Terre A. Lane"

Principal Mining Engineer

Global Resource Engineering, Ltd.

Denver, Colorado

Date of Signing: January 6, 2025

CERTIFICATE OF QUALIFIED PERSON

I, J. Todd Harvey, of 17301 W Colfax Ave, Suite 400, Golden, Colorado, 80401, the co-author of the report entitled “Preliminary Economic Assessment NI 43-101 Technical Report, Moonlight-Superior Copper Project, Plumas County, CA, USA” with an effective date of December 16, 2024, and an issue date of January 6, 2025 (the “Technical Report”), DO HEREBY CERTIFY THAT:

1. I am currently employed as Principal Process and Mining Engineer by Global Resource Engineering, Ltd.
2. I graduated with Ph.D. in Mining Engineering from the Queen’s University at Kingston in 1994, a Master's degree in Mining Engineering from the Queen’s University at Kingston in 1990 and a Bachelor’s degree in Mining Engineering in 1988 all with a specialization in mineral processing. I also hold a degree in Metallurgical Engineering and Computer Science from Ryerson University in Toronto Canada graduating in 1986 as well as an MBA from the University of New Brunswick in Saint John Canada graduating in 2001.
3. I have worked as a Process Engineer for over 35 years since my graduation from university. My relevant experience includes process due diligence/competent persons evaluations of developmental phase and operational phase mines throughout the world, including mines in the USA, Canada, Kazakhstan, Brazil, Mexico, and Africa to name a few. I have a wide range of experience in multiple mineral fields including precious metal processing and base metals such as copper, lead, and zinc.
4. I am a Registered Member (No. 04144120) of the Society for Mining, Metallurgy & Exploration Inc. (SME). I am also a member of the Association for Mineral Exploration (AME), Minerals Engineering Journal Review Board, and the Journal of Hydrometallurgy Review Board.
5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of National Instrument 43-101.
6. I have not visited the Moonlight-Superior Property.
7. I am responsible for Sections 1.9, 1.12, 12.6, 13, 17, and 26.1.3.
8. I am independent of US Copper as described in section 1.5 by National Instrument 43-101.
9. I have not previously worked on the Moonlight-Superior Project.
10. I have read National Instrument 43-101 and Form 43-101F1. The Technical Report has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

J. Todd Harvey

“J. Todd Harvey”

Metallurgist

Global Resource Engineering, Ltd.

Denver, Colorado

Date of Signing: January 6, 2025

CERTIFICATE OF QUALIFIED PERSON

I, J. Larry Breckenridge, P.E., of 17301 W Colfax Ave, Suite 400, Golden, Colorado, 80401, the co-author of the report entitled "Preliminary Economic Assessment NI 43-101 Technical Report, Moonlight-Superior Copper Project, Plumas County, CA, USA" with an effective date of December 16, 2024, and an issue date of January 6, 2025 (the "Technical Report"), DO HEREBY CERTIFY THAT:

1. I am currently employed as principal environmental engineer by Global Resource Engineering, Ltd.
1. I am a graduate of Dartmouth College with a degree in Engineering Modified with Environmental Science (BA) and from the Colorado School of Mines with a Masters' degree in Environmental Engineering.
2. I am a Qualified Person under NI 43-101 because I am a registered Environmental Engineer in the State of Colorado, USA, No. 38048.
3. I have practiced area of water management, geochemistry, and environmental management -- exclusively for precious and base metals projects for over 25 years. I have worked with Global Resource Engineering in my same role for the last 12 years. I have participated in the permitting process for numerous mines in the United States and in Latin America. I have evaluated geochemical risk and permitting requirements for precious metals projects and also performed water availability studies.
4. I have read the definition of "qualified person" set out in National instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Moonlight-Superior Property.
6. I am responsible for Sections 4.4, 4.5, 20, and 26.1.5.
7. I am independent of US Copper as described in section 1.5 by National Instrument 43-101.
8. I have not previously worked on the Moonlight-Superior Project.
9. I have read National Instrument 43-101 and Form 43-101F1 and confirm the sections of the Technical Report for which I am responsible (as listed above) have been prepared in compliance with that instrument and form.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

Mr. J. Larry Breckenrdige

"J Larry Breckenridge"

Principal Environmental Engineer
Global Resource Engineering, Ltd.
Denver, Colorado
Date of Signing: January 6, 2025