Preliminary Economic Assessment & NI 43-101 Technical Report for the CuMo Project, USA

Prepared for

American CuMo Mining Corp.

Prepared by

srk consulting

SRK Consulting (Canada) Inc.
2CM027.001
May 2020

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Important Notice

Acronyms and Abbreviations

Distance	
μm	micron (micrometre)
mm	millimetre
cm	centimetre
m	metre
km	km
in	inch
ft	foot
Area	
m^2	square metre
km^2	square km
ac	acre
Ha	hectare
Volume	
L	litre
m^3	cubic metre
ft^3	cubic foot
bcm	bank cubic metres
Mbcm	million bcm
bcy	bank cubic yards
Mbcy	million bcy
Mass	
kg	kilogram
g	gram
g/t	g/ metric tonne
t or st	short ton
kst	thousand short tons
Mst	million short tons
Bst	billion short tons
lb	pounds
mmlbs	millions of lbs
oz	troy ounce
wmt	wet metric tonne
dmt	dry metric tonne
Pressure	
psi	pounds per square inch
Pa	pascal
kPa	kilopascal
MPa	megapascal
Elements and Compounds	
Mo	molybdenum
MoS_2	molybdenite
Cu	copper
Au	gold
Ag	silver
S	sulfur
CN	cyanide
NaCN	sodium cyanide
Other	
°F	degrees Fahrenheit
°C	degrees Celsius
cfm	cubic feet per minute
elev	elevation
m AMSL	metres elev. above mean sea level
hp	horsepower
hr	hour
s	second (unit of time)
kW	kilowatt
kWh	kilowatt hour
M	Million or mega
mph	miles per hour
ppb	parts per billion
ppm	parts per million
s.g. or SG	specific gravity
V	volt
W	watt
$k	thousand US dollars
$M	million US dollars
$Bn	billion US dollars
tph or stph	short tons per hour
tpd or stpd	short tons per day
mtpa or mstpa	million short tons per annum
Ø	diameter
Acronyms	
SRK	SRK Consulting (Canada) Inc.
CIM	Canadian Institute of Mining
NI 43-101	National Instrument 43-101
ABA	Acid- base accounting
LOM	life of mine
AP	Acid potential
NP	Neutralization potential
ML/ARD	Metal leaching/ acid rock drainage
PAG	Potentially acid generating
non-PAG	Non-potentially acid generating
RC	reverse circulation
IP	induced polarization
COG	cut-off grade
NSR	net smelter return
NPV	net present value
Conversion Factors	
1 ton	2,000 lb
1 tonne	2,204.62 lb
1 troy oz	31.10348 g

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1 Executive Summary

1.1 Introduction

American CuMo Mining Corp. (CuMoCo; TSX-V:MLY) retained SRK Consulting (Canada) Inc. to conduct a preliminary economic assessment (PEA) on the CuMo project in Boise County, Idaho, USA, and to present its outcomes in this National Instrument (NI) 43-101 independent technical report. Other contributors to the PEA and report are Sacré-Davey Engineering and Giroux Consultants Ltd.

SRK based the PEA on the resource model generated by qualified person (QP), Gary Giroux, and reported on in 2015 (Giroux et al, 2015). SRK and Sacré-Davey both based certain aspects of their scope (infrastructure and mineral processing respectively) on an earlier PEA conducted by Ausenco Canada (Ausenco) on the CuMo project (Braun et al, 2009). Original work by SRK includes the evaluation of bulk sorting, generation of mine designs, schedules and costing, selection and evaluation of a tailings management strategy, and updating the economic analysis. Original work by Sacré-Davey included evaluation of particle sorting.

Site visits for the purposes of personal inspections of the CuMo property have been undertaken by Mr. Gary Giroux, resource QP (June 2015); Mr. Bob McCarthy, SRK mining QP (October 2018); Mr. Andy Thomas, SRK pit geotechnical QP (October 2018), and Mr. Calvin Boese, SRK waste management QP (October 2018).

Note: Throughout this report, all currency is 2019, non-escalated United States dollars and all units are imperial, unless otherwise specifically noted.

1.2 Property Description, History, and Ownership

The CuMo deposit is a molybdenum-copper deposit situated 37 miles, equivalent to 60 kilometers (km), northeast of Boise, Idaho, USA. The project is situated in the southern section of the Boise Mountains which are characterized by north-northwest trending mountain ranges separated by alluvial filled valleys. Topographic elevations on the CuMo claims range from 5,400 feet (1,700 meters) to 7,100 ft (2,400 m) above sea level.

Situated in a historic lode gold camp with a recorded production of 2.8 million ounces, molybdenite (MoS_2) mineralization was not discovered in this area until 1963 by Amax Exploration (Amax). After conducting surface sampling in 1964, Amax relinquished rights to the property. It was subsequently explored by Curwood Mining Company, Midwest Oil Corporation (later Amoco Minerals Company), Amax (a second time), and then Climax Molybdenum Company (a subsidiary of Amax Inc.). The Historic Drilling was done between 1969 and 1982 for a total of 10,981 m (36,026 ft) in 23 diamond drill holes and three reverse circulation holes. Note: Reverse circulation holes are not used in the resource calculation.

The property was re-staked in 1998 by CuMo Molybdenum Mining Inc. and optioned to Mosquito Consolidated Gold Mines Ltd., (now CuMoCo) in 2004.

Presently, the CuMo project is held by a wholly owned USA subsidiary of CuMoCo, Idaho CuMo Corporation (ICMC).

1.3 Exploration

After CuMoCo had optioned the property in 2004, Kobex Resources Ltd. (Kobex) optioned it from CuMoCo in 2005 and commenced drilling in 2006. Kobex drilled one complete hole and 50% of a second hole 1,087 m (3,565 ft). In late 2006, CuMoCo resumed control and completed the 2006 to 2011 exploration drilling programs, including the incomplete hole by Kobex. CuMoCo completed 20,187 m (66,230 ft) of drilling in 32 diamond drill holes in that program. During 2012, CuMoCo drilled nine additional holes totaling 4,713 m (15,464 ft), aimed at improving the resource categorization and gaining a better understanding of the extent of the deposit.

1.4 Geology and Mineralization

The CuMo deposit is located at the southwestern end of the Idaho-Montana Porphyry Belt. Igneous complexes in this belt are interpreted to be related to an Eocene, intra-arc rift, and are characterized by alkalic rocks in the northeast, mixed alkalic and calc-alkalic rocks in the middle, and calc-alkaline rocks in the southwest. The CuMo deposit is typical of large, dispersed, low grade molybdenum ± copper porphyry deposits that are associated with hybrid magmas typified by fluorine-poor, differentiated monzogranite igneous complexes. Due to their large size, the total contained economic molybdenum in these types of deposits can be equivalent to or exceed that of high-grade molybdenum deposits.

CuMoCo's work has resulted in the interpretation and modelling of three distinct mineralized zones within the deposit. These zones were previously interpreted by Amax as distinct shells that were produced by separate intrusions. Re-interpretation of down-hole histograms for copper (Cu), silver (Ag) and molybdenite (MoS_2[1]) suggests the mineralized zones are part of a single, large, concentrically zoned system with an upper copper-silver zone (named Cu-Ag Zone), underlain by a transitional copper-molybdenum zone (named Cu-Mo Zone), in turn underlain by a lower molybdenum-rich zone (named Mo Zone). Three-dimensional modeling of the above zonation indicates the current area being drilled is located on the north side of a large system extending 4.5 km (15,000 ft) in diameter, of which 1.5 km (3,000 ft) has been drilled.

1.5 Mineral Resource Estimate

A resource estimate update was completed (November 2015), based on a total of 65 diamond drill holes totaling 36,166 m (118,654 ft). Note that the three reverse circulation holes were not used in the resource estimate. Nine of the 65 diamond drill holes were completed in 2012. As no additional drilling has been completed since the 2015 resource was estimated, it is considered current.

A geological model separating the CuMo Deposit into four mineralize zones with an oxidized layer on top was developed by CuMoCo geologists. In addition, major fault blocks were identified both by assay data and by marker beds. Assays were tagged as one of four mineralized zones: a near surface Cu-Ag Zone, a deeper Cu-Mo Zone and a still deeper Mo Zone and an underlying potassic-silica zone (MSI). Statistical analysis of each variable in each zone led to the capping of assays based on the grade distribution within each zone. Uniform down-hole 50 feet (ft) composites were produced for each zone. For variography, the major post mineral fault blocks were rotated back to their original position using marker beds. Semi-variograms were produced for each variable within each zone based on the

[1] The convention for the CuMo project has been to measure percent elemental molybdenum (%Mo) in assays and to calculate %MoS_2 by multiplying %Mo by 1.6681. Both %Mo and %MoS_2 are stored in the project's database, and the latter, %MoS_2, is used in resource estimates and mine planning.

samples' original pre-fault locations. A block model with block dimensions of 50 ft was superimposed on the mineralized zones. Grade was interpolated into blocks by ordinary kriging. A tonnage factor was determined for each zone based on multiple specific gravity determinations. Individual blocks were classified as measured, indicated or inferred resource based on their location relative to drill-hole composites.

Note: Mineral resources that are not mineral reserves do not have demonstrated economic viability.

To take into account the four primary potentially economic minerals estimated, a form of metal equivalent or recoverable value (RCV) was calculated for each block based on reasonable commodity prices and estimated recoveries in each of five zones; the oxide zone (a combination of altered Cu-Ag and Cu-Mo Zones), Cu-Ag Zone, Cu-Mo Zone, Mo Zone and MSI Zone. The 2015 resource estimate is summarized below for RCV cut-offs.

The metal prices used for resource estimation are provided in Table 1-1.

Table 1-1: Metal prices for resource estimation

Metal	Price
Copper (Cu) (per lb)	\$3.00
Molybdenum oxide (MoO3) (per lb)	\$10.00
Molybdenum Metal (Mo) (per lb)	\$15.00
Silver (Ag) (per oz)	\$12.50

The metal recoveries used were a function of mineralized zones as follows in Table 1-2.

Table 1-2: CuMo metal recoveries by zone

Zone	Cu Recovery (%)	Mo Recovery (%)	Ag Recovery (%)
OX	60	80	65
Cu-Ag	68	86	75
Cu-Mo	85	92	78
Mo	72	95	55
MSI	72	95	55

In 2012, Snowden Mining Consultants (Snowden) used Geovia's Whittle^{TM} pit optimizer to determine a constraining open pit shell for the CuMo deposit. Optimization parameters were from Thompson Creek mine (a comparable open pit molybdenum project located in Idaho). The optimization parameters included mill feed, mining and processing costs of \$7.52 per processed ton, overall pit slope angles of 45°, metallurgical recoveries as shown above and appropriate dilution and offsite costs and royalties. The commodity prices used in 2012 by Snowden for restraining the resource were Mo at $25/lb, Cu at $3/lb, Ag at $20/oz and W at $10/lb. This pit constraint is still valid.

Since the infill drill holes completed in 2012 were all within the conceptual pit, this resource update uses the Snowden 2012 optimum pit shell to constrain the estimate.

In the mineral resource estimate tables below (Table 1-3, Table 1-4, Table 1-5, and Table 1-6), the base case $5.00/t RCV cut-off is highlighted and is selected based on operating costs and the results of grade improvement using a mineral sorting process. The $5.00 cut-off is suggested to separate waste from material that is fed into the sorters. From the sorters, only mill feed above an economic cut-off would be sent for immediate processing.

It should be noted that since the convention for the CuMo project has been to work with %MoS_2, as calculated from measured %Mo, the %MoS_2 values in the resource estimate tables are 1.6681 times greater than %Mo.

In August 2018, an estimate for rhenium (Re) and sulfur (S) associated with the MoS_2 was completed using linear regression of MoS_2 vs. Re and MoS_2 vs S to show the average grades of Re and S that would be contained with MoS_2 within each block. The Re and S were not used to determine the RCV value of resources shown in Table 1-3, Table 1-4, Table 1-5, and Table 1-6 below.

Note: Regression analysis is not industry standard practice in calculating overall resources. However, the fact that rhenium is entirely and almost all the sulfur are contained within the material containing MoS_2, (note; a minor amount of sulfur is contained in pyrite) which has been estimated by kriging, means that regression is a valid method of obtaining a reasonable estimate of the rhenium and sulfur contents at the level of precision of this study. Due to the large number of samples involved in the regression analysis, the confidence of this particular regression estimate is comparable to that obtained by the method of ordinary kriging.

Table 1-3: Measured resource within pit shell

Cut-off RCV ($)	Grade > RCV Cut-off					Contained Metal				
	Quantity (Mt)	MoS_2 (%)	Cu (%)	Ag (ppm)	RCV ($)	Re (ppm)	S (%)	Mo (mmlbs)	Cu (mmlbs)	Ag (Moz)
2.5	308.4	0.079	0.074	2.09	17.32	0.029	0.233	292.1	456.5	18.8
5.0	297.2	0.081	0.076	**2.09**	**17.83**	**0.03**	**0.229**	**288.6**	**451.7**	**18.1**
7.5	282	0.085	0.076	2.06	18.48	0.031	0.223	287.4	428.7	16.9
12.5	227.9	0.097	0.075	2	20.50	0.036	0.217	265	341.8	13.3
15.0	195.4	0.105	0.072	1.9	21.71	0.039	0.212	246	281.3	10.8
17.5	159.7	0.115	0.067	1.8	23.04	0.043	0.207	220.1	213.9	8.4
20.0	122.9	0.125	0.063	1.7	24.50	0.047	0.202	184.1	154.8	6.1

Source: Giroux et al, 2015, modified 2019

Table 1-4: Indicated resources

Cut-off RCV ($)	Grade > RCV Cut-off					Contained Metal				
	Quantity (Mt)	MoS_2 (%)	Cu (%)	Ag (ppm)	RCV ($)	Re (ppm)	S (%)	Mo (mmlbs)	Cu (mmlbs)	Ag (Moz)
2.5	2216.1	0.049	0.079	2.48	12.32	0.018	0.277	1301.9	3501.4	160.3
5.0	**1972.3**	**0.053**	**0.085**	**2.57**	**13.40**	**0.019**	**0.269**	**1253.3**	**3352.9**	**147.8**
7.5	1708.3	0.059	0.088	2.59	14.55	0.021	0.258	1208.4	3006.5	129
12.5	1050.6	0.076	0.09	2.55	17.67	0.027	0.235	957.4	1891.1	78.1
15.0	798.5	0.083	0.09	2.56	19.06	0.03	0.231	794.6	1437.2	59.6
17.5	541.6	0.093	0.088	2.49	20.60	0.034	0.226	603.9	953.2	39.3
20.0	301.3	0.106	0.082	2.36	22.49	0.039	0.219	383	494.2	20.7

Source: Giroux et al, 2015, modified 2019

Table 1-5: Measured and indicated resources

Cut-off RCV ($)	Grade > RCV Cut-off					Contained Metal				
	Quantity (Mt)	MoS_2 (%)	Cu (%)	Ag (ppm)	RCV ($)	Re (ppm)	S (%)	Mo (mmlbs)	Cu (mmlbs)	Ag (Moz)
2.5	2524.6	0.053	0.079	2.43	12.93	0.019	0.272	1604.3	3988.9	178.9
5.0	**2269.6**	**0.057**	**0.084**	**2.5**	**13.98**	**0.021**	**0.264**	**1551.1**	**3812.9**	**165.5**
7.5	1990.4	0.063	0.086	2.51	15.10	0.022	0.253	1503.5	3423.5	145.7
12.5	1278.6	0.079	0.087	2.46	18.17	0.029	0.232	1211.1	2224.8	91.7
15.0	993.9	0.088	0.087	2.43	19.58	0.032	0.227	1048.7	1729.5	70.4
17.5	701.4	0.098	0.083	2.33	21.16	0.036	0.221	824.1	1164.2	47.7
20.0	424.3	0.112	0.077	2.17	23.07	0.041	0.214	569.8	653.4	26.9

Source: Giroux et al, 2015, modified 2019

Table 1-6: Inferred resources (molybdenum, copper, silver, rhenium and sulfur)

Cut-off RCV ($)	Grade > RCV Cut-off					Contained Metal				
	Quantity (Mt)	MoS_2 (%)	Cu (%)	Ag (ppm)	RCV ($)	Re (ppm)	S (%)	Mo (mmlbs)	Cu (mmlbs)	Ag (Moz)
2.5	3373.6	0.04	0.057	1.93	9.55	0.014	0.304	1617.9	3845.9	189.9
5.0	**2556.6**	**0.048**	**0.067**	**2.13**	**11.48**	**0.017**	**0.282**	**1471.4**	**3425.9**	**158.8**
7.5	1996	0.056	0.07	2.23	13.07	0.02	0.261	1340.1	2794.4	129.8
12.5	996.4	0.078	0.064	1.98	16.74	0.028	0.231	931.8	1275.4	57.5
15.0	637	0.086	0.074	2.16	18.63	0.03	0.244	656.8	942.7	40.1
17.5	384.8	0.094	0.084	2.34	20.49	0.032	0.259	433.7	646.4	26.3
20.0	190.2	0.109	0.078	2.37	22.80	0.037	0.262	248.6	296.8	13.1

Source: Giroux et al, 2015, modified 2019

Note on Inferred Mineral Resources

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

1.6 Project Development and Operations

CuMo is to be developed as an open pit mining operation, mining waste and providing feed to a primary crusher which then supplies crushed material to a mineral sorting plant. The sorting plant consists of both bulk and particle sorting and produces feed for a mill and flotation plant. The project plan includes an off-site roaster to convert molybdenum concentrate (as MoS_2) to saleable MoO_3. Major by-products are copper and silver, and minor by-products are rhenium and sulfuric acid. Potential for the production of tungsten as a minor by-product may also exist.

The proposed operation is for a 150,000 short tons per day (stpd) feed rate to the mill. This requires a sort feed rate averaging about 200,000 stpd, with a maximum of about 265,000 stpd. Mining rates to achieve this feed average about 400,00 stpd, reaching a maximum mining rate of 500,000 stpd.

The overall process is for material to be mined at the mining rate, and a grade control cut-off is applied to that material to determine what material is sent to the sort plant (sorter feed); the remaining material is sent to waste dumps. The sort plant consists of an initial three-stage bulk sort process, where for each stage, a pair of cut-offs is applied to produce mill feed, waste, and middlings streams. The middlings stream for each sort stage becomes the feed for the next sort stage. After the third sort, the middlings are sent to a stockpile which is the feed source for a particle sorting process. The product from the particle sort process is combined with the mill feed product of the bulk sort process in a coarse material stockpile for feed to the mill.

The mill is a conventional semi-autogenous grinding circuit and flotation circuit creating an interim copper-molybdenum concentrate which is then further processed in a molybdenum flotation circuit to separate copper and molybdenum concentrates. Molybdenum concentrate is transported to the project roaster for production of MoO_3. Copper concentrate is shipped to market.

The tailings storage facility (TSF) will be located at the headwaters of the Clear Creek watershed, in a natural basin formed by the surrounding ridgeline. The TSF will have capacity to store the 1,582M tons (~900M m^3) of tailings produced over the 28 year mine life, with an ultimate crest elevation of 6,950 ft. Tailings containment will be provided by the natural topography on the valley sides and an engineered dam that will be buttressed by the Clear Creek waste rock facility (WRF) constructed immediately downstream of the TSF. A starter dam will be constructed to elevation 6300 ft to facilitate early mine production, followed by an additional five raises spread out over the life of the mine.

1.7 Social and Environmental

At this time, no issues were identified that would materially impact the ability to eventually extract mineral resources at the project.

The proposed mine will be located on public land administered by the United States Forest Service (USFS) and private land owned and controlled by ICMC. The permitting path will involve multiple state and federal agencies. Permits likely to be required for the project are presented in Table 20-1. An environmental impact statement will be required at the level of NEPA analysis for mine development, operations, and closure. Reclamation bonds will be required by both federal and state agencies. The reclamation liability for the proposed mine will have to be determined based on third-party costs, and the bond amount will have to be posted using an approved financial instrument

ICMC has initiated consultation with various stakeholders namely: government officials at all levels and local communities in regard to the potential social and community impacts or improvements that may occur as the project progresses. All groups are provided regular updates as the project is proceeding. Local communities and officials have come out in strong support of the project and are actively working with the project on both the Grimes Creek project and future planning (Hilscher et al, 2018).

The mine will be located in an area used for weekend summer dispersed recreation and fall big-game hunting and is well-known in the Boise area. Organized environmental groups such as the Idaho Conservation League and Sierra Club are keeping their constituents informed so as to coordinate opposition to the project. As such, well-funded, organized opposition to mining activities should be anticipated.

At the current time the United States Forest Service (USFS) is working on a Supplemental Red Line Environmental Assessment that will allow the Company to proceed to the next round of drilling and road access construction on the property. The authorization is expected in 2020, and no surface disturbing activities can proceed on the property until the authorization is received. This is the only ongoing permitting activity.

1.8 Project Costs

Operating costs were derived from the mining operation based on comparison to similar size operations and the authors' experience. Modifications were made to account for varying haul profiles that are expected during the mine life. General and administrative costs were similarly based on similar size operations and the authors' experience.

Processing costs were based on prior work by Ausenco (Ausenco, 2009) and compare well with more recent studies and so continue to apply.

Capital costs for mining were based on evaluation of mining equipment fleet requirements and application of unit equipment prices used in recent studies. Pre-production mining (pre-stripping) was also capitalized for the purpose of economic analysis. Capital costs for the sorting plant were estimated based on its material handling focus (conveyors).

The capital costs for infrastructure and mineral processing from the earlier Ausenco work (Ausenco, 2009) were reviewed and were deemed reasonable in comparison to more recent studies of large porphyry projects though some cost escalation was applied.

1.9 Project Economics

1.9.1 Cautionary Statements

Certainty of Preliminary Economic Assessment

The preliminary economic assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Mineral Resources are Not Reserves

Mineral resources are not mineral reserves and do not have demonstrated economic viability.

1.9.2 Economic Summary

The project as presented and under the current assumptions has the potential to be economic. The after-tax NPV is positive and is robust across a range of sensitivities with respect to capital costs, operating costs and revenue (price). A summary of the project economics is shown in Table 1-8.

Table 1-7: Summary of project economics

Project Metric	Units	Value
Pre-Tax NPV @ 5%	$M	2,470
Pre-Tax NPV @ 8%	$M	800
Pre-Tax NPV @ 10%	$M	113
Pre-Tax IRR	%	10%
After-Tax NPV @ 5%	$M	1,709
After-Tax NPV @ 8%	**$M**	**356**
After-Tax NPV @ 10%	$M	-205
After-Tax IRR	%	9%
Undiscounted After-Tax Cash Flow (LOM)	$M	7,032
Payback Period from Start of Processing	years	8.0
Initial Capital Expenditure	$M	3,071
LOM Sustaining Capital Expenditure	$M	972
Closure	$M	150
LOM C-1 Cash Costs After By-product Credits	$/lb Mo	4.67
Nominal Flotation Process Capacity	stpd	150,000
Mine Life (years @ > 90% of full production)	years	28
LOM Flotation Mill Feed	kst	1,582,526
LOM Grades		
Molybdenite (MoS_2)	%	0.074%
Molybdenum (elemental Mo)	%	0.044%
Copper	%	0.105%
Silver	grams per tonne	3.00
LOM Waste Volume	kst	2,425,101
LOM Strip Ratio (Waste:Sort Feed)	ratio	1.11
Mass Pull to Mill from Sort Feed	%	72%
LOM Strip Ratio (Waste:Mill Feed)	ratio	1.53
First Five Years Average Annual Metal Production		
Molybdenum (Mo)	klbs/yr	34,976
Copper	klbs/yr	93,394
Silver	kounces/yr	3,940
LOM Average Annual Metal Production		
Molybdenum (Mo)	klbs/yr	43,072
Copper	klbs/yr	84,229
Silver	kounces/yr	3,575
LOM Average Mill Process Recovery		
Molybdenum (Mo)	% contained metal	91.87%
Copper	% contained metal	76.33%
Silver	% contained metal	70.42%

1.10 Project Risks

1.10.1 Mineral Resource

The mineral resource is supported by exploration results, test-work and modelling. As with any mineral resource estimate there is uncertainty inherent in the estimation process. There is a risk that the grades and metallurgical recoveries may be lower than currently modelled. There is also a risk that the interpretation of the results is inaccurate and that less mineralized material is present than is currently modelled.

Additional exploration and test-work will potentially reduce this risk as the project is advanced.

1.10.2 Mining

The mining concepts for CuMo are largely proven. The adoption of autonomous equipment does possess some risk in that federal and local regulators may require extensive efforts by proponents to ensure the safety of their operations.

The CuMo open pit is envisioned to be a large, deep pit (up to 3500 ft deep). With this comes the potential geotechnical risk for wall failures. While the author has assumed a relatively flat overall wall angle for the PEA (37°), there may be risks associated with yet unknown rock mass or structural geology conditions that may require consideration of even flatter slopes in places.

1.10.3 Mineral Sorting

The technology envisioned in this PEA for bulk sorting, prompt gamma neutron activation analysis (PGNAA), has had limited application to molybdenum-copper deposits. While demonstrated for some low-grade copper deposits, testing is required to verify that molybdenum is measurable at the specific grades envisioned for CuMo.

As with bulk sorting technology, additional testing is required to better estimate the final results expected from particle sorting.

1.10.4 Processing

There is a risk that achieved recoveries could be lower than estimated, that throughputs will not be achieved and that costs may be higher than modelled. The process recovery, throughput and cost estimates will be refined as part of the pre-feasibility study.

1.10.5 Project Infrastructure

The planned mine will be a green-fields site and requires construction of mine and process-related infrastructure including the TSF. Access roads in and around the project site will be required. There is a risk that the designs, costs and implementation timelines for the provision of this infrastructure may not be as anticipated, increasing costs and schedule.

1.10.6 Permitting

At this time, no issues were identified that would materially impact the ability to eventually extract mineral resources at the project. Previous environmental analyses have identified the presence of a rare plant Sacajawea's bitterroot (*Lewisia sacajawa),* and potential habitat for Endangered Species

Act wildlife, and USFS sensitive species. These potential issues will need to be analyzed and disclosed in NEPA documents and potentially mitigated.

The mine will be located in an area used for weekend summer dispersed recreation and fall big-game hunting and is well-known in the Boise area. Organized environmental groups such as the Idaho Conservation League and Sierra Club are keeping their constituents informed citing issues of potential pollution of the Boise river which supplies drinking water to the city of Boise. As such, well-funded, organized opposition to mining activities should be anticipated.

Under the 1872 Mining Law as amended, ICMC has the legal right to develop the mineral resources on their mining claims. The USFS has a requirement to manage ICMC's activities in accordance with its mining regulations at 36 CFR 228A and must ensure compliance with the requirements of the National Environmental Policy Act. As defined in law and regulations, the USFS is limited in that it may not deny ICMC's mining plan of operations provided that the activities proposed are reasonably incident to mining, not needlessly destructive, and comply with applicable federal, state, and local laws and regulations. The USFS does not have the authority to impose unreasonable requirements that would have the effect of denying the statutory right to explore and develop the mineral resource, provided the mining plan of operations otherwise meets the intent of applicable laws and regulations (USFS 2018).

There is a risk that the mining plan of operations would identify and characterize issues that may lengthen the timeline and increase the costs of the permitting the project. Note that the PEA described in this report does not quantify the timeline and costs for the pre-construction and permitting activities.

Table 20-1 in Section 20.2 summarizes the federal, state, and local authorizations and permits that will be required for mining. No applications for mining authorizations and permits have been filed with federal, state, and local agencies. Reclamation bonds will have to be posted with the state of Idaho and the USFS.

1.10.7 Economic Risks

Project Strategy Risk

Overall, the author considers that the likelihood of a major revision to project strategy emerging from the pre-feasibility study to be moderate. Mineral sorting as contemplated in this study is not a mature technology, and there is a risk that the assumptions used may not prove accurate. Elimination of the mineral sorting pre-process from the strategy has the potential to materially reduce the economic proposition of the project.

Commodity Price Risk

There is a risk that commodity prices may not be consistent with assumptions made in this study. In particular, molybdenum, which contributes to the majority of the project value is historically subject to significant price volatility.

Capital Cost Risk

There is a risk that the capital required to build and operate the project may be higher than that forecast in this study. The author recommends that the precision of the estimates be refined at the pre-feasibility study and feasibility study before commitment to project construction is made.

Operating Cost Risk

There is a risk that the operating costs incurred to operate the project may be higher than that forecast in this study. The author notes that variability in the operating cost drivers (productivity, input costs and labor costs) over time is expected. The analysis assumes constant conditions but is best thought of as reflecting an expectation of average costs. The author recommends that the precision of the estimates be refined in the pre-feasibility and feasibility studies before commitment to project construction is made.

Schedule Risk

There is a risk that the schedule to build the project may vary from that assumed in the study. This is an asymmetrical risk, with significantly more downside scope than upside. This risk is exacerbated by the seasonality of the location, with somewhat difficult construction conditions occurring in some winter months. Small delays have the potential to be more significant than, might otherwise be the case, if they push critical path activities into winter months, thereby incurring a much longer delay.

Process Recovery Risk

There is a risk that achieved recoveries could be lower than estimated, reducing the revenue and economic returns of the project. The process recovery estimates will be refined as part of the feasibility study.

Permitting and Pre-construction Schedule Risk

This was not explicitly considered for the purposes of this study in the economic analysis as the analysis is conducted only from the commencement of construction. Nevertheless, the risk of longer-than-anticipated permitting timeline will reduce the project value is considered from “today” forward.

1.11 Project Opportunities

1.11.1 Mineral Resource

The exploration drilling and thus mineral resource model for CuMo is constrained on the western extents of the deposit. There is potentially an opportunity with increased exploration to expand the resource to the west, thus offering either more process feed within the current envisioned open pit or increasing the size of the open pit to the west.

1.11.2 Mining

With increased knowledge of the rock mass and structural geology, through additional geotechnical field programs and investigation, there is potential to steepen the wall angles for CuMo, potentially reducing and/or deferring some mining costs.

Further consideration of high angle conveying solutions in combination with semi-mobile crushing and conveying (IPCC) concepts could highlight opportunities for cost savings at CuMo. Applying IPCC to sort feed, which needs to be crushed either way and is up to 50% of the mined material, poses the greatest opportunity.

1.11.3 Mineral Sorting

The bulk sorting analysis was conducted on drill core that was sampled on a standard 10 ft interval. Thus, heterogeneity could only be assessed down to this scale. With multiple stage sorting and splitting, smaller size packets of material could be measured. As heterogeneity increases with reduced scale, there is potential that better segregation of waste, mill feed and middlings is possible. The opportunity would be for increased waste rejection and ultimately reduced middlings fractions to improve the economics of the project.

Ultimately, the potential for exploitation of the heterogeneity of the deposit may not be firmly quantified by way of studies conducted on exploration-level data. Much higher-resolution sampling and sorting may be possible at an operational scale. This has the potential to enhance project economics, but the quantum of that improvement is difficult to quantify.

The field of mineral-sorting is the subject of significant research and development. There exists an opportunity for this project to exploit improvements in this technology.

1.11.4 Processing

Additional metallurgical work to determine optimum grind size (the current assessment is based on the finest grind tested to date), analyze recoveries of the various metals in the proposed unit operations, and analyze the effects of the higher grade coming from the mineral sorters on metal recoveries. This has the potential to improve project economics.

Optimization of reagents to reduce costs and improve metallurgical recoveries has the potential to improve recoveries.

Further testing is required to determine if there is an opportunity to economically recover tungsten from the mineralized material.

1.11.5 Project Infrastructure

Further studies may allow for optimization of infrastructure design, costing and schedule. Whilst optimization is worth pursuing, the author views modification to the infrastructure concepts to be unlikely to materially affect the economic proposition at a strategic level for the project.

1.11.6 Economic Opportunities

Real Option Value

In the case of a large, long-life open-pit mine such as is contemplated for the CuMo project, there exists significant optionality that can be leveraged to improve project cashflows and values. The simple sensitivity analysis conducted in Section 22.12 assumes a constant operating strategy, even as assumptions are varied. In practice, management has the option to alter strategy in response to those variations. Downsides can be mitigated, and upsides can be leveraged for greater returns.

It is also expected that the mine would run using a dynamic cut-off policy where sorting strategies and cut-offs, mill-feed cut-offs, stockpiling strategies and mining rates will all be varied in real time to maximize returns as prices and costs vary. The benefits of this strategy are not reflected in the central estimate approach to valuation summarized in this report.

Project Strategy Opportunity

While the probability of a major revision to project strategy can be considered moderate, careful consideration and revision of the strategic decisions should be a feature of studies going forward. In particular, effort should be made to enhance the optionality of the project, particularly where this is a low-cost investment.

Commodity Price Opportunity

There is a risk that commodity prices may not be consistent with assumptions made in this study. Higher prices, both realized and forecast, would lead to re-optimization of the mine and processing plans with a potential to create additional value beyond that shown by the sensitivity analysis summarized in Section 22.11.

Capital Cost Opportunity

Opportunities to reduce or defer capital expenditure may be realized in future studies. Care should be taken when considering the relationship between lower capital opportunities and technical risk to the project.

Operating Cost Opportunity

Operating costs may be lower than forecast for the purposes of this study. Lower costs should feed into both strategic and short-term mine planning, to allow optimization of stockpiling, sorting and mill feed strategies.

Schedule Opportunity

This risk is highly asymmetric. The author considers that the opportunity to execute a significantly shorter construction program is low. The author cautions that optimized schedules with multiple critical or near-critical path activities will contain additional embedded risks.

Process Recovery Opportunity

Further metallurgical test-work will allow for optimization of the process flow sheet and plant design in the pre-feasibility and feasibility studies. Better than planned recoveries are possible.

Pit Slope Angle Opportunity

This is not considered to be a significant opportunity from an economic perspective. Strip ratios are relatively low, and incremental change in waste-movement volumes do not impact the overall project economics significantly.

1.12 Conclusions and Recommendations

1.12.1 Mineral Resources

Exploration work consisting mainly of drilling is required to reach feasibility. It is estimated that a total of 33 additional holes for 71,000 ft plus an additional five geotechnical holes for 12,000 ft on the deposit plus additional 74,800 ft allocated to condemnation drilling of waste dump, mill site and tailings pond areas, making a total of 157,8000 ft of drilling budgeted. This drilling is broken into the following categories:

- In-fill drilling,
- Delineation drilling,
- Orientated geotechnical drilling – requires orientated core recovery system,
- Drilling for metallurgical sample – large diameter hole (PQ size) recommended, and
- Condemnation drilling waste dump, mill and tailings site.

The shortest time to complete this work will be two seasons using four drill rigs each season.

1.12.2 Pit Geotechnical

The author provides these recommendations for the next steps of geotechnical assessment:

- Geotechnical database for quality assurance and quality check assessment (to address the inconsistencies and potentially poor data observed in the existing data set)
  - Select a sub-set (~10%) of resource drill holes that give good spatial coverage of the proposed pit walls, and from multiple drilling campaigns
  - Undertake quantitative basic geotechnical logging using the full core photographs of these drill holes (total core recovery (TCR), solid core recovery (SCR), rock quality designation (RQD) and fracture frequency - FF/m)
  - FF/m vs RQD plots for both data-sets
  - Comparison of the values in the database with the photo-logged values
  - Assessment of differences in order to determine whether variance is systematic or random, and consequently decide on the respective approach to address (e.g. apply correction factor, re-logging more of the drill holes)
  - Qualitative assessment of the rock susceptibility to deterioration by comparing core in the photos (fresh), to the current condition of the stored core (aged)
- Major structures assessment
  - Log the photos of the core for major structures
  - Develop conceptual integrated litho-structural 3-D model
- Geotechnical-specific diamond-cored drill holes targeted to provide coverage of the proposed interim and ultimate pit walls, and compatible with the pit depth
  - Geotechnical logging to RMRB89 system (historical logging to RMRL90 which is typically for underground mine applications)
  - Field (empirical and point load) and laboratory (uniaxial and triaxial compressive strength and direct shear) testing of fresh core to determine intact rock strength
  - Calculate RMR values and conduct comparison with lithology, alteration and mineralogy zones of the 3-D geology model to establish broad geotechnical domains

– Establish pit sectors and domain-representative sections to conduct pit slope stability analyses and select pit design angles

1.12.3 Mining

The author recommends further study of the application of high angle conveying of sort feed at CuMo.

The author further recommends the continued consideration of autonomous haulage for CuMo, with commensurate refinement of performance parameters and costs.

1.12.4 Mineral Sorting

The author recommends that CuMoCo engage with bulk sorting technology providers to advance testing of penetrative technologies (e.g. PGNAA) for the measurement of molybdenum in lower grade applications.

Additional scanning of the existing core to examine heterogeneity at a finer level than the 10 ft intervals used in the current study is recommended. Further testing of existing particle sorting technologies/machines to look for improvements in throughput is recommended.

1.12.5 Processing

Metallurgical aspects to be studied were highlighted in the preliminary metallurgical analysis, some of which require larger samples to finalize the detailed flow sheet and determine how many cleaning stages will be required.

A critical part of the analysis is a grinding-versus-recoverability study, since in the previous study, only two grinding sizes were studied: coarse and fine. The fine grind promised to be more economically favorable despite the increase in costs. Further study with multiple grinding size options is required to determine an optimum grinding system. An intermediate grind for example in the range between 71 to 106 microns P_{80}, would allow single stage SAG milling to be evaluated for reduced comminution energy cost, lower operating and maintenance labor in comminution and dewatering, and easier discard of a coarser tailing product when compared to the present grind P_{80} of 63 microns. The single stage SAG milling concept also allows for low cost future expansion of the initial lines provided by simply adding a ball mill to each SAG mill line to create about double the tonnage capability, without adding feed bins or conveyors. SAG mills with diameter/length aspect ratios over two are needed to make this work.

Work will consist of collecting and analyzing sufficient large bulk samples to determine the optimum flow sheet for the deposit. This work is expected to be further supported by a variability study to analyze variations within the deposit. Typically, a total of 100 to 150 twenty-kilogram samples will be used for the variability study.

1.12.6 Tailings Management

Engineering studies, including TSF design and potential water management and treatment design, including:

- Updating the TSF and Clear Creek Waste Facility designs based on field investigation results

- Developing tailings deposition plan and waste placement sequence to match pit development and mill output
- Detailed analysis of the water and load balance to predict the accumulation of mill reagents in the process water circuit from the tailings

1.12.7 Permitting

At this time, no issues were identified that would materially impact the ability to eventually extract mineral resources at the project. A mining plan of operations and reclamation cost estimate must be prepared to identify locations of the mine, waste rock dumps, roads (haul and access), power and water line corridors from the source to the point of use, mill, tailings storage facility, and other support facilities. Operating plans must be developed in conjunction with the mining plan of operations. ICMC should develop robust reclamation and closure plans for the facilities. ICMC should also begin acquiring any necessary water rights. Stakeholder outreach should continue.

Once the facility locations have been determined, ICMC should coordinate with state and federal agencies to identify the baseline studies that will need to be completed to support the development of an environmental impact statement and initiate those studies.

Previous environmental analyses have identified the presence of a rare plant Sacajawea's bitterroot (*Lewisia sacajawa),* and potential habitat for Endangered Species Act wildlife, and USFS sensitive species. These potential issues will need to be analyzed and disclosed in NEPA documents and potentially mitigated.

Organized environmental groups such as the Idaho Conservation League and Sierra Club are keeping their constituents informed so as to coordinate opposition to the project. As such, well-funded, organized opposition to mining activities should be anticipated.

1.12.8 Plan and Budget for Additional Work

Table 1-8 sets out a summary of work expected to be completed prior to a commitment to construction. The estimated time frame for this work program is three years.

Table 1-8: Budget for additional work

Item	Additional Information	Budget (000s $)
Diamond Drilling		
Delineation, infill, metallurgy	48,097 m (157,800 ft) @ $100/ft	15,780
Road Construction	2 km @ $50,000/km	100
Sample Preparation and Analysis	8,800 @ $60 each	528
Metallurgical Testing	Sample Collection, etc.	125
	Batch Round of Testing	1,000
	Variability Test-work	1,200
Land Acquisition and Staking Costs		8,000
Environmental Studies	Environmental Assessment	713
	Baseline Studies Startup	12,500
	Environmental Plan of Operations	800
	Environmental Impact Statement	23,500
	Permitting	3,000
Engineering Studies Scoping	Mill Site, Tailings Site Analysis	550
	Intergoverment Task Force Creation	500
	Mining Plan of Operations	1,200
	Pre-feasibility Study	5,500
Mobilization-Demobilization		427
Road Maintenance		325
Supervision and Project Management	Supervision	225
	Corporate Manager	360
	Project Manager	240
	Assistant Geologist(2)	364
	Technicians (12)	1,174
Vehicles	5 Vehicles	150
Accommodation and Food	30 Personnel	760
Travel		42
Project Office and Warehouse		1,225
Land Filing Fees	Current BLM: $155/claim/year	87
Land Filing Fees	Projected Additional Filing Fees	256
Consultants	(Mining, Metallurgical and Marketing)	575
Resource Modeling		1,650
Public Relations and Project	Public Relations and Legal, etc.	2,550
Presentation	Liaison County and State Officials	1,250
Subtotal		**86,655**
Contingency		13,345
Total		**100,000**

2 Introduction and Terms of Reference

2.1 Issuer

The CuMo project is an early-stage molybdenum-copper exploration project, located approximately 37 miles northeast of Boise, Idaho, USA. CuMoCo, which is listed on the TSX Venture Exchange (MLY), holds its interest in the CuMo project through its direct subsidiary, ICMC, a USA corporation. For the purposes of this report, both CuMoCo and ICMC are referred to as proponents of the CuMo project.

2.2 Terms of Reference

In November 2018[2], CuMoCo contractually commissioned SRK to update a PEA for the CuMo project and to issue an independent NI 43-101 report. The services were rendered from November 2018 to June 2019, leading to the preparation of this technical report, the summary results of which were disclosed publicly by CuMo in a separate news release.

This technical report includes, and is based on, a mineral resource statement for the CuMo project prepared by Mr. Gary Giroux of Giroux Consultants Ltd. and reviewed by the author. That mineral resource statement was published as part of a technical report in October 2015. This current technical report re-states the mineral resource for CuMo as well as incorporates the results of the 2019 PEA.

This technical report was prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 (NI 43-101) and Form 43-101F1. The mineral resource statement reported herein was prepared in conformity with generally accepted CIM "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines."

2.3 Responsibility

A summary of responsibilities by Qualified Person in the preparation of this report is shown in Table 2-1.

[2] Some site visits were undertaken in October, prior to formal commencement of services.

Table 2-1: QP responsibilities

Name	Company	QP Responsibility
Bob McCarthy	SRK	Sections 1.1, 1.2, 1.6, 1.8, 1.10.2, 1.10.3, 1.11.2, 1.11.3, 1.12.3, 1.12.4, 1.12.8, 2 to 5 (except 4.2, 4.3, 4.4, 4.5), 13.2.2, 15, 16 (except 16.2.1), 17.2, 18.1, 21.1.1, 21.2.1, 21.2.2, 21.2.4, 23, 24, 25.1.2, 25.1.3, 25.2.2, 25.2.3, 25.3.2, 25.3.3, 26.3, 26.4, 26.8, 27 & 28
Gilles Arseneau	SRK	Sections 1.3, 1.4, 6 to 10
Gary Giroux	Giroux Consultants	Sections 1.5, 1.10.1, 1.11.1, 1.12.1, 11, 12, 14, 25.1.1, 25.2.1, 25.3.1, 26.1 & Appendices 2-4
John Starkey	Sacré-Davey	Sections 1.10.4, 1.11.4, 1.12.5, 13 (except Section 13.2.2), 17 (except 17.2), 21.1.2 (Processing), 21.1.4, 21.2.3, 25.1.4, 25.2.4, 25.3.4 & 26.5
Andy Thomas	SRK	Sections 1.12.2, 16.2.1 & 26.2
Neil Winkelmann	SRK	Sections 1.9, 1.10.5, 1.10.7, 1.11.5, 1.11.6, 4.2, 4.3, 18.2 to 18.5, 19, 21.1.2 (Infrastructure), 22, 25.1.5, 25.2.5, 25.2.7, 25.3.5, 25.3.6 & Appendix 1
Calvin Boese	SRK	Section 1.12.6, 18.6,& 21.1.3 & 26.6
Valerie Sawyer	SRK	Sections 1.7, 1.10.6, 1.12.7, 4.4, 4.5, 20, 25.2.6 & 26.7

2.4 Work Program – Preliminary Economic Assessment

The PEA reported in this technical report was undertaken in the SRK Vancouver office during the months of November 2018 to March 2020. It included evaluating the potential technical and economic merit of the CuMo project as an open pit mining operation.

The PEA described herein is preliminary in nature and is partly based on inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary assessment based on these mineral resources will be realized.

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

2.5 Basis of Technical Report

This report is based on information collected by the authors during site visits performed as set out in Section 2.7 and on additional information provided by CuMoCo throughout the course of the PEA study.

The authors have no reason to doubt the reliability of the information provided by CuMoCo. The authors have not performed verification studies with respect to information provided by CuMoCo other than as described explicitly in this report.

Note: Throughout this report, all currency is 2019, real (i.e. non-escalated for future values) United States Dollars unless otherwise specifically noted.

2.6 Site Visit

In accordance with NI 43-101 guidelines, some QPs have visited the CuMo project site. Mr. McCarthy and Mr. Thomas visited the property from 30 to 31 October 2018, while Mr. Boese visited on only 30 October. All were accompanied by Joey Puccinelli of ICMC. The purpose of the site visit was to observe the mining area as well as project infrastructure, including access, rail, and water supply. Drill core was also inspected to provide context and observations for mining and geotechnical purposes.

Mr. Giroux last visited the site in June 2015, when all appropriate requirements of a current inspection were met for data verification and resource estimation. The purpose of the site visit was to observe the drilling area and the drill core handling and storage facilities. Drill core was inspected, core processing facilities were viewed, demonstrations of the core cutting and handling procedures were discussed and the separate in-warehouse core handling facility was observed. No drilling related work has been performed on site since that time.

In 2017, all core, rejects and information was moved from Garden Valley to a larger secure warehouse in nearby Horseshoe Bend, which has the same level of security as the one in Garden Valley with the exception that the geologist, as of the date of the current report, no longer lives at the property. The author has viewed photos of the facility which were also part of the site visit of SRK personnel in 2018 (Section 2.6).

Mr. Starkey, Mr. Winkelmann, and Ms. Sawyer have not visited the site.

The QPs were given full access to relevant data.

2.7 Acknowledgement

SRK and the team would like to acknowledge the support and collaboration provided by CuMoCo personnel for this assignment. Their collaboration was greatly appreciated and instrumental to the success of this project.

3 Reliance on Other Experts

In relation to the information contained in Section 4.2 and Section 4.3, The authors have not performed an independent legal review and verification of land title and tenure information. The authors did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties. The authors have relied upon information collated by CuMoCo with regard to legal matters relevant to this report. This reliance is on information as to claim ownership and mineral rights as provided by the United States Bureau of Land Management.

The authors have relied on the USFS and NEPA to examine procedures and status for Sections 4.4 and 20, and various accounting firms have been contacted to confirm the current US Mine tax system used in Sections 21 and 22.

With respect to Section 4.5 The author was informed by CuMoCo that there are no known litigations potentially affecting the CuMo project.

The author has no reason to believe that any of the information as provided by CuMoCo and outlined above is inaccurate or misleading.

4 Property Description and Location

4.1 General

The CuMo property is located approximately 37 air miles northeast of the city of Boise, Idaho, USA (Figure 4-1). It is situated in the northern portion of the Grimes Pass area on the USGS 1:62,500 Placerville Quadrangle (15' Series) within T7N and T8N, R5E and R6E, in Boise County, Idaho (Figure 4-2). The Latitude at the approximate center of CuMo property is 44 degrees, 2'N and the Longitude is 115 degrees 47' 30" W or UTM coordinates of 597,500E, 4,876,000N (NAD 27 CONUS).

4.2 Mineral Tenure

The property consists of 185 unpatented and un-surveyed contiguous mining lode claims covering an area of approximately 3,260 acres and 41 fully patented claims covering an area of 739 acres. Most of the claims consist of full-sized, 600 ft by 1500 ft claims (20.66 acres each). However, the total includes 27 fractional claims where the new claims were staked over existing claims. The claims are shown in Figure 4-2 and the claim information is listed in Appendix 1. Unpatented claims have the mineral rights with the surface owned by the federal government. Patented claims are private property and cover both the surface and mineral rights.

In Idaho, staked claims expire annually on September 1. An annual fee of $155/claim must be paid to the BLM prior to Aug 31, 2019 or all claims will expire on Sept 1, 2019. At $155/claim, CuMoCo must make annual payments to the BLM of $28,675 to keep all claims in good standing.

For patented claims, since they are owned outright, taxes are assessed by the county on a yearly basis. Currently the yearly tax bill for the patented claims is approximately $450. It varies as it is dependent upon assessed value and the county tax rate which changes from year to year.

4.3 Ownership Agreements

On October 13, 2004, CuMoCo completed an "Option to Purchase Agreement" with CuMo Molybdenum Mining Inc. to purchase eight unpatented mineral claims located in Boise County, Idaho, USA known as "CuMo Molybdenum Property". As part of the original CuMo and CuMoCo agreement, all claims acquired within five miles of the CuMo 1-8 claims became part of the option deal. Therefore, all the new claims referred to in this report as part of the CuMo Molybdenum Property are automatically subject to the terms outlined in that agreement.

Terms of the agreement are:

1. Advance royalty payments:
   - $10,000 upon signing (completed)
   - $10,000 after 60 days (completed)
   - $5,000 after 6 months (completed)
   - $20,000 1st year anniversary (completed)
   - $20,000 2nd year anniversary (completed)

- $15,000 3rd year anniversary (completed)
- $15,000 every 6 months thereafter (up-to-date)

These payments are to be credited against a 1.5% net smelter return (NSR) which reduces to 0.5% NSR after cumulative payments of $3,000,000.

2. Work requirements:

   - $25,000 during the first year (completed)
   - At least $50,000 each year thereafter (up-to-date)
   - On January 21, 2005, CuMoCo entered into an option agreement with Kobex Resources Ltd. (Kobex), whereby Kobex could acquire a 100% interest in the CuMo Molybdenum Property and another property in Australia. Under the terms of the Agreement, Kobex would earn a 100% undivided interest in these properties in consideration of cash payment of $5,000,000, 12,500,000 treasury shares and $10,000,000 of work expenditure commitment.

On October 6, 2006, Kobex surrendered all rights and interests in the CuMo property to CuMoCo.

CuMoCo has completed all payments since 2006 and the property is in good standing.

February 14, 2017 CuMoCo announced it has purchased 20 claims in the area around the CuMo project from a group of local prospectors. The 20 unpatented mining claims cover an area of approximately 400 acres. The purchase price of 100% ownership the claims was one $250,000 silver unit plus one million shares of CuMoCo.

Note: A silver unit is a seven-year exchange approved debenture that can be converted into the right to buy silver for $5 per ounce from any future production at CuMo. The debenture pays 8.75% interest per annum.

In April 25, 2017, CuMoCo announced that its wholly-owned subsidiary, ICMC, has completed an option to purchase agreement for 36 patented mining claims, covering an area of approximately 640 acres adjacent to the CuMo project. Patented claims contain the surface rights as well as the mineral rights. The consideration payable for the claims is as follows:

- Upon closing date of the agreement, the sum of $320,000 in cash, two (2) Silver Units in the aggregate principal amount of $500,000 and such number of CuMoCo shares having a value of $322,500 (with the CuMoCo shares being issued at a price equal to the 10-day weighted average trading price of the CuMoCo shares on the TSXV as of the last business day prior to the Closing Date)
- Upon the first anniversary of the Closing Date, $320,000 in cash, one (1) Silver Unit in the aggregate principal amount of $250,000 and such number of CuMoCo shares having a value of $322,500 (with the CuMoCo shares being issued at a price equal to the 10-day weighted average trading price of the CuMoCo shares on the TSXV as of the last business day prior to the first anniversary of the Closing Date)
- Upon the second anniversary of the Closing Date, $320,000 in cash, one (1) Silver Unit in the aggregate principal amount of $250,000 and such number of CuMoCo shares having a value of

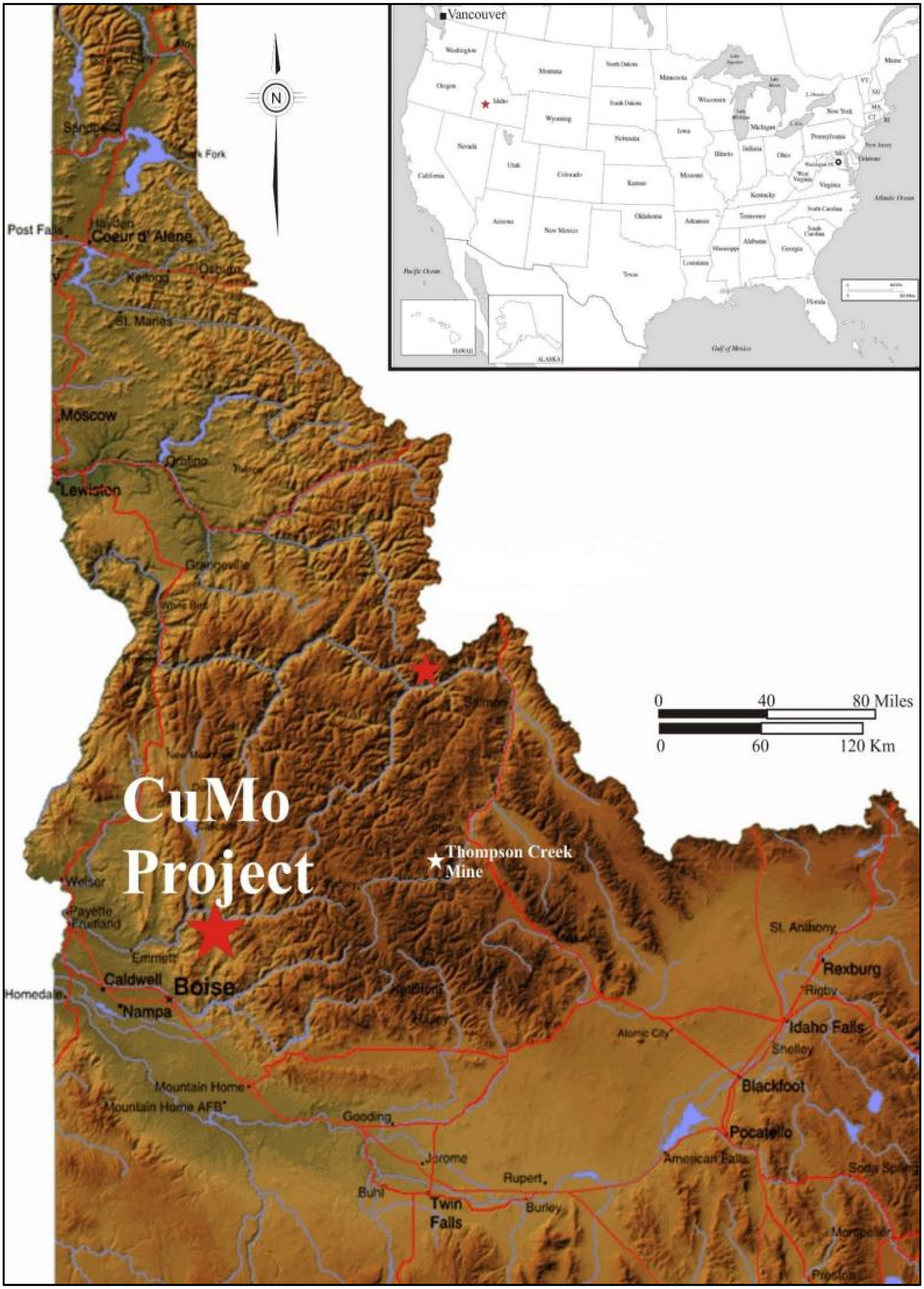
$322,500 (with the CuMoCo shares being issued at a price equal to the 10-day weighted average trading price of the CuMoCo shares on the TSXV as of the last business day prior to the second anniversary of the Closing Date)

- Upon the third anniversary of the Closing Date, $320,000 in cash, one (1) Silver Unit in the aggregate principal amount of $250,000 and such number of CuMoCo shares having a value of $322,500 (with the CuMoCo shares being issued at a price equal to the 10-day weighted average trading price of the CuMoCo shares on the TSXV as of the last business day prior to the third anniversary of the Closing Date)

Payment of the Option Payments (except for the issuance of the American CuMo shares) may be accelerated at CuMo's option.

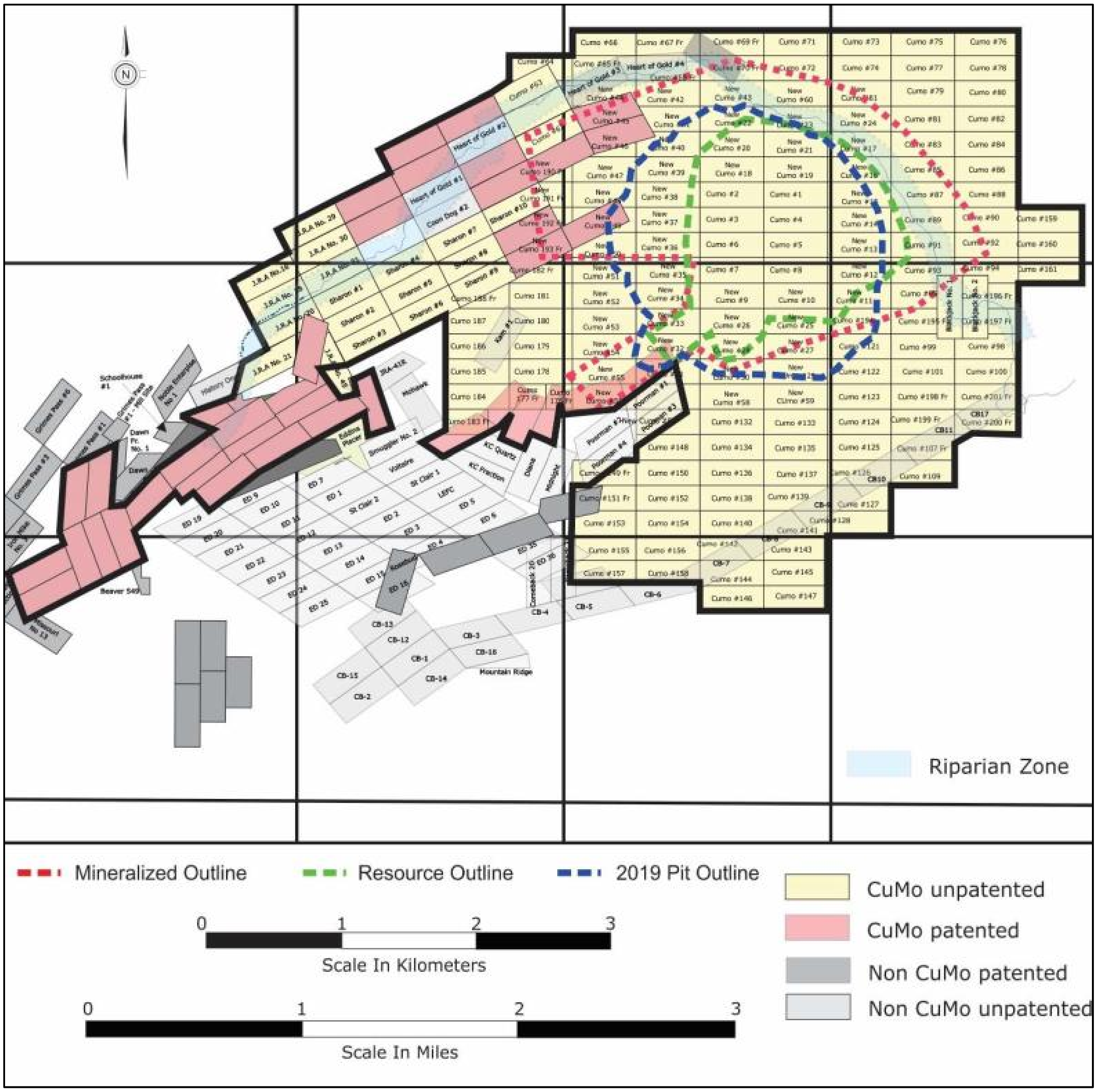
In July 17, 2017, CuMoCo announced that it had signed a mining claims purchase agreement effective as of July 6, 2017 (the "Purchase Agreement") between CuMoCo and its wholly-owned subsidiary, ICMC, and CuMo Molybdenum Mining Inc., Western Geoscience Inc. and Thomas Evans (collectively, the "Parties"). CuMoCo is to acquire from the Parties a 100% interest, including any Net Smelter Royalties owned by the parties, in the CuMo project which is currently under option, pursuant to an option agreement between CuMoCo and CuMo Molybdenum Mining Inc. dated October 13, 2004 and amended on January 14, 2005 (the "Option Agreement").

As of the effective date of this report all agreements remain in place as described above.



Source: Giroux et al, 2015

Figure 4-1: CuMo property location map



Source: Giroux et al, 2015 modified 2019

Note: The mineralized outline of the deposit in this figure is the outline of the geologically-defined potentially mineralized zone[3]. For clarity, it is **not** the outline of the current resource, **nor** of the resource constraining pit (resource outline) which are inside the mineralized outline of the deposit. It is also **not** the outline of the pit design (2019 pit outline) used in the preliminary economic analysis.

Figure 4-2: Claim location map for the CuMo property

The change since the 2015 report in the outline of the deposit is a result of reconnaissance work performed in 2017 by CuMo field geologists on the recently acquired claims.

[3] Defined according to geology, lithology and/or mineralogy without explicit consideration of grades or economic potential

4.4 Environmental

4.4.1 Environmental Regulations

The CuMo project will be subject to federal, state of Idaho, and local regulations. Key regulations to which the project will be subject governing exploration and mining design, operations, and reclamation include:

- General Mining Act of 1872, 30 U.S.C. §§ 22-42
- Mining and Minerals Policy Act of 1979
- 36 Code of Federal Regulations Part 228 administered by the USFS
- Idaho Administrative Procedures Act (IDAPA) 20.03.02 Rules Governing Exploration, Surface Mining, and Closure of Cyanidation Facilities administered by the Idaho Department of Environmental Quality
- IDPA 16.01.02, Water Quality Standards and Wastewater Treatment Requirements
- IDPA 20.03.02, Rules Governing Exploration and Surface Mining Operations in Idaho
- IDAPA 58.01.01 Rules for the Control of Air Pollution, administered by the Idaho Department of Environmental Quality
- IDAPA 58.01.02 Water Quality Standards, Anti-Degradation, administered by the Idaho Department of Environmental Quality
- IDAPA 58.01.11 Ground Water Quality Rule, administered by the Idaho Department of Environmental Quality

4.4.2 Environmental Liabilities

There are currently no known environmental liabilities on the property. The company has a $300,000 reclamation bond on deposit once the permits are re-issued.

It is possible, that with the development of a detailed mining plan of operations and the more detailed investigation of aspects of the property that are associated with that plan, that as-yet unknown environmental liabilities and/or issues may be identified.

4.4.3 Other Significant Factors and Risks

At this time, no specific issues have yet been identified that would materially impact the ability to eventually extract mineral resources at the project. That is, any other significant factors and risks that may affect access, title, or the right or ability to perform work on the property are not yet known.

However, ICMC should be prepared to address potential issues associated with but not limited to the following aspects:

- Water including supply, water rights, and delivery system and potential impacts
- Water management (stormwater, contact/non-contact water, water quality)
- Geochemistry of ore, waste rock, tailings solids and solution, and post-mining pit lake

- Threatened, endangered, and special status plant and animal species
- Jurisdictional waters
- Transportation and access
- Reclamation and closure

4.5 Permits

Exploration on federal lands requires an authorization to conduct exploration except for sampling of rocks and soils by hand and other activities that create no land disturbance. There are three levels of permits reflecting increasing disturbance:

- The lowest level of authorization is a Categorical Exclusion. This is the least intense disturbance and requires some public notification. The authors understand that track mounted auger/rotary drilling with no new road clearing would fit in this category according to USFS personnel.
- Environmental Assessment requires an in-depth study with 30 days for public comment, plus additional time for appeal. The authors understand that drilling with a reverse circulation (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 an authorization. Spot 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.

Approval for a diamond drilling program has been obtained from the USFS, to be carried out from the existing network of drill access roads and was permitted under a Categorical Exclusion, issued in 2008. An application for a Water Use Permit for diamond drilling purposes was originally filed with the Idaho Department of Water Resources in 2008, that permit is renewed annually.

In January 2007, an exploration plan of operations was submitted for an expanded exploration program involving construction of new roads for drill access, and the USFS gave notice that an Environmental Assessment is required for that program. Note: This exploration plan of operations was filed while the Categorical Exclusion was active and that no mining plan of operations has been prepared.

On June 14, 2010, the Environmental Assessment was completed and submitted for public review and hearing during a mandated 90-day period. On February 14, 2011, A Finding of No Significant Impact was delivered by the USFS. During the mandated 45-day appeal period, one environmental group (Idaho Conservation League) submitted an appeal of the USFS decision.

On May 17, 2011, the USFS denied the appeal allowing CuMoCo to begin work under the new exploration permit following a mandatory 15-day stay period which ended on June 7, 2011. The permit covers all exploration work required to produce the information necessary to produce a feasibility study and lasts for up to five years.

The Idaho Conservation League et al, filed a challenge in the “United States District Court for the District of Idaho” on December 15, 2011: “Plaintiffs Idaho Conservation League, Idaho Rivers United, and Golden Eagle Audubon Society seek summary judgment reversing and remanding the USFS’s

February 2011 approval of the CuMo Mine Exploration Project, in the upper Grimes Creek watershed of the Boise National Forest." The USFS was named as defendant while CuMoCo was named as Intervener Defendant. CuMoCo has worked through the litigation process and filed a response brief and reply brief. The USFS has also filed response and reply briefs. The Idaho Conservation League also filed a reply brief.

On August 29, 2012, the judge in the case dismissed four of the five claims by the opponents but remanded the section on groundwater over for further study. As a result, on February 7, 2013 the USFS initiated a Supplemental Environmental Assessment in order to address the judge's concerns. This worked culminated on April 13, 2015 with the re-issuance of a draft Finding of No Significant Impact.

In January 2016, the Idaho Conservation League and others filed a challenge in the United States District Court for the District of Idaho of the new April 15 decision.

On July 13, 2016, the judge in the case accepted the work on the groundwater but remanded the decision to give USFS time to study the effects of the 2014 Grimes Creek fire on a sensitive plant species. As a result, on August 7, 2016, the USFS initiated a Supplemental Environmental Assessment in order to address the judge's concerns. Note: In 2018 this was renamed by the USFS to the Supplemental Redline Environmental Assessment. In 2016, another fire effected the property area and additional studies were required. The USFS is currently in process of preparing the updated report, which is expected to lead to a final decision in early 2020.

As of the effective date of this report all agreements remain in place as described in this Section 4.

At the current time there are no active permits as ICMC is waiting on the Supplemental Redline Environmental Assessment report and the final decision notice and finding of no significant impact.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

International air travel is available from Boise, Idaho. The property is accessed by road from Boise by taking US State Highway 55 northerly for approximately 40 miles (65 km) to the town of Banks, Idaho, and then east on the Banks Lowman Road towards the town of Garden Valley for approximately 10 miles (16 km). One mile east of Garden Valley is a secondary road heading south across the Payette River. Following this secondary road, the western most edge of the CuMo claim block is approximately 10 miles (16 km) from Garden Valley.

Alternatively, access can be gained by traveling northeast from Boise along Highway 21 past the towns of Idaho City and Centerville, along Grimes Creek, and then over the Grimes Pass.

The project is situated in the southern section of the Salmon River Mountains which lie immediately west of the Rocky Mountains and are characterized by north-northwest trending mountain ranges separated by alluvial filled valleys. Topographic elevations on the CuMo claims range from 5,100 ft (1700 m) to 7,200 ft (2,400 m) above sea level.

The climate is defined by summer temperatures to a maximum of 100° F (38°C) and cold, windy winters with lows to -10° F (-23°C). Precipitation is moderately light with an average rainfall of 30 inches (<1 meter) and an average snowfall of approximately 140 inches (3.6 m). Vegetation in the project area consists of cedar, lodgepole pine, mountain mahogany, and juniper.

The area is serviced by the Idaho Power Company which supplies electricity to residents of Garden Valley, Lowman and Pioneerville. The nearest rail line is the Idaho Northern & Pacific line formerly operated by Union Pacific that runs through the town of Banks, approximately 20 road miles (32 km) to the west of the property.

Equipment, supplies, and services for exploration and mining development projects are available at Boise. There is also a trained mining-industrial workforce available in Boise.

Exploration and mining at the property can be conducted year-round, due to the established road system and its proximity to other infrastructure. The property is large enough to accommodate the current CuMo deposit exploration or mining operations including facilities. Potential waste disposal and tailings storage areas, currently outside the property, will require permits from federal and state agencies as discussed in Section 20 of this report.

The project will be located on patented claims owned or optioned by ICMC and public land administered by the USFS. The extent of public land used for mining purposes will be identified in the mining plan of operations. In the USA, with the exception of the patented claims owned or optioned by ICMC, all surface rights in the area of the current design are administered by the USFS and are not available for purchase but for lease. The NEPA process will disclose the potential impacts from construction, mining, closure, and reclamation activities and identify mitigation to avoid or ameliorate impacts prior to authorization of the mining plan of operations. These surface rights are granted at the time of a record of decision to place the mine into commercial production and they form part of the permitting process.

6 History

6.1 Exploration

The Boise Basin was first explored following the discovery of placer gold deposits in 1862. Several lode gold deposits were discovered and developed immediately following the initial alluvial gold rush, with significant production occurring in the late 1800's and early 1900's. There are a number of lode prospects within approximately two miles of the CuMo property, some of which have recorded minor past production of base and precious metals. No production has occurred on CuMo itself.

The first interest in the CuMo property was shown during aerial reconnaissance by Amax Exploration in 1963. Follow-up geochemical rock and soil sampling indicated anomalous molybdenum and copper values. Forty claims were then staked, and three previously existing claims were optioned. A 2.5 mile (4 km) rough access road was constructed in 1964 to facilitate collection of rock samples and geological mapping. The property was subsequently relinquished due to the combination of contemporary economic conditions and initial sample grades.

In 1968, Curwood Mining Company staked 12 claims and undertook detailed mapping and geochemical rock sampling. This work indicated roughly coincident anomalies in copper, molybdenum and silver. Several trenches were excavated, and one line of dipole-dipole array IP geophysical survey was conducted.

In 1969, Midwest Oil Corp. optioned the property and conducted exploration drilling through 1972 (four short rotary holes (less than 100 ft) initially, which were later depended using diamond drilling, followed by six cored holes). Midwest also performed an IP survey in 1971 and an airborne magnetic survey in 1973. The IP survey indicated a pyrite halo on the north side of the deposit, although an alternative interpretation concluded "the combined IP data may indicate a halo effect but more probably shows an east-west trend to the rock types and mineralization" (Baker, 1983). The CuMo deposit did not have a strong magnetic signature, being somewhat of a plateau with surrounding highs.

In 1973, Midwest formed a joint venture with Amax and then subsequently Midwest was merged with AMOCO resulting in an Amax-AMOCO joint venture (JV) with AMOCO as operator. During the period 1973 to 1981, the Amax-AMOCO JV completed 30,822 ft of drilling (summarized in Table 6-1), surface geological mapping, re-logging of the core, road construction, an aerial topographic survey, and age dating. In 1980, Amax Exploration Inc. transferred its interest to Climax Molybdenum Company (Climax), also a subsidiary of Amax Inc. In 1982, Climax collected more than 300 soil geochemical samples from three different grids.

Table 6-1: Summary of historic drilling

Year	Company	Holes	Footage	Meters	Comments
1969	Midwest	4	378	115	Rotary holes shallow due to water included in core
1970	Midwest	0	653	199	2 rotary holes deepened with core to 400' depth
1971	Midwest	1	2,251	686	One core hole deepened further to 1,884 ft
1972	Midwest	3	1,892	577	One core hole deepened from 810-1,416 ft
1974	Amax	1	805	245	Hole 9-9A
1975	Amax	1	2,382	726	Hole 10
1976	Amax	2	4,343	1,324	One vertical, other 1,340 ft @ -45
1977	Amax	3	5,861	1,786	3 vertical DDH 1,804-2,124 ft deep
1978	Amax	3	6,774	2,065	3 vertical DDH 2,132-2,361 ft deep
1979	Amax	2	4,823	1,470	Vertical DDH to 2,543 ft depth
1980	Amax	3	2,630	802	RC holes
1981	Amax	3	3,204	977	Vertical DDH 1,000 to 1,193 ft depths
Total		**26**	**35,996**	**10,971**	

A total 23 diamond holes and three RC holes were drilled on the property (Table 6-2). Most RC holes were pre-collars to diamond drill holes with only the diamond drill component of the holes being used for resource modelling and sampling. The historic holes were sampled mostly at a 20 ft sample interval.

A Skelton core representation of the historic drill holes (one four-inch piece of core for each 10-foot interval), and all the sample rejects were delivered directly from Climax's secure facility in Colorado and are stored in the project secure warehouse facility for use by the project.

Table 6-2: List of historic drill holes

Hole	Northing	Easting	Elevation	Dip	Azimuth	Length (ft)
71-01	120,990	219,904	6026	-90	0	1,884 completed
71-02	120,575	219,820	6,060	-70	0	405 completed
71-03	120,250	219,905	6,165	-90	0	70 completed
C71-04	120,785	219,940	6,045	-90	0	113 completed
C72-05	120,525	220,570	6,202	-90	0	1,416 completed
C72-06	121,749	219,919	5,902	-90	0	663 completed
C72-07	121,491	219,823	5,962	-90	0	275 completed
C72-08	118,890	220,025	6,467	-90	0	379 completed
C74-09	121,438	220,687	5,890	-60	168	804.6 completed
C75-10	119,756	221,220	6,341	-90	0	2,381 completed
C76-11	120,456	221,250	5,996	-90	0	3,003 completed
C76-12	120,955	221,432	5,742	-43	190	1,340 completed
C77-13	119,472	219,903	6,426	-90	0	1,804 completed
C77-14	119,085	221,271	6,613	-90	0	2,123.8 completed
C77-15	119,772	221,951	6,339	-90	0	1,933.2 completed
C78-16	119,210	219,148	6,248	-90	0	2,131.7 completed
C78-17	118,712	219,887	6,544	-90	0	2,281.5 completed
C78-18	119,823	222,649	6,168	-90	0	2,361 completed
C79-19	120,178	219,887	6,170	-90	0	2,280 completed
C79-20	120,878	220,787	6,105	-90	0	2,543 completed
RC80-21	120,511	220,541	6,202	-90	0	1,000 completed
RC80-22	119,913	220,412	6,239	-90	0	670 completed
RC80-23	120,695	219,420	5,827	-90	0	960 completed
C81-24	120,671	222,009	6,070	-90	0	1,000 completed
C81-25	119,890	219,290	6,019	-90	0	1,011 completed
C81-26	121,338	221,433	5,768	-90	0	1,193 completed

Notes: C holes are diamond and RC are reverse circulation.
Holes contained in the above list represent individual holes that may have been drilled across more than one year, while table 6-1 shows the actual footage drilled in each year according the records.

6.2 Historical Resource Estimate

The estimate summarized here was undertaken by Climax prior to the inception of NI 43-101 and does not follow the Standard nor adhere to the categories outlined in NI 43-101. The “Amax Resource” is considered an historical estimate, and not a “Resource” in accordance with NI 43-101. A technical report on the property was never filed. A qualified person has not done sufficient work to classify the historical estimate as a current mineral resource. CuMoCo is not treating the historic estimate as current mineral resources. it is included here for historic completeness only. The resource for the property is only as defined in Section 14 of this report.

Based on the 26 drill holes a resource block model was constructed in 1983, extending between local grid coordinates 17,000 to 25,000 east and 16,000 to 23,000 north. The individual blocks were 100 ft in both the north-south and east-west directions and were 50 ft in height. Blocks were located from 7,000 ft down to 3,050 ft above sea level. Grades were estimated using 50 ft drill hole assay composites and mineralized zone boundaries. Kriging was performed within a 1,500 ft horizontal search limited to 300 ft vertically.

Table 6-3: CuMo historical results, 1982 Amax block model

Cut-off Grade (MoS_2%)	Million Tons	Average Grade (MoS_2%)
0.02	2,100	0.072
0.03	1,900	0.078
0.04	1,600	0.084
0.05	1,500	0.092
0.06	1,100	0.097
0.08	730	0.116
0.10	470	0.131
0.12	280	0.145
0.14	140	0.170

Source: Baker, 1983

In 1983, Climax transferred its interest in the property to Amax Exploration Inc. and no further work appears to have been done on the property.

7 Geological Setting and Mineralization

7.1 Regional Geology

The regional tectonic setting consists of a basement of amalgamated Archean and Paleoproterozoic crystalline terrains that were joined during the Paleoproterozoic Trans-Montana orogeny, and are overlain discontinuously by sedimentary rocks of Mesoproterozoic, Neoproterozoic, and Paleozoic ages; and volcanic and sedimentary rocks of Eocene and Miocene ages. Voluminous tonalite to granite bodies of the Idaho batholith and later granitic plutons of Eocene age intrude the older rocks. Major deformational episodes superimposed on the Precambrian basement include the Cretaceous Sevier orogeny, which mainly involved east-vergent “thin-skinned” thrusting; Eocene extensional deformation, which resulted in development of metamorphic core complexes; and basin and range-type faulting (Sims et al, 2005), as opposed to the Laramide orogeny’s “basement cored” uplifts which partially overlapped the Sevier orogeny in time and space.

The regional geology has been compiled at 1:1,000,000 to form the digital map of Idaho (Johnson and Raines, 1996). The CuMo deposit is situated within the Idaho batholith and is part of a regional scale belt of porphyry and related deposits identified as the Idaho-Montana Porphyry Belt (Rostad, 1978). This belt is part of a magmatic arc that formed on the northeast margin of the North American Craton (Figure 7-1) during Laramide time (Late Cretaceous-Early Tertiary). The Idaho-Montana Porphyry Belt lies within the much longer, 1,500 km, Great Falls tectonic zone (Figure 7-2), which was distinguished by brittle structures and intrusions of Phanerozoic age that are interpreted to have been controlled by the reactivation of basement structures. (O’Neill and Lopez, 1985). Two sets of basement structures, in particular, provided zones of weakness that were repeatedly rejuvenated (Sims et al, 2005):

(1) Northeast-trending ductile shear zones developed on the northwest margin of the Archean Wyoming province during the Paleoproterozoic Trans-Montana orogeny; and

(2) Northwest-trending intra-continental faults of the Mesoproterozoic Trans-Rocky Mountain strike slip fault system.

The Trans-Montana orogeny comprises a deformed, north-facing, passive continental margin and subsequent fore-deep assemblages overlying an Archean basement that is juxtaposed with accreted conjoined terrains. The juncture is the linear deformed belt between the Great Falls and Dillon shear zones. The fold-and-thrust belt of the Trans-Montana orogeny coincides in part with the Great Falls tectonic zone.

The Trans-Rocky Mountain fault system is a major, deep-seated, northwest trending, intracontinental strike-slip fault system of Mesoproterozoic age. It consists principally of west northwest-striking strike-slip faults (principal displacement zones), branching and en-echelon northwest-trending faults, and widely spaced, more local north-trending faults.

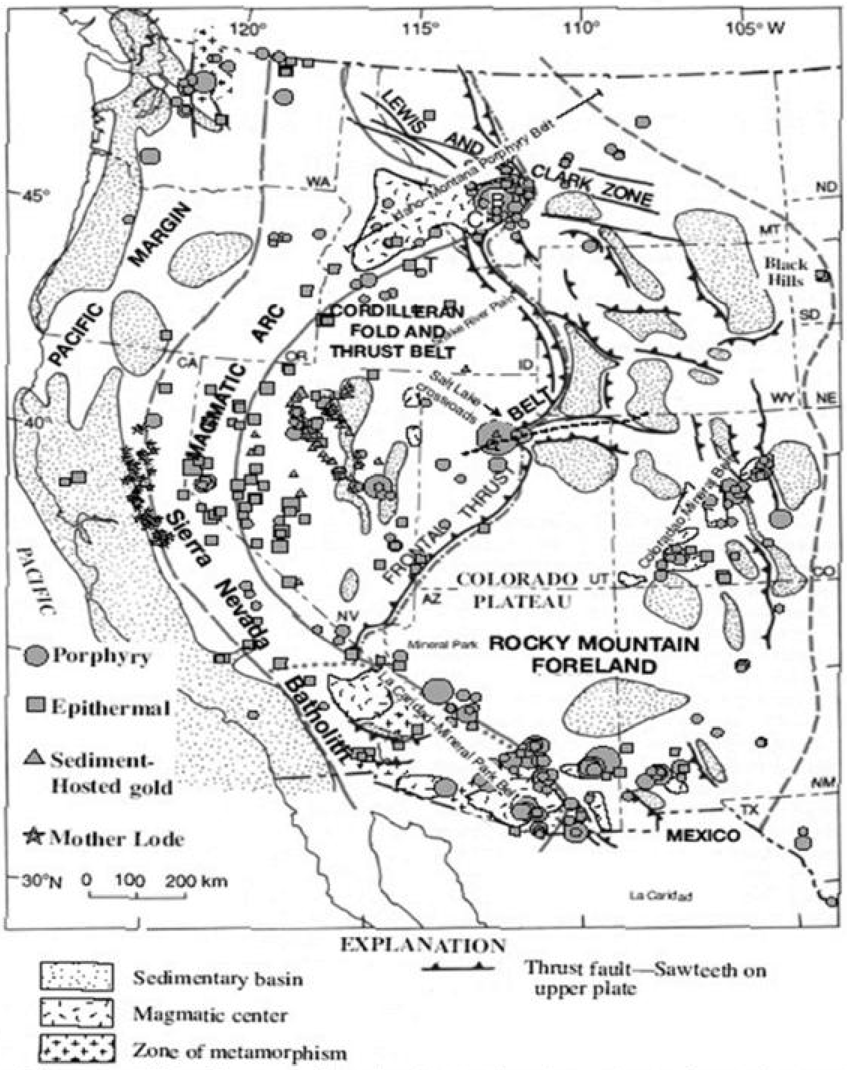
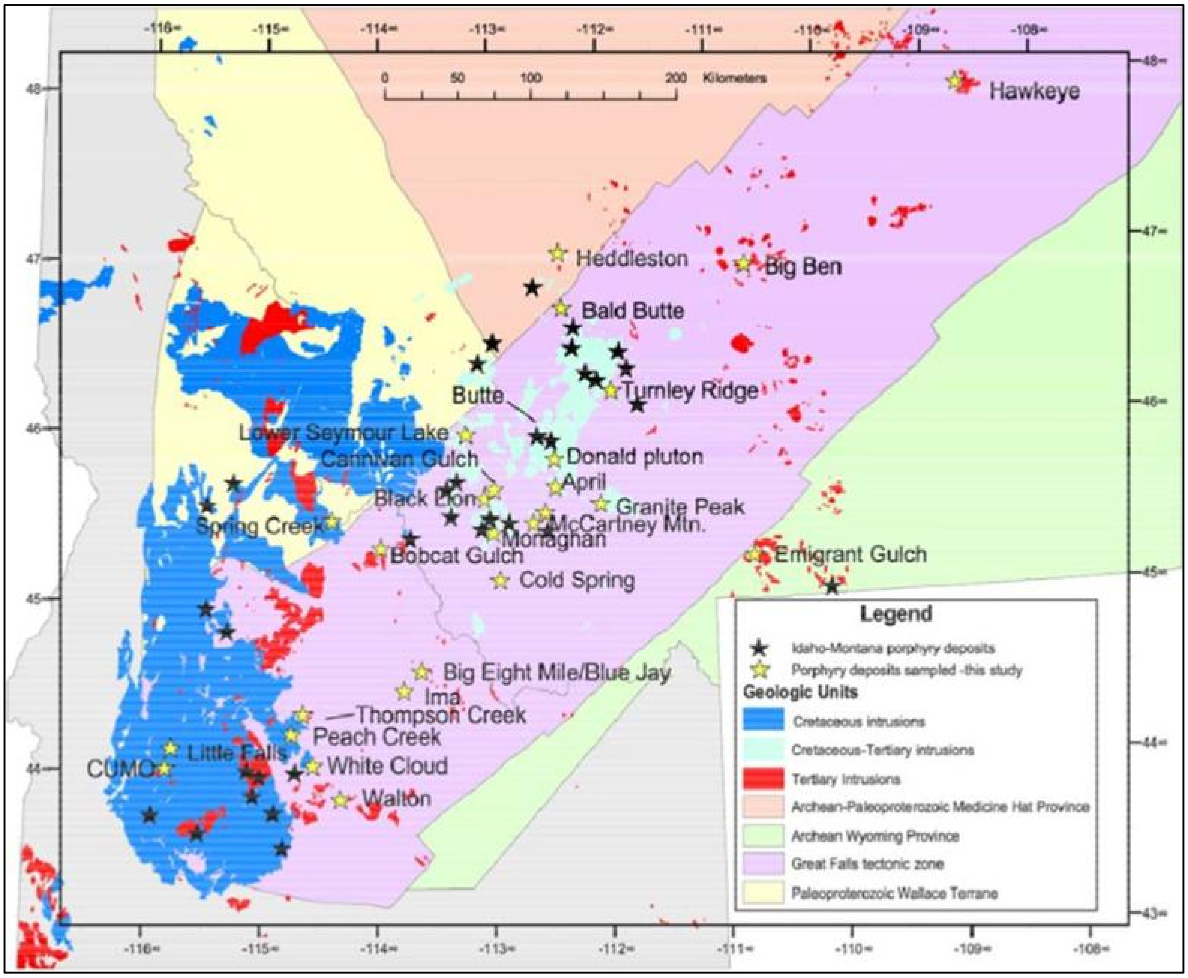


FIG. 1. Map of the western United States cordillera showing ore deposits superimposed on major tectonic elements and Laramide igneous zones, sedimentary basins, and metamorphic belts. The western United States is divided into four generalized geologic provinces (boundaries shown as heavy solid and dashed lines): Pacific margin, Magmatic arc, Cordilleran fold and thrust belt and the Rocky Mountain foreland. The smallest and largest ore deposit symbols represent gross values of about $20 million and $60 billion, respectively. Intermediate sizes of symbols are based linearly on deposit gross values lying between these extreme values. The short dashed line in northern Utah and southern Wyoming shows a segment of the boundary between the Archean basement on the north and Proterozoic basement on the south. It should be noted that although Jurassic accretion and magmatism resulted in complex geologic terranes along the Pacific coastal states, during the Laramide these regions experienced downwarping and basin development. Specific deposits discussed in the text include: B = Butte and C = Cannivan Gulch deposits in Montana; T = Thompson Creek deposit in Idaho (Modified from Miller et al., 1992).

Source: Hildenbrand et al, 2000

Figure 7-1: Tectonic map of the western United States



Source: Lund et al, 2005

Figure 7-2: Distribution of Idaho-Montana porphyry deposits in relation to the great falls tectonic zone

Mineral deposits in the Idaho-Montana Porphyry Belt (also called the Transverse Porphyry Belt of Idaho-Montana by Carten et al, 1993) are related to Eocene granitic intrusions. The distribution of deposits along this belt from northeast to southwest follows a progression from alkalic rocks (intra-arc rift-related), to mixed alkalic and calc-alkalic, and finally calc-alkalic intrusive rocks, a pattern that is similar to the distribution of igneous rocks from south to north along the proto Rio Grande rift (Carten et al, 1993). The CuMo deposit is located at the southwestern end of this belt and is associated with a calc-alkalic monzogranite, reported as 45-52Ma age (Carten et al, 1993) that intrudes Cretaceous equigranular intrusive rocks of the Atlanta Lobe of the Idaho Batholith.

The Idaho batholith is a composite mass of granitic plutons covering approximately 15,400 square miles. The northern part is called the "Bitterroot" lobe and the southern part the "Atlanta" lobe. Most of the southern lobe was emplaced 75 to 100 million years ago (Late Cretaceous); whereas the northern lobe was emplaced 70 to 80 million years ago. Older plutons of Jurassic age occur on the northwest side of the Bitterroot lobe and many Eocene plutons have intruded the eastern side of the Atlanta lobe of the batholith. Although radiometric dates and field relationships restrict the age of the Idaho Batholith to between 180 and 45 million years, the dominant interval of emplacement was early to Middle Cretaceous. There is a general west-to-east decrease in age for plutons of the batholith.

On the west side of the batholith the rocks are tonalites or quartz diorites, whereas on the east side they range from granodiorites to granites. The boundary between the two composition types also coincides with the 0.704 Sr87/Sr 86 boundary and also the boundary between the Mesozoic and Paleozoic eugeoclinal accreted rocks on the west with the continental Precambrian rocks on the east side (Digital Atlas of Idaho: http://imnh.isu.edu/digitalatlas/geo/bathlith/bathdex.htm).

The CuMo deposit is situated within the Atlanta Lobe of the Idaho batholith. The western margin of the Atlanta lobe is strongly folded and metamorphosed into gneissic rocks, which are well exposed near McCall. The western side is composed of tonalite, 95 to 85 million years old. The batholith core is biotite granodiorite; and the eastern side lobe is muscovite-biotite granite approximately 76 to 72 million years old[4].

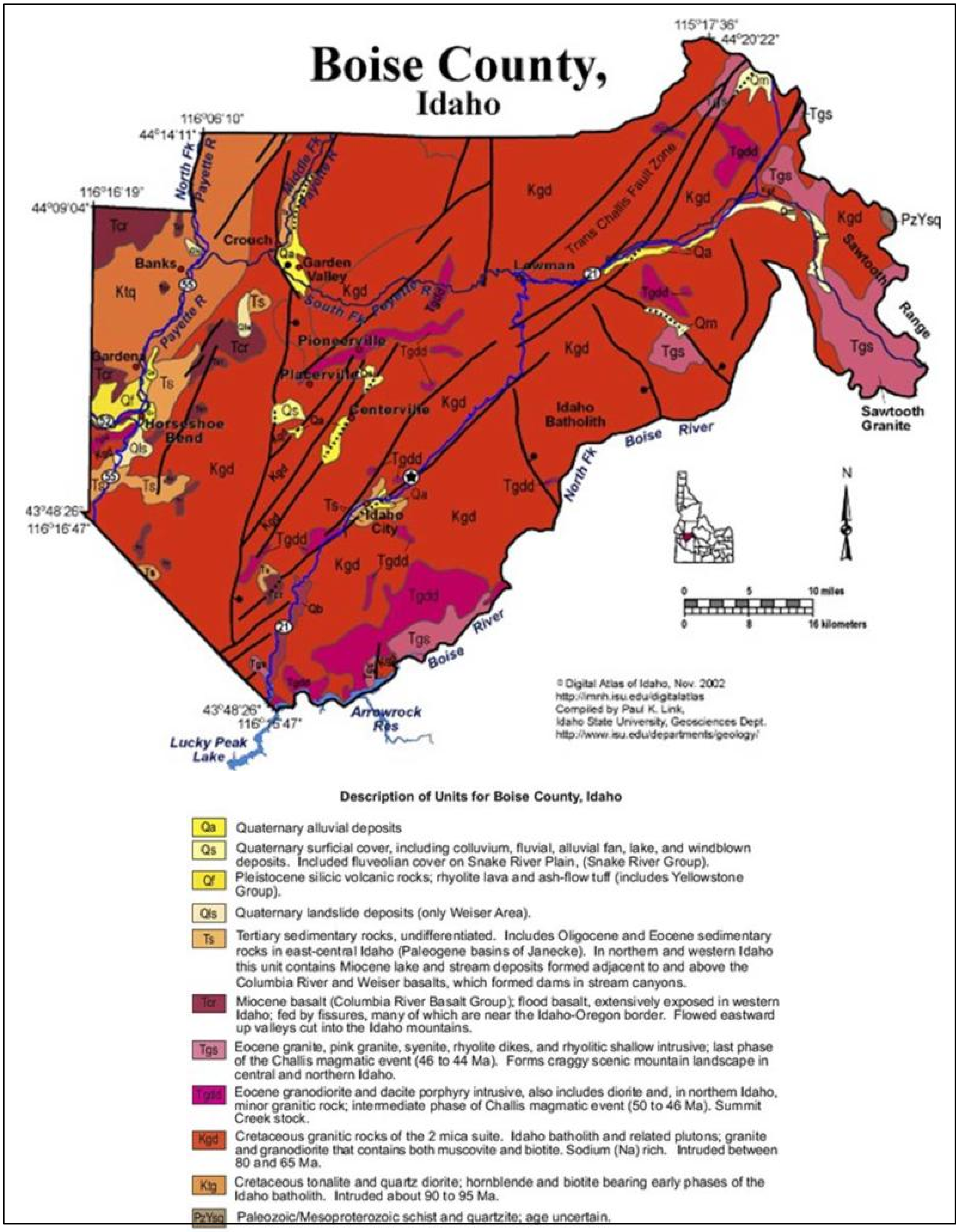
7.2 Local Geology

The geology of the area around the CuMo deposit was mapped and originally compiled at 1:24,000 scale by Anderson (1947). This mapping has been incorporated into the 1:100,000 scale Deadwood River 30 x 60 quadrangle map (Kilsgaard et al, 2006), and adjoining Idaho City 30 x 60 quadrangle map (Kilsgaard et al, 2001), and compiled into the Boise County map of the digital Atlas of Idaho (Figure 7-3).

The CuMo area is underlain by biotite granodiorite, the most common rock type of the Atlanta lobe of the Idaho batholith (Unit Kgd) (Killsgaard et al, 1985). This unit was mapped by Anderson (1947) as quartz monzonite: (Unit Kqm) – in part porphyritic and including granodiorite. The rock is light grey, medium to coarse-grained and equigranular to porphyritic. Biotite averages about 5% and sericite alteration of feldspar is common. Killsgaard et al (1985) report the age of this unit at 82-69Ma based on potassium-argon dating.

Tertiary plutonic rocks intruded into the batholith in the area of CuMo include Eocene diorite and hornblende biotite granite forming the Boise Basin and Long Gulch Stocks and associated dikes (Unit Tgdd) (Killsgaard et al, 2005). These units were identified as diorite and quartz monzonite porphyry, respectively, by Anderson (1947). The Eocene granites are generally characterized by pink color due to potassium feldspar as a major component, miarolitic cavities that may be lined with smoky quartz, high radioactivity relative to the Idaho batholith, the presence of perthitic feldspar, myrmekite and granophyric texture indicating high temperature crystallization complicated by quenching, and a high content of large cation elements including molybdenum, high fluorine content, and high-iron biotite (Killsgaard et al, 1985).

[4] Digital Atlas of Idaho: http://imnh.isu.edu/digitalatlas/geo/bathlith/bathdex.htm.



Source: Modified from: http://imnh.isu.edu/digitalatlas/counties/boise/geomap.htm

Figure 7-3: Geology of Boise County, Idaho, showing geological setting of CuMo deposit

Hypabyssal equivalents of the granites include numerous rhyolite dikes that are concentrated along the trans-Challis fault system (Killsgaard et al, 1985). Rhyolite dikes are generally less than 25 ft thick and may exhibit flow banding, whereas rhyolite porphyry dikes can reach 200 ft in thickness and have prominent quartz phenocrysts (Anderson, 1947).

Extensive placer gold workings and lode deposits in the area are situated along the northeast trending trans-Challis fault system (Killsgaard et al, 1989; Bennett, 1986). As shown in Figure 7-3, a north-trending Basin and Range fault, down on the east, bounds the system of northeast-striking trans-Challis faults to the west of CuMo (Link, 2002).

7.3 Property Geology

Amax completed detailed bedrock mapping on the CuMo property between 1964 and 1981. Earlier periods of mapping outlined five general rock types, including quartz monzonite of the Idaho Batholith, rhyolite porphyry, lamprophyre, dacite and diabase dykes. Subsequent mapping through to1982 resulted in subdivision of those five units into 17 (Table 7-1).

Table 7-1: Summary of rock units present at the CuMo property

Unit	Age	Rock Type	Texture	Grain Size
Tl	Tertiary	Lamprophyre	Porphyritic	Fine
Td	Tertiary	Diabase	Massive, amygdaloidal	Aphanitic
Tr	Tertiary	Rhyolite	Massive to flow-banded	Aphanitic to fine
TpE	Tertiary	Biotite quartz monzonite porphyry	Porphyritic	Fine
Tbx	Tertiary	Intrusion to intrusive breccia	Breccia	Aphanitic to fine
Trp	Tertiary	Biotite quartz monzonite porphyry	Porphyritic	Aphanitic to fine
TpF	Tertiary	Biotite quartz latite to rhyolite porphyry	Porphyritic	Aphanitic
TpB	Tertiary	Biotite quartz latite to rhyolite porphyry	Porphyritic	Aphanitic
TpA	Tertiary	Biotite quartz latite to quartz monzonite porphyry	Porphyritic	Aphanitic to fine
TpD	Tertiary	Biotite quartz monzonite to quartz latite porphyry	Porphyritic	Aphanitic to fine
TpC	Tertiary	Biotite quartz latite to quartz monzonite porphyry	Porphyritic	Aphanitic to fine
Tbhqmp	Tertiary	Biotite hornblende quartz monzonite porphyry	Porphyritic	Fine
Tbdp	Tertiary	Biotite dacite porphyry	Porphyritic	Aphanitic
Tgd	Tertiary	Granodiorite	Equigranular	Fine-medium
Ta	Tertiary	Andesite	Porphyritic	Aphanitic
Kg	Cretaceous	Gabbro	Equigranular – diabasic	Fine
Kqm	Cretaceous	Biotite-quartz monzonite	Equigranular to porphyritic	Coarse-medium

Baker (1983) noted that the “ranges of textures in the various dike types (TpA-TpF) overlap, but show a general trend from early, phenocryst-rich porphyries with large phenocrysts, to young, phenocryst-poor porphyries with small phenocrysts”.

Three main intrusive types were observed in the holes drilled to date, including equigranular quartz monzonite, quartz monzonite porphyry, and intrusive breccia. Mafic dikes were also intersected locally. The equigranular quartz monzonite is considered to be the Idaho batholith (Unit Kqm) and locally contains K-feldspar megacrysts. The intrusive breccia is comprised of fragments of porphyry and equigranular quartz monzonite. All of the felsic intrusive phases contain molybdenite (MoS_2) mineralization. Examples of the main rock types are shown in Figure 7-4. The quartz monzonite porphyry (Unit Tbqmp) varies considerably in proportion and size of phenocrysts, with at least four varieties recognized (Figure 7-4). The first and possibly earliest phase (Unit Tbqmp Type I) is dark to medium grey, with 10-15%, <7 mm feldspar phenocrysts, 1-2% fine-grained biotite, and <5% quartz set in a fine-grained groundmass. The second phase (Unit Tbqmp Type II) is medium to light grey, with 30% feldspar phenocrysts and minor biotite set in a medium-grained groundmass. The third phase (Unit Tbqmp Type III) is similar to Type II but contains K-feldspar megacrysts. The fourth phase and possibly most recent is a crowded porphyry variant of Type III containing >30% feldspar phenocrysts set in a medium-grained groundmass. Type I through IV phases may correlate with Amax units TpD, TpB, TpA and TpC, respectively, and appear to follow a general pattern of early, phenocryst poor phases intruded by later phenocryst-rich phases, which is opposite to the general progression observed by previous workers.

Structure may be an important factor on the distribution of mineralization at the CuMo property. A strong northeast to east-northeast structural trend, characteristic of the trans-Challis fault system, is evident in the area of the property. The Tertiary dyke system trends in this same orientation with steep to moderate dips to the south. Faults and mineralized structures identified to date dominantly trend to the northeast as well. These include numerous small base and precious metal occurrences that occur in the area and surrounding the CuMo deposit with most of the major lodes striking east-northeast (N70E) whereas subordinate lodes are oriented northeasterly (N35E, N10-20E and N30-60E). Several fault zones, marked by sections of broken core, were logged in the drilling, which appear to offset the interpreted mineral zones. The full significance of these fault structures to the deposit geometry remains to be determined.

a) Porphyry unit Tbqmp1 (Amax TpF) C40-08: 158ft

b) Porphyry unit Tbqmp2 (Amax TpC) C41-08: 376ft

c) Porphyry unit Tbqmp3 (Amax TpA) C35-08: 2505.5ft

d) Porphyry unit Tbdp C42-08: 342ft

e) Porphyry unit Tbhqmp (surface sample of Boise Basin Stock)

f) Porphyry unit Tbhqmp (DDH C36-08, 2409.5ft)

Source: Giroux et al, 2015

Note: All core pieces are 2.4 inches in width

Figure 7-4: Core photographs of felsic porphyry types recognized in drill core

7.4 Mineralization

7.4.1 Description of Mineralized Zones

The CuMo deposit is located in an historic gold mining camp. Gold was discovered in the Boise Basin in 1862 and lode mining began within a year. As of 1940, total gold production amounted 2.8 million ounces of which 74% was from placer operations (Anderson, 1947). More gold has been produced from the Boise Basin than any other mining locality in Idaho (Killsgaard et al, 1989). Although they are primarily gold deposits, considerable silver and minor copper, lead and zinc were produced as by-products from the lodes.

Anderson (1947) recognized two separate mineralizing events that he referred to as early Tertiary and early Miocene. The first event consists of gold-quartz veins containing minor sulfide minerals that occur within the Idaho batholith and are associated with weak wall rock alteration. Associated sulfide minerals include pyrite, arsenopyrite, sphalerite, tetrahedrite, chalcopyrite, galena and stibnite. The second mineralizing event occurs within porphyry dikes and stocks as well as in the batholith, and is characterized by relatively abundant sulfide mineralization, subordinate quartz and widespread wall rock alteration. Base metal mineralization consists of pyrite, sphalerite, galena, tetrahedrite, chalcopyrite, minor quartz and siderite with local occurrences of pyrrhotite and enargite. The gold-quartz veins generally occur relatively distal to the CuMo deposit (within 4 to 6 miles/6 to 10 km), whereas the base-metal-gold lodes occur in a belt that follows the “porphyry belt” from Quartzburg through Grimes Creek, proximal to and coincident with the CuMo deposit. This “porphyry belt” is what the CuMoCo refers to as the older copper-gold porphyry system which is characterized by the chalcopyrite-silver-gold bearing veins.

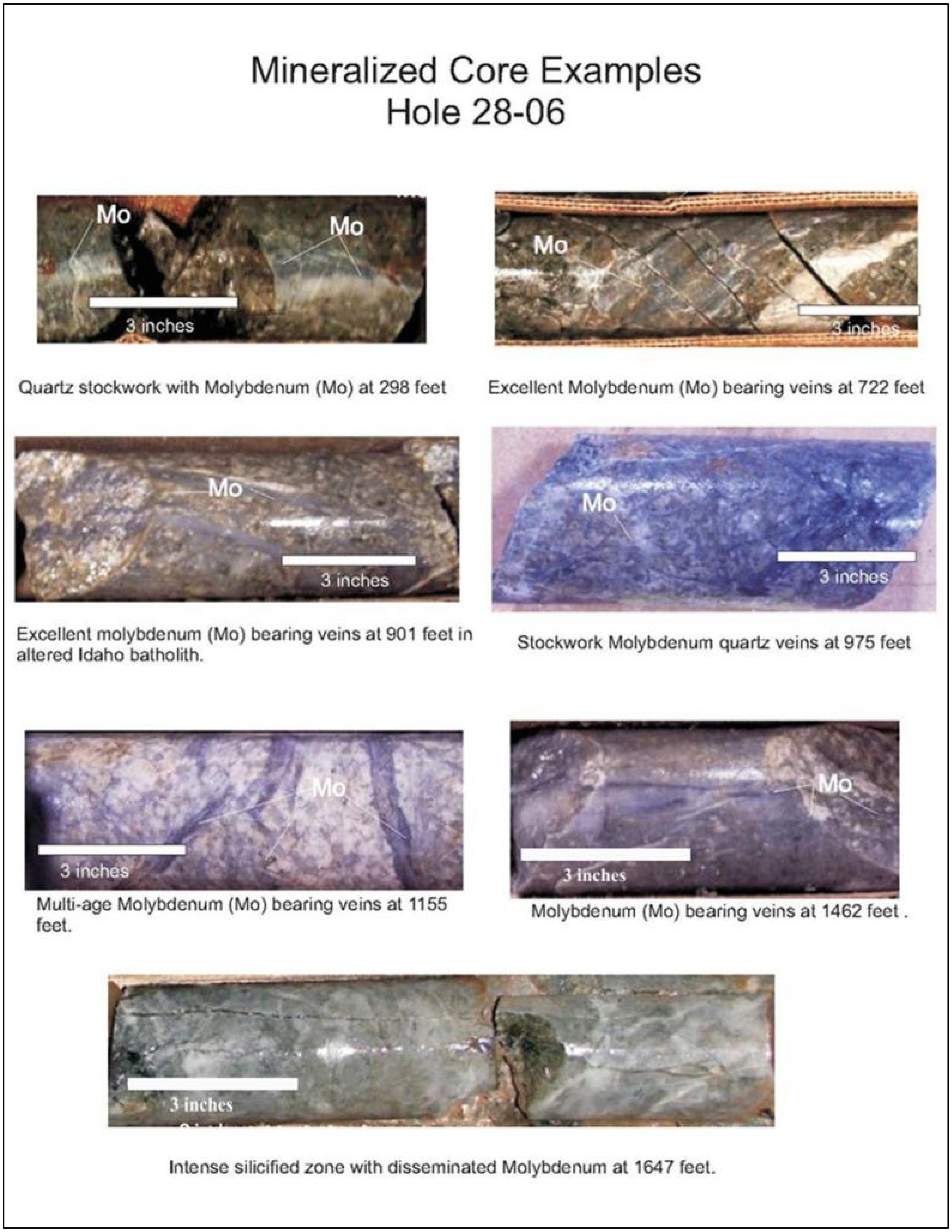
7.4.2 Property Mineralization

Molybdenum mineralization was discovered at CuMo in 1963. The only other molybdenum showing in Boise County is the Little Falls molybdenum prospect, which is situated just to the northeast of CuMo.

Mineralization on the property occurs in veins and veinlets developed within various intrusive bodies. Molybdenite (MoS_2) occurs within quartz veins, veinlets and vein stockworks. Individual veinlets vary in size from tiny fractures to veinlets five centimeters in width, with an overall thickness averaging 0.3-0.4 cm. Pyrite and/or chalcopyrite are commonly associated with molybdenite although molybdenite can occur alone without other metallic mineralization.

Chalcopyrite occurs in quartz-pyrite + molybdenite veinlets, in magnetite + pyrite as well as in pyrite-biotite + quartz + magnetite veins with secondary biotite halos. Scheelite is common on the property and closely parallels the distribution of molybdenite (Baker, 1983).

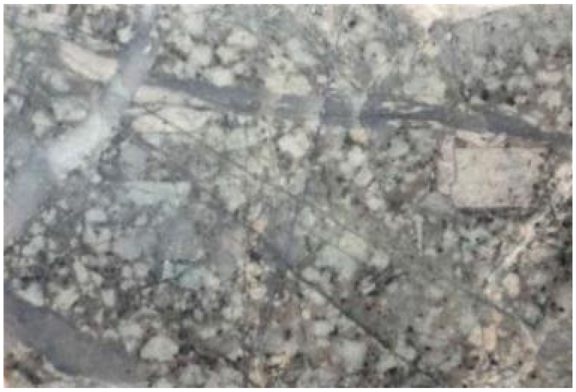
Figure 7-5 and Figure 7-6 show examples of mineralization at CuMo from the previous drill holes.



Source: Giroux et al, 2015

Note: in this older figure, “Molybdenum” is referring to molybdenite mineralization.

Figure 7-5: Photographs of mineralized core from the CuMo 2006 program, hole C06-28


a) Quartz - molybdenite veinlets in porphyry unit Tbqmp3 C35-08 (2291 ft)


b) Stockwork quartz - molybdenite veinlets in quartz monzonite unit Kqm C35-08 (2496 ft)


c) Quartz - molybdenite veinlet in intrusive breccia unit Tbx C08-37 1896.5 ft


d) Coarse molybdenite in white quartz veinlet. C36-08 (1566.5 ft)

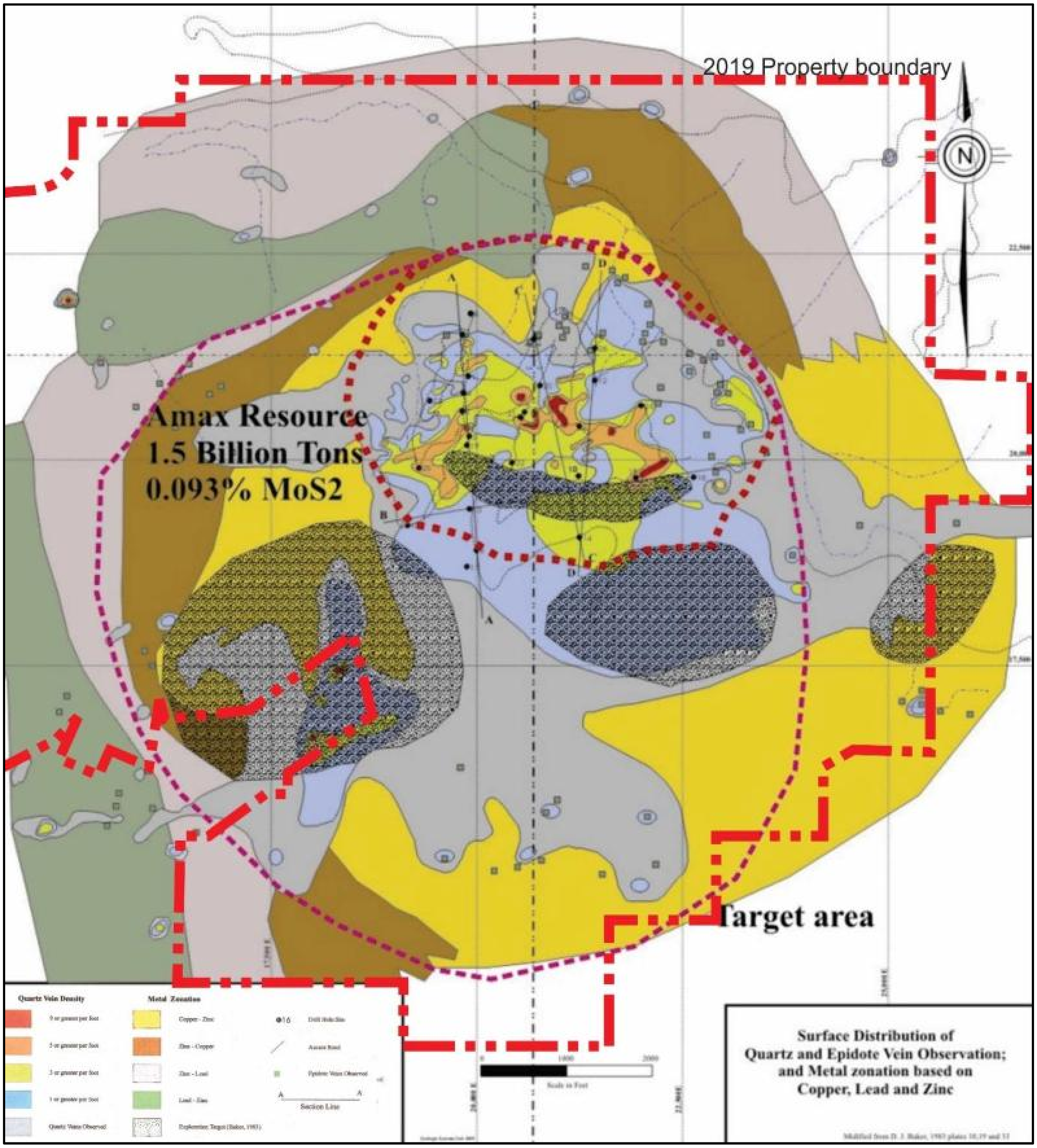
Source: Giroux et al, 2015

Note: All core pieces are 2.4 inches in width

Figure 7-6: Photographs of molybdenite mineralization in 2008 drill core

Compilation of Amax data on the frequency of veins mapped on surface as well as their mineral constituents was presented by Giroux et al (2005) and is shown graphically in Figure 7-7.

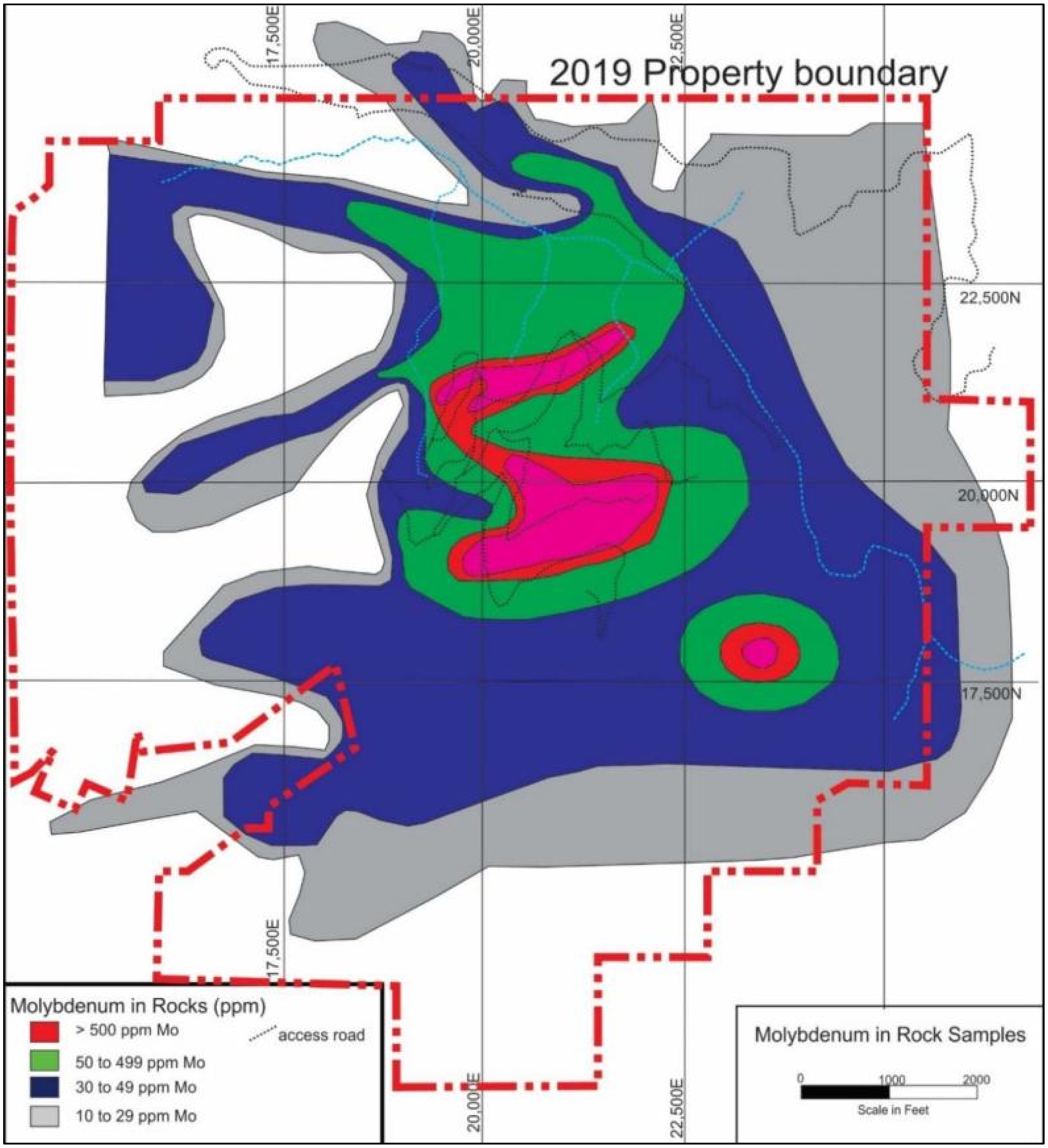
A concentric pattern is clearly evident, which is also shown by the distribution of anomalous Mo and Cu rock geochemical results – Figure 7-8 and Figure 7-9. The area drilled to date occupies only a portion of the central area; Amax had identified prospective target areas to the southeast and east of the area drilled.



Source: Giroux et al, 2015 modified 2019

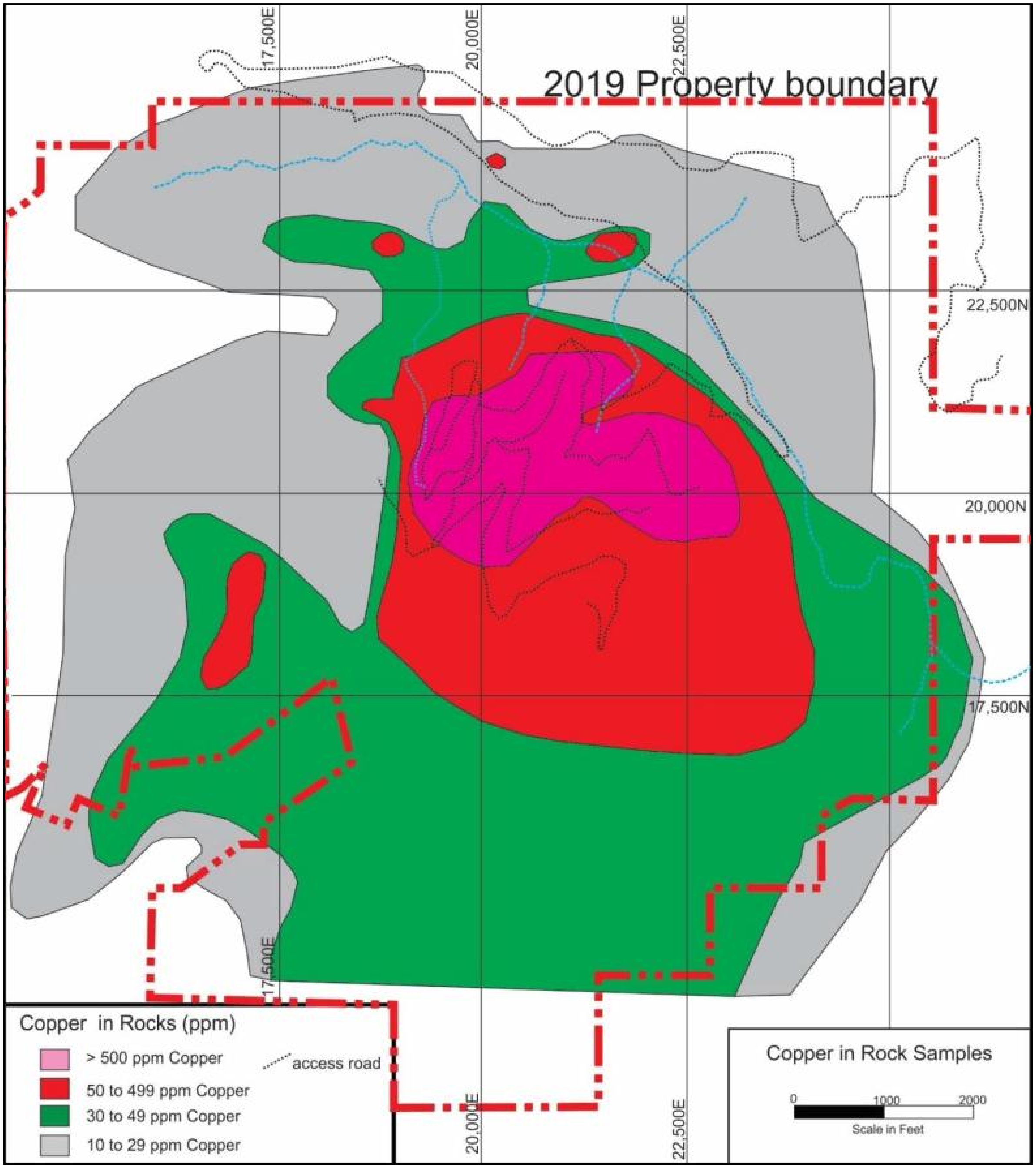
Note: The “Amax Resource” indicated on the figure is considered an historical estimate, and not a “Resource” in accordance with NI 43-101. A technical report on the property was never filed. A qualified person has not done sufficient work to classify the historical estimate as a current mineral resource. The Company does not consider the Amax resource as current.

Figure 7-7: Surface distribution of quartz and epidote veinlets and metal zonation



Source: Giroux et al, 2015 modified 2019

Figure 7-8: Geochemical distribution of Mo in surface rock chip samples



Source Giroux et al, 2015 modified 2019

Figure 7-9: Geochemical distribution of Cu in surface rock chip samples

Amax interpreted two shells of molybdenite mineralization, with the upper shell being richer in copper and silver, but of lower molybdenite (MoS_2) grade, and the lower shell being molybdenite (MoS_2)-rich and depleted in copper and silver (Baker, 1983). Amax interpreted this pattern of metal zoning to have formed above and peripheral to two or more source intrusions (of which only one was recognized physically).

8 Deposit Types

The CuMo deposit is a porphyry type deposit and has been classified as a porphyry copper molybdenum deposit (Klein, 2004; Spanski, 2004), or as a porphyry molybdenum-copper (low fluorine type) deposit (Mutchler et al, 1999).

Typical porphyry molybdenum-copper deposits are cylindrical, stock-like composite bodies having elongate outcrops 1.5 x 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 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 CuMo deposit is primarily of economic interest for its Mo content but contains significant values of Cu and Ag. Low-grade zones of copper enrichment typically form above and partially overlap with molybdenum shells in porphyry molybdenum deposits (Carten et al, 1993). The CuMo deposit is classified as a porphyry Mo-Cu deposit (%Mo greater than 0.04% and Cu being potentially economically significant).

The CuMo deposit is typical of large, dispersed, low-grade molybdenum ± copper deposits. These systems are associated with hybrid magmas typified by fluorine-poor, differentiated monzogranite igneous complexes, characteristic of continental arc terranes. Due to their larger size, the total contained potentially economic molybdenum in these types of deposits can be equivalent to or exceed that of high-grade molybdenum deposits such as Henderson or Climax (Carten et al, 1993).

9 Exploration

Since obtaining the property in 2005, work has been solely focused on drilling on the CuMo property.

Only reportable exploration conducted by CuMoCo outside of drilling on the CuMo property was a dump sample taken during reconnaissance work undertaken by the project geologists in 2017. The dump is located approximately 1,731 m west of the western most drill hole to date, hole 10-47, The sample although taken to represent the material occurring on the dump (Coon Dog, located in Figure 10-1) which was estimated to contain 15 tons of material, the sample is considered a grab and not representative of any sort of size, width or extent of material. The sample which was assayed by ALS Chemex using ICP-M61 technique assayed 3.12% Cu, 783 ppm Ag, and 0.986 ppm Au.

Note: *The reader is cautioned that grab sample assays represent prospecting samples and may not be representative of the grade or width of the mineralization. There is presently insufficient data with respect to the size and extend of the mineralization represented by the sample to determine its significance. Future drilling is designed to determine that significance, if any.*

Note: Sample was taken by collecting approximately 10 kg of random broken rocks pieces from the area of the dump

The reconnaissance worked involved two geologists examining surface exposures looking for additional indications of mineralization. Several indications were identified including the Coon Dog Dump. The work resulted in an increase to the extent of mineralization (deposit outline) as can be seen in Figure 4-2. A drill program for the area has been proposed for the 2020 field season.

The reader will encounter several outlines of various aspects of the CuMo project that are explained in the pertinent section, but for clarity are summarized here.

The first outline is the deposit or mineralized outline, this is shown in Figure 4-2 (Mineralized outline) and represents the extent of the CuMo deposit based on the geology, alteration and mineralization. It is the largest and most extensive boundary.

The next outline encountered is the conceptual pit or 2015 block model boundary, this is the outline of the location of all blocks that are placed around the drill holes that are within a conceptual pit design. As drilling proceeds, more and more of these blocks are converted into resources.

The next outline is the 2015 resource boundary (resource outline in figure 4-2) , this outlines the area of blocks that were actually calculated in the current 2015 resource; it amounts to 60% of the previous block model boundary. Reader should note that not all blocks within the block boundary have been actually calculated.

The final boundary is the actual 30-year pit boundary (2019 pit outline in figure 4-2) that contains the blocks within the 2015 resource that are proposed to be mined during the 30 years.

10 Drilling and Trenching

10.1 Summary of Programs by Year

Between 2006 and 2012, CuMoCo has drilled a total of 25,486.82 m in 42 holes (Table 10-1).

Table 10-1: Summary of holes drilled by CuMoCo

Year	No Holes	Length (m)
2006	1	1,085.1
2007	7	3,872.5
2008	11	8,159.7
2009	9	5,687.8
2010	3	1,312.8
2011	2	1,156.7
2012	9	4,213.3
Total	**42**	**25,487.9**

10.2 Sampling and True Thickness Adjustments

All drill holes completed by CuMoCo were sampled at 10 ft intervals for the entire hole. The deposit is a stockwork type. No preferred orientation of veins has been identified. No systemic adjustment of sampling intervals or intercept lengths to reflect “true thickness” has been applied, nor is it considered warranted.

10.3 2006 Drill Program

In 2006, diamond drilling was done by Kettle Drilling Inc. of Coeur d’Alene on behalf of Kobex Resources Ltd. and CuMoCo Resources Corp. Kobex commenced drilling in August 2006 and completed one hole. On October 6, 2006, Kobex delivered a notice of termination in respect of the CuMo property. The option on the project was terminated when the second hole was at a depth of 600 ft, and the action was taken before any assays were received. ICMC (wholly owned US subsidiary of CuMoCo.) assumed control of the project on October 10, 2006 and completed this hole to a depth of 1,710 ft before the program was halted due to the onset of winter conditions.

10.4 2007 to 2011 Drill Program

Between 2007 and 2011, diamond drilling was done by Kirkness Drilling (Kirkness) of Carson City, Nevada. Kirkness drilled 33 diamond drill holes. Table 10-2 provides details of the drilling undertaken from 2006 to 2011.

Table 10-2: Summary of 2006 to 2011 diamond drilling at CuMo

Hole	Northing	Easting	Elevation	Dip	Azimuth	Length (ft)
27-06	120,032	220,208	6,351	-90	0	1,849 completed
28-06	119,540	220,817	6,321	-90	0	1,716 completed
29-07	119,779	221,247	6,344	-70	140	2,281.7 completed
30-07	119,732	219,617	6,213	-90	0	2,411.5 completed
31-07	119,792	221,243	6,342	-70	45	2,104 completed
32-07	119,558	220,823	6,324	-70	190	2,044 completed
33-07	118,477	221,227	6,797	-90	0	2,095 stopped
34-07	118,658	220,487	6,534	-70	95	1,769 stopped
35-08	118,655	220,480	6,533	-90	0	2,817 completed
36-08	119,335	219,449	6,275	-90	0	2,488 completed
37-08	119,780	221,247	6,341	-70	335	2,195 completed
38-08	118,655	220,480	6,533	-70	180	2,441 completed
39-08	118,918	220,813	6,575	-90	0	2,688 completed
40-08	119,530	220,791	6,321	-70	225	2,252 completed
41-08	119,630	218,962	6,220	-90	0	3,018 completed
42-08	118,749	219,911	6,549	-70	270	2,707 stopped (winter)
43-08	120,613	220,053	6,174	-80	40	1,308 stopped by fault
44-08	118,085	221,516	6,739	-65	75	3,047 completed
45-08	119,802	218,821	6,184	-80	330	1,796 stopped (winter)
46-09	118,914	220,811	6,575	-75	110	959 stopped
47-09	120,687	219,422	5,833	-90	0	2,530 completed
48-09	120,690	219,425	5,826	-70	305	2,576 completed
49-09	119,095	221,746	6,645	-90	0	2,847 completed
50-09	121,548	219,844	5,833	-75	270	1,826 completed
51-09	121,535	219,860	5,829	-90	0	1,593.5 completed
52-09	118,500	221,251	6,791	-75	20	2,772 completed
53-09	119,804	218,831	6,183	-75	15	2,461 completed
54-09	119,535	219,005	6,196	-75	15	1,096 completed
55-10	117,560	218,422	6,724	-65	0	2,479 completed
56-10	117,560	218,422	6,724	-65	305	1,294 completed
57-10	117,559	218,422	6,724	-90	0	534 stopped (winter)
58-11	119,143	219,970	6,451	-90	0	1,885 completed
59-11	119,096	221,746	6,645	-75	0	1,910 completed

Note: Hole 27-06 was started in 2006 and completed in 2007. With footage recorded in Table 10-1 in both 2006 and 2007.

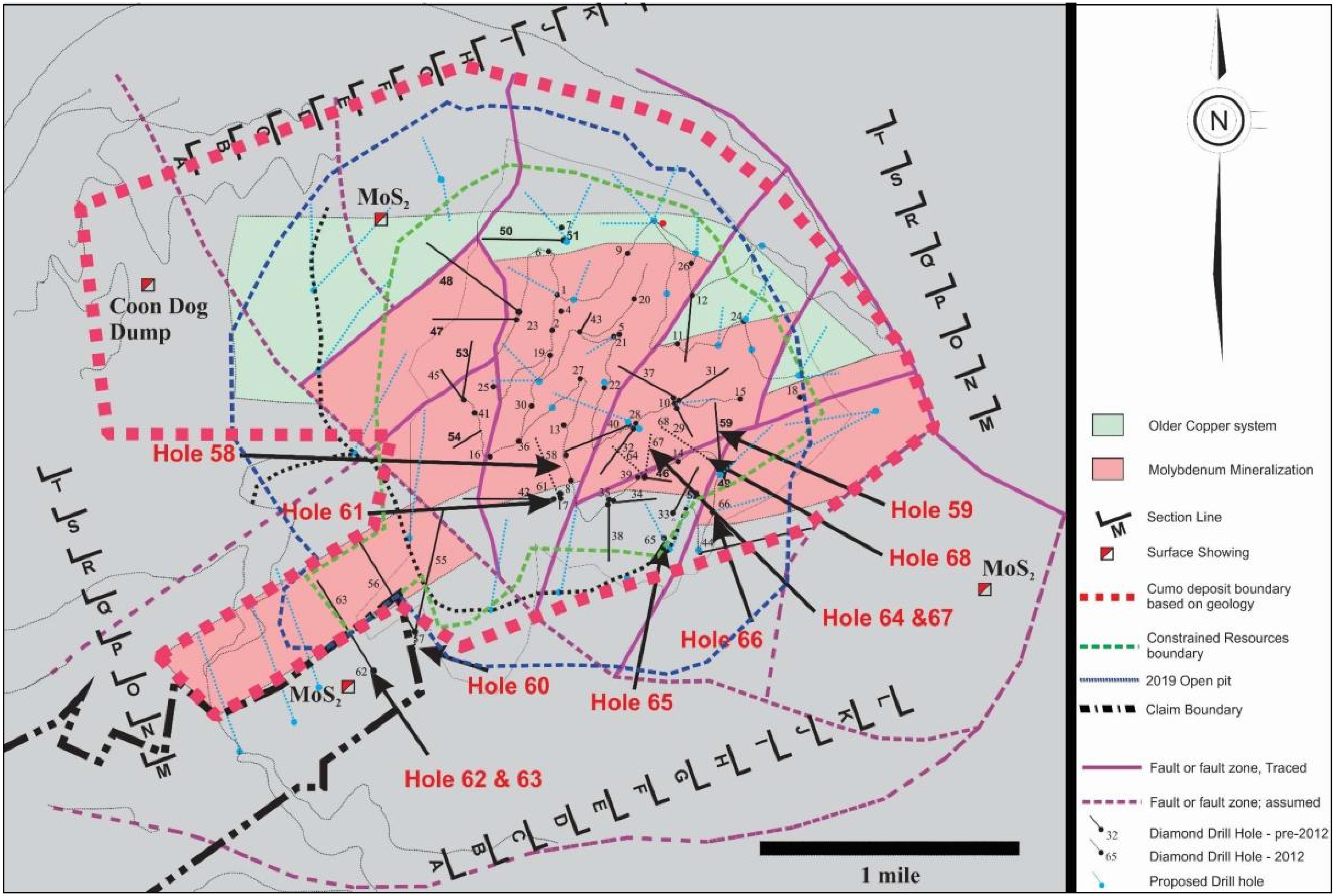
All CuMoCo drilling programs were directly supervised by onsite geology staff located in Garden Valley, Idaho. Drilling consisted of both HQ and NQ diameter core with holes being started with HQ diameter and then reducing at a major fault intersection or at 1000 feet which ever was less. Core recoveries were monitored and were excellent (90%+)

All CuMoCo holes were surveyed down-the-hole at regular intervals (100 feet) using a Reflex survey instrument.

All core was collected at the drill site by the diamond drillers under supervision of onsite geology staff and delivered to the secure warehouse facility in Garden Valley where they were logged, analyzed and samples collected. All drill sites were surveyed using a total field station in order to accurately locate the holes. Section 11.1 gives more details on the sampling procedures and core box handling methods employed.

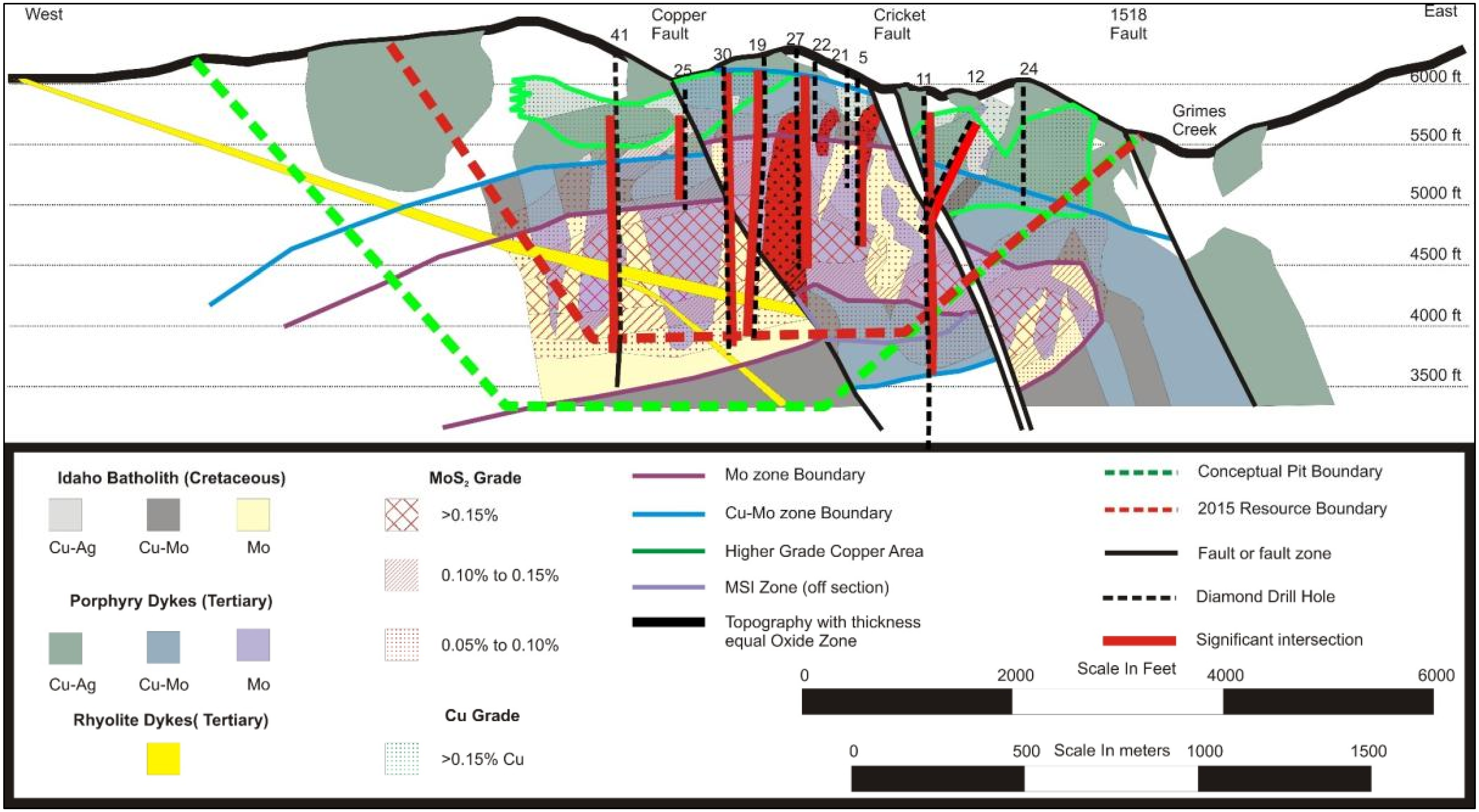
Figure 10-1 shows the locations of all holes drilled to date in the deposit, as well as the future locations of the 33 drill-holes proposed in the recommendations in Section 26 of this report. Figure 10-2 and Figure 10-3 show typical sections through the deposit.

A summary of significant intersections for all the CuMo drilling undertaken by CuMoCo is given in Table 10-3.



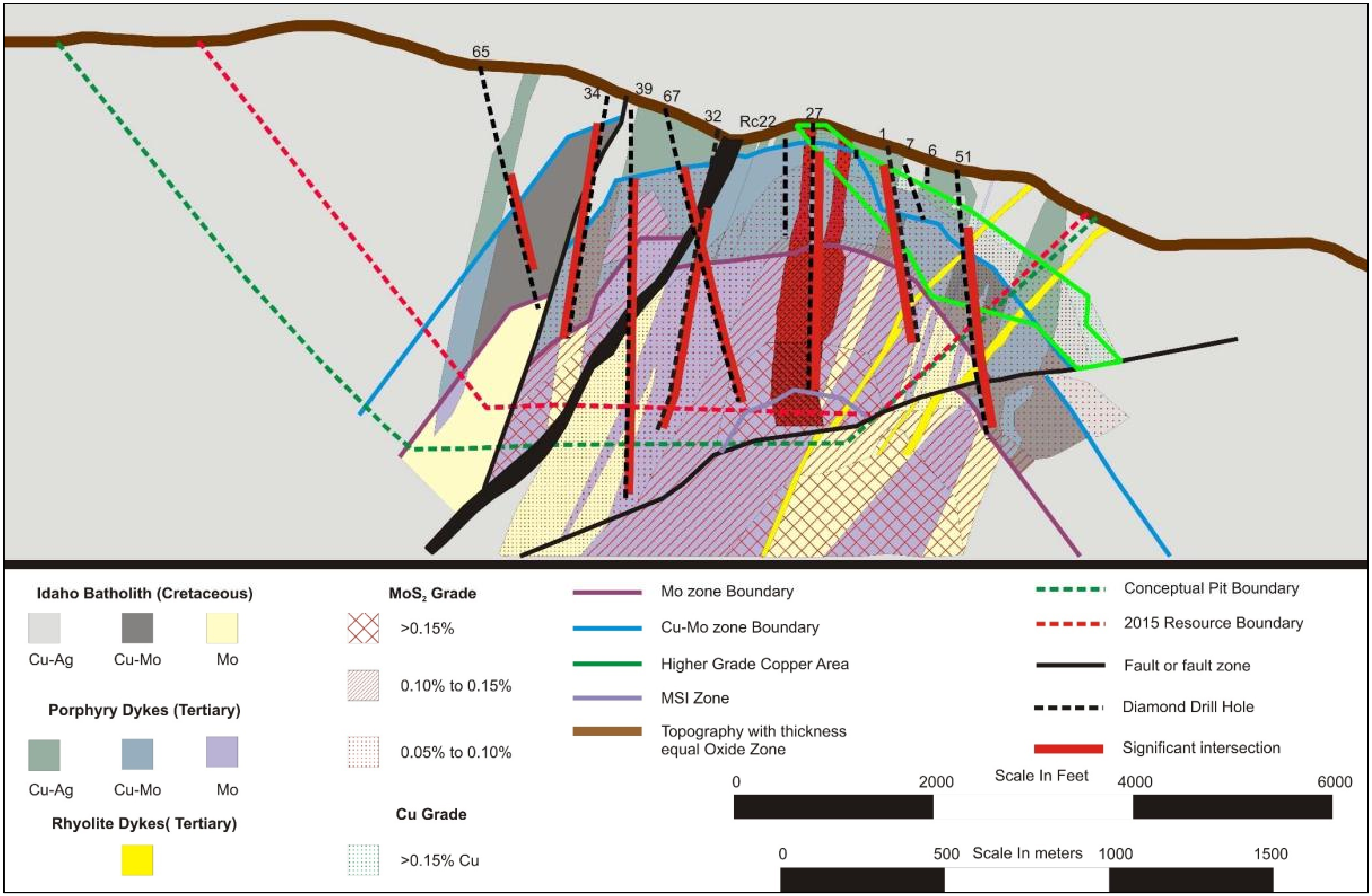
Source: Giroux et al, 2015 modified 2019

Figure 10-1: Map showing the location of completed and proposed drill holes



Source: Giroux et al, 2015 modified 2019

Figure 10-2: CuMo deposit Q-Q cross section



Source: Giroux et al, 2015 modified 2019

Figure 10-3: CuMo deposit H-H cross section

Table 10-3: Significant intersections from CuMo drilling

Hole (Name)	From (meters)	To (meters)	Length (meters)	Zone	Cu equiv. %	MoS_2 equiv. %	MoS_2 (%)	Cu (%)	Ag (ppm)	Re (ppm)	W (ppm)
C71-01	70.4	574.2	503.8	main	0.38	0.088	0.059	0.12	2.59	0.00	46
C71-01	118.9	143.3	24.4	Incl.	0.53	0.122	0.099	0.14	2.56	0.00	44
C71-01	518.2	574.2	56.1	Incl.	0.49	0.114	0.100	0.08	1.21	0.00	54
C72-05	137.2	431.6	294.4	main	0.43	0.099	0.060	0.13	4.46	0.00	75
C74-09	140.2	245.2	105.0	main	0.54	0.126	0.077	0.12	7.16	0.00	71
C75-10	67.1	658.4	591.3	main	0.47	0.109	0.099	0.05	1.43	0.00	48
C76-11	42.7	740.1	697.5	main	0.36	0.084	0.074	0.05	1.55	0.00	36
C76-11	396.2	597.4	201.2	Incl.	0.55	0.128	0.127	0.03	0.77	0.00	58
C76-12	29.9	435.9	405.9	main	0.25	0.058	0.041	0.06	1.66	0.00	45
C77-13	207.3	549.9	342.6	main	0.51	0.119	0.111	0.05	1.98	0.00	49
C77-14	237.7	647.3	409.6	main	0.53	0.124	0.114	0.06	1.84	0.00	65
C77-14	365.8	597.4	231.6	Incl.	0.68	0.158	0.151	0.06	1.91	0.00	74
C77-15	182.9	589.2	406.4	main	0.53	0.123	0.113	0.06	1.73	0.00	57
C77-15	384.0	573.0	189.0	Incl.	0.64	0.150	0.153	0.02	0.75	0.00	69
C78-16	304.8	649.7	344.9	main	0.44	0.102	0.093	0.04	1.86	0.00	32
C78-17	353.6	695.4	341.8	main	0.37	0.086	0.064	0.08	2.55	0.00	40
C78-18	426.7	719.6	292.9	main	0.62	0.144	0.129	0.08	2.71	0.00	41
C79-19	36.6	694.9	658.4	main	0.51	0.118	0.101	0.08	2.27	0.00	49
C79-20	50.3	548.6	498.3	main	0.43	0.099	0.069	0.11	3.83	0.00	52
C81-25	57.9	308.2	250.2	main	0.43	0.101	0.070	0.13	2.42	0.00	58
C81-25	225.6	308.2	82.6	Incl.	0.53	0.124	0.090	0.14	2.98	0.00	84
C81-26	9.1	228.6	219.5	main	0.41	0.094	0.034	0.18	7.58	0.00	28
C06-27	36.6	563.6	527.0	main	0.42	0.097	0.084	0.06	1.60	0.02	49
C06-27	329.2	563.6	234.4	Incl.	0.58	0.136	0.133	0.04	0.99	0.04	59
C06-28	15.2	515.1	499.9	main	0.47	0.110	0.097	0.07	1.92	0.05	54
C06-28	256.0	378.0	121.9	Incl.	0.70	0.162	0.162	0.03	0.98	0.09	68
C07-29	57.9	679.7	621.8	main	0.52	0.121	0.103	0.08	2.13	0.05	53
C07-29	359.7	545.6	185.9	Incl.	0.74	0.171	0.169	0.04	1.2	0.08	37
C07-30	12.2	727.3	715.1	main	0.52	0.122	0.108	0.06	2.05	0.04	41
C07-30	359.7	605.9	246.3	Incl.	0.80	0.187	0.185	0.04	1.46	0.07	37
C07-31	6.7	641.3	634.6	main	0.34	0.079	0.064	0.07	1.76	0.02	43
C07-31	237.7	469.4	231.6	Incl.	0.40	0.092	0.081	0.05	1.45	0.03	45
C07-32	6.7	641.3	634.6	main	0.55	0.129	0.109	0.09	2.26	0.04	61
C07-32	237.7	469.4	231.6	Incl.	0.65	0.151	0.129	0.10	2.62	0.05	77
C07-33	220.0	638.3	418.2	main	0.20	0.048	0.026	0.07	2.01	0.01	48
C07-33	603.5	638.3	34.7	Incl.	0.48	0.111	0.084	0.10	2.68	0.03	67
C07-34	42.7	539.2	496.5	main	0.25	0.058	0.034	0.08	2.30	0.01	53
C07-34	472.4	539.2	66.8	Incl.	0.41	0.096	0.074	0.09	2.36	0.02	67
C08-35	36.6	804.7	768.1	main	0.31	0.072	0.057	0.06	1.73	0.02	37
C08-35	128.0	804.7	676.7	Incl.	0.33	0.077	0.062	0.07	1.69	0.02	39
C08-35	527.3	804.7	277.4	Incl.	0.43	0.100	0.089	0.05	1.37	0.03	35

Hole (Name)	From (meters)	To (meters)	Length (meters)	Zone	Cu equiv. %	MoS_2 equiv. %	MoS_2 (%)	Cu (%)	Ag (ppm)	Re (ppm)	W (ppm)
C08-36	170.7	758.3	587.7	main	0.43	0.100	0.088	0.05	1.42	0.03	34
C08-36	280.4	758.3	477.9	Incl.	0.39	0.090	0.103	0.04	1.04	0.03	33
C08-37	18.3	669.0	650.7	main	0.43	0.100	0.084	0.05	1.67	0.03	42
C08-37	237.7	649.2	411.5	Incl.	0.40	0.094	0.104	0.02	1.17	0.04	41
C08-38	51.8	744.0	692.2	main	0.46	0.106	0.029	0.06	4.40	0.00	32
C08-39	94.5	819.3	724.8	main	0.24	0.056	0.099	0.06	1.38	0.03	52
C08-39	274.3	728.5	454.2	Incl.	0.47	0.109	0.122	0.04	1.09	0.04	57
C08-40	18.3	686.4	668.1	main	0.54	0.127	0.115	0.06	3.79	0.04	46
C08-40	118.9	634.0	515.1	Incl.	0.57	0.133	0.129	0.06	4.27	0.05	45
C08-40	338.3	554.7	216.4	Incl.	0.64	0.150	0.142	0.04	7.78	0.06	45
C08-41	259.1	862.6	603.5	main	0.75	0.173	0.067	0.08	2.23	0.02	43
C08-41	454.2	618.7	164.6	Incl.	0.38	0.088	0.107	0.08	2.99	0.03	38
C08-41	759.0	862.6	103.6	Incl.	0.56	0.129	0.077	0.06	1.53	0.03	34
C08-42	167.6	825.1	657.5	main	0.38	0.089	0.044	0.06	5.81	0.01	25
C08-42	289.6	825.1	535.5	Incl.	0.33	0.077	0.047	0.07	6.78	0.01	27
C08-42	600.5	825.1	224.6	Incl.	0.36	0.084	0.063	0.05	1.61	0.01	21
C08-43	50.3	397.2	346.9	main	0.32	0.075	0.044	0.09	4.23	0.02	52
C08-43	201.2	249.9	48.8	Incl.	0.48	0.053	0.07	0.11	3.14	0.03	45
C08-44	342.9	865.6	522.7	main	0.71	0.078	0.03	0.02	0.89	0.01	29
C08-44	780.3	819.9	39.6	Incl.	0.15	0.035	0.06	0.02	1.47	0.01	20
C08-45	51.8	547.4	495.6	main	0.27	0.062	0.02	0.15	3.08	0.00	42
C08-45	307.8	547.4	239.6	Incl.	0.27	0.062	0.03	0.18	3.05	0.00	40
C09-46	91.4	292.3	200.9	main	0.33	0.077	0.03	0.09	2.61	0.01	55
C09-47	88.4	529.3	440.9	main	0.27	0.062	0.07	0.18	4.29	0.02	20
C09-47	292.6	865.6	573.0	main	0.42	0.097	0.05	0.18	5.03	0.02	20
C09-48	463.3	737.6	274.3	Incl.	0.40	0.094	0.08	0.05	1.70	0.03	17
C09-49	246.9	464.7	217.8	main	0.38	0.087	0.11	0.06	1.91	0.04	17
C09-49	158.5	478.5	320.0	main	0.48	0.112	0.03	0.15	5.29	0.01	20
C09-50	271.3	823.0	551.7	main	0.31	0.072	0.04	0.15	4.86	0.02	19
C09-51	545.6	804.7	259.1	Incl.	0.34	0.080	0.09	0.07	1.69	0.03	18
C09-52	243.8	753.2	509.3	main	0.43	0.100	0.14	0.05	1.29	0.06	17
C09-52	460.2	753.2	292.9	Incl.	0.63	0.147	0.09	0.19	4.07	0.02	18
C09-53	179.5	334.1	154.5	main	0.42	0.098	0.12	0.15	3.68	0.03	19
C09-53	70.1	128.0	57.9	main	0.49	0.113	0.11	0.05	1.69	0.03	17
C09-54	362.7	365.8	3.0	Incl.	0.20	0.045	0.03	0.07	35.44	0.00	21
C10-55	67.1	152.4	85.3	main	0.25	0.057	0.04	0.01	0.42	0.01	21
C10-55	91.4	149.4	57.9	main	0.49	0.071	0.07	0.02	3.80	0.02	21
C10-56	67.1	152.4	85.3	main	0.15	0.035	0.03	0.01	0.01	0.01	25
C10-57	91.4	149.4	57.9	main	0.35	0.082	0.07	0.02	0.02	0.02	18

Note: description of how the equivalent values are calculated is provided in Section 10.6 below

The 2006-2011 results confirmed the extent and grade of mineralization on the property as indicated by previous drilling and demonstrated continuity of mineralization between the original wide-spaced holes.

The 2006-2011 drilling data supports the presence of three distinct mineralized zones within the deposit. Amax previously interpreted these zones as distinct shells that were produced by separate intrusions. Re-interpretation of the geology, alteration and down-hole histograms for Cu, Ag and Mo have confirmed the mineralized zones are a part of a single, large, concentrically zoned system with an upper Cu-Ag Zone, underlain by a transitional Cu-Mo Zone, in turn underlain by a lower molybdenum-rich Mo Zone (Figure 10-2).

10.5 2012 Drill Program

In 2012, a total of 4,213.3 m (15,463 ft) in nine holes were completed (Table 10-4). The holes were located to infill gaps in the existing drilling coverage and were drilled along existing tracks and roads. All holes were surveyed down-the-hole at regular intervals using a Reflex survey instrument. Co-ordinates, elevations and lengths are in feet.

Table 10-4: Summary of 2012 diamond drilling

Hole	Year	Easting	Northing	Elevation	Dip	Azimuth	Length	Comment
12-60	2012	218,422	117,560	6,724	-50	180	1455	Completed
12-61	2012	219,911	118,749	6,549	-75	335	1318	Stopped
12-62	2012	218,041	116,866	6,629	-50	135	1484	Completed
12-63	2012	218,042	116,867	6,629	-60	330	807	Completed
12-64	2012	220,811	118,914	6,575	-75	25	2139	Completed
12-65	2012	221,118	118,149	6,786	-80	315	1908	Completed
12-66	2012	221,688	118,674	6,690	-90	0	2241	Completed
12-67	2012	220,811	118,914	6,575	-70	340	1978	Completed
12-68	2012	221,746	119,096	6,645	-70	310	2134	Completed

A summary of significant intersections for all the CuMo drilling undertaken by CuMoCo is given in Table 10-5 and Table 10-7.

Table 10-5: Significant intersections from 2011-2012 CuMo drilling

Hole Name	From (metres)	To (metres)	Length (metres)	Zone	MoS_2 (%)	Cu (%)	Ag (ppm)	Re (ppm)	W (ppm)
C11-58	213.4	574.5	361.2	main	0.08	0.07	0.07	0.03	41
C11-59	152.4	582.2	429.8	main	0.07	0.13	0.13	0.02	109
C12-60	70.1	118.9	48.8	main	0.05	0.02	0.02	0	7
C12-61	121.9	401.4	279.5	main	0.03	0.11	0.11	0.01	28
C12-62	No significant intersections: hole drilled away from deposit								
C12-63	184.4	189	4.6	main	0	0.21	130.6	0	7
C12-64	91.4	667.5	576.1	main	0.08	0.07	1.77	0.03	47
C12-64	301.8	573	271.3	Incl.	0.12	0.07	1.6	0.04	59
C12-65	167.6	478.5	310.9	main	0.02	0.05	1.23	0.01	44
C12-66	121.9	401.4	279.5	main	0.02	0.06	1.58	0	40
C12-66	163.1	401.4	238.4	Incl.	0.02	0.07	1.69	0	45
C12-67	173.7	600.5	426.7	main	0.1	0.09	2.11	0.04	56
C12-67	277.4	600.5	323.1	Incl.	0.12	0.08	1.66	0.05	61
C12-68	277.4	548.6	271.3	main	0.1	0.08	1.85	0.04	73
C12-68	402.3	548.6	146.3	Incl.	0.13	0.07	1.77	0.06	65

Note: The convention for the CuMo project has been to measure percent elemental molybdenum (%Mo) in assays and to calculate %MoS_2 by multiplying %Mo by 1.6681.

Table 10-6: Recoverable equivalent grades for significant intersections from 2011-2012 CuMo drilling

Hole Name	Length (metres)	RecG* MoS_2 equiv. (%)	RecG* Cu equiv. (%)
C11-58	361.2	0.100	0.43
C11-59	429.8	0.125	0.54
C12-60	48.8	0.068	0.29
C12-61	279.5	0.061	0.27
C12-62	No significant intersections: hole drilled away from deposit		
C12-63	4.6	0.556	2.39
C12-64	576.1	0.097	0.41
C12-64	271.3	0.130	0.56
C12-65	310.9	0.031	0.14
C12-66	279.5	0.031	0.14
C12-66	238.4	0.034	0.15
C12-67	426.7	0.119	0.51
C12-67	323.1	0.130	0.56
C12-68	271.3	0.112	0.48
C12-68	146.3	0.143	0.61

* - RecG = Recoverable grades expressed as recoverable equivalent-metal grades (Section 10.6).

Notes: These values are NOT additive and are simply different ways of expressing the poly-metallic material in terms of recoverable equivalent grade. Each value reflects all relevant metal grades in the intersections.

The convention for the CuMo project has been to measure percent elemental molybdenum (%Mo) in assays and to calculate %MoS_2 by multiplying %Mo by 1.6681.

10.6 Metal Equivalent Calculations

Because of the multi-element nature of the mineralization and mineral zoning, it was decided to calculate both a copper and molybdenum equivalent for the intercepts. The following outlines the calculations involved:

Metal equivalents for mineral equivalent calculations were based on metal prices outlined in Table 10-7.

Table 10-7: Metal prices used to calculate copper and molybdenum equivalent

Metal	Price ($US)	Unit
Copper	2.50	lb
Molybdenum trioxide	10.00	lb
Silver	0.35	ppm

Estimated metallurgical recoveries used in the calculations are outlined in Table 10-8.

Table 10-8: Metallurgical recoveries used to calculate copper and MoS_2 equivalent

Mineral Zone	Mo%	Cu%	Ag %
OX	80	60	65
Cu-Ag	86	68	75
Cu-Mo	92	85	78
Mo	95	72	55
MSI	95	72	55

Recovery (Rec) is taken from the above table for each assay in a particular mineral zone and applied in the following formula to derive the equivalents:

$$\%Cu\ Equiv. = (\%Cu \times 20 \times \$(Cu) \times Rec(Cu) + \%MoS_2 \times 20 \times \$(MoO_3) \times (1.5/1.6681) \times Rec(Mo) + Ag \times \$(Ag) \times Rec(Ag)) \,/\, (\$(Cu) \times Rec(Cu) \times 20)$$

$$\%MoS_2\ Equiv. = (\%Cu \times 20 \times \$(Cu) \times Rec(Cu) + \%MoS_2 \times 20 \times \$(MoO_3) \times (1.5/1.6681) \times Rec(Mo) + Ag \times \$(Ag) \times Rec(Ag)) \,/\, (\$(MoO_3) \times Rec(Mo) \times 20 \times 1.5/1.6681)$$

Note that since the convention on the CuMo project has been to work with $\%MoS_2$ for resource estimation, in the foregoing equivalency formulae, %MoS2 is converted back to %Mo by dividing by 1.6681. %Mo is then converted to $\%MoO_3$ by multiplying by 1.5. Also, the $\%MoS_2$ Equiv values would be 1.6681 times greater than %Mo Equiv.

Table 10-9: Terms used in formulae for equivalent grade calculations

Term	Definition
%Cu	Copper grade in %
$(Cu)	Copper price per pound
Rec(Cu)	Copper recovery
%MoS_2	Molybdenum disulfide (molybdenite) grade in %
$($MoO_3$)	Molybdenum oxide price per pound
Rec(Mo)	Molybdenum recovery
Ag	Silver grade in ppm
$(Ag)	Silver price per gram
Rec(Ag)	Silver Recovery
%Cu. Equiv.	Copper equivalent in-situ grade
%MoS_2 Equiv.	Molybdenite equivalent in-situ grade

Note: Only molybdenum (as grade of MoS_2 and conventional pricing of MoO_3), copper and silver are used in the equivalent calculations.

11 Sample Preparation, Analyses, and Security

The QP has reviewed the procedures followed by CuMoCo and by third parties on behalf of CuMoCo, and believes these procedures are consistent with industry best practices and acceptable for use in geological and resource modelling.

11.1 General sampling

Sampling was restricted during 2006 to 2012 to diamond drill hole (DDH) core and metallurgical sampling of previously drilled DDH core. Standard core sampling methods were employed for both drill core and metallurgical samples.

At the time of drilling, each core box was clearly labeled by the driller's helper with the DDH hole number, core box number, and "to" and "from" drill core footages. Wooden core boxes were used at all times, and full core boxes were sealed with a lid. The driller(s) and/or geologist(s) then delivered the core boxes to the secure core storage warehouse located in Garden Valley, Idaho. The core boxes were laid out in sequence upon long tables specifically made for core logging purposes. A geologist then logged the core for lithology, structure, alteration and mineralization. Geotechnical measurements for RQD were recorded. Each core box was additionally labelled using a metal Dymo® labelling tool for long-term preservation of identification. The core was photographed, two boxes at a time, using a mounted Nikon digital camera. It was then delivered to the core-cutting technician. The photographs were downloaded onto computer files specific to each drill hole.

A core technician using a standard rock saw sampled the core using typical procedures. Half-core was collected at regular 10 ft intervals for analysis. Sample lengths were adjusted to lithological contacts in cases where barren dikes were intersected.

Half core sample intervals were placed in ether cloth or heavy plastic sample bags with the sample number placed on the outside of the bag in black permanent marker. Individual sample interval tags were included in each sample bag. The bag was then secured with a wire tie and placed within a plastic transport crate for shipping.

MoS_2 loss from soft fracture fillings being washed away when the core was sawed in half have been noted at CuMo. Although there is no physical way to eliminate this problem at present, other than schooling the technicians on the extra care needed when sawing a soft fracture zone, geologists at CuMo have addressed possible inadvertent contamination of other core from MoS_2 enriched water from the rock saw's water recirculation tank. The cut core was given a second clear water bath prior to being bagged or stored and the recirculation tank was voided and refilled based upon clarity.

The half core was sent for analysis, and the other half was retained and stored at the core storage warehouse in Garden Valley, Idaho. The retained core was replaced in their original core boxes which were sealed with a plywood cover and stacked upon a standard pallet. Each plywood cover was clearly labelled with the core's information. The pallet was then strapped with a metal banding tool and stored within the archive section of the core storage warehouse in Garden Valley, Idaho.

Blanks and standards were inserted into the sample stream at a frequency of one every 20 samples. The core-cutting technician selected the exact intervals and noted them on the sample logs. The core technician inserted the blanks whereas the standards were selected and inserted by the geologist-in-charge.

Standards were selected from three bulk standards (low, medium and high grade) that were prepared from historic CuMo drill core samples. Standards were selected on the basis of appropriate grade to match the estimated grade of the core adjacent to each standard sample interval.

The standards were prepared and packaged by CDN Resource Laboratories Ltd. (CDN Labs) of Surrey, British Columbia. Each bulk sample was pulverized in a large rod mill, screened through 200 mesh using an electric sieve, and homogenized in a large rotating mixer. Each standard was sealed in plastic to prevent gravity separation and oxidation. The standards were certified by Smee & Associates Consulting Ltd. of North Vancouver, British Columbia, based on round-robin analysis at five laboratories using a four-acid digestion and ICP-ES finish (Table 11-1). Note that the Mo ppm values reflect measured Mo metal (not MoS_2).

Table 11-1: Certified standards prepared for CuMo project

Standard	Element	Certified Mean	Standard Deviation (between lab)
Standard 1	Total Cu	1138 ppm	65 ppm
	Total Mo	367 ppm	19 ppm
Standard 2	Total Cu	151 ppm	8 ppm
	Total Mo	995 ppm	41 ppm
Standard 3	Total Cu	840 ppm	35 ppm
	Total Mo	54.0 ppm	3.7 ppm

The bagged core samples were string or wire tied and then stored temporarily in holding pallets at the core storage warehouse in Garden Valley. When enough samples were accumulated, the samples were delivered by CuMoCo personnel to ALS-Chemistry (ALS Chemex) in Elko, Nevada for preparation and analysis.

11.2 Density Determinations

Historical specific gravity determinations were made by Amax for CuMo for each mineralized zone. The measurements were made using the weight in air/weight in water procedure by Skyline Laboratories of Colorado. CuMoCo, prior to 2012, had occasional density measurements at ALS-Chemex's lab.

In 2012, CuMoCo initiated a regular density measurement program where 4 to 6-inch skeletons of half-cores from each sample interval that are representative of the 10 ft interval are analyzed. The following equipment was used in the analysis which has been added to the regular core processing routine: 4000 grams (g) Sartorius Extend Series Digital Scale, with hook attachment, stand for scale, bucket distilled water, bricks, computer with MS EXCEL®, 2000 g calibration weight.

The density calculations are as follows:

Weight in air / (Weight in air – Weight in water)

The following data were recorded on the EXCEL® spreadsheet in accordance with the example structure shown in Table 11-2.

Table 11-2: Density data example

Hole	Sample	Dl	Mg	Ml	Ds	Diameter	Scanner Max	Scanner Avg	Code
C08-41	95.5	1	396.53	240.82	2.55	16	0.101	0.048	Cu-Ag

The hole number is listed along with the depth of the sample. Dl is the density of the distilled water, Mg is the mass of the sample in air, Ml is the mass of the sample in water, and Ds is the density of the solid. A zone code is also added to identify the mineralized zone of the sample.

A total of 4,339 density measurements were completed on holes C08-41 to C12-68.

An additional density measurement of the bulk sample delivered to SGS was done as part of the metallurgical study, the density obtained by SGS confirmed the earlier density measurements done by Amax.

Table 11-3 outlines the density values for each of the different mineralized zones plus dykes.

Table 11-3: Density measurement results summary

Grade Domain Code	Density (tonnes per m^3)	Sample Count
OX	2.50	578
Cu-Ag	2.58	1496
Cu-Mo	2.58	1458
Mo	2.57	638
MSI	2.57	91
DYKE	2.52	78

11.3 Assay Techniques

Samples submitted by Kobex were routinely analyzed by the ALS-Chemex ME-ICP61 procedure code for 39 elements using a four-acid digestion with analysis by Plasma Emission Spectroscopy (ICP-AES).

Samples submitted by CuMoCo were routinely analyzed by ALS-Chemex , an independent ISO 9002 certified laboratory, ME-MS ICP61 procedure code for 47 elements using a four-acid digestion with analysis by Inductively Coupled Plasma Mass Spectrometry (ICP-MS).

Samples submitted by CuMoCo for inter-laboratory check analysis were analyzed by SGS, an independent ISO 9002 and ISO 17025 accredited laboratory, by the SGS ICM40B for 50 elements using a four-acid digestion/ICP-AES and ICP-MS.

The assay methods report the main element results as follows:

- Molybdenum as ppm Mo, which is stored as both ppm Mo metal and molybdenum disulphide (MoS_2%) in the database to reflect the actual natural material.

- Copper in ppm Cu, which is stored in the database as both ppm and percent copper (Cu%).
- Silver in ppm Ag and stored as ppm, grams/tonne and ounces/ton in the database.
- W in ppm and stored in the database as ppm.
- Rhenium is reported in ppm and stored in the database as ppm.

11.4 Security

A contemporary, well-kept, large steel building was used to warehouse CuMoCo's core, samples, sampling equipment and field office at the CuMo project headquarters in Garden Valley, Idaho. The building is well-lit and insulated with heavy metal doors that have security locks.

The building is located on the property of a nearby landowner and is on a state highway, which local law enforcement regularly patrols. Additionally, a geologist lives on the property for most of the year in an apartment that adjoins the metal building. Core is stored on pallets that are stacked two high and bound by metal strapping. Bagged samples waiting to be shipped are kept in high-walled pallets in a central location within the building.

The area where the samples are kept is well-lit, well ventilated and easy to observe by staff. The floor is reinforced concrete and the walls are steel. There are few windows. CuMoCo personnel are present on a nearly 24-hour basis in season. Off-season, a local watchman lives adjacent to the property and provides security for the building and its contents.

In 2017, all core, rejects and information was moved from Garden Valley to a larger secure warehouse in nearby Horseshoe Bend, which has the same level of security as the one in Garden Valley with the exception that the geologist, as of the date of the current report, no longer lives at the property. The author has viewed photos of the facility which was also part of the site visit of SRK personnel in 2018 (Section Site Visit2.6).

11.5 QA/QC Programs

11.5.1 Historical Checks

In a June 2005 report (Giroux et. al, 2005), there were six data sets available to verify the original Skyline MoS_2 assay data base (pre-CuMoCo involvement in project). The original Skyline assays were re-assayed by Skyline at three stages of the sampling procedure; from core duplicate samples, from splits of rejects and from splits from pulps. Three inter lab sets of duplicates are also available to compare with the Skyline original assays; a pulp sent to Amax Lab in Climax from diamond drill hole assays, a second split at the drill of reverse circulation drill cuttings and a selected set of samples sent to Hazen Laboratory. The results from all comparisons are presented as scatter plots in Appendix 5 and discussed below.

It should be noted that the convention for the CuMo project has been to measure percent elemental molybdenum (%Mo) in assays and to calculate %MoS_2 by multiplying %Mo by 1.6681. Both %Mo and %MoS_2 are stored in the project's database, and the latter, %MoS_2, is used in resource estimates and mine planning. Consequently, in the following descriptions, where %MoS_2 grades are compared, these are the calculated MoS_2 grades based on the underlying %Mo assays.

The first set compares 64 original MoS_2 values with duplicate core samples both run at Skyline Laboratory. Figure A5.1 shows a very slight proportional bias indicated with the best fit regression line pulled below the equal value line, but more than likely this is simply a function of the wide random scatter in the data. The correlation coefficient is only 0.7061 indicating the amount of scatter about the best fit regression line seen in Figure A5.1. The average precision of ±78% again points to a high sampling variability between core samples.

A second subset of checks consisted of 25 original Skyline assays compared to re-splits of rejects also run by Skyline. This data set shows a very slight fixed bias with the best fit regression line, seen in Figure A5.2, a constant 0.0011 % MoS_2 below the equal value line. The coefficient of correlation is a very good 0.9666. The average precision of this test is ±20%. This indicates good reproducibility between samples after initial crushing.

A third test was on a total of 408 samples that were reanalyzed by Skyline by taking a second assay from the pulp. This comparison is shown in Figure A5.3. The best fit regression line through the data mirrors the equal value line, indicating no bias. The correlation coefficient is 0.9891. The precision on the estimate is very good at ±23%.

The fourth check on the historic data consisted of a total of 303 sample pulps from diamond drill holes that were analyzed for MoS_2 by both Skyline and Amax. The results are shown in Figure A5.4. There is no bias indicated with the best fit regression line pulled slightly below the equal value line by a single high valued outlier. There appears to be an equal number of samples falling on either side of the equal value line. The correlation coefficient was excellent at 0.9671. The precision which is a measure of the reproducibility of a result by repeated attempts was ±56.5%.

A fifth set of duplicates compared original MoS_2 results to an Amax second split of cuttings in a total of 57 samples from reverse circulation drill holes (see Figure A5.5). This subset shows no apparent bias, with the best fit regression line and equal value line being very close. The correlation coefficient of 0.5989 is lower than previous comparisons and the data shows much more scatter. The average precision is a much higher ±101%. This is a very poor comparison between two laboratories with a large degree of random scatter that is probably more the result of the two splits from the RC cuttings than a comparison of the laboratories.

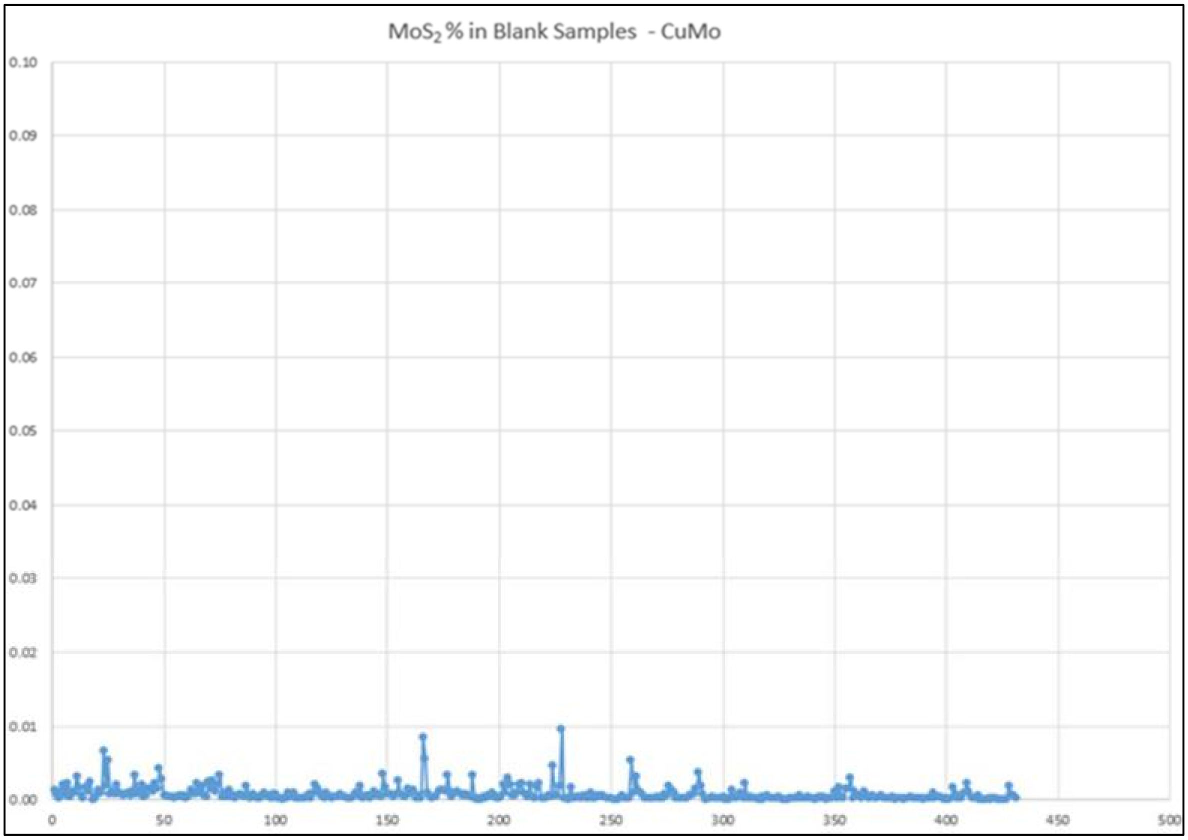
A final test on Skyline original samples was a set of 10 samples sent to Hazen Research. While this is hardly a representative sample, the results shown in Figure A5.6 show a pronounced proportional bias with the best fit regression line pulled below the equal value line. On average, the original Skyline assays for MoS_2 were 0.70 of the value indicated by Hazen. A total of 8 out of 10 samples returned higher values at Hazen. The correlation coefficient was poor at 0.7281 and the average precision was ±55%.

In general, the results from the first three tests at Skyline showed increasing precision and lower sampling variability from splits of core to splits from rejects to splits from pulps as would be expected. The inter-lab test from diamond drill pulps between Skyline and Amax showed good agreement and no bias. A similar check between Skyline and Amax on two splits from RC cuttings again showed no bias but higher sampling variability. The final limited test of 10 samples comparing original Skyline results with Hazen checks showed a strong bias with Hazen overestimating MoS_2 relative to Skyline.

The sample preparations and analyses done historically were made by a large, professional international mining company, Amax, who ostensibly used professional sampling and assaying laboratories for their samples taken in the project area. There is no reason to suspect any irregularities or question the results of the historic sampling.

11.5.2 Blanks

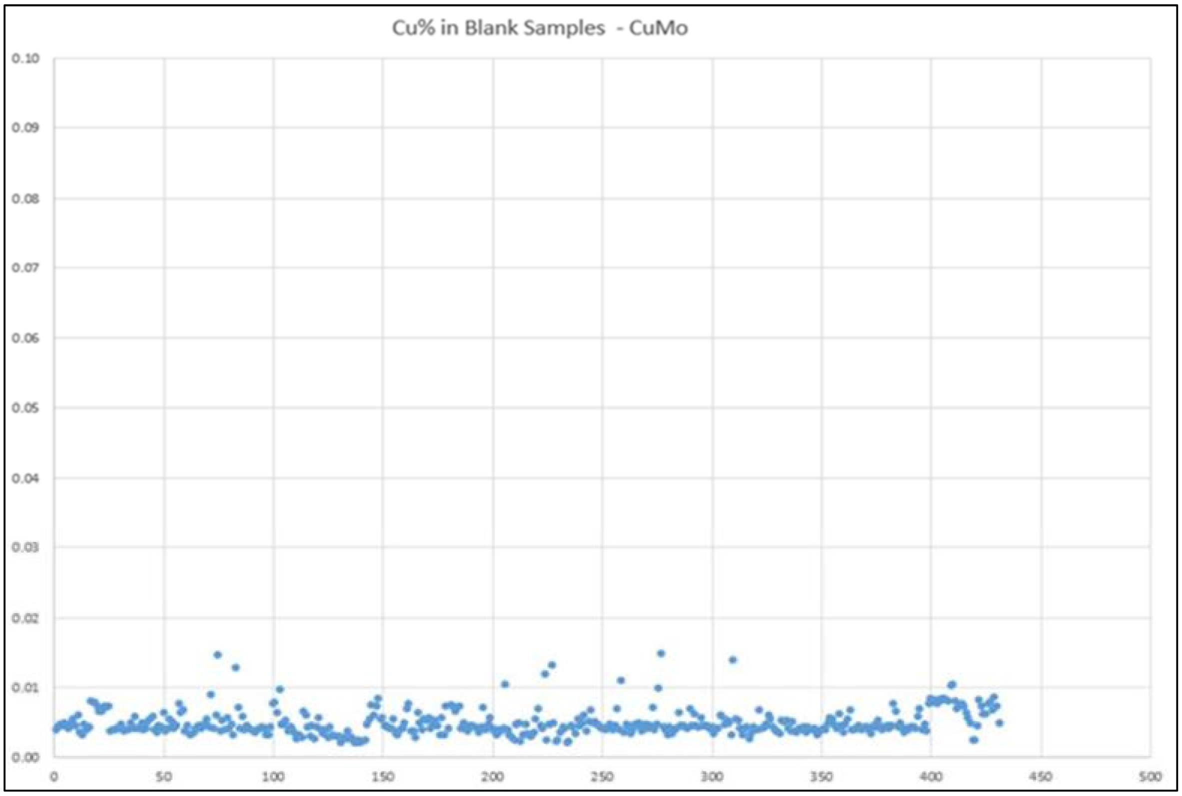
During CuMoCo's diamond drill programs blank samples were inserted in the sample stream at or about a 1 in 20 frequency. A total of 431 were analyzed for Mo, Cu, Ag, Re, Ga, W, Fe and S. The results were very good with no anomalies produced. The graphs for MoS_2 (calculated) and Cu are shown in Figure 11-1.



Source: Giroux et al, 2015

Note: MoS_2 grade is on the y-axis and sample number on x axis

Figure 11-1: MoS_2 in blank samples from CuMoCo drill programs at CuMo



Source: Giroux et al, 2015

Note: Cu grade is on the y-axis and sample number on x axis

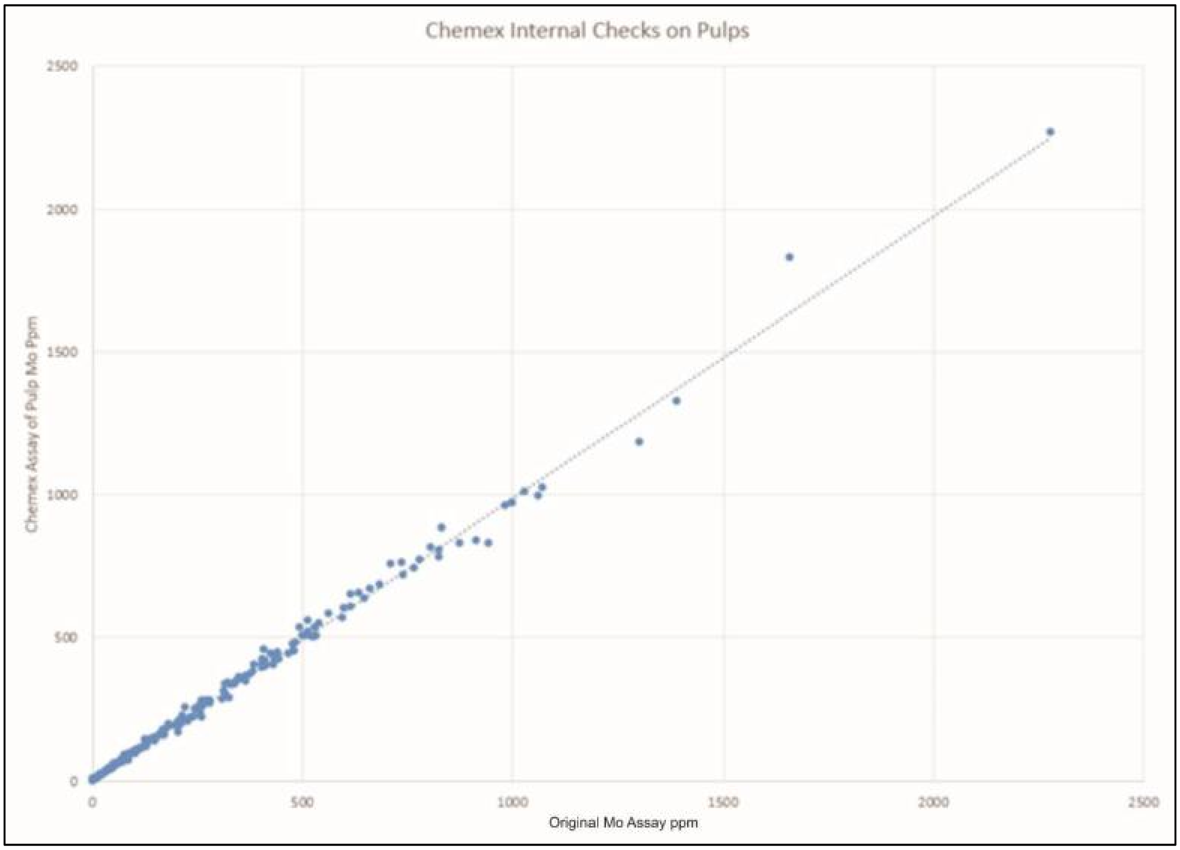
Figure 11-2: Cu in blank samples from 2008 drill program CuMo

11.5.3 Internal Lab Standards

The primary laboratory, ALS Chemex, inserted a blank and standard with every batch run during 2008. The policy was that unless the correlation results were excellent the batch was redone. A total of 180 blanks and 346 standard results were provided with the analysis.

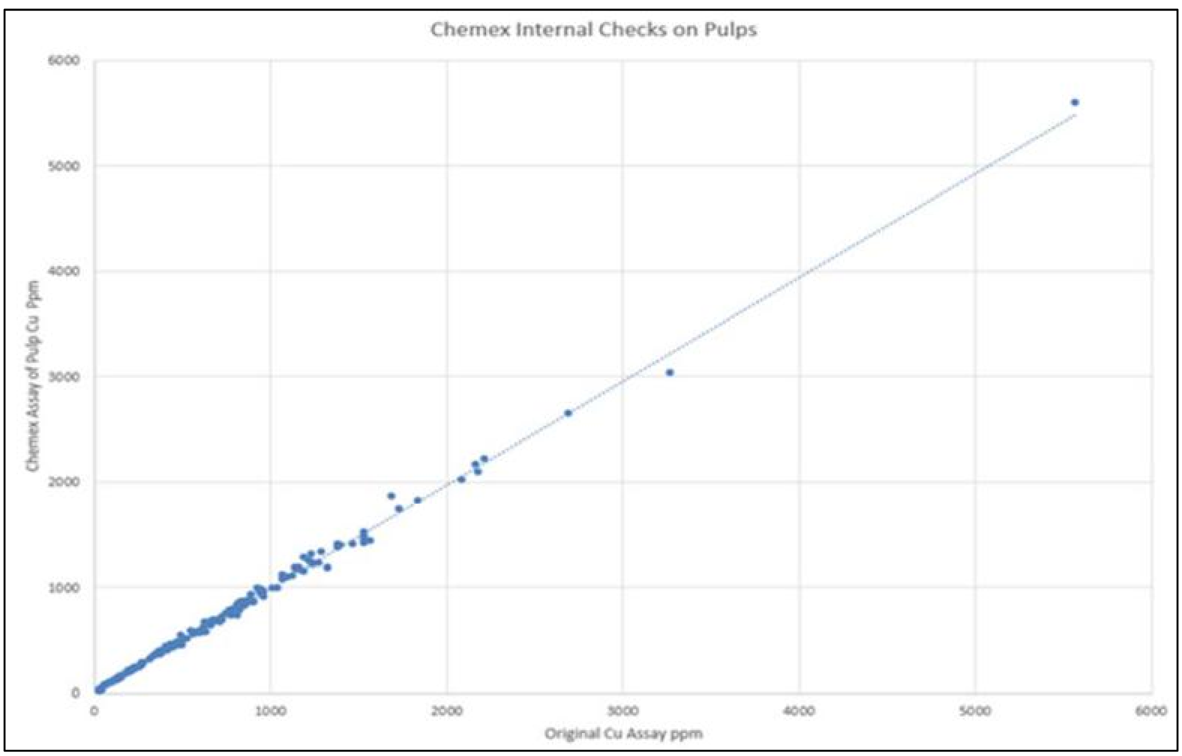
11.5.4 Internal Pulp Checks

ALS Chemex also routinely ran duplicate checks on sample pulps. Over the 2007-2012 drill program a total of 143 check samples were run for Mo. Figure 11-3 and Figure 11-4 shows the results are excellent with all but a few samples falling on an equal value line. The best fit regression line mirrors the equal value line.



Source: Giroux et al, 2015

Figure 11-3: Scatter plot of Chemex internal duplicates for Mo ppm (Mo metal)

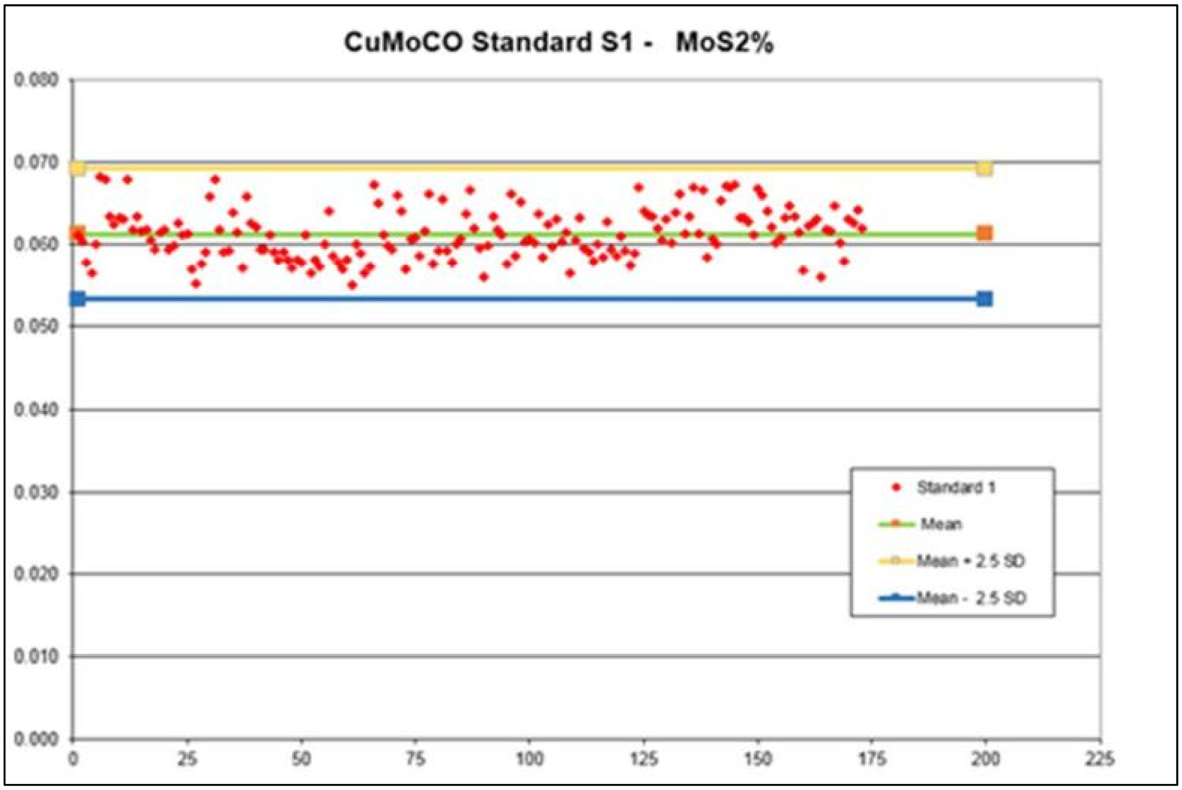


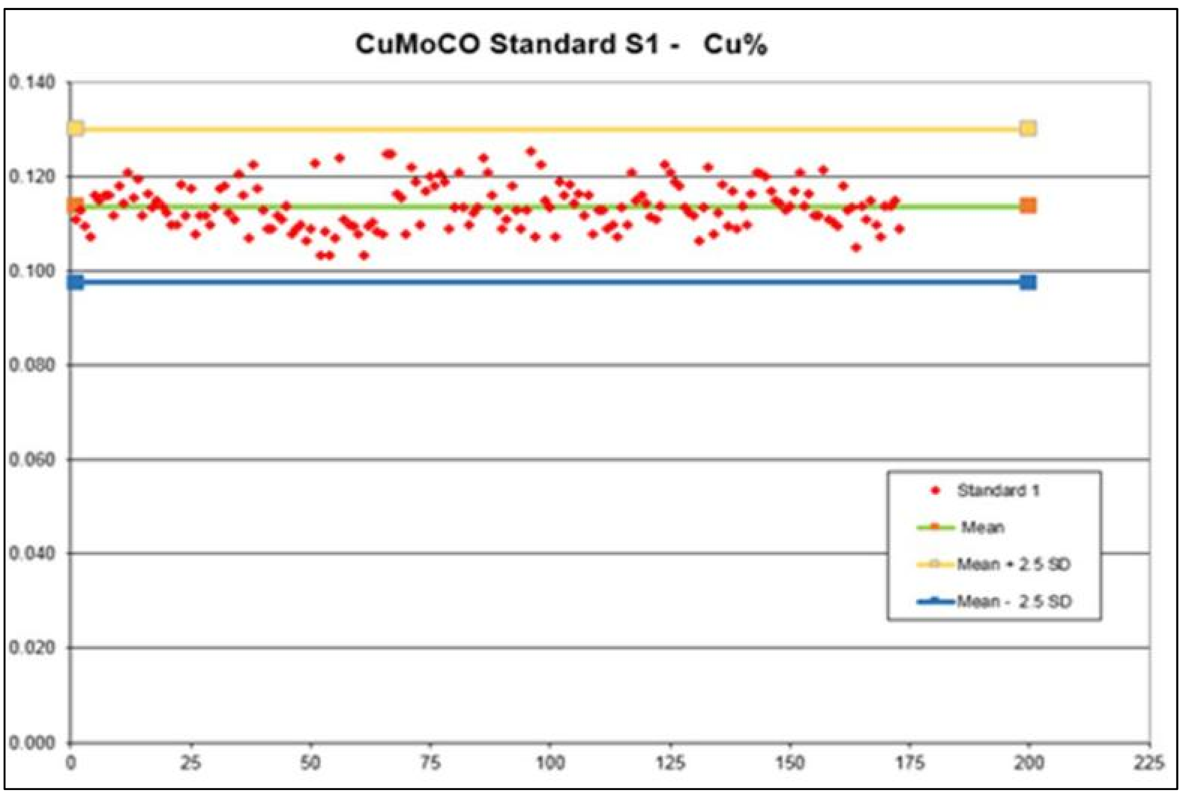
Source: Giroux et al, 2015

Figure 11-4: Scatter plot of Chemex internal duplicates for Cu ppm

11.5.5 CuMoCo Standards

As explained in Section 11.1, CDN Labs prepared a set of standards using drill core from the CuMo property. Results for Standard 1 (see Figure 11-5), the medium grade standard for MoS_2 (calculated) and highest grade for Cu, show results are reasonable with most falling between the mean ± 2.5 standard deviations.

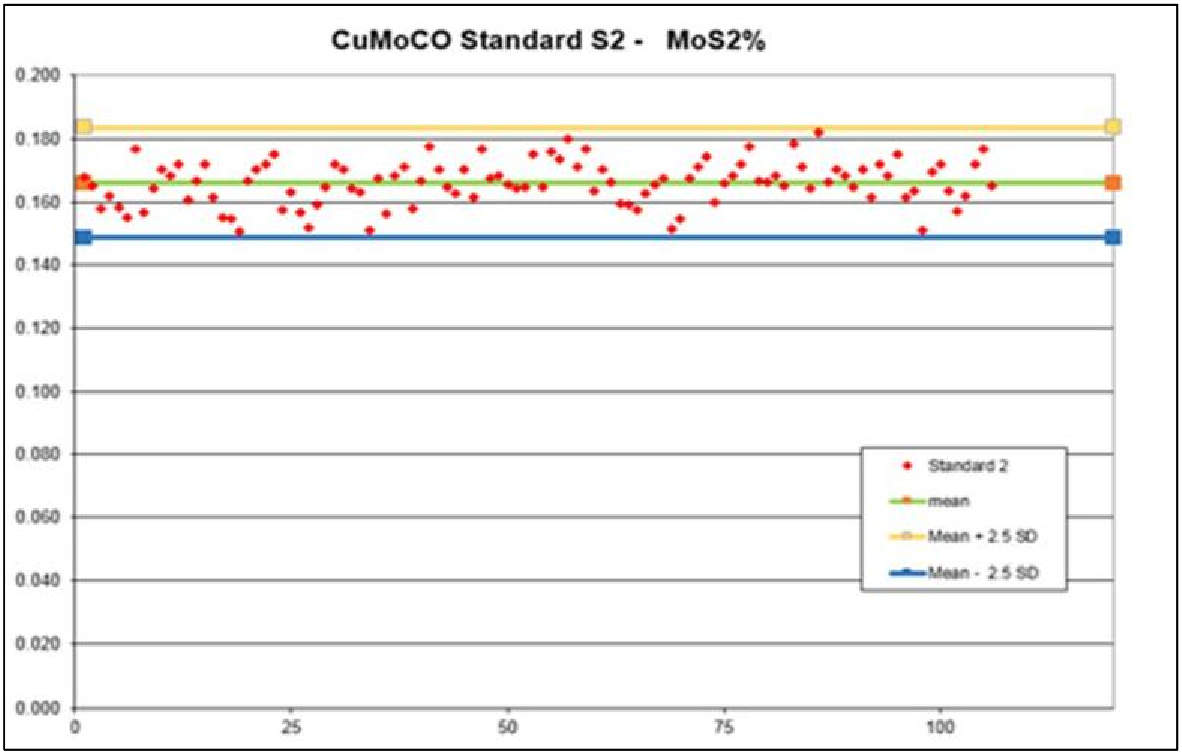


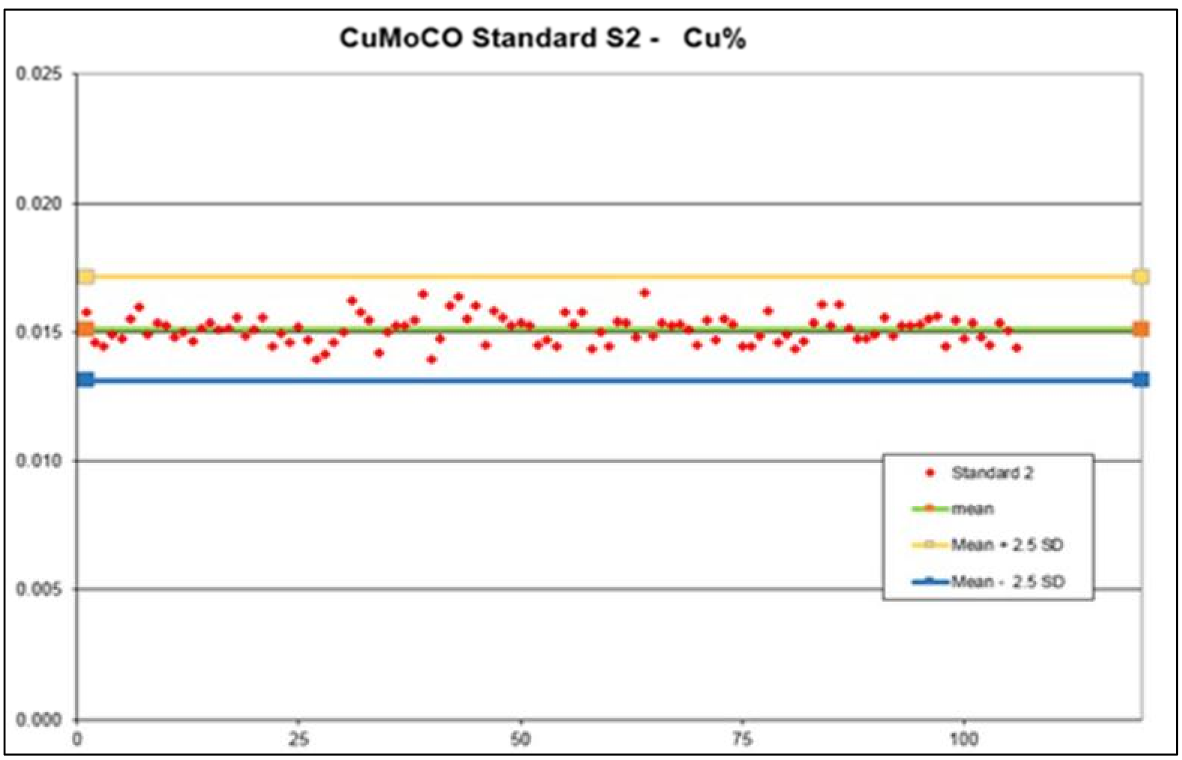


Source: Giroux et al, 2015

Figure 11-5: Results for Standard S1

Results for Standard S2, a higher grade MoS_2 and low grade Cu standard, show reasonable results for Cu and MoS_2 assays (see Figure 11-6) with all falling between the mean ± 2.5 standard deviations.

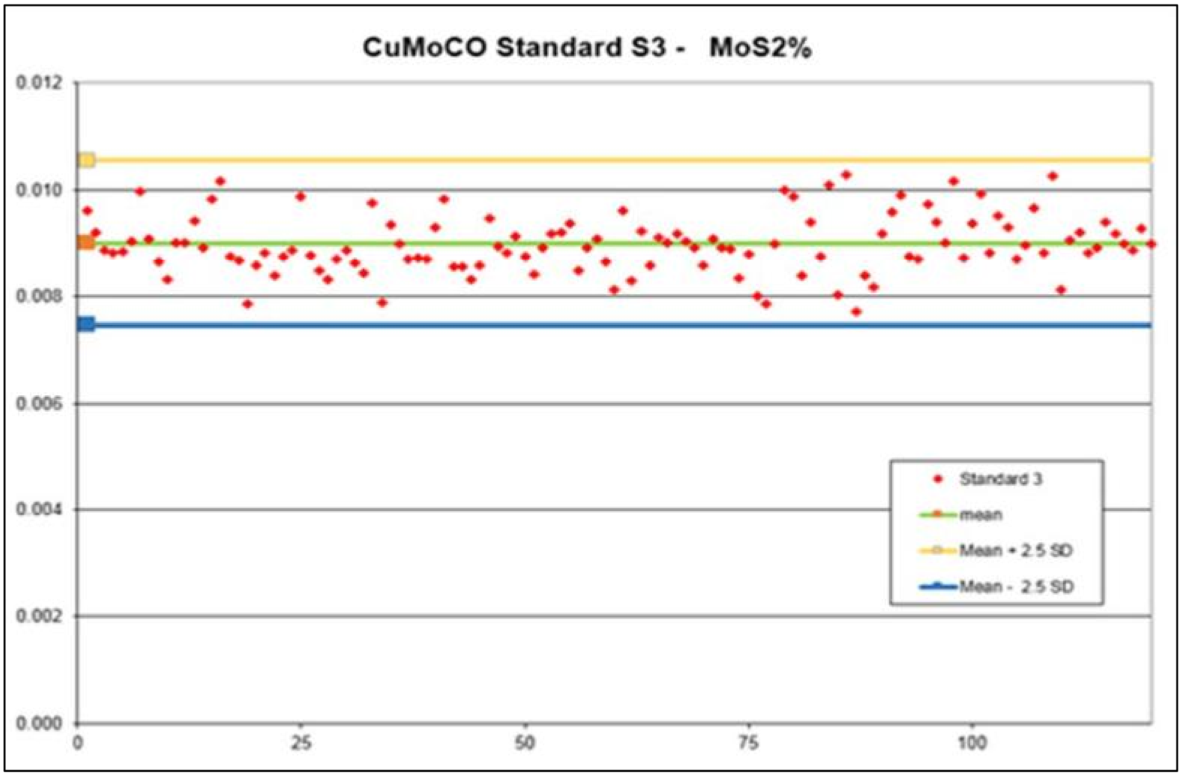


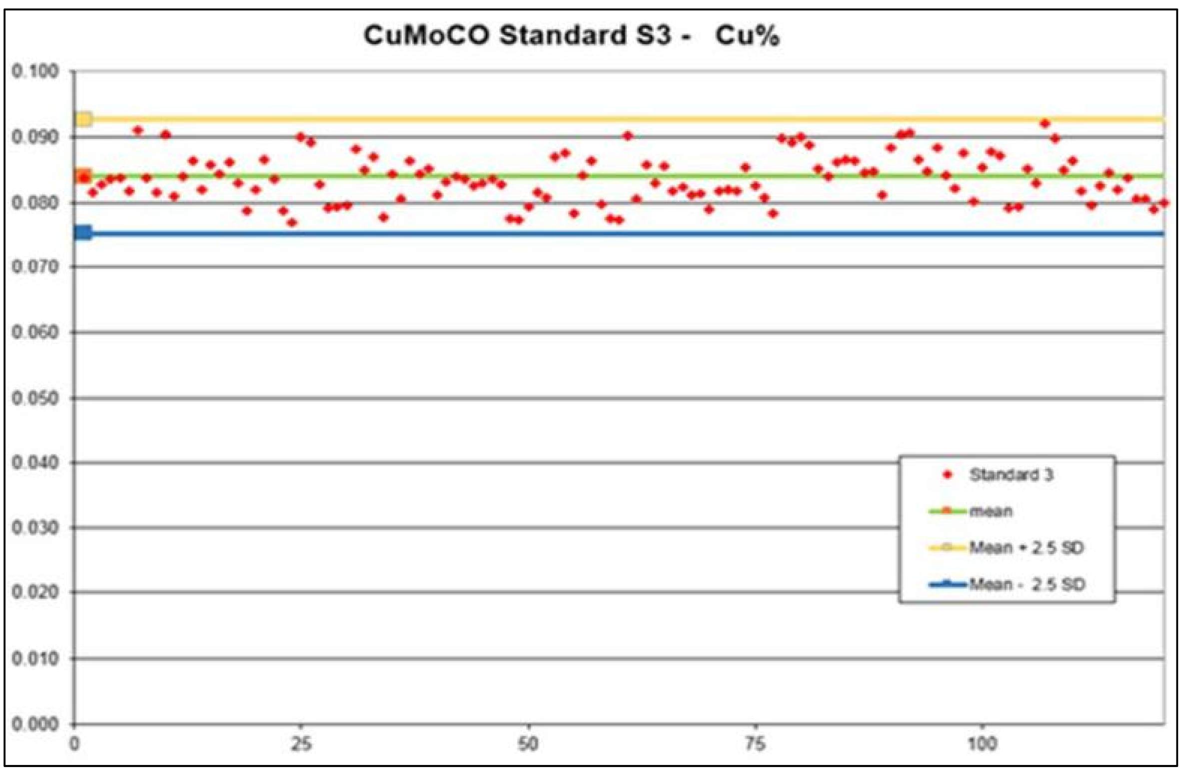


Source: Giroux et al, 2015

Figure 11-6: Results for Standard S2

The results for Standard S3 are also reasonable with more noise in the analysis, due to the low grade values encountered, but no large variations are observed. See Figure 11-7.





Source: Giroux et al, 2015

Figure 11-7: Results for Standard S3

11.5.6 Coarse Reject Duplicates

Coarse reject duplicate samples are duplicate samples that are taken after first crushing. At the ALS Chemex Laboratory in Elko, where the diamond drill hole core samples were crushed in the first step in the preparation stage, two duplicate samples were taken for roughly every 20th sample being analyzed by splitting the crushed half core. CuMoCo have been taking coarse reject duplicates since 2006. Coarse reject duplicates were submitted to measure the precision of the sample preparation and analysis process. The first duplicate underwent the same analytical procedure as the original sample (ICP-MS61), while the second duplicate was analyzed for molybdenum and copper using x-ray fluorescence (XRF) technique. Doing this confirmed not only the sample variability but variability in analytical techniques.

708 duplicate samples were submitted in between 2008 and 2012, for a submission frequency rate of 1 in 20 samples. The results are presented as a series of scatter plots with all variables reported in ppm and are shown in Appendix 2.

Overall, the results of the CuMoCo coarse crushed duplicates from drill core samples show good precision and no evidence of sampling bias. Silver duplicate analyses tend to show some scatter but are within acceptable tolerance limits. Precision plots yield good results, with an average of 80% of the data plotting within 20% of their respective duplicate samples, whilst an average of 55% of the data plot is within 10%. The results of the field duplicate samples are shown in Appendix 2.

11.6 Survey Validation

In 2007, CuMoCo established a survey control network completed by Geoterra Integrated Resource System Ltd. which included 24 control points surveyed by a licensed legal land surveyor, Shelby H. Griggs of Boise, Idaho. The survey was established using NAD83(1999)(HARN) UTM Zone 11 coordinates and NAVD88 elevations. Points included several drill holes completed before Hole 30. Monument control points were permanently marked with aluminum land survey pins. Future drill holes sites were surveyed using a total GPS station tying into the original survey points. In 2012, Sacré-Davey conducted a re-survey of previous holes and also surveyed 2012 holes and found no discernible difference in older hole locations.

All CuMoCo drill holes (i.e. 2008 and later) were surveyed down-the-hole using a Reflex survey instrument. Holes prior to 2008 were surveyed by either Troparia and/or single shot Sperry Sun survey tools.

The QP examined the survey database, survey reports and data base to confirm data was valid and visited and checked some of the drill sites during a site visit.

11.7 Verification of Drilling Data

Data prior to 2008 was verified and validated by Ausenco who compared and checked the data for errors in the compiled data from the header, survey, assay, geology and geotechnical tables are validated for missing, overlapping or duplicated intervals or sample numbers, and for matching drill hole lengths in each table. Drill hole collars and traces were viewed on plan view and in section as a visual check on the validity of the collar and survey information.

In 2012, Snowden repeated the same process on all data prior to 2012.

12 Data Verification

The section discusses the procedures completed by the author to verify the data. The qualified person has reviewed the procedures used by CuMoCo and produced a description and an analysis of the results as contained in Section 11. These are standard data verifications with no limitations.

All assay results used in the verification process by the qualified person were obtained from fully certified analytical laboratories with signed assay certificates.

The QP has reviewed the data collection and verification procedures followed by CuMoCo and by third parties on behalf of CuMoCo, and believes these procedures are consistent with industry best practices and acceptable for use in geological and resource modelling.

Sections 11.5 through 11.7 describe data verification done by previous qualified persons as well as the current author. These have been subsequently reviewed by the author and determined to be valid in order to demonstrate the validity of the data.

In 2015, the author completed the survey validation steps described in Section 11.7 on the 2012 drilling data and assays and also analyzed the original data set supplied directly from Snowden. The author found no errors in the pre-2012 data and a few minor discrepancies which were corrected in the 2012 data.

13 Mineral Processing and Metallurgical Testing

Unless otherwise stated, the sub-sections in this section were previously provided in the report, "Summary Report on the CUMO Molybdenum Property, Boise County, Idaho" (Giroux, Dykes, Place, 2015). The primary QP for this section, John Starkey has reviewed the underlying data, analytical work, and technical reports and takes responsibility for this summary. Also, Mr. Starkey re-interpreted comminution test results in terms of kW/t to assess grinding requirements.

Mr. Starkey has added content and has rewritten those parts that require confirmation after the addition of ore sorting to the process.

13.1 Metallurgical Testing (2009, 2015)

13.1.1 Introduction

This sub-section includes some new content for this current PEA.

The test-work undertaken to date is limited, with three composite samples tested for comminution characteristics and preliminary flotation testing to produce bulk copper/molybdenum concentrates. Despite limitations, the existing test-work data are considered suitable for a conceptual study and the comminution data are considered adequate for a preliminary engineering assessment of the grinding circuit design. No copper/molybdenum separation or ferric chloride leaching of molybdenum concentrates has been undertaken to allow determination of final concentrate grades and recoveries achievable into saleable concentrates. Where no test-work data are available, reasonable assumptions, based on typical industry values or data from other similar projects (e.g. Sierrita, and Thompson Creek) have been used to develop the process design criteria used in plant design.

Thompson Creek is a primary molybdenum mine of similar mill feed grade to CuMo, and thus has similar recovery processes. Sierrita is a copper-molybdenum processing operation that produces separate copper and molybdenum concentrates from a bulk concentrate. Both are directly applicable to CuMo. As mentioned, these cover the basis for assumptions for copper/molybdenum separation by flotation, and production of saleable concentrates using more flotation cleaning stages beyond those tested here, that the other operations have in place, and ferric chloride leaching of molybdenite (MoS_2) flotation concentrate, which follows flotation. This use of typical industry values for copper/molybdenum separation is recommended by John Starkey as a reasonable approach for this PEA.

The CuMo mineralized material is of moderate hardness and is amenable to grinding in a conventional SAG/ball milling circuit with or without pebble crushing. The mineralogy is fine grained and test-work done indicates the requirement for a fine target grind size to achieve adequate liberation for flotation.

Acid Based Accounting testing indicates that the tailings are potentially acid neutralizing (PAN) due to the presence of carbonate and low pyrite content. SGS concludes that "the tailings tested were not acid generating". Further studies are required, but if confirmed, this will lead to significant cost savings in the tailings handling circuit and a major reduction in the environmental impact of the project.

The three composite samples which were tested are labelled for the mineralized zones: Cu-Ag, Cu-Mo and Mo. The Cu-Ag and Cu-Mo labelled composites comprise both the oxide and sulfide parts of the system; oxide is not separated. The Mo composite consists of both Mo and MSI Zones. To arrive at the recoveries for the oxide and MSI Zone from the mixed samples, polished sections were examined, and factors were calculated to reduce the recoveries obtained for the Cu-Ag sample. This is a conservative approach as the inclusion of the lower recovery oxide within the Cu-Ag and Cu-Mo Zones effectively reduces the overall recovery below what could otherwise be expected.

13.1.2 Sample Selection

CuMoCo began collecting metallurgical samples for grinding and flotation testing in December 2007. One fourth of the core (quarter core) was used from continuous samples of the mineralized zones (an upper copper-silver zone, underlain by a transitional copper-molybdenum zone, in turn underlain by a lower molybdenum-rich zone) from drill holes CO6-27, CO6-28 and CO6-29 and collected as individual 10 ft samples of quarter core selected as representative of the three mineralized zones. Technicians supervised by geological staff collected the samples and prepared them for shipment. A bonded carrier took the samples from Garden Valley, Idaho to Vancouver, British Columbia. The samples were taken to SGS Canada, Kent Corporate Center, Kent Avenue N., Vancouver, British Columbia, for the metallurgical study. The test-work results are detailed in an independent private report entitled “An Investigation into the recovery of molybdenum, copper and silver from CuMo samples prepared for Mosquito Consolidated Gold Mines Ltd. Project 50004-001”.

13.1.3 Test-work Program

The flotation and grinding metallurgical test-work program used as the basis for this report consisted of comminution and flotation test-work on three separate metallurgical composites; copper/silver, copper/molybdenum and molybdenum, that were assembled to represent the three known mineralized types in the CuMo deposit at the time of testing. The test-work results are reported in “An Investigation into the recovery of molybdenum, copper and silver from CuMo samples prepared for Mosquito Consolidated Gold Mines Ltd Project 50004-001” (SGS, 2009).

Two main phases of flotation and grinding metallurgical testing were undertaken on the CuMo samples:

- Bench scale comminution testing, consisting of SAG Performance Index (SPI®) and Bond ball mill work index testing
- Bench scale flotation testing consisting of rougher kinetic flotation, cleaner flotation and locked-cycle tests, supplemented with mineralogical examination

Comminution Test-work Suite

The current comminution dataset consists of three SPI® and Bond ball mill work index tests, one on each of the composites. Table 13-1 summarizes the outcomes of the comminution laboratory test-work undertaken for this study, the table also shows the selected design case, which typically corresponds to copper/silver. To date, no samples have had SAGDesign Testing, Drop Weight Index Testing (either by the JK Drop Weight Test or SAG Media Competency Test), Bond Crushing Index, Bond Rod Mill Index or Abrasion Index testing. Values for these metrics have been estimated from the available data or from typical values for similar mineralization.

Table 13-1: Summary of comminution test-work data

Comminution Characteristics		Cu-Ag	Cu-Mo	Mo	Design
Specific Gravity	g/cm³	2.64	2.60	2.60	2.64
SPI®	min	84.5	73.0	70.8	84.5
SMC DWI	kW/m³	n/a	n/a	n/a	7.4
Crushing work Index	kWh/mt	n/a	n/a	n/a	15.8
Bond rod mill work index	kWh/mt	n/a	n/a	n/a	15.8
Bond ball mill work index (closing screen 106 pm)	kWh/mt	15.8	15.7	12.6	15.8
Bond Abrasion index		n/a	n/a	n/a	0.25

Due to the preliminary status of the test-work and the composite nature of the samples tested, the most competent sample results have been used as the basis for design. It has been assumed that this will provide a similar design point as the 80th percentile competency and ensure a robust design. This premise will need to be confirmed through additional testing using SPI tests on geometallurgical samples, and confirmed by SAGDesign testing of composite samples, in the next phase of study as more detailed mine schedule information and material comminution characteristics become available.

Flotation Test-work Results

Flotation test-work was completed prior to the commencement of the present study, commencing with rougher kinetic flotation testing and culminating with locked cycle testing of the major material types. Only bulk sulfide flotation with multi stage cleaning has been undertaken to produce a copper/molybdenum concentrate. No copper/molybdenum separation has been undertaken to date. Analysis of the test-work has been used to develop the plant process design criteria and estimates of concentrate grade, copper, molybdenum and silver recovery.

13.1.4 Conceptual Study Flotation Test-work

The flotation test-work program was divided into three phases: rougher flotation; open circuit cleaner flotation; and locked cycle flotation.

Rougher Flotation

Initially, a series of rougher flotation tests were conducted to determine the sensitivity of the material types to grind size and reagent scheme. These tests were supplemented with mineralogical examination by QEM*SCAN (Quantitative Mineralogy by Scanning Electron Microscopy) to determine fundamental mineral liberation and mineral speciation. These tests indicated the following:

- Copper mineralogy in the Cu-Ag Zone is fine grained and exhibited sensitivity to primary grind size, with highest recovery at a grind size of 80% passing 63 µm. Molybdenum and silver content exhibit little sensitivity to grind size.
- Target elements showed little sensitivity to grind size for the Cu-Mo Zone, with only a slight change in recovery between a grind size of 80% passing 106 and 63 µm for copper, molybdenum and silver.

- The copper and silver minerals in the Mo Zone exhibited significant sensitivity to grind size. Although the sensitivity of molybdenum was lower, the finer grind resulted in an increase in molybdenum recovery.
- Sulfur assays on the concentrates from the Cu-Ag Zone and Cu-Mo Zone indicate the presence of a floatable sulfide gangue mineral; most likely pyrite (no sulfur assays were available for the Mo Zone).

The results of these tests are summarized in Table 13-2.

Table 13-2: Baseline flotation results for CuMo composite samples

Mineral Zone	Test No.	Feed		Concentrate grade			Concentrate Recovery		
		% Cu	ppm Mo	% Cu	% Mo	ppm Ag	% Cu	% Mo	% Ag
Cu-Ag	VF1-1	0.16	213	1.22	0.18	39	76.5	87.7	78
	VF1-2	0.16	179	1.71	0.27	53	58.7	81.6	70.3
Cu-Mo	VF2-1	0.12	435	2.11	0.79	42	89.7	92.4	74
	VF2-2	0.11	398	1.54	0.61	36	89.3	92.9	74.5
Mo	VF3-1	0.03	1135	0.47	1.99	13	77	94.4	64.4
	VF3-2	0.03	1135	0.44	1.75	12	83.1	96.9	71.8

Note: to be clear, the grades in this table referring to Mo are all elemental Mo (not MoS_2).

The tests indicate that the mineralization is amenable to flotation, resulting in good recovery of target mineral species into a low mass concentrate stream. The sensitivity of the mineralization to primary grind size indicates that a fine grind for all the types will be required to ensure good recovery. Additional grind sensitivity test-work should be included in subsequent testing to optimize the mineral recovery with grind size.

Open Circuit Flotation

Cleaner flotation was conducted at the finer target primary grind size of 80% passing 63 μm and incorporated a rougher concentrate regrind stage to increase mineral liberation. Varying regrind times and reagent dosages were trialed to determine optimum flotation conditions.

The cleaner flotation reagent scheme was changed from that trialed in the rougher tests; a molybdenum specific activator (Moly Oil) and a copper molybdenum specific collector (Aero 3302). Despite the presence of pyrite, reporting to final concentrate, a non-specific sulfide collector (SIBX) was used for the cleaner flotation testing.

The fine grain structure of the mineralization identified by the QEM*SCAN testing and the increase in rougher grade and recovery indicated that regrinding of rougher concentrates would be required to achieve adequate concentrate grades. Concentrate regrinding was therefore incorporated in all subsequent cleaner and locked cycle testing. The target regrind size was arbitrarily selected at 90-95% passing 20 μm and achieved by grinding for a set time per test. Multiple stages of cleaning were incorporated to target high concentrate grades, typically with an elevated pH level in the final stage of cleaning. The results from selected optimization tests are summarized in Table 13-3.

Table 13-3: Cleaner flotation results for CuMo composite samples

Mineral Zone	Test No.	Feed		Concentrate grade			Concentrate Recovery		
		% Cu	ppm Mo	% Cu	% Mo	ppm Ag	% Cu	% Mo	% Ag
Cu-Ag	VF1-3	0.14	176	19.8	3.32	596	49.6	68.2	49
	VF1-4	0.16	185	15.3	2.3	462	64	81.3	64.9
	VF1-5	0.15	175	16.4	2.68	539	55.6	79	41.2
Cu-Mo	VF2-3	0.12	392	18	6.31	344	85.5	93.7	76.8
	VF2-4	0.12	416	17.3	6.53	354	81.8	92.6	74.8
	VF2-5	0.11	315	16.6	4.88	365	85.4	90.4	70.3
Mo	VF3-3	0.03	1048	5.9	24.4	151	79.6	95.9	52.2
	VF3-4	0.03	1025	6.1	24.8	150	79.8	95.8	50.7
	VF3-5	0.03	958	5.7	21.3	168	79.8	95.3	56.2

Note: to be clear, the grades in this table referring to Mo are all elemental Mo (not MoS_2).

The concentrate grades achieved in the majority of these tests indicate the presence of significant levels of diluents in the final concentrate. The absence of mineralogy or sulfur assays on the final concentrates makes determination of the nature of these diluents difficult to determine. However, the most likely explanation for this is the presence of floatable pyrite in the mineralization that has not been depressed in the flotation circuit and is reporting to final concentrate. This issue will require further evaluation and testing during subsequent studies.

Following the completion of the open circuit cleaner flotation test-work phase, a locked cycle test was conducted on each of the major types. This phase was aimed at testing the best flow sheet conditions in a locked cycle test to determine the closed-circuit grade recovery performance of each of the types for project evaluation.

Locked Cycle Test-work at Design Conditions

Flotation results from the optimization test-work highlighted the benefit of fine regrinding and multiple stages of concentrate cleaning on improving concentrate grade. A flow sheet incorporating rougher concentrate regrinding and multiple stages of cleaning, similar to that from the open circuit cleaner testing was selected for the conceptual study. To test the flow sheet performance on all types, a series of locked cycle tests was conducted.

Locked cycle tests are used to determine the effects of recycling intermediate streams, like scavenger concentrates, on the overall grade recovery performance of the type. By retaining these streams and combining them with concentrates from a subsequent flotation test, an assessment can be made of the overall performance from a full-scale plant operation.

Locked cycle tests were undertaken for the main mineralized zones and the results are summarized in Table 13-4.

Table 13-4: Locked cycle test results

Mineralized Zone	Test No.	Feed		Concentrate grade			Concentrate Recovery		
		% Cu	ppm Mo	% Cu	% Mo	ppm Ag	% Cu	% Mo	% Ag
Cu-Ag	VF1-LCT1	0.16	190	13	2	357	62.5	82	71.7
Cu-Mo	VF2-LCT1	0.12	401	16.4	5.66	324	90.7	93.8	80
Mo	VF3-LCT1	0.04	1065	5.1	21.6	122	71.6	99.6	59.3

Note: to be clear, the grades in this table referring to Mo are all elemental Mo (not MoS_2).

Analysis of these results indicate that recoveries of target minerals are acceptable and are generally in line with those achieved in the open circuit cleaner testing. However, the final concentrate grades are again lower than required to produce saleable concentrates after copper/molybdenum separation. Additional test-work will be required to determine the nature of the concentrate diluents and ways to maximize their rejection whilst maintaining target recoveries.

Tungsten Recovery

This sub-section was added for this current PEA.

SGS 2009 conducted a preliminary tungsten separation test on rougher tailing of the lock cycle test of Composite 3. The test consisted of feeding the rougher tailings to a Falcon Concentrator whose concentrate was upgraded on a Mozley table. The results of the test are as shown in Table 13-5.

Table 13-5: Tungsten recovery test results

Stream	Wt%	WO_3 - %	
		Assay	Distribution
Mozley Concentrate	0.04	4.61	26.34
Falcon Concentrate	2.85	0.093	40.55
Calculated Feed	100	0.003	100.00

Source: SGS 2009

The sample used in the test, Composite 3, is from the Mo Zone which has the lowest grade of tungsten compared to the other zones. The average grade of tungsten for the Mo Zone is 21 ppm, while the Cu-Ag Zone has an average of 34 ppm, and the Cu-Mo Zone has an average of 41 ppm.

However, the SGS report states that as before (for the other two composites), the tungsten grades were too low for reliable assaying (of the tungsten values).

Based on the SGS report, the possibility to recover tungsten in an economic process has not yet been established.

13.1.5 Grade and Recovery Predictions

This sub-section includes some new content for this current PEA.

Analysis of the locked cycle tests has been undertaken to determine flotation performance predictions. The design recoveries of the target metals are generally in line with or slightly lower than those achieved in the locked cycle tests suggesting a degree of conservatism in the selected recoveries. The numbers were selected as generally being lower than the actual test-work values with the exception of the Cu-Ag Zone, as this sample consisted of both oxidized and non-oxidized material.

Analysis of the locked cycle tests has been undertaken to determine flotation performance predictions. The design recoveries of the target metals are generally in line with or slightly lower than those achieved in the locked cycle tests suggesting a degree of conservatism in the selected recoveries. The numbers were selected as generally being lower than the actual test-work values with the exception of the Cu-Ag Zone, as this sample consisted of both oxidized and non-oxidized material.

A review of the specified recoveries indicates that they are reasonable for a bulk concentrate from the CuMo mineralized zones. However, the concentrate grades achieved directly from the current tests do not reflect those required to achieve saleable concentrates and have been adjusted for the plant design and preliminary economic evaluation on the assumption that additional test-work will confirm and further optimize flotation metallurgy, allowing higher concentrate grades to be achieved with minimal impact on recovery.

It should be noted that the SGS (2009) report concludes the following in regard to saleable concentrates from the tests:

In the case of the Cu-Ag Zone sample: *"However, the upgrading ratios indicate that a saleable grade of Cu concentrate can be made from this composite." (page 6)*

In the case of the Cu-Mo Zone sample: *"The upgrading ratios assured that saleable Cu and Mo concentrates can be made by added cleaning stages." (page 7)*

And finally in the case of the Mo Zone sample: *"The upgrading ratios indicate that Cu and Mo concentrates of saleable grades can be made by added cleaning stages." (page 8)*

These assertions support the general assumptions with respect to concentrate grades and process design details and will be required when advanced level studies are done on the project in the future.

In order to derive a process design and capital and operating cost estimate, it has been assumed that a selective molybdenum flotation phase with copper depression, followed by a ferric chloride leach on the molybdenum concentrate to remove residual copper, is required. The design and grade recovery performance of these process units have been estimated from operating and test-work data from other similar studies and operating plants. These include Las Pelambres, Andina, Collahuasi, Gibraltar and Sierrita to developing projects (2009) such as Pebble, Prosperity and Mirador. All these have or have examined copper-molybdenum separation circuits.

The recoveries of target metals into their respective concentrates have been reduced to reflect metal misreporting during the separation stages. The final concentrator recoveries that have been assumed for the PEA of CuMo are shown in Table 13-5. These figures include bulk concentrate recovery, copper/molybdenum flotation separation and ferric chloride leach recovery.

Table 13-6: Grade/recovery predictions for CuMo

Material Type	Concentrate	Concentrate Grade		Concentrate Recovery		
		% Cu	% Mo	% Cu	% Mo	% Ag
Cu-Ag	Molybdenum	0.1	52	0.02	83	
	Copper	19	0.1	64	2.4	70
Cu-Mo	Molybdenum	0.1	51	0.04	92	
	Copper	22	0.1	85	0.7	78
Mo	Molybdenum	0.02	49	0.1	95	
	Copper	20	0.8	72	1	55

Note that the recovery predictions shown in Table 13-6 for Cu-Ag Zone material were based on samples that also contained oxide material. Segregation of this oxide material results in the adjusted and slightly higher recovery predictions for non-oxide material referenced elsewhere in this report (Table 14-13).

In addition to the primary elements listed, the study also analyzed the final concentrate from the lock cycle tests for gallium, osmium and rhenium, while the rougher tails were analyzed for gallium. Rhenium was the only metal present in quantities above detection limit returning values of 0.9 ppm, 2.9 ppm and 15 ppm respectively in the molybdenum concentrates from the three material types.

No test work was completed to determine the actual recovery of rhenium and sulfuric acid during the roasting process, though there are no indications that it cannot be achieved based on current technology and existing roasters. This test work is recommended for the next stage of development for this project.

13.2 Mineral Sorting

The following sections provide original text for the current PEA

13.2.1 Particle Sorting

The opportunity for preconcentration using sensor-based sorting was evaluated in 2015 where Sacré-Davey conducted a preliminary investigation with 100 rock samples from the deposit. The purpose of this test was to get an indication of the sample response to various sensors. Since this test showed a potential for preconcentration, a second set of testing was done with an XRF device using 400 samples. The initial study was completed in November 2016.

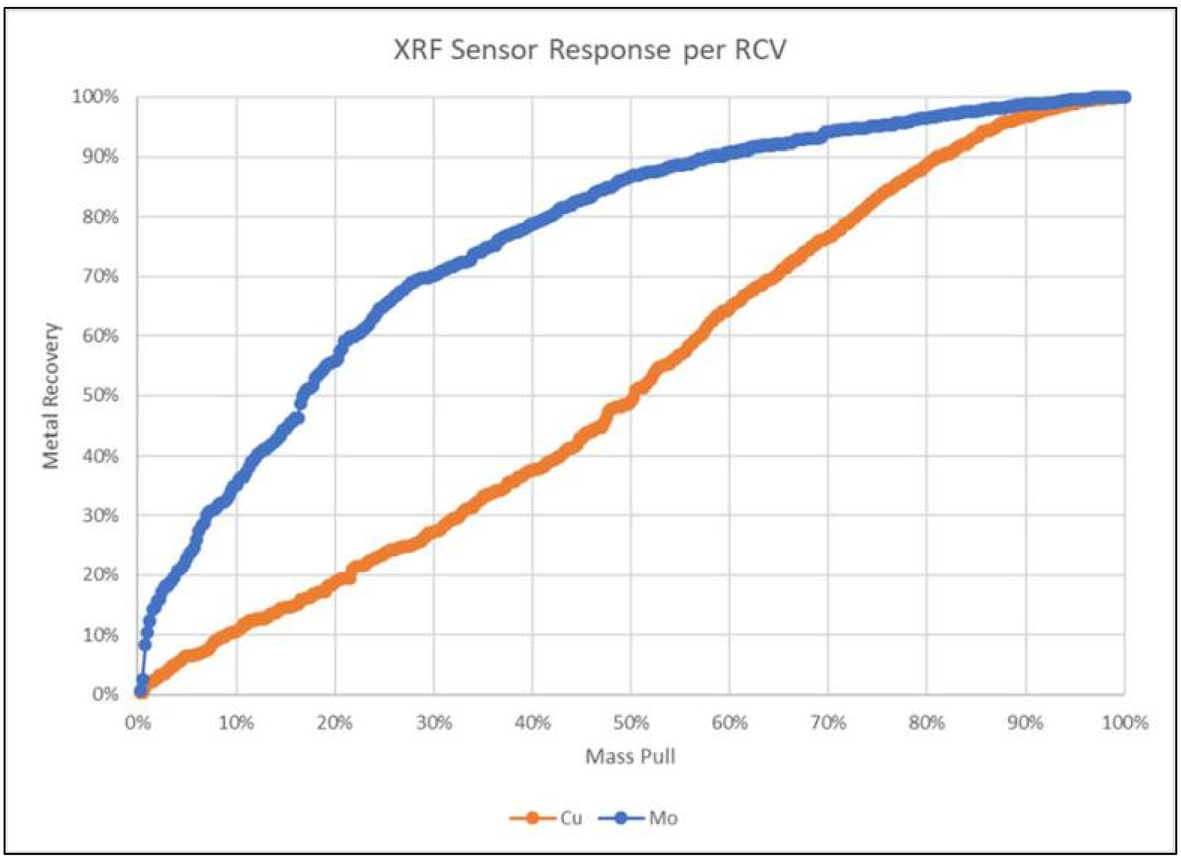
Samples from quarter core were used from continuous samples of the four mineralized zones; The samples assembled were selected to represent the four known non oxide mineralized zones in the CuMo deposit, namely: Cu-Ag, Cu-Mo, Mo and MSI. A total of 400 random samples of 1-5” size were sent and tested at the Coal and Mineral Processing Laboratory at the University of British Columbia. The samples were cleaned with high-pressure air and then scanned on the XRF device, followed by the EM device. Testing was initially conducted under the supervision of Brent Hilscher from Sacré-Davey. Following that, the samples were sent to MetSolve Laboratories Inc. for Cu and Mo assays.

Heterogeneity assessments of the Cu and Mo grade analysis were conducted based on the assay results to confirm initial confidence in the potential application of mineral sorting. Next, correlation studies between the assay result and sensor-based result were carried out upon observation of the provided rock samples. The outcome of the studies was then used in building several economic models to demonstrate the benefit for mineral sorting. Finally, John Starkey examined the assays, scanning records and previous reports to confirm that there was a basis for what is stated herein.

The purpose of the particle sorting study was to understand the deposit's amenability to mineral sorting. The study conducted was a scoping level preliminary evaluation to understand the possible opportunity. Detailed bulk sample test-work would be necessary to accurately measure the impact on the potential project economics.

The study demonstrated that there is significant variability in the deposit providing an opportunity to reject the low-grade rocks and upgrade the accepted mass. The sensors also showed positive response for upgrading the mill feed; however, due to the low concentration of Cu and Mo, further testing and validation is necessary.

The interpreted results are presented in Figure 13-1. This shows the recovery of Mo and Cu as a function of sorting mass pull. The sorting mass pull is the cumulative RCV percent of test samples from highest RCV to lowest, based on the XRF measurements of Cu and Mo.



Source: SRK, 2019

Figure 13-1: Particle sort XRF test results

Further testing and studies will be required at the pre-feasibility and feasibility stages to capture representative samples and the impact of the sorter on individual mineralized zones. The particle sort study was conducted to understand the deposit's heterogeneity on a rock-by-rock basis. Due to the large production rates of the project, a combination of bulk and particle sorting may be more suitable.

Mineral sorting products have not yet been tested for changes in the work index or flotation recovery. After sorting, most base metals operations experience an improvement in both the grinding specific energy and flotation recovery. These changes will be quantified as part of future studies.

13.2.2 Bulk Sorting

The success of the particle sorting test program, combined with recognition that currently available particle sorting technology on its own would not be able to handle the processing rates envisioned for CuMo, prompted further investigation into the viability of bulk sorting.

SRK, under the direction of Mr. McCarthy, the relevant QP, undertook a heterogeneity study of the CuMo deposit by analyzing exploration drill hole data. Two approaches were undertaken:

- Observing the effect of measurement scale on different heterogeneity parameters
- Assessing the relationship between bench composite grades and sample grades that make up those composites

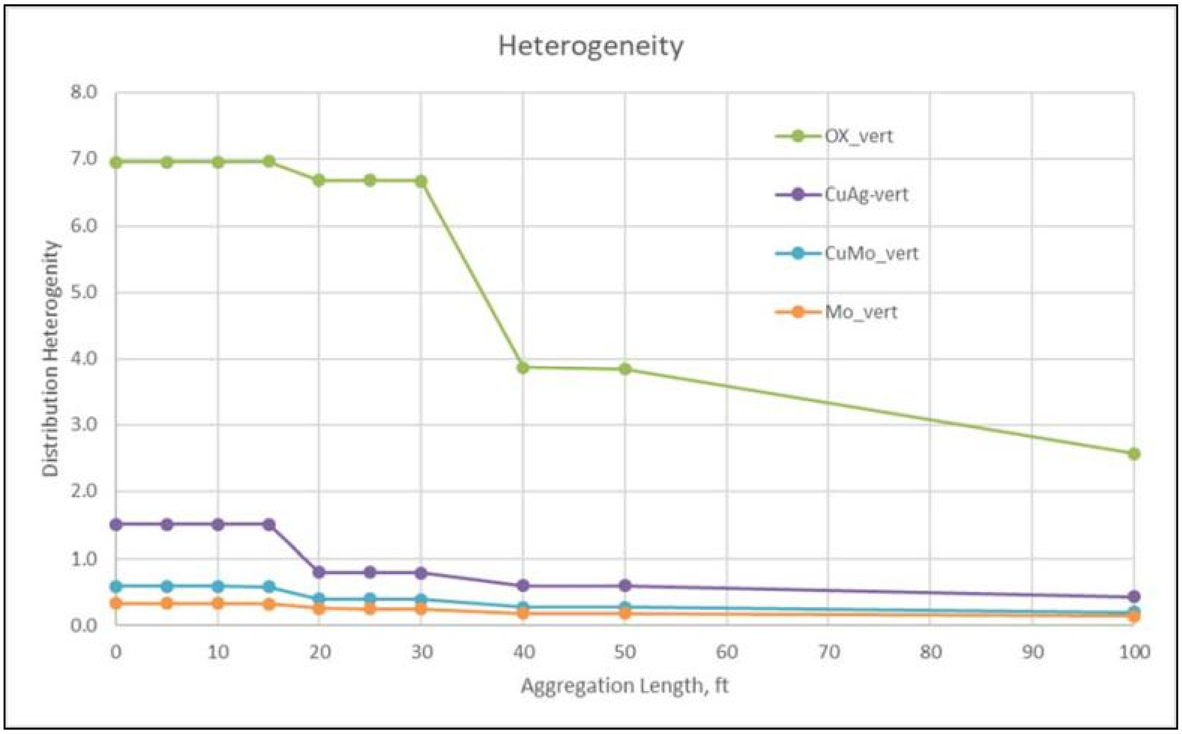
The following sections discuss the results of these assessments.

Heterogeneity and Scale

An SRK-developed approach to assess how mineral deposit heterogeneity is influenced by observation or sampling scale was applied to CuMo. It involves the analysis of exploration drill core data, to see the impact of varying aggregation lengths on key parameters, including most notably assay grades. In polymetallic deposits, NSR or equivalent is used (RCV in the case of CuMo).

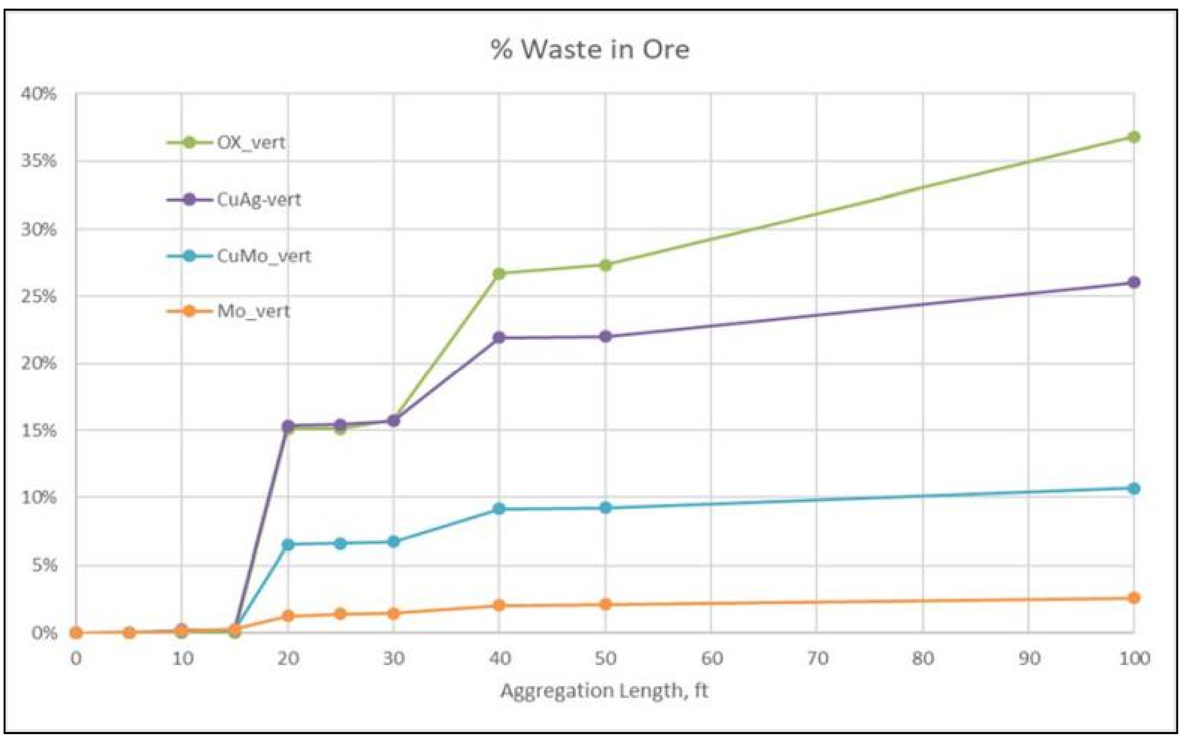
For CuMo, SRK assessed the main mineralized zones – oxide, Cu-Ag, Cu-Mo, and Mo. The drill holes were de-surveyed and sample intervals were assessed in the vertical direction – a proxy for mining bench height. Intervals were combined over increasing aggregation lengths, up to a maximum of 100 ft. Statistics and comparative analyses were run on the resulting aggregations. Select results are presented in Figure 13-2 and Figure 13-3.

One way to look at the impact of scale on heterogeneity is to calculate the distribution heterogeneity for different aggregation lengths. Distribution heterogeneity for a dimensionless lot (Pitard, 1993) was used here. It is a unitless parameter relating mass and grade (or NSR) of a group (aggregation) to the overall population or lot. It is apparent in Figure 13-2 that for all mineralized zones at CuMo, there is a decrease in heterogeneity with increasing scale. The OX zone was the most affected, and the Mo Zone was the least impacted by increases in scale.



Source: SRK, 2019

Figure 13-2: Impact of scale on distribution heterogeneity



Source: SRK, 2019

Figure 13-3: Impact of scale on "Waste in Ore" ratio

Figure 13-3 provides another measure of heterogeneity that the author finds very informative. It is "Waste in Ore[5]", which compares sample intervals that are below a cut-off but are still within aggregations whose average grades are above the cut-off. Figure 13-3 shows that increasing aggregation length results in increasing % waste in above cut-off material and that such increases happen quickly. They happen within the mining scale (e.g. 50 ft benches), but then largely flatten off for longer aggregation lengths. This suggests there may be benefits to selectively mining or processing material at smaller scales in order to reject waste that is inherent in a mineral deposit.

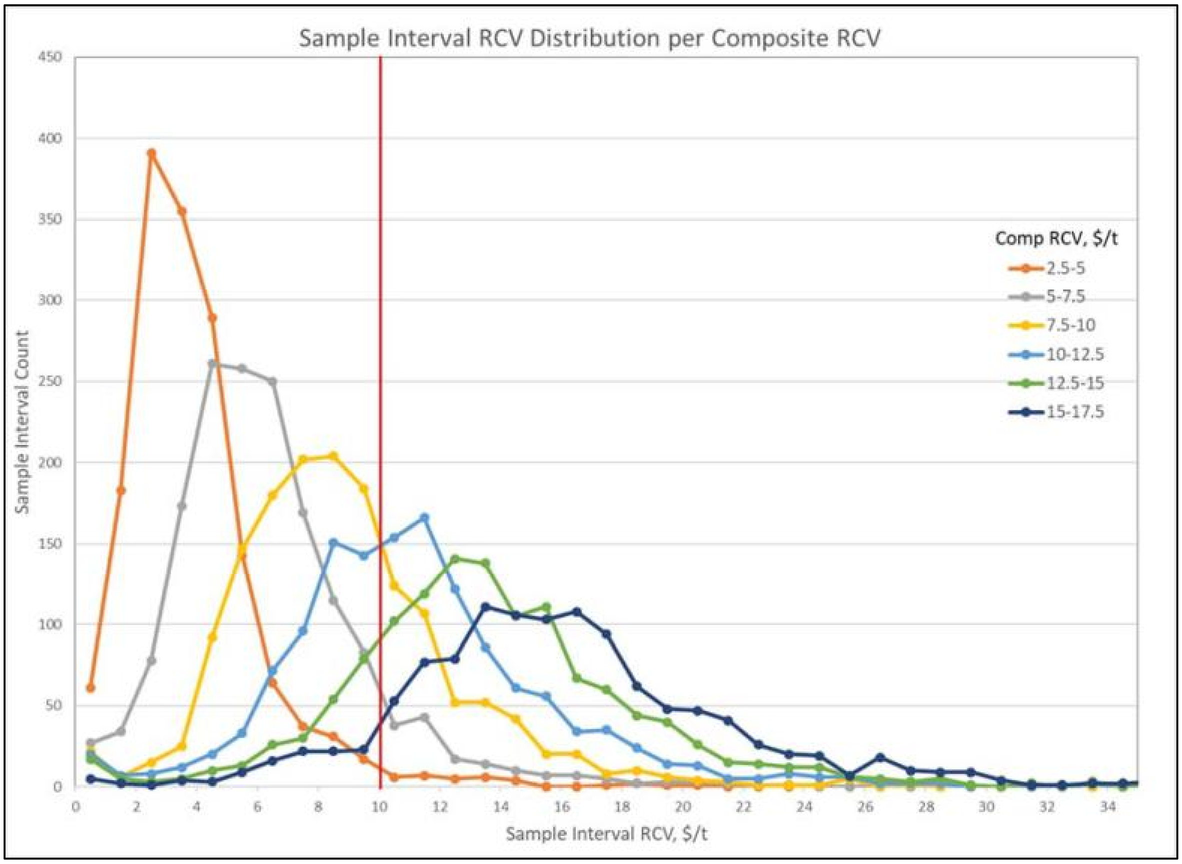
The main findings of this analysis for CuMo are that heterogeneity diminishes with increasing scale (or conversely, it increases with decreasing scale) and that the different mineralized zones at CuMo exhibit differing heterogeneity characteristics. Whilst this is generally accepted for all such analysis, notably in this case, significant change in heterogeneity for several zones occurred at around the scale of the mining bench dimensions and potentially smaller selective mining unit dimensions, raising the possibility of benefits from more selective in-pit pre-selection or bulk mineral sorting.

Composite-Sample Relationship

The other technique for assessing heterogeneity from drill holes interrogates the composite-sample relationship inherent in drill hole data. For this, the author developed bench composites of all the drill holes, based on an expected 50 ft bench height. Then, the RCV of the composites were calculated from the samples falling within the composites. For CuMo, RCV is determined as the product of the price and the mill recovery for the metal of interest. It is calculated for each of the mineralized zones in the deposit (see Section 14.9).

The resulting relationship can be plotted as the number of samples versus the sample interval grade (RCV) for each of multiple bench composite RCV ranges. This relationship is referred to as the "composite-sample relationship".

[5] "Ore" is used here in the generic sense and does not imply that the mineralized material at CuMo is "ore" under CIM Definition Standard, nor that the material constitutes a mineral reserve which can be synonymous with the use of the word "ore". The material does not have demonstrated economic viability. The CuMo property has no mineral reserves.



Source: SRK, 2019

Figure 13-4: CuMo composite-sample relationship

Figure 13-4 shows the composite-sample relationship for select composite RCV ranges. These ranges are set with $2.50/t increments and within each range the sample interval RCVs are counted in $1.00/t bins.

A red vertical line has been drawn at the $10/t RCV point, approximating the cut-off NSR for CuMo. Only six of the composite RCV ranges are shown – three on either side of $10/t RCV.

There are two important observations of the composite-sample relationship for CuMo:

- Composite RCV ranges below the $10 cut-off ($2.50-5.00; $5.00-7.00; $7.50-10.00), which should all be waste, have sample intervals within them that are above the $10 cut-off. This is more pronounced for composite ranges nearer the cut-off.
- Composite RCV ranges above the $10 cut-off ($10.00-12.50; $12.50-$15.00; $15.00-17.50), which should all be selected as above cut-off mill feed, have sample intervals within them that are below the $10 cut-off. Again, this is more pronounced for composite ranges nearer the cut-off. There tends to be more "waste in above-cut-off material" than "above cut-off material in waste" in general and as one moves away from the cut-off.

These observations effectively point to the opportunity for mineral sorting, if one can segregate material at the sample interval scale (or smaller, per the conclusion of the heterogeneity and scale analysis), one can remove waste from the mill feed and recover valued mineralized material from what would be otherwise waste.

SRK used these composite-sample relationships to test the impact of using different cut-offs to segregate different fractions of potential mill feed in a bulk sorting context. This is discussed further in Section 16.2.2.

14 Mineral Resource Estimates

14.1 Introduction

In 2015 at the request of CuMoCo, Giroux Consultants Ltd. was retained to produce a resource estimate on the CuMo project in Southern Idaho. A total of 68 drill holes covering the various mineralized zones were provided. While the Cu-Mo-Ag-W resource was estimated in April 2015, the effective date for this estimate is August 30, 2018, when estimates for Re and S were completed.

G.H. Giroux was the qualified person responsible for the resource estimate. Mr. Giroux is a qualified person by virtue of education, experience and membership in a professional association. He is independent of CuMoCo applying all of the tests in section 1.5 of National Instrument 43-101. Mr. Giroux visited the property, reviewing drill core and drill sites, on June 2, 2015.

This 2015 CuMo resource estimate represents an update of the 2012 estimate by Snowden Mining Industry Consultants (Jones, et al) and the 2009 resource estimate (Holmgren and Giroux), based on an additional 11 new diamond drill holes completed in 2011-2012.

The mineral resources estimated may ultimately be affected by a broad range of environmental, permitting, socio-economic (as discussed in Section 20), legal, title (as discussed in Section 4), marketing and political factors (as discussed in Section 19). At this time the authors are unaware of any of these factors that could materially affect the mineral resource estimate. Of course, going forward, relevant factors that could influence the resource estimate include changes to the geological, geotechnical or geometallurgical models, infill drilling to convert mineral resources to a higher classification, drilling to test for extensions to known resources, collection of additional bulk density data and significant changes to commodity prices. It should be noted that all these factors pose potential risk and opportunities to the current mineral resource.

14.2 Data Analysis

A total of 65 DDHs and three RC drill holes, over a combined total of 121,280 ft, were provided with 1,001 downhole surveys and 10,456 assays for Mo and Cu. For this resource estimation, the three RC holes were not used (see Appendix 3 for a list of drill holes used in the estimate), leaving only the 65 diamond drill holes as being used. For the 65 diamond drill holes used, the total length was 36,165.7 m (118,654 ft)

The provided data was checked for sample overlaps, gaps in sample intervals and assays within allowable intervals. No errors were found.

The basic assay statistics for DDHs are presented below in Table 14-1

Table 14-1: Summary of MoS_2 and Cu contents

	MoS_2 (%)	Cu (%)
Number	10,456	10,456
Mean	0.053	0.077
Standard Deviation	0.058	0.069
Minimum	0.0005	0.001
Maximum	1.09	0.920
Coefficient of Variation	1.09	0.89

Note: MoS_2 here is calculated from the assays for Mo by multiplying by 1.6681.

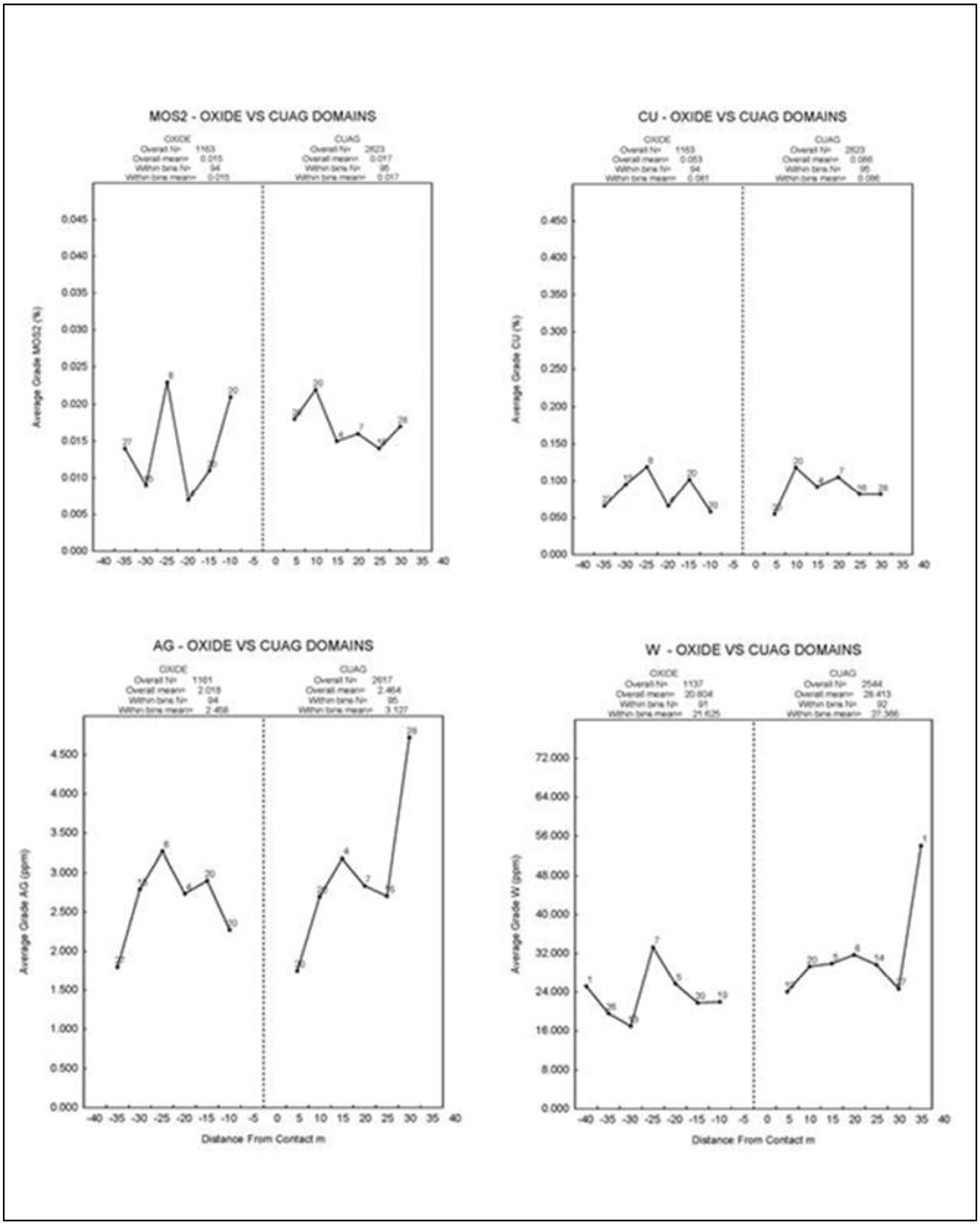
The molybdenum and copper mineralization at CuMo lies in four distinct mineral zones with an oxidized layer on top. More or less from top to bottom there occurs in most drill holes an oxide zone, a Cu-Ag Zone, a Cu-Mo Zone and a Mo Zone. Within one fault block, the Cu-Ag Zone is missing, and the oxide sits on top of the Cu-Mo Zone. These zones are underlain by a potassic-silica zone with lower grade copper and molybdenum material called the MSI Zone. While the oxide zone has been modeled for metallurgical reasons, it has been combined with the Cu-Ag Zone or in a few cases the Cu-Mo Zone for estimation purposes. While no test-work has been completed on the oxide zone at this time, experience with other such deposits indicates that metal recoveries tend to be lower in oxidized zones as compared to primary zones and as a result it was modelled separately, and a lower recovery was applied. This is a conservative approach and will be useful for future work.

Contact plots for each variable in Figure 14-1 show there is no difference in average grade across the oxide – Cu-Ag Zone contact. There are also several post mineral dykes that are large enough and continuous enough to be modeled. The Cu and MoS_2 grade statistics are shown in Table 14-2, sorted by zone. Silver and tungsten assays are shown in Table 14-3 for the same mineral zones. Values for MoS_2 and Cu reported as 0.000 were assigned values of 0.0005% and 0.001%, respectively. Silver values reported as 0.000 were set to 0.01 ppm while tungsten values reported as 0.000 were set to 0.1 ppm.

Table 14-2: Summary of assay statistics for Cu and MoS_2 sorted by zone

Item	Cu–Ag Zone		Cu-Mo Zone		Mo Zone		MSI Zone		Dykes	
	MoS_2 (%)	Cu (%)	MoS_2 (%)	Cu (%)	MoS_2 (%)	Cu (%)	MoS_2 (%)	Cu (%)	MoS_2 (%)	Cu (%)
Number	3,813	3,813	3,509	3,509	2,677	2,677	330	330	128	128
Mean	0.017	0.076	0.049	0.103	0.113	0.053	0.057	0.028	0.005	0.016
Standard Deviation	0.019	0.074	0.045	0.072	0.066	0.042	0.029	0.038	0.014	0.038
Minimum	0.0005	0.001	0.0005	0.001	0.0005	0.001	0.0010	0.001	0.0005	0.001
Maximum	0.315	0.77	1.09	0.92	0.99	0.59	0.17	0.20	0.13	0.18
Coefficient of Variation	1.15	0.97	0.92	0.70	0.58	0.80	0.51	1.34	2.62	2.36

Note: MoS_2 here is calculated from the assays for Mo by multiplying by 1.6681.



Source: CuMoCo 2015

Figure 14-1: Contact plots for oxide-Cu-Ag Zone contact

Table 14-3: Summary of assay statistics for Ag and W sorted by zone

Item	Cu–Ag Zone		Cu-Mo Zone		Mo Zone		MSI Zone		Dykes	
	Ag (ppm)	W (ppm)	Ag (ppm)	W (ppm)	Ag (ppm)	W (ppm)	Ag (ppm)	W (ppm)	Ag (ppm)	W (ppm)
Number	3,806	3,791	3,492	3,497	2,653	2,654	327	330	128	121
Mean	2.88	32.3	3.07	46.7	1.78	45.9	1.65	37.1	0.62	9.8
Standard Deviation	16.28	108.9	13.35	33.8	9.81	38.3	10.39	109.3	1.23	11.9
Minimum	0.01	0.1	0.01	0.1	0.01	0.1	0.01	3.3	0.01	0.1
Maximum	838.0	5400	744.0	470.0	494.0	890.0	182.0	1980	8.6	65.0
Coefficient of Variation	5.65	3.37	4.35	0.72	5.51	0.83	6.28	2.95	1.99	1.21

To determine if capping was required and if so, at what level, the distribution of grades for each variable within each mineralized zone was examined using lognormal cumulative frequency plots. In all cases, multiple overlapping lognormal populations were present. Cap levels were set to minimize the effects of a small number of erratic outliers.

A similar strategy was applied to Cu, Ag and W. The capping levels for each variable are shown in Table 14-4, Table 14-5, and Table 14-6.

Table 14-4: Summary of capping levels by mineralized zone

Domain	Variable	Cap Level	Number Capped
Cu-Ag Zone	MoS_2	0.16 %	4
Cu-Mo Zone	MoS_2	0.40 %	2
Mo Zone	MoS_2	0.48 %	7
MSI Zone	MoS_2		0
Dykes	MoS_2	0.05 %	1
Cu-Ag Zone	Cu	0.83 %	0
Cu-Mo Zone	Cu	0.62 %	4
Mo Zones	Cu	0.27 %	6
MSI Zones	Cu		0
Dykes	Cu	0.15 %	3
Cu-Ag Zone	Ag	115 ppm	6
Cu-Mo Zone	Ag	102 ppm	4
Mo Zone	Ag	24 ppm	4
MSI Zone	Ag	8 ppm	3
Dykes	Ag	4.0 ppm	3
Cu-Ag Zone	W	452 ppm	5
Cu-Mo Zone	W	277 ppm	6
Mo Zone	W	275 ppm	6
MSI Zone	W	118 ppm	3
Dykes	W		0

The results from capping are tabulated below with some significant reductions in the coefficient of variation for some variables.

Table 14-5: Summary of capped assay statistics for Cu and MoS_2 sorted by zone

Item	Cu–Ag Zone		Cu-Mo Zone		Mo Zone		MSI Zone		Dykes	
	MoS_2 (%)	Cu (%)	MoS_2 (%)	Cu (%)	MoS_2 (%)	Cu (%)	MoS_2 (%)	Cu (%)	MoS_2 (%)	Cu (%)
Number	3,813	3,813	3,509	3,509	2,677	2,677	330	330	128	128
Mean	0.017	0.076	0.049	0.103	0.112	0.053	0.057	0.028	0.005	0.016
Standard Deviation	0.018	0.074	0.040	0.070	0.063	0.041	0.029	0.038	0.009	0.036
Minimum	0.0005	0.001	0.0005	0.001	0.0005	0.001	0.0010	0.001	0.0005	0.001
Maximum	0.16	0.77	0.40	0.62	0.48	0.27	0.17	0.20	0.05	0.15
Coefficient of Variation	1.10	0.97	0.83	0.68	0.56	0.78	0.51	1.34	2.04	2.31

Note: MoS_2 here is calculated from the assays for Mo by multiplying by 1.6681.

Table 14-6: Summary of capped assay statistics for Ag and W sorted by zone

Item	Cu–Ag Zone		Cu-Mo Zone		Mo Zone		MSI Zone		Dykes	
	Ag (ppm)	W (ppm)	Ag (ppm)	W (ppm)	Ag (ppm)	W (ppm)	Ag (ppm)	W (ppm)	Ag (ppm)	W (ppm)
Number	3,806	3,791	3,492	3,497	2,653	2,654	327	330	128	121
Mean	2.56	29.8	2.88	46.4	1.58	45.5	0.99	31.3	0.57	9.8
Standard Deviation	5.82	32.4	4.63	31.7	1.78	32.8	1.28	20.7	1.00	11.9
Minimum	0.01	0.1	0.01	0.1	0.01	0.1	0.01	3.3	0.01	0.1
Maximum	115.0	452.0	102.0	277.0	24.0	275.0	8.0	118.0	4.00	65.0
Coefficient of Variation	2.27	1.09	1.61	0.68	1.13	0.72	1.29	0.66	1.75	1.21

14.3 50-Foot Composites

The bulk of the historic drill holes (1969 to 1982) were assayed on 10 or 20 ft intervals while those assayed by CuMoCo (2006-2012) were assayed on 10 ft intervals. A 50 ft composite length was chosen to match a reasonable mining bench for this scale of deposit. This differs from the 2012 resource estimate where a 20 ft composite was used. The statistics for 50 ft composites are shown in Table 14-7 Samples coded as oxide were combined with Cu-Ag composites for estimation purposes.

Table 14-7: Summary of 50 ft composite statistics

	MoS_2 (%)	Cu (%)	Ag (ppm)	W (ppm)
Cu-Ag Zone				
Number	810	810	810	807
Mean	0.016	0.076	2.68	29.8
Standard Deviation	0.013	0.062	4.77	28.1
Minimum	0.001	0.001	0.01	0.1
Maximum	0.101	0.432	92.39	365.1
Coefficient of Variation	0.80	0.82	1.78	0.94
Cu-Mo Zone				
Number	813	813	808	810
Mean	0.048	0.103	2.88	45.8
Standard Deviation	0.027	0.057	2.81	23.4
Minimum	0.003	0.003	0.22	5.4
Maximum	0.226	0.366	42.50	190.6
Coefficient of Variation	0.56	0.55	0.98	0.51
Mo Zone				
Number	639	639	631	631
Mean	0.112	0.053	1.64	46.7
Standard Deviation	0.046	0.037	1.27	24.1
Minimum	0.016	0.003	0.09	10.0
Maximum	0.302	0.218	10.68	160.0
Coefficient of Variation	0.41	0.69	0.77	0.52
MSI Zone				
Number	81	81	80	81
Mean	0.056	0.027	1.04	31.8
Standard Deviation	0.023	0.037	1.08	16.7
Minimum	0.003	0.002	0.05	6.4
Maximum	0.104	0.150	5.00	101.7
Coefficient of Variation	0.42	1.35	1.04	0.53
Dykes				
Number	37	37	37	35
Mean	0.004	0.014	0.55	10.5
Standard Deviation	0.005	0.026	0.80	12.1
Minimum	0.001	0.001	0.01	1.5
Maximum	0.019	0.082	3.00	60.0
Coefficient of Variation	1.40	1.90	1.46	1.16

Note: %MoS_2 statistics here are derived from assay grades calculated by multiplying %Mo by 1.6681.

14.4 Variography

For variogram analysis, the composite data was adjusted to accommodate post mineral faulting. Fault blocks were moved back to pre-fault locations based on marker beds displaced across fault boundaries. Semi-variograms were produced using these pre-fault locations. For estimation, the original locations of composites were used.

Pairwise, relative semi-variograms were used to determine grade continuity for MoS_2, Cu, Ag and W in 50 ft composites. Nested spherical models were fit to all directions with the model parameters tabulated below and the models shown in Appendix 4.

Table 14-8: Parameters for semi-variogram models at CuMo

Variable	Domains	Direction	C0	C1	C2	Short Range (ft)	Long Range (ft)
MoS_2	Cu-Mo and Mo Zone	Az 60 Dip 0	0.06	0.12	0.12	200	1800
		Az 330 Dip -35				400	500
		Az 150 Dip -55				300	1300
	Cu-Ag Zone	Az 0 Dip 0	0.16	0.16	0.20	200	1200
		Az 270 Dip 0				200	400
		Az 0 Dip -90				400	800
Cu	Cu-Ag and Cu-Mo Zone	Az 60 Dip 0	0.08	0.08	0.10	250	1600
		Az 330 Dip -35				500	700
		Az 150 Dip -55				300	1600
	Mo Zone	Az 60 Dip 0	0.05	0.15	0.15	400	1200
		Az 330 Dip 0				300	400
		Az 0 Dip -90				300	500
Ag	Cu-Ag and Cu-Mo Zone	Az 70 Dip 0	0.12	0.05	0.09	200	1000
		Az 340 Dip 0				50	200
		Az 0 Dip -90				120	500
	Mo Zone	Az 60 Dip 0	0.06	0.15	0.14	300	1200
		Az 330 Dip 0				300	500
		Az 0 Dip -90				450	700
W	Cu-Mo and Mo Zone	Az 0 Dip 0	.06	.02	0.15	150	1000
		Az 270 Dip 0				50	500
		Az 0 Dip -90				100	800
	Cu-Ag Zone	Az 30 Dip 0	0.08	0.11	0.17	160	1100
		Az 300 Dip 0				200	1200
		Az 0 Dip -90				300	400

There were insufficient composites within the MSI Zone to model so the models for the Mo Zone were applied to estimate this zone.

14.5 Block Model and Grade Estimation

A block model with blocks 50 x 50 x 50 ft in dimension was superimposed over the mineralized zones with the proportion of each block below surface topography and within the various mineralized solids recorded. The block model origin was as follows:

Lower Left Corner

- 214,600 E Column Size – 50 ft 207 Columns
- 114,250 N Row Size – 50 ft 179 Rows

Top of Model

- 7075 Elevation Level Size – 50 ft 76 Levels

The grades for the four variables namely: MoS_2, Cu, Ag and W were interpolated into each block containing some proportion of mineralized solid by ordinary kriging. Kriging was completed for each variable separately within two mineralized zones. A combination of soft and hard boundaries was used to estimate MoS_2, Cu, Ag and W to reflect the metal zonation present at the CuMo deposit.

Table 14-9: Estimation boundary summary

Mineral/Metal	Estimation Boundary Information
MoS_2	Estimated for Cu-Ag Zone using only composites from Cu-Ag and oxide zones
	Estimated for Cu-Mo and Mo Zones using only composites from Cu-Mo and Mo Zones
Cu	Estimated for Mo Zone using only composites from Mo Zone
	Estimated for Cu-Ag and Cu-Mo Zones using only composites from Cu-Ag, Cu-Mo and oxide Zones
Ag	Estimated for Mo Zone using only composites from Mo Zone
	Estimated for Cu-Ag and Cu-Mo Zone using only composites from Cu-Ag, Cu-Mo and Oxide Zone
W	Estimated for Cu-Ag Zone using only composites from Cu-Ag and Oxide Zones
	Estimated for Cu-Mo and Mo Zones using only composites from Cu-Mo and Mo Zones

Each kriging run was composed of four passes. The dimensions for the search ellipse, within each pass, were a function of the semi-variogram ranges. Pass 1 required a minimum of four composites within a search ellipsoid with dimensions equal to one quarter of the semi-variogram range for each direction. For blocks not estimated, the search ellipse was expanded to half the semi-variogram range in Pass 2 and again a minimum of four composites were required to estimate the block. Pass 3 expanded the search ellipse to the entire range, and a final fourth pass used double the range. In all cases, the maximum number of composites from a single hole was set to three to ensure that a minimum of two drill holes were used in each estimate. The maximum number of composites used was set to 16, and if more than 16 composites were found, the closest 16 were used. The search parameters for each run are listed below in Table 14-10. Pass 4 for Ag and W used larger search ellipses to produce a value for all blocks estimated for MoS_2 and Cu. This was due to the under-sampling of Ag and W relative to Mo and Cu.

A grade for each of the four variables was estimated in a total of 734,490 blocks.

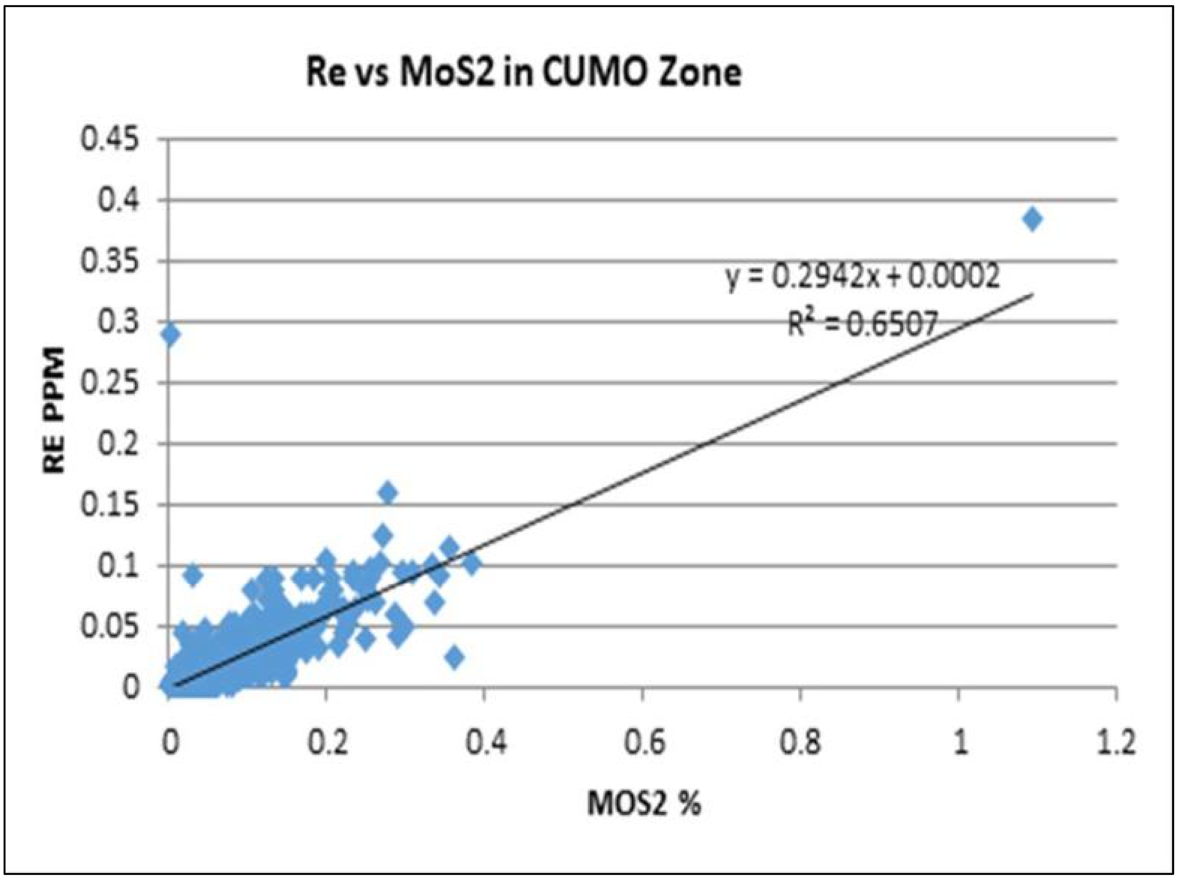
Table 14-10: Summary of kriging search parameters for each mineralized zone

Domain	Variable	Pass	Number Of Blocks Estimated	Az/Dip	Dist. (ft)	Az/Dip	Dist. (ft)	Az/Dip	Dist. (ft)
Cu-Ag	MoS_2	1	4,614	0 /0	300	270 / 0	100	0 / -90	200
		2	26,207		600		200		400
		3	83,342		1,200		400		800
		4	252,646		2,400		800		1,600
Cu-Mo & Mo	MoS_2	1	35,447	60 / 0	450	150 / -55	325	330 / -35	125
		2	110,887		900		650		250
		3	121,147		1,800		1,300		500
		4	59,784		3,600		2,600		1,000
Cu-Ag & Cu-Mo	Cu	1	50,852	60 / 0	400	150 / -55	175	330 / -35	400
		2	128,958		800		350		800
		3	235,739		1,600		700		1,600
		4	139,891		3,200		1,400		3,200
Mo	Cu	1	1,789		300		100		125
		2	22,307	60 / 0	600	330 / 0	200	0 / -90	250
		3	58,857		1,200		400		500
		4	80,068		2,400		800		1,000
Cu-Ag & Cu-Mo	Ag	1	1,859		250		50		125
		2	18,305	70 / 0	500	340 / 0	100	0 / -90	250
		3	94,108		1,000		200		500
		4	441,174		2000		400		1,000
Mo	Ag	1	3,067		300		125		175
		2	31,146		600		250		350
		3	63,317	60 / 0	1,200	330 / 0	500	0 / -90	700
		4	65,491		2,400		1,000		1,400
Cu-Ag	W	1	14,288		275		300		100
		2	51,953	30 / 0	550	300 / 0	600	0 / -90	200
		3	122,565		1,100		1,200		400
		4	179,224		2,200		2,400		800
Cu-Mo & Mo	W	1	4,799		250		125		200
		2	59,057	0 / 0	500	270 / 0	250	0 / -90	400
		3	130,570		1,000		500		800
		4	144,312		2000		1,000		1,600

Note: Distances shown in the table represent one quarter (Pass 1), one half (Pass 2), full (Pass 3) and twice (Pass 4) the semi-variogram range in the three principal directions.

Rhenium and sulfuric acid are credits received by the project during the roasting process at a roaster controlled by the project. A roaster and sulfur recovery plant have been built into the capital cost section of this report. Rhenium and sulfuric acid are contained solely within the molybdenite (MoS_2) – rhenium as an impurity within the molybdenite (MoS_2) structure, and sulfuric acid is produced from sulfur when the MoS_2 is converted to MoO_3. Due to the irregular nature of impurities and the sulfur content within the molybdenum, these cannot be estimated in blocks by kriging. Instead statistical linear regressions were applied to 7,485 analyses related to rhenium content in the molybdenite (MoS_2) within the various geological domains (mineralized zones) to determine the actual amount of these products produced. The results of the statistical linear regression are lower and more conservative than the rhenium recovery reported by SGS (2009).

Scatter plots were produced for each mineralized zone, plotting Re and S against MoS_2, and from these, a linear regression equation was used to estimate the amount of Re (ppm) and S (%) present on a block by block basis (see Figure 14-2 for an example plot showing Re vs MoS_2 in Cu-Mo Zone).



Source: Giroux et al , 2015

Figure 14-2: Scatter plot showing Re vs MoS_2 in the Cu-Mo Zone

For blocks containing more than one mineralized zone, a weighted average was produced. The two commodities are considered not as by-products of a producing mine but as smelter/processing credits from the concentrates. Smelter credits and penalties are common within the mining industry and in many cases, the credit or penalty element is not contained in the current resources or reserves of a project. The author has included the commodities to provide full disclosure as circuits to recover and produce these products are built into capital and operating costs. Re and S values have not been used to determine the RCV of blocks. The contribution of these commodities to the overall economic analyses is small and well within the accuracy of the PEA level of study, with rhenium contributing 0.37% of the overall revenue and sulfuric acid 0.49%. Rhenium is of special interest to the development of the property as it is now on a list of minerals that are critical to the USA.

Note: Regression analysis is not industry standard practice in calculating overall resources. However, the fact that rhenium and sulfur are contained almost entirely within the material containing MoS_2, which has been estimated by kriging, means that regression is a valid method of obtaining a reasonable estimate of the rhenium and sulfur contents at the level of precision of this study. Due to the large number of samples involved in the regression analysis, the confidence of this particular regression estimate is comparable to that obtained by the method of ordinary kriging.

14.6 Bulk Density

A total of 4,539 specific gravity determinations were made for CuMo in all mineralized zones. This total includes 4,339 determinations made during the 2011 drill program. The measurements were made using the weight in air/weight in water procedure. The results are summarized in Table 14-11.

Table 14-11: Summary of density parameters for each mineralized zone

Zone	Number of SG Determinations	SG Minimum	SG Maximum	Average SG (gm/cc)	Average TF (cu.ft/ton)
Ox	578	2.08	2.74	2.50	12.80
Cu-Ag	1,505	2.28	3.70	2.58	12.42
Cu-Mo	1,524	2.25	2.85	2.58	12.40
Mo	763	2.30	2.75	2.57	12.45
Msi	91	2.40	2.73	2.57	12.48
Dyke	78	2.19	2.75	2.52	12.71
Total	**4,539**	**2.08**	**3.70**	**2.57**	

The tonnage factor for each block was a weighted average based on the mineralized zone's tonnage factor and the amount of that zone within the block.

14.7 Reasonable Prospects of Eventual Economic Extraction

Reasonable prospects of eventual economic extraction have been established by constraining the resource estimate to within a conceptual open pit design using reasonable parameters from an analogous nearby molybdenum deposit. An RCV in non-oxide material of $5.00/t has been highlighted as a possible open pit cut-off based on similar size mines at a feasibility or production stage. In the mineral resource tables at the end of this section, the $5.00 cut-off for the assumed price is highlighted and is selected based on operating costs. The $5.00 cut-off is suggested to separate waste from material that is fed to the sorters. From the sorters, only mill feed above an economic cut-off would be sent for immediate processing.

In 2012, Snowden used Geovia's Whittle™ pit optimizer to determine a constraining open pit for the CuMo deposit. Optimization parameters were from the Thompson Creek mine (a comparable open pit molybdenum project located in Idaho). The optimization parameters included mill feed mining and processing costs of $7.52 per processed ton, overall pit slope angles of 45°, metallurgical recoveries as shown in Table 14-12 and appropriate dilution and offsite costs and royalties. The metal prices used in 2012 by Snowden for pit optimization were Mo at $25/lb, Cu at $3/lb, Ag at $20/oz and W at $10/lb.

Since the infill drill holes completed in 2011-12 were all within this conceptual pit this resource update uses the Snowden 2012 optimum pit shell to both constrain the estimate and demonstrate reasonable prospects of eventual economic extraction.

14.8 Resource Classification

At CuMo, geological continuity has been established through diamond drilling. The concentric zonation and faults have been used to constrain the mineralization in a series of mineralized zones. Grade continuity within the mineralized zones has been determined by semi-variograms for each variable.

Semi-variograms are an aspect of data analysis that assist in defining the correlation and range of influence of a grade variable in various directions in three dimensions. Semi-variograms are a graphical geostatistical tool used to determine the direction and range over which samples show continuity. The semi-variogram plots the mean squared difference between samples as an increasing function of distance between samples, and as the distance between samples increases, it reaches a point (the range) where samples are no longer correlated.

In this case, the semi-variogram analysis was completed after moving major fault blocks back to pre-fault positions. The kriging procedure was completed on fault blocks in their current positions; thus by using the range in each of the major directions, the grade continuity can be quantified. This in turn can be used to establish classification levels.

The resource is classified in accordance with the 2014 CIM Definition Standards.

Measured

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

Contiguous blocks within the Cu-Ag and Cu-Mo Zones estimated in Pass 1 (using one quarter of the semi-variogram range) for both MoS_2 and Cu were classified as measured. For the Mo and MSI Zones where Cu, Ag and W provide little of the economic benefit contiguous blocks estimated in Pass 1 for MoS_2 were classified as measured.

Indicated

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

Indicated blocks were established from unclassified blocks estimated for Cu or MoS_2 in Pass 1 or 2 using search ellipses up to a maximum of one half the semi-variogram range.

Inferred

All other blocks were classified as inferred.

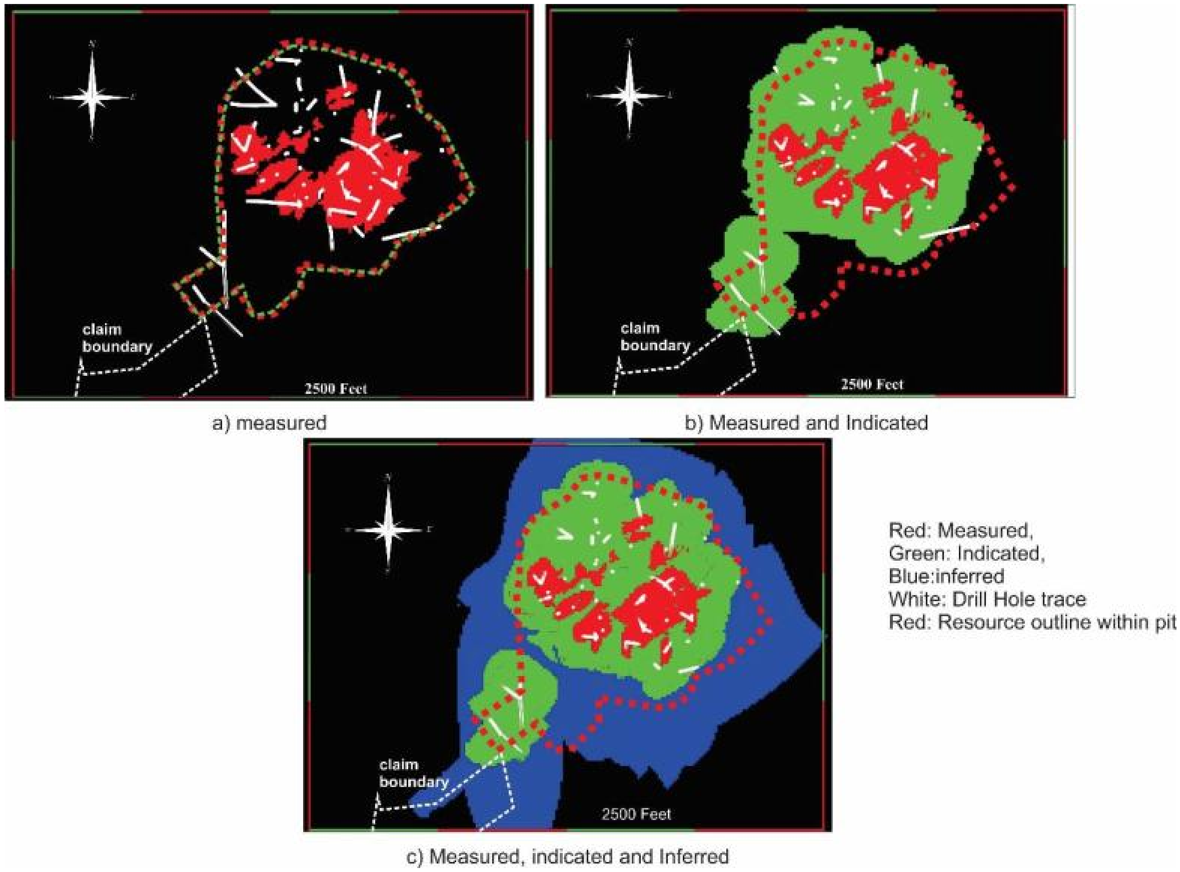
An inferred mineral resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

The author in classifying the mineral resource into the three different categories has examined the characteristics of the mineralization and associated reports and preliminary data and information concerning mining, metallurgy, economics and social and environmental sensitivity and has determined that the classification meets the requirement of reasonable prospects of eventual economic extraction in regard to this PEA study.

Specifically, it must be noted that metallurgical tests to separate a combined Cu-Mo concentrate into separate saleable concentrates have yet to be completed. However, SGS 2009, as outlined in section 13.1.5, indicates that there is no reason that separate saleable copper and molybdenum concentrates cannot be produced. In addition, based on similar operations at Las Pelambres, Andina, Collahuasi, Gibraltar and Sierrita, there is no reason to indicate that this concentrate separation cannot be produced with additional metallurgical testing prior to a pre-feasibility study. Given this information, the author is confident that the metallurgical work would allow the application of modifying factors to support future detailed mine planning and the final evaluation of the economic viability of the deposit.

Figure 14-3 shows indicative plan views of the measured, indicated and inferred blocks at CuMo.

Note: As with the 2015 resource estimate, the current resource is constrained within the 2012 Snowden pit. Figure 10-1, Figure 10-2 and Figure 10-3 show the outline of the 2012 Snowden constraining pit, and a projection of categorized blocks.



Source: Giroux et al , 2015, modified 2019

Note: The above shows all blocks estimated. The outline of the 2012 Snowden constraining pit has been added. The blocks within this constraining pit are summarized in the various tables.

Figure 14-3: Plan views of the measured, indicated and inferred blocks at CuMo

14.9 Recovered Value

To properly evaluate the CuMo deposit with four metals occurring in different zones, A factor named recovered value, or RCV, was used. This calculation used metal prices in US dollars and metal recoveries.

The RCV calculations were based on the set of prices defined in Table 14-13.

Table 14-12: Metal prices for resources

Metal	Price
Copper (Cu), $/lb	3.00
Molybdenum trioxide (MoO_3), $/lb	10.00
Molybdenum Metal (Mo), $/lb	15.00
Silver (Ag), $/ounce	12.50

Molybdenum is sold as molybdenum trioxide (MoO3) which has higher Mo content. The price used in this study for MoO_3 is $10/lb. MoO_3 is calculated from MoS_2 by the following:

- Pounds Mo = MoS_2 * 20 / 1.6681
- Pounds MoO_3 = Pounds Mo * 1.5

The metal recoveries used to calculate RCV were a function of mineralized zones as follows:

Table 14-13: Metal recoveries sorted by mineralized zone

Metal	%Recoveries in Oxides	%Recoveries in Cu-Ag Zone	%Recoveries in Cu-Mo Zone	%Recoveries in Mo & MSI Zones
Cu	60.0	68.0	85.0	72.0
Mo	80.0	86.0	92.0	95.0
Ag	65.0	75.0	78.0	55.0

Note: The recoveries for all metals in the MSI Zone were similar to the Mo Zone

Factors to use in RCV equation were as follows:

MoS_2 Factor ($/ton) = MoS_2 % * Mo Recovery % * 2000 lbs/ton * $/lb MoO_3 * 1.5/1.6881

Cu Factor ($/ton) = Cu % * Cu Recovery % * 2000 lbs/ton * $/lb Cu

$$\text{Ag Factor (\$/ton)} = \frac{\text{Ag ppm * Ag Recovery \% * \$/oz Ag}}{\text{31.1035 g/oz * 1.1023 tons/tonne}}$$

The equations to calculated RCV for each mineralized zone were as follows:

RCV (oxides) = (Cu% * 36.0) + (Ag(ppm) * 0.24) + (MoS_2% * 143.88)

RCV (Cu-Ag) = (Cu% * 40.8) + (Ag(ppm) * 0.27) + (MoS_2% * 154.67)

RCV (Cu-Mo) = (Cu% * 51.0) + (Ag(ppm) * 0.28) + (MoS_2% * 165.46)

RCV (Mo) = (Cu% * 43.2) + (Ag(ppm) * 0.20) + + (MoS_2% * 170.85)

RCV (MSI) = (Cu% * 43.2) + (Ag(ppm) * 0.20) + + (MoS_2% * 170.85)

For blocks overlapping the mineralized zone boundaries, a weighted average RCV was produced.

A complete series of tables for each zone (5), each classification (4) and for each price regime (3) plus an overall set were produced (a total of 72 tables). For the purposes of this summary report, the mineral resource estimate described next was for all zones.

14.10 Mineral Resource Estimate

Table 14-14. Table 14-15, Table 14-16, and Table 14-17 report the overall mineral resource estimated within the Snowden 2012 open pit shell at a variety of RCV cut-offs. The $5.00/t cut-off is highlighted as an appropriate RCV cut-off based on grade improvements using mineral sorting processes. The base case $5.00 cut-off is suggested to separate waste from material that is fed to the sorters. The actual cut-off used in economic analysis and mine design will vary depending on numerous conditions at the time of the calculation: including metal prices, recoveries and operating costs.

It should be noted that since the convention for the CuMo project has been to work with %MoS_2, as calculated from measured %Mo, the %MoS_2 values in the resource estimate tables are 1.6681 times greater than %Mo.

Table 14-14: Measured resources

Cut-off RCV ($)	Grade > RCV Cut-off					Contained Metal				
	Quantity (Mt)	MoS_2 (%)	Cu (%)	Ag (ppm)	RCV ($)	Re (ppm)	S (%)	Mo (mmlbs)	Cu (mmlbs)	Ag (Moz)
2.5	308.4	0.079	0.074	2.09	17.32	0.029	0.233	292.1	456.5	18.8
5.0	297.2	0.081	0.076	**2.09**	**17.83**	**0.03**	**0.229**	**288.6**	**451.7**	**18.1**
7.5	282	0.085	0.076	2.06	18.48	0.031	0.223	287.4	428.7	16.9
12.5	227.9	0.097	0.075	2	20.50	0.036	0.217	265	341.8	13.3
15.0	195.4	0.105	0.072	1.9	21.71	0.039	0.212	246	281.3	10.8
17.5	159.7	0.115	0.067	1.8	23.04	0.043	0.207	220.1	213.9	8.4
20.0	122.9	0.125	0.063	1.7	24.50	0.047	0.202	184.1	154.8	6.1

Source: Giroux et al, 2015, modified 2019

Table 14-15: Indicated resources

Cut-off RCV ($)	Grade > RCV Cut-off					Contained Metal				
	Quantity (Mt)	MoS_2 (%)	Cu (%)	Ag (ppm)	RCV ($)	Re (ppm)	S (%)	Mo (mmlbs)	Cu (mmlbs)	Ag (Moz)
2.5	2216.1	0.049	0.079	2.48	12.32	0.018	0.277	1301.9	3501.4	160.3
5.0	**1972.3**	**0.053**	**0.085**	**2.57**	**13.40**	**0.019**	**0.269**	**1253.3**	**3352.9**	**147.8**
7.5	1708.3	0.059	0.088	2.59	14.55	0.021	0.258	1208.4	3006.5	129
12.5	1050.6	0.076	0.09	2.55	17.67	0.027	0.235	957.4	1891.1	78.1
15.0	798.5	0.083	0.09	2.56	19.06	0.03	0.231	794.6	1437.2	59.6
17.5	541.6	0.093	0.088	2.49	20.60	0.034	0.226	603.9	953.2	39.3
20.0	301.3	0.106	0.082	2.36	22.49	0.039	0.219	383	494.2	20.7

Source: Giroux et al, 2015, modified 2019

Table 14-16: Measured and indicated resources

Cut-off RCV ($)	Grade > RCV Cut-off					Contained Metal				
	Quantity (Mt)	MoS_2 (%)	Cu (%)	Ag (ppm)	RCV ($)	Re (ppm)	S (%)	Mo (mmlbs)	Cu (mmlbs)	Ag (Moz)
2.5	2524.6	0.053	0.079	2.43	12.93	0.019	0.272	1604.3	3988.9	178.9
5.0	**2269.6**	**0.057**	**0.084**	**2.5**	**13.98**	**0.021**	**0.264**	**1551.1**	**3812.9**	**165.5**
7.5	1990.4	0.063	0.086	2.51	15.10	0.022	0.253	1503.5	3423.5	145.7
12.5	1278.6	0.079	0.087	2.46	18.17	0.029	0.232	1211.1	2224.8	91.7
15.0	993.9	0.088	0.087	2.43	19.58	0.032	0.227	1048.7	1729.5	70.4
17.5	701.4	0.098	0.083	2.33	21.16	0.036	0.221	824.1	1164.2	47.7
20.0	424.3	0.112	0.077	2.17	23.07	0.041	0.214	569.8	653.4	26.9

Source: Giroux et al, 2015, modified 2019

Table 14-17: Inferred resources (molybdenum, copper, silver, rhenium and sulfur)

Cut-off RCV ($)	Grade > RCV Cut-off					Contained Metal				
	Quantity (Mt)	MoS_2 (%)	Cu (%)	Ag (ppm)	RCV ($)	Re (ppm)	S (%)	Mo (mmlbs)	Cu (mmlbs)	Ag (Moz)
2.5	3373.6	0.04	0.057	1.93	9.55	0.014	0.304	1617.9	3845.9	189.9
5.0	**2556.6**	**0.048**	**0.067**	**2.13**	**11.48**	**0.017**	**0.282**	**1471.4**	**3425.9**	**158.8**
7.5	1996	0.056	0.07	2.23	13.07	0.02	0.261	1340.1	2794.4	129.8
12.5	996.4	0.078	0.064	1.98	16.74	0.028	0.231	931.8	1275.4	57.5
15.0	637	0.086	0.074	2.16	18.63	0.03	0.244	656.8	942.7	40.1
17.5	384.8	0.094	0.084	2.34	20.49	0.032	0.259	433.7	646.4	26.3
20.0	190.2	0.109	0.078	2.37	22.80	0.037	0.262	248.6	296.8	13.1

Source: Giroux et al, 2015, modified 2019

Note: Mineral resources that are not mineral reserves do not have demonstrated economic viability.

15 Mineral Reserve Estimates

No mineral reserve estimates, as defined by CIM Definition Standards, currently exist for the CuMo project.

16 Mining Methods

16.1 Mining Approach

The CuMo project is envisaged to be developed using open pit mining techniques. The scale of the deposit is such that ultra-class mining equipment (e.g. 400-ton trucks) has been considered for the purposes of this PEA. As well, to improve the head grade of the flotation mill feed, the author has considered the implementation of mineral sorting for the project. Specifically, the author envisions a multi-stage bulk sorting process accompanied by a final particle-sorting stage to upgrade the mil feed. These are described further in Section 17.2 and 17.3. The result of the mineral sorting strategies is a reduction of waste fed to the mill, thereby improving feed head grade. There is however added cost, but this is all taken into consideration in determining the potential mineable resource for the project.

16.2 Pit Optimization

Under supervision of the author, SRK applied Lerchs Grossman pit optimization techniques using Geovia's Whittle™ software to generate potential pit shells for mining. The inputs and outcomes of this process are described herein.

16.2.1 Pit Geotechnical Considerations

The author conducted a basic, PEA level, geotechnical assessment to define pit wall slope inputs for the pit design. The assessment comprised:

- A review of the existing geotechnical data sources
- A site visit to view the proposed pit footprint and evaluate the historical core
- An assessment of the extents and confidence level of with the data
- Processing of the data for rock mass characterization and classification
- Formulation of pit wall recommendations

Data Sources

A summary of the reviewed data types provided by CuMoCo pertinent to this geotechnical assessment is presented in Table 16-1.

Table 16-1: Summary of the reviewed data types

Data Type	Details
Technical Report	NI 43-101 Resource Estimate Update (Snowden), dated 12 June 2012
Geology Model	2-D schematic map and sections illustrating the deposit major geology, alteration, mineralization structural regime
Drill hole Database	Drill hole database comprising; • exploration drill holes from the 2006 to 2012 drilling campaigns, including; lithology, alteration, mineralized zone and RQD. Included three • three geotechnical drill holes from the 2010 drilling campaign with detailed properties logged to $RMR_L(90)$.
Core Photographs	Core photographs from geotechnical drill hole C10-55
Topography	Site topographic surface
Pit Shell	Snowden resource pit shell

On 30 October 2018, a senior geotechnical engineer from SRK (the “author” of this sub-section) visited the CuMo project site, the area of the proposed pit footprint and the project core facility. Observations were made of the site setting, rock exposures and the core from geotechnical drill hole C10-55 was viewed. These observations were considered for the analysis and design herein.

Snowden Report

Snowden conducted a resource estimate update and technical report in 2012 (Snowden, 2012). There was no geotechnical assessment undertaken as part of the study. Pit wall slopes used in the PEA were given for ground elevation intervals and became shallower with pit depth i.e. the upper interval was 45°, the next 40°, and the north south and west walls had a lower interval of 35°. The resultant overall slope angles (OSA) calculated using this configuration are shown in Table 16-2.

The pit-shell used to constrain the resource estimation is not the same pit-shell as used to derive the mine plan used for this PEA. The design parameters for the PEA mine plan and resulting pits are discussed in Sections 16.2.3 16.2.4 and 16.2.5.

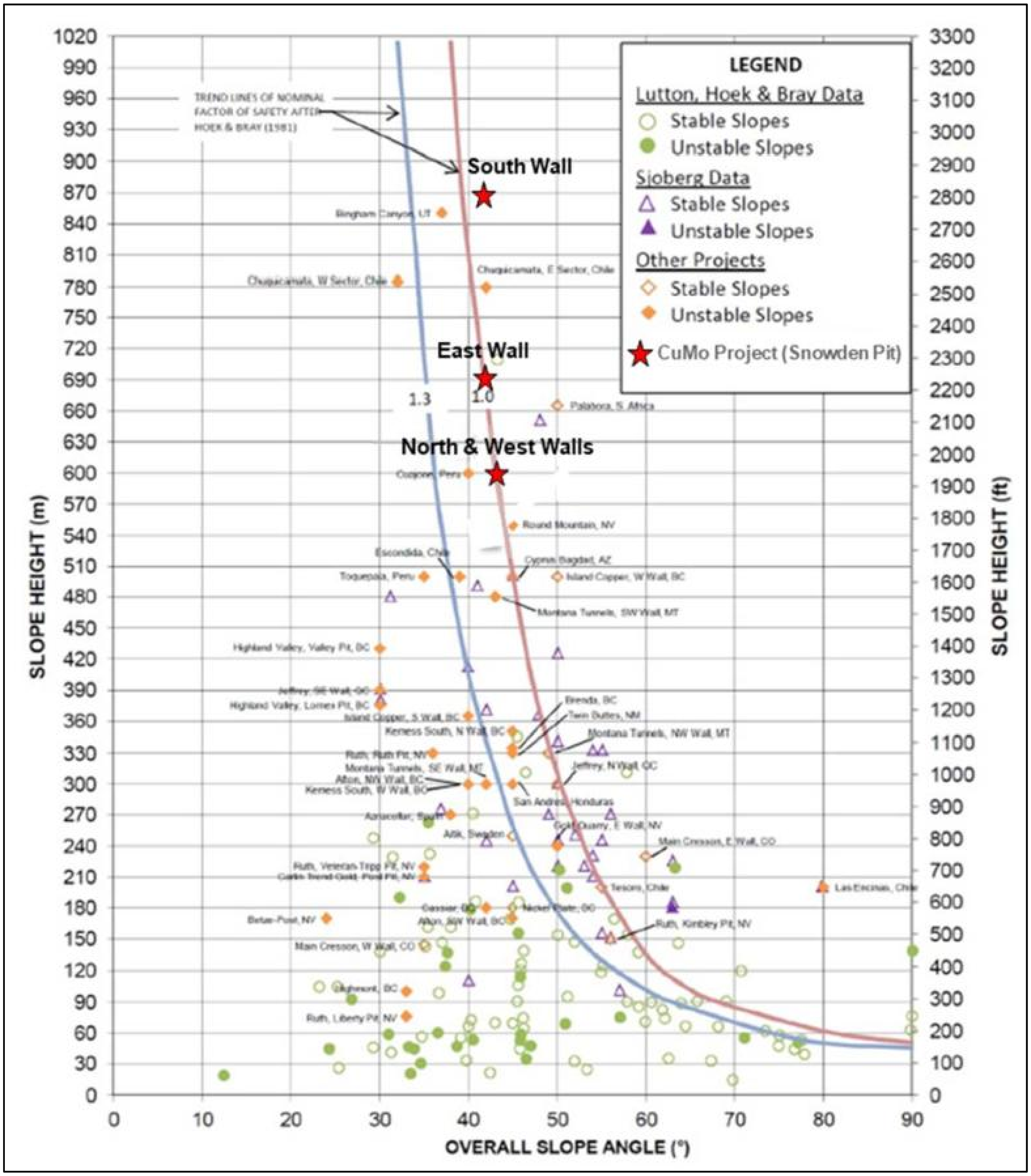
Table 16-2: Pit slope design details in Snowden (2012)

Pit Sector	Snowden Resource Pit Shell Depth (m)	Bench Height (m)	Calculated OSA (°)
North wall	670	15	42
East wall	590	15	43
South wall	850	15	42
West wall	670	15	42

These wall heights and design slope angles were plotted on an industry-recognized empirical chart showing various operations around the world Figure 16-1. Also plotted on the chart are separately-derived ‘trend lines of nominal factor of safety’ (after Hoek and Bray, 1981). Note that pit slope stability depends on a variety of site-specific factors which makes it difficult to directly compare with other sites, but the chart is still useful for benchmarking at early project design stage.

At an ultimate depth of over 600 m, the conceptual design walls are significantly higher than most operating mines in North America. The precedent for cases is very limited but the pits that are of those heights have all encountered slope stability problems in some areas of the mine. The plot suggests that these proposed OSAs fall around FoS of equity and may not be achievable.

As a result, the author proceeded to investigate and study further the wall heights and slope angles.



Source: SRK, 2019

Figure 16-1: Empirical pit wall chart with the Snowden (2012) walls added

Data Processing

The author undertook these tasks with the project data:

- Viewed the project drill hole traces relative to the Snowden (2012) ultimate pit shell
- Modelled the project RQD data to produce wireframes of binned values broadly equating to; very poor (<20), poor (21-40), fair (41-60), good (61-80), very good (81-100)
- Processed the detailed geotechnical data for RMR_L90 rock mass rating and viewed the colour-coded values over the full depth of the hole
- QA/AC check of the core photos against the calculated RMR_L90 values
- Basic statistical analysis of the RQD and RMR_L90 data

Key Observations and Findings

The author makes these observations and findings from the data review and processing:

- The three geotechnical drill holes are clustered together in the south western area of the deposit, hence spatial (and depth) coverage is very limited.
- The primary geotechnical drill hole (C10-55) is aligned roughly parallel with the pit wall, hence geotechnically-important structures aligned this way will be invisible or under-represented.
- The core from drill hole C10-55 showed a deep weathering profile, was generally highly fractured largely due to medium to high micro-defect intensity, and included damage zones, gauge, breccia and rubble probably associated with large-scale brittle structures e.g. faults, shear zones.
- The model of the project RQD data shows that most of the core was logged as ‘poor’ to ‘very poor’ RQD, with small core zones which are ‘fair’. The mean RQD value for the data set was 18.
- The RMR_L90 values in drill hole C10-55 showed a large spread over the range of 25 to 70, with a mean value of 46. The colour-coded plot showed no obvious zonation or increase in values with depth.
- The QA/QC check found that there were sections of core that had similar RMR_L90 values and yet had distinctly different rock quality in the core photos. This points to possible errors, or inconsistencies, in the logging practices and the project geotechnical data set.

Based on these observations and findings, the author concludes that pit slope stability of the deposit will likely be controlled by rock mass strength and major structures.

Pit Slope Evaluation

These tasks were conducted to reach an evaluation of the possible achievable pit slopes:

- Using professional judgement and experience, the author selected and applied mining adjustment factors for; structures orientation, induced stresses, blasting and weathering, to get Mining RMR (MRMR) ratings for drill hole C10-55.

- Selected a pit design acceptance criterion of FoS=1.3 which is common for inter-ramp slopes (although in more advance design stages a higher factor of safety may be used locally around critical pit infrastructure such as ramps).
- Using the well-known and industry-recognized Haynes and Terbrugge (1990) chart and considering an inter-ramp stack height of around 350 ft, the MRMR values were plotted to find pit slope angles. Reflective of the small data set and low confidence in the values, the lower half of the calculated MRMR range was plotted and used for the assessment.

Pit Slope Recommendations

For the PEA pit design, the author recommends the following pit wall design criteria:

- maximum bench height of 50 ft
- bench width of 26 ft
- inter-ramp wall angles of 42°
- geotechnical berm of 65 ft every seven benches

To allow for geotechnical berms and a spiral ramp to reach the pit bottom, derived an overall slope angle to be used in pit shell definition of 37°.

Note these major limitations of the geotechnical assessment:

- It was largely based on data from one geotechnical drill hole in one area of the deposit only. It does not provide coverage of the geology, alteration, mineralization units and regimes present over the site, nor of the open pit depth extent.
- The site groundwater regime, or phreatic surface/s, were not considered. Porewater pressures can have a significant strength reduction and destabilizing effect on slopes.
- The potential presence of lower-angle major fault structures could impact the overall slopes and may require specific design recommendations and/or mitigation strategies including flattening of the slopes.

16.2.2 Bulk Sorting

Sort Analysis of Drill Hole Data

As mentioned, mineral sorting is being considered for the CuMo project to improve the grade of the mill feed. The description here is for the adoption of bulk sorting at CuMo.

In preparing the drill hole data for a sort analysis, the author applied factors to account for expected sorting conditions or inefficiencies. In particular, two factors were considered – dilution zone thickness at sample interval contacts and minimum thickness of sample interval. The first represents possible mixing that can occur during blasting or in material handling. The grades in this zone are the average of adjacent sample intervals. The second factor typically considers thin intersections of sample intervals after bench compositing. Considered values for dilution and minimum thicknesses at CuMo ranged from zero to two feet. In the end, a 2-ft dilution zone per interval was used, with no consideration of minimum sample interval thickness.

The bulk sort analysis starts with considering grade control in the mine, whereby the author selected a sort feed cut-off to determine what goes to the sort plant versus what goes to waste. Then, using the composite-sample relationships discussed in Section 13.2.2, the author ran several scenarios examining the impact of multiple cut-off RCVs. In addition to the grade control cut-off, the author considered two RCV cut-offs for a given stage of sorting. Material below the lower RCV cut-off would be rejected as waste in the sort process, and material above the upper RCV cut-off would represent feed to the mill. Material between the cut-offs is referred to as “middlings”.

It was possible with these simulations of sorting, conducted directly on the drill hole data, to assess which combinations of cut-off grades produced the best results in terms of improved metal grades of the mill feed fraction and increased waste rejection. A final sort analysis however needed to be applied to the resource blocks to be able to assess preliminary economics that balance metal recoveries and waste rejection. The drill hole analysis results however provided a good starting point.

Sort Analysis of Resource Block Model

The composite-sample analysis discussed in Section 13.2.2 provides relationships between bench composite RCV and sample interval RCV. However, for sort analysis of blocks, selecting drill hole composite-sample relationships based on matching RCV grades is not possible due to the volume-variance effect. Block models generally have lower grades than the underlying drill hole data. To overcome this, the drill hole composite-sample relationships were re-expressed on a percentile RCV basis. Ranges or “bins” of percentile RCV (in 10% intervals) were thus set up for the drill hole composites and the corresponding composite-sample relationships were re-estimated for the drill hole data.

Then, using the 3-D mineral resource block model described in Section 14, the author performed a sort analysis for blocks contained within the pit shell used by Snowden (Snowden, 2012) to constrain the mineral resource. The percentile RCV of a block is compared to the percentile RCV ranges for the drill hole composites to select the applicable composite-sample relationship for sorting. Again, by applying a cut-off RCV for waste and another for mill feed, the block could be segregated into three products, waste, mill feed, and middlings, according to the composite-sample relationship.

To maximize the benefit of bulk sorting, and to take advantage of increased heterogeneity at smaller scales, multiple stages of bulk sorting were considered. The middlings portion became the feed for the subsequent sorting stages. As well, the middlings product streams were split in two to further reduce the volume of batches for sorting and thus increase the heterogeneity (per conclusions of Section 13.2.2).

To determine the composite-sample relationships that would apply to subsequent stages of sorting, the RCV of the middlings was re-calculated from the reported Cu, Mo, and Ag grades. This RCV value was compared to the drill hole composite analysis to derive the corresponding composite percentile RCV range. The composite-sample relationship for this range was then used to predict the results of bulk sort.

Using the original composite-sample relationship at each sort stage is seen to be conservative. As was observed for CuMo (Section 13.2.2), the smaller the scale observed, such as at a later stage bulk sort, the greater is the heterogeneity, thus improving discrimination around cut-off grades. The limitation for the CuMo project is the drill hole sample length (10-ft) which precludes shorter interval heterogeneity analysis.

Final Bulk Sort Parameters

For this PEA, three stages of bulk sorting were run on the block model. The grouping of cut-offs which appear to produce the best economic results are as follows:

- Grade control cut-off RCV = $7.50/t
- Stage 1 Bulk Sort – Lower cut-off = $7.50/t; upper cut-off = $20.00/t
- Stage 2 Bulk Sort – Lower cut-off = $7.50/t; upper cut-off = $17.50/t
- Stage 3 Bulk Sort – Lower cut-off = $7.50/t; upper cut-off = $15.00/t

Re-use of the same cut-offs (e.g. the lower cut-off segregating waste from middlings) is allowed as it is recognized that any bulk sort is not precise and that sort products will continue to contain a mix of material across the full range of sample grades.

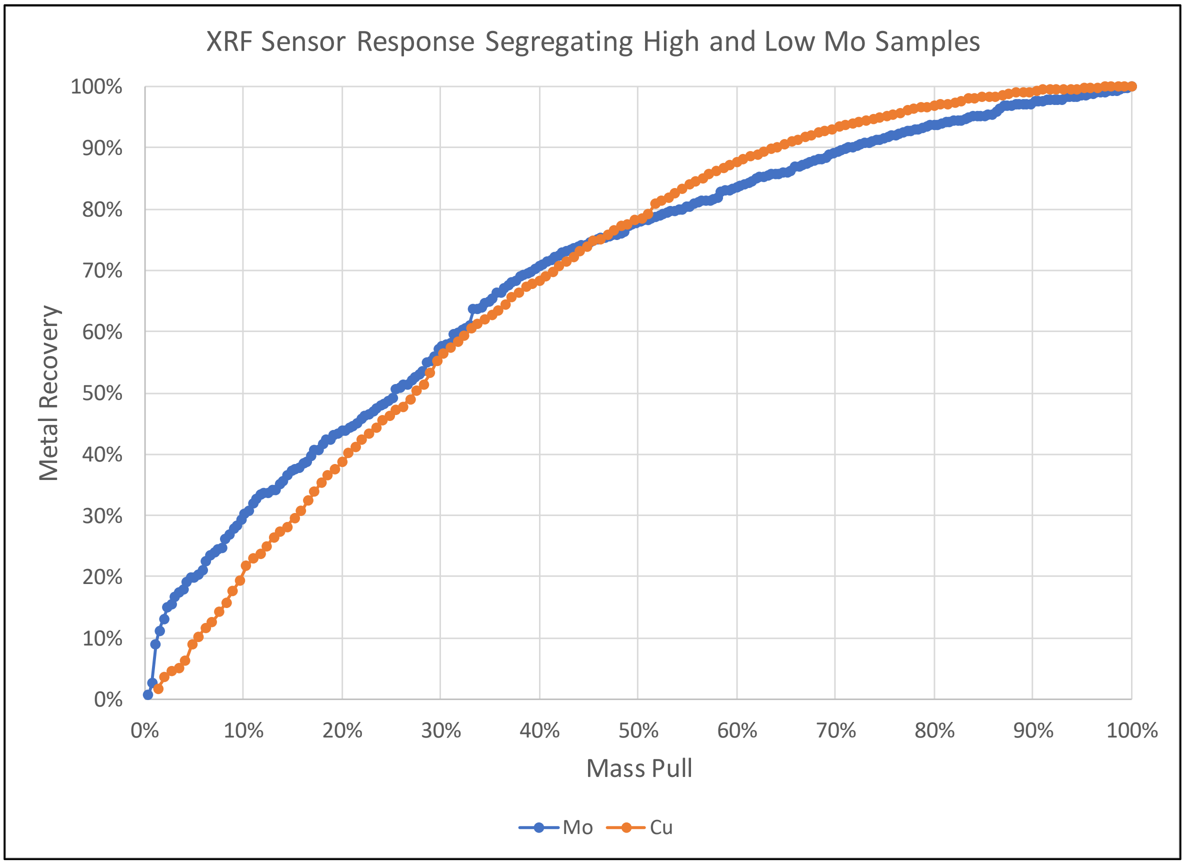
The outcome of the block sorting analysis are blocks coded with tonnages of waste, mill feed, and middlings. Grades were calculated for each of these fractions in each block. As well, sorting costs were determined by applying $0.10/t for each bulk sort stage as well as an initial primary crushing cost of $0.20/t, which applies to all material fed to the sorting plant. This version of the block model was then used for mine planning.

16.2.3 Particle Sorting

A review of the particle sorting test work (Section 13.2.1) highlighted that the recovery of copper is not as good as molybdenum when RCV alone is the primary measure for sorting. Consequently, the author undertook a limited bivariate analysis of the test results, whereby the test samples were segregated into Mo-rich and Mo-poor samples. This would allow sorting based on Mo XRF grade for the Mo-rich samples and on Cu XRF grade for the Mo-poor samples.

In addition, as it was recognized that particle sorting was to come after bulk sorting, it was appropriate to cap the value of samples to be used in the analysis. A review of the samples showed that an RCV cap of $60/t was appropriate for the feed to particle sorting. Lastly, the particle sort analysis was weighted by the portions of the mineralized zones contained within the eventual PEA pit (Note that to reduce complexity, the benefit of particle sorting was not applied in pit optimization, but rather prior to economics. This is a more conservative but acceptable approach.)

Multiple Mo grades were tested as cut points to segregate the samples into Mo-rich and Mo-poor populations. It was found that a 100 ppm Mo cut point had the best outcomes, which are provided in Figure 16-2. For this figure, the Mo-rich samples are ranked (sorted) based on Mo grade, while the Mo-poor samples are ranked by Cu grade. As can be seen, the Cu response is considerably improved (vs Figure 13-1), while the Mo response is somewhat muted. This is fine as it was found that through the bulk sorting analysis, most of the higher Mo grade material was pulled to mill feed, leaving Cu with greater potential for particle sorting.



Source: SRK, 2019

Figure 16-2: Particle sort analysis splitting between Mo-rich and Mo-poor samples

A variety of different cut points and mass pulls were subsequently tested in the project economics, with the most promising being the 100 ppm Mo cut point and a mass pull of around 40%. The specific sort parameters are provided below:

- Mass pull - 40.8%
- Mo recovery – 56.9%
- Cu recovery – 52.6%

Again, these parameters were only used in project economics, not pit optimization, which is discussed further below.

16.2.4 Pit Optimization Input Parameters

The 3-D resource block model was imported to MineSight^{TM} mine design software in order to populate the blocks with the results of the sorting analysis. The new updated model was transferred to Whittle^{TM} optimization software to carry out the pit optimization work in order to generate conceptual mining and processing schedules for the Preliminary Economic Analysis contained in this report. The pit shells that resulted are new, and are contained within, but are not be confused with, the 2012 Snowdon resource-constraining shell used for resource estimation.

In consultation with CuMoCo, assumptions were made for metal pricing (Mo, Cu, Ag) and offsite costs. Open pit mining costs were estimated to reflect expected haul destinations for waste and mill feed, taking advantage reduced haulage expected in early years. As well, to model the impact of a potential pre-strip period, which would be capitalized in the economic analysis, material above a selected elevation (6,100 ft) was assigned zero cost. Processing costs were based on a prior Ausenco trade-off study for plant throughput (Ausenco, 2009). A mill feed of 150,000 stpd was considered. Another assumption is that the project will build a roaster to treat the molybdenite (MoS_2) concentrate.

A summary of the input parameters used is presented in Table 16-3.

Whittle™ open pit optimization software was then used to generate new pit shells for mine planning, using the resource block model updated with sorting results. The economically defined pit shell limits included measured, indicated and inferred mineral resources.

An inferred mineral resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

Inferred mineral resources must be excluded from estimates forming the basis of feasibility or other economic studies.

16.2.5 Optimization Results

A series of optimized pit shells were generated for the CuMo deposit based on varying revenue factors (base metal price multiples). The results of the pit optimization evaluation on the deposit for varying revenue factors values are presented in Figure 16-4. Note the NPV in this optimization summary does not take into account capital costs and is used only as a guide in shell selection and determination of the mining shapes. The actual NPV of the project is summarized in the economics section of this report (Section 22).

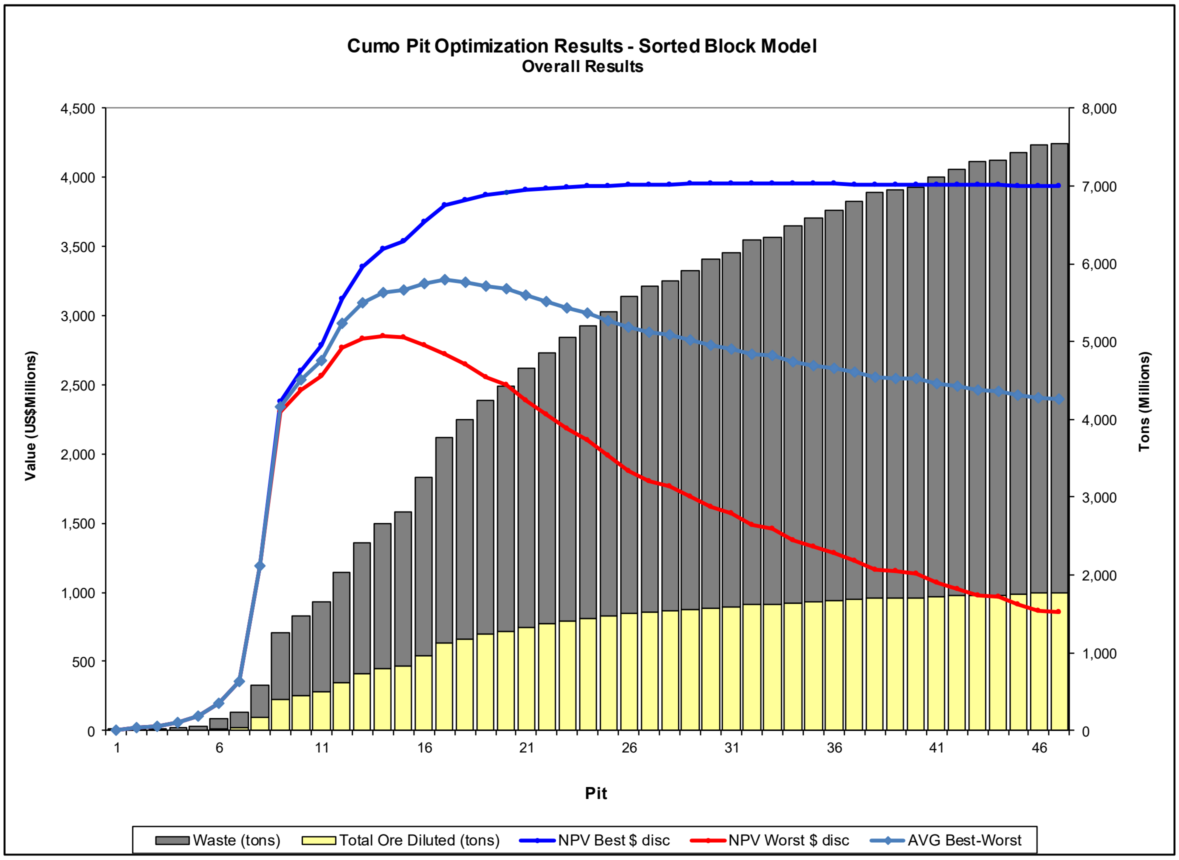
Whittle™ produces both "best case" (i.e., mine out shell 1, the smallest shell, and then mine out each subsequent shell from the top down, before starting the next shell) and "worst case" (mine each bench completely to final limits before starting next bench) scenarios. These two scenarios provide a bracket for the range of possible outcomes. The shells were produced based on varying revenue factors (0.3 through to 1.3 of base case) to produce the series of nested shells with the NPV results shown.

Note that in the pit optimization analysis undertaken, no value was assigned to the middlings from the third and final stage of bulk sorting. However, the decision was taken later in the project to feed the middlings to particle sorters. This has been reflected in the overall preliminary economic evaluation of the deposit, but not in the selection of pit shells for mineable resources.

Table 16-3: Pit optimization input parameters

Item	Unit	Value
Revenue		
Mo Price	$/lb	14.00
Cu Price	$/lb	3.00
Ag Price	$/oz	17.50
Metal Recoveries	%	Varies; see Table 13-5
Technical Constraints		
Pit slope angles	Overall degrees	37
Mining dilution	%	3%
Mining recovery	%	98%
Processing rate	tpd	150,000
Offsite Costs / Inputs		
Molybdenum		
Concentrate grade	%Mo in MoS_2 conc	52%
Concentrate moisture	%	0
Transport to roaster	$/t	5
Roasting Cost	$/lb concentrate	0.50
Roaster recovery	%	99%
Transport to market	$/t MoO_3	0
Copper		
Concentrate grade	% Cu	23
Concentrate moisture	%	10%
Payable Cu	%	96.5%
Transport to smelter	$/t concentrate	39.00
Smelter cost	$/t concentrate (dry)	75.00
Refining cost	$/lb	0.08
Silver		
Payable Ag	%	90%
Ag refining cost	$/oz	0.40
Other offsite costs[6]	%	1.0
Costs		
Mining cost	$/t mined	Modeled by bench
Processing cost	$/t milled	4.45
G&A Cost	$/t milled	0.50
Sustaining capital costs	$/t	$1.14

[6] loss, insurance, commission



Source: SRK, 2019

Figure 16-3: Pit optimization results

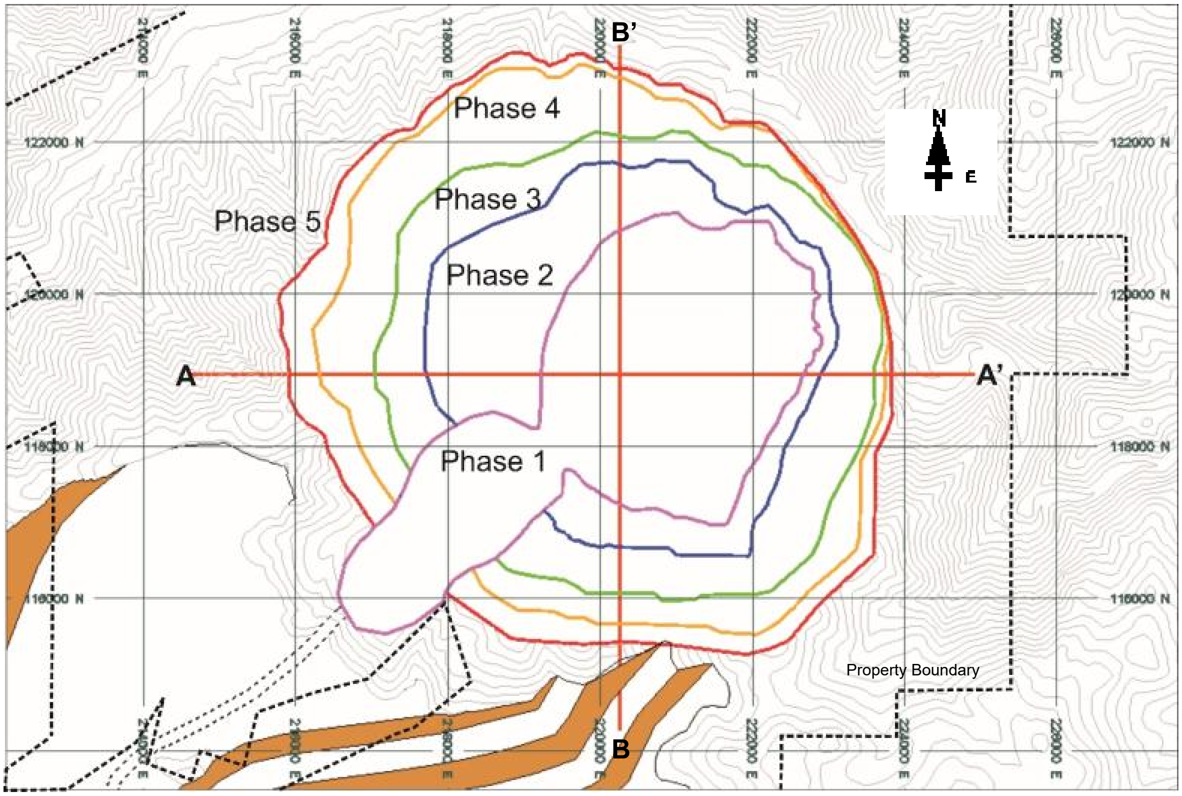
16.2.6 Pit Shell Selection

The author reviewed the pit optimization results and with consideration of the pit shell NPVs as well as their shapes and quantities, selected the appropriate pit shells for the development of conceptual production schedules. No specific mine designs were created, nor were fully detailed schedules developed. The author considers this appropriate for schedules in a PEA. The estimates of mined quantities for the phases representing the increment between pit shells are provided in Table 16-4. Mill feed after mineral sorting is also shown. This includes not only the mill feed product from bulk sorting, but also the same from particle sorting, using a mass pull on the bulk sort middlings of 40.8%.

Table 16-4: CuMo mined quantities

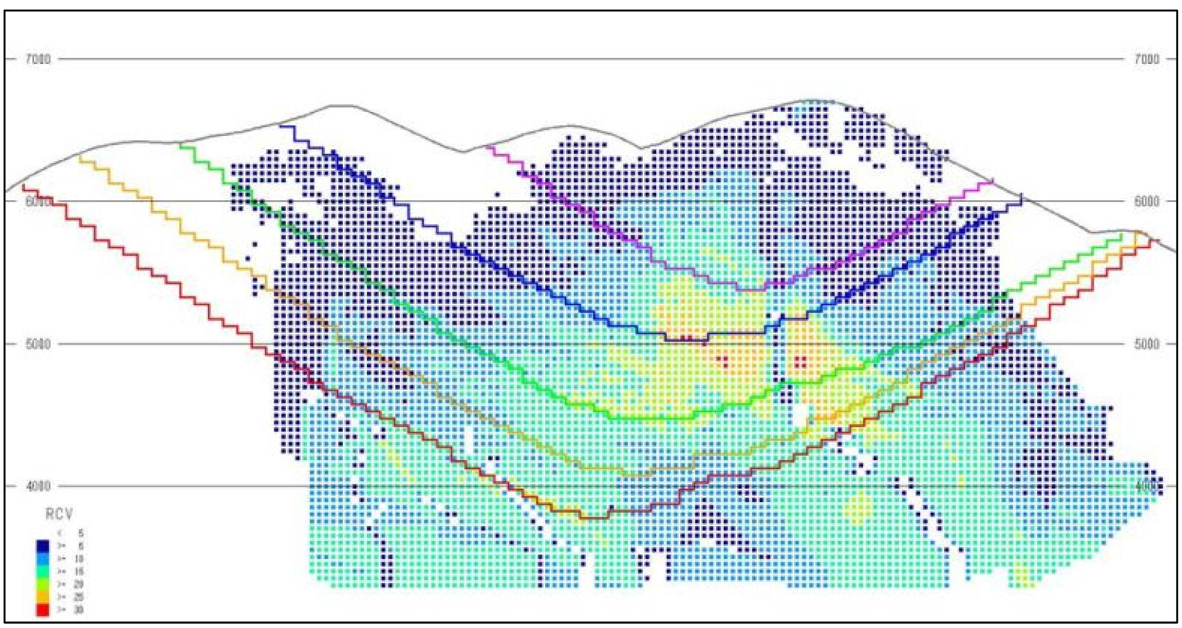
Phase	Shell	Total	Waste	Sort Feed	Strip Ratio	Mill Feed			
		Mt	Mt	Mt		Mt	%MoS_2	%Cu	ppm Ag
1	8	575	307	268	1.14	194	0.07	0.11	3.07
2	9	673	306	367	0.83	272	0.08	0.11	3.09
3	13	1,144	603	541	1.11	391	0.08	0.10	2.80
4	18	1,339	673	666	1.01	475	0.07	0.11	3.29
5 (final)	23	883	536	347	2.07	250	0.08	0.09	2.61
Total		**4,615**	**2,425**	**2,190**	**1.12**	**1,582**	**0.07**	**0.10**	**3.00**

The pit shells representing the five phases are illustrated in Figure 16-3 to Figure 16-6.



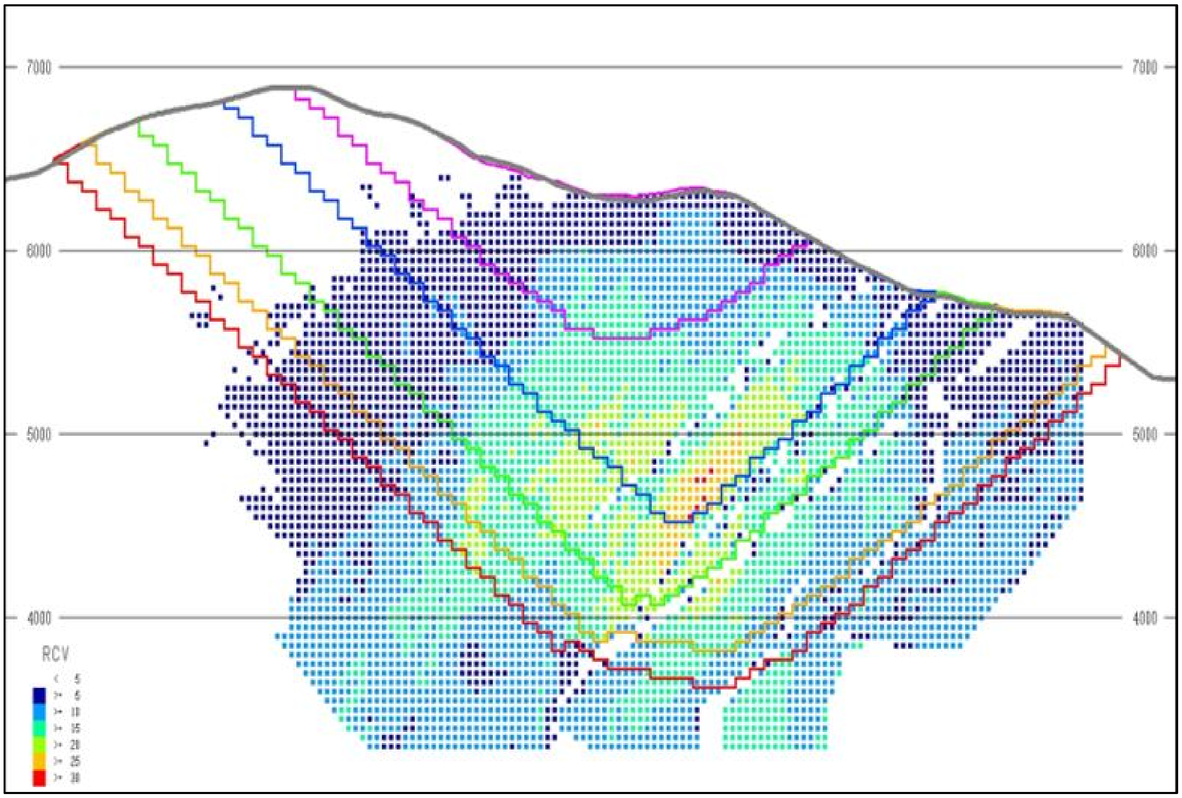
Source: SRK, 2019

Figure 16-4: CuMo pit phase shell outlines



Source: SRK, 2019

Figure 16-5: CuMo pit phase shell east-west cross-section A-A'



Source: SRK, 2019

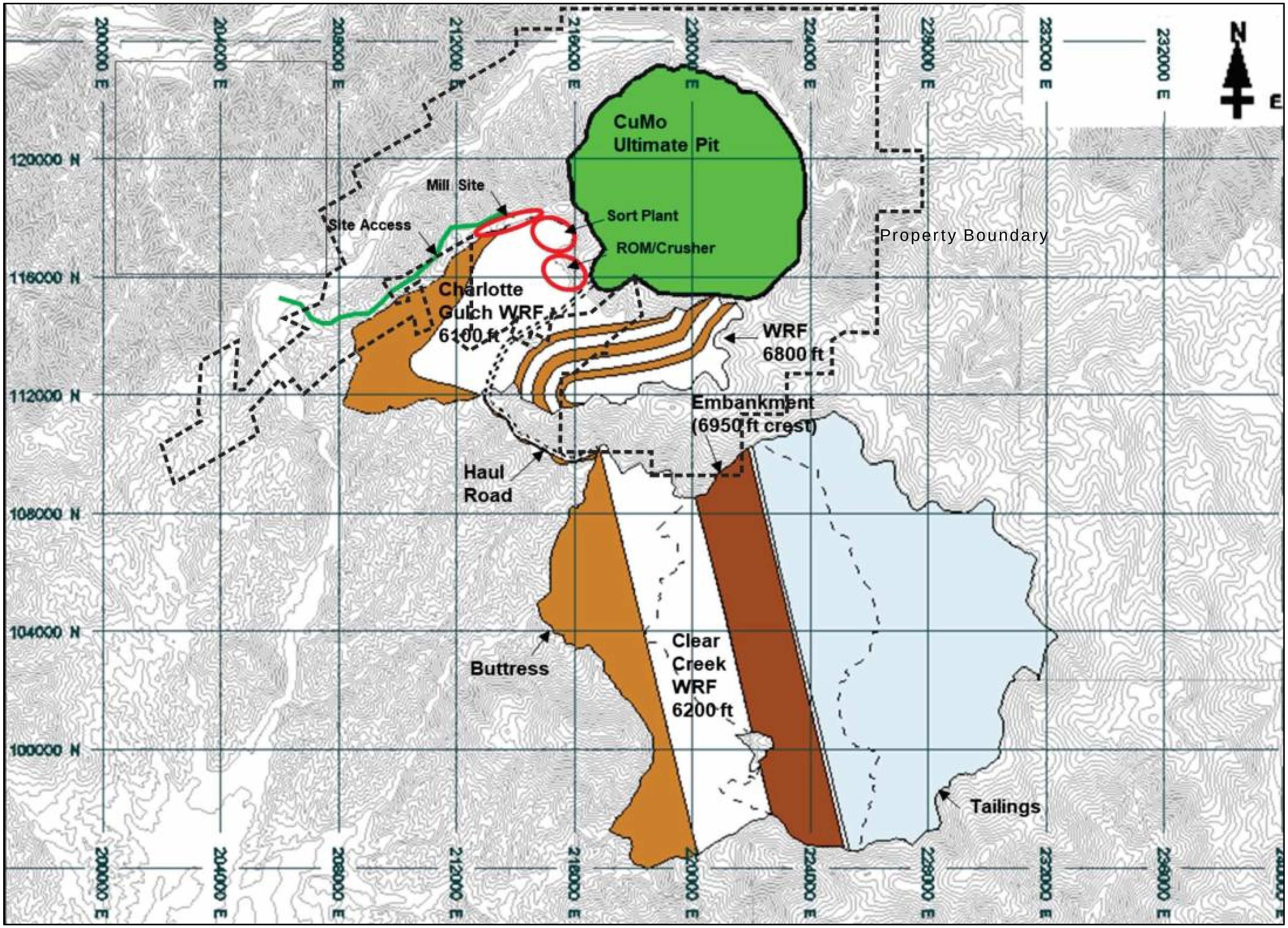
Figure 16-6: CuMo pit phase shell north-south cross-section B-B'

The pit shells are mathematical derivations. During the pit design at later stages in the project, it is envisioned that some pit shells will be combined in certain sectors of the pit to ensure adequate mining widths. For CuMo, the northeast sector, constrained by Grimes Creek, would be such an instance.

16.3 Waste Rock Facilities and Stockpile Design

Waste rock is produced from two sources, run of mine waste and sort waste. Sort waste is generated during the mineral sorting process, both bulk and particle sorting, and will be used in construction of the TSF embankment, discussed in Section 18.6. Run of mine waste is transported from the pit to WRF in Charlotte Gulch and Clear Creek and is also used as construction material in the TSF embankment (refer to Figure 16-7).

WRFs are designed to ensure physical stability throughout the mine life and into perpetuity. Benching, drainage, geotechnical stability, operational efficiency, and closure are all factors considered during design of waste rock facilities. At the time of the PEA, there was limited information available for geotechnical or geochemical assessments, but these are recommended for future study work.



Source: SRK, 2019

Figure 16-7: CuMo mine layout

The WRF in Charlotte Gulch, immediately to the south of the CuMo pit, is constructed by two methods. Upper bench waste from the initial phases of mining are placed in platforms following the east and south walls of Charlotte Gulch. The initial platform is at an elevation of 6,800 ft, followed by two wraparound platforms at elevations of 6,600 ft and 6,300 ft respectively.

The bulk of the waste rock in Charlotte Gulch is to be placed as a single platform at 6,100 ft elevation which is the approximate elevation of pit access. Toward the central and south portions of the WRF, the platform will increase to 6,200 ft in elevation to clear a height of land.

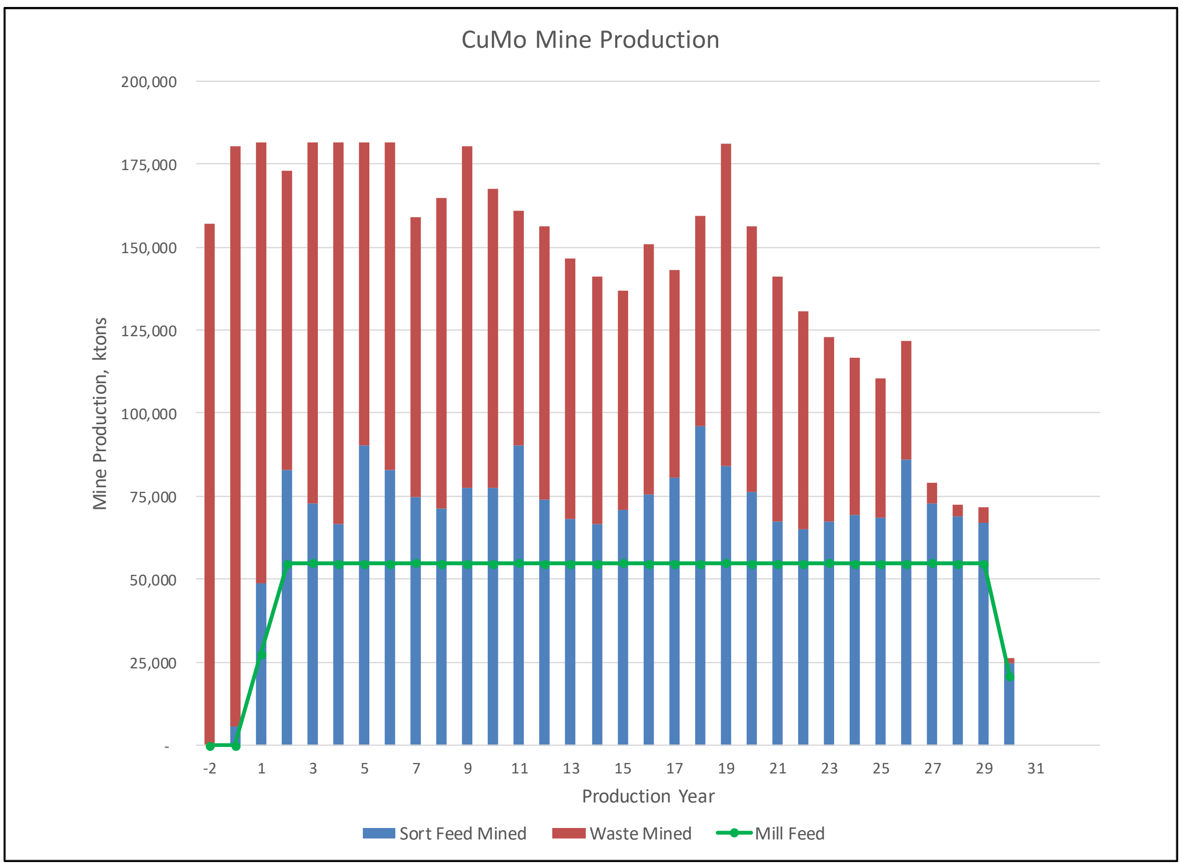
Clear Creek Waste Rock Facility

Run of mine waste rock will be hauled from the pit to the Clear Creek WRF which is buttressed against the tailings embankment also in Clear Creek. The buttress is designed to 6,200 ft elevation with a 3V:1H downstream slope. The WRF is intended to both provide additional waste storage capacity and to facilitate tailings embankment geotechnical stability and drainage.

16.4 Production Schedule

The author developed a life-of-mine (LOM) production schedule based on satisfying a mill feed, after sorting of 150,000 stpd. After an initial build-up of 2.7 million tons, the mine plan maintained a stockpile inventory ahead of the sort plant of 0.2-0.8 Mt. Note that wherever possible, mill feed inventory should be maximized at the face in the pit to ensure heterogeneity is maintained. Another scheduling criterion was balancing haul truck hours, ensuring no spikes in required trucks.

The resulting mine production schedule is provided in Figure 16-8.



Source: SRK, 2019

Figure 16-8: CuMo LOM production schedule

The production schedule shows two years of pre-stripping, followed by a ramp-up year in mill feed (27.4 Mt). Steady stated production of 54.7 Mt or 150,000 stpd is achieved in year 2. Full production lasts 28 years, with a tail-off in year 30 of production.

16.5 Equipment Selection and Fleet Requirements

Owing to the magnitude of mine production, ultra-class mine equipment is to be considered at CuMo. As part of this, and in keeping with current trends in mine haulage, the author has considered the deployment of an autonomous haulage fleet. While extra costs are incurred for hardware on the trucks, a central control system, and associated licensing and technical support, the benefits of labor savings, increased utilization, and improved tire life and maintenance costs were applied. Additionally, the author considered the use of semi-autonomous drills, wherein one operator can operate three drills drilling autonomously.

The envisioned fleet of primary mining equipment at steady state production is provided in Table 16-5.

Table 16-5: CuMo primary mine equipment fleet

Equipment Type	Size	Basis	Fleet Size
Rotary Blast Hole Drill	15 in	Hole Diameter	5
Electric Cable Shovel	100 t	Bucket Size	4
Autonomous Trucks	400 t	Payload	25-27
Track Dozer	21 ft	Blade Width	6
Rubber Tire Dozer	21 ft	Blade Width	3
Grader	24 ft	Blade Width	4
Water Truck	45,000 gal	Water Tank	3
Backhoe	5.0 yd^3	Bucket Size	2

In addition to this primary mine equipment, ancillary equipment consisting of utility (small) earthmoving equipment, mobile equipment maintenance vehicles, light vehicles, dewatering pumps, and portable lighting are to be included for the project. But at this level of study, their costs will be factored from the primary equipment.

17 Recovery Methods

Unless otherwise stated, the sub-sections in this section were previously provided in the report, "Summary Report on the CUMO Molybdenum Property, Boise County, Idaho" (Giroux, Dykes, Place, 2015). The authors have reviewed the underlying data, analytical work, and technical reports and have taken responsibility for this summary making edits as necessary. As well, John Starkey has re-interpreted comminution test results in terms of kW/t to simulate and confirm grinding mill sizes.

Mr. Starkey has accordingly confirmed or rewritten Section 17, with the exception of Section 17.2.

17.1 General

The CuMo processing facilities and associated service facilities will process ROM (run-of-mine) feed delivered to the primary crusher, to produce separate copper and molybdenum concentrates, waste rocks, and tailings. The proposed process encompasses crushing the ROM feed, bulk sorting, particle sorting, grinding, bulk rougher and cleaner flotation, regrinding, molybdenum separation and dewatering of copper and molybdenum concentrates. Molybdenum concentrates will be further processed downstream in a roaster to produce a saleable molybdenum trioxide product. The roaster would comprise of a standard multiple hearth gas fired roasting furnace heating the concentrate to approximately 600 degrees centigrade. In order to protect air quality, the flue gases and dust from the roasting are processed to produce sulfuric acid, and rhenium if it is economic to do so. In the case of sulfuric acid, it is recovered through water with the use of absorption towers. In the case of rhenium, it would be recovered through solvent extraction to produce ammonium perrhenate. The copper concentrate will be trucked from site for downstream processing at another facility outside the scope of this report. The flotation tailings will be thickened before placement in the TSF.

The present design incorporates three 50,000 stpd grinding lines with the ability to expand flotation and further downstream processes as needed. The process after mining comprises two stages: stage 1 includes a gyratory crusher, bulk sort conveyor diversion system, stockpile feed conveyor, and bulk sorted stockpiles, particle sort system, and another stockpile conveyor; stage 2 includes the sorted product stockpile, SAG and ball mill grinding circuit, bulk flotation circuit including regrind and cleaner flotation, copper and molybdenum separation flotation circuit, copper concentrate dewatering and load-out; molybdenum concentrate thickening, ferric chloride leach circuit, (molybdenum) filtration, drying, bagging and load-out, and tailings thickening and pumping facilities. Bagged molybdenum concentrate is roasted in a separate off-site roasting plant for conversion to molybdenum trioxide.

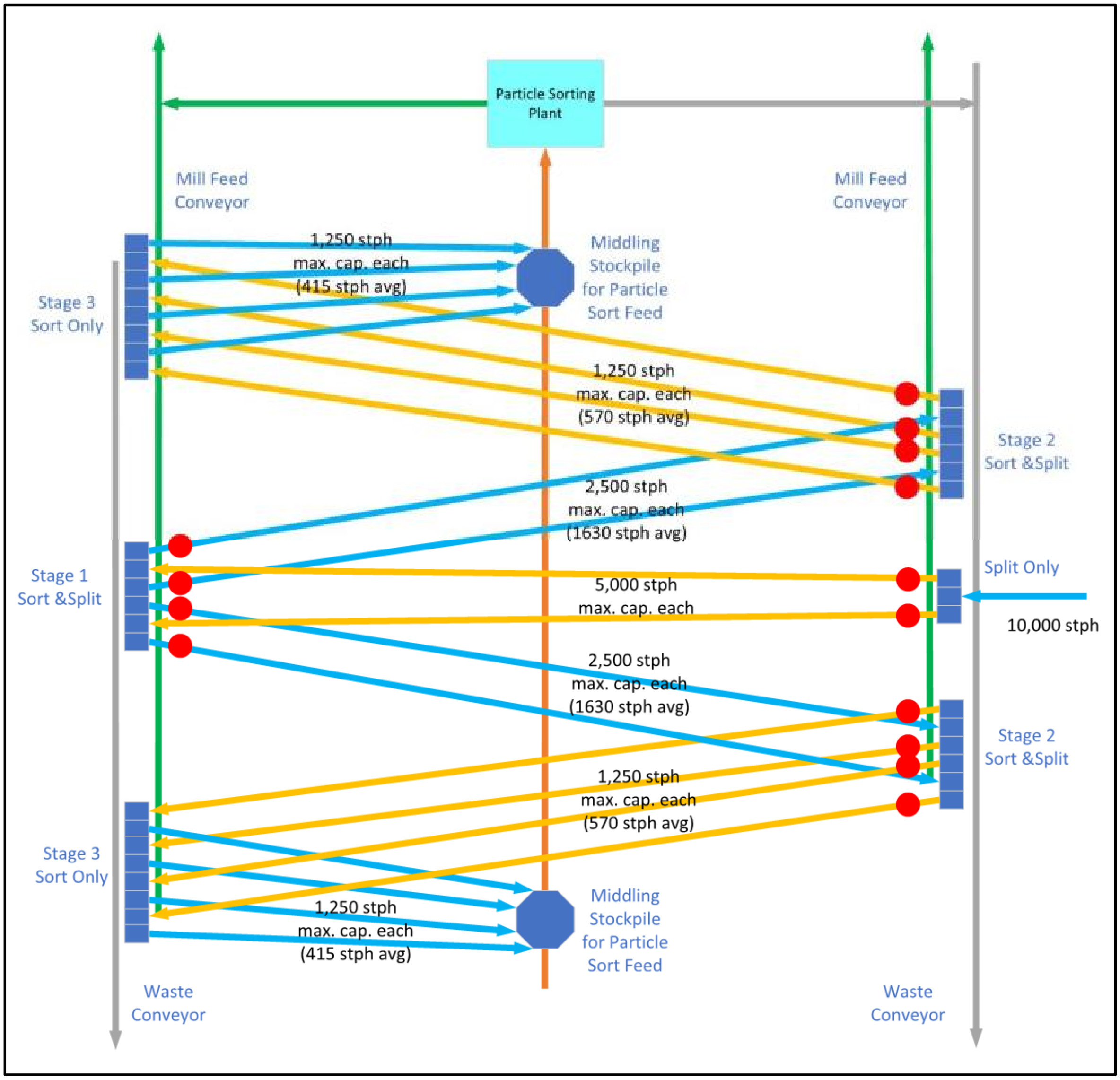
The concentrator will use a conventional grinding and flotation flow sheet and industry standard equipment. Plant operation will be monitored using a control system from a centrally located control room. Sampling and stream assay monitoring will be via an automated system linked to the control system.

17.2 Bulk Sorting

The following is original text for the current PEA.

The bulk sorting plant, located downstream of the primary crushers, would consist of a series of stages of splitting of streams, measuring their metal content, and then sorting. The schematic in Figure 17-1 shows the elements of a three-stage bulk sort plant.

However, prior to the sorting plant, there would be a diversion mechanism that would allow the crushed material to bypass the sorting plant. This would be for emergencies, to not disrupt the flow of material to the mill. A future improvement may be to place an analyzer on the conveyor belt after the primary crusher to determine whether crushed material needs to go to the sorting plant or not.

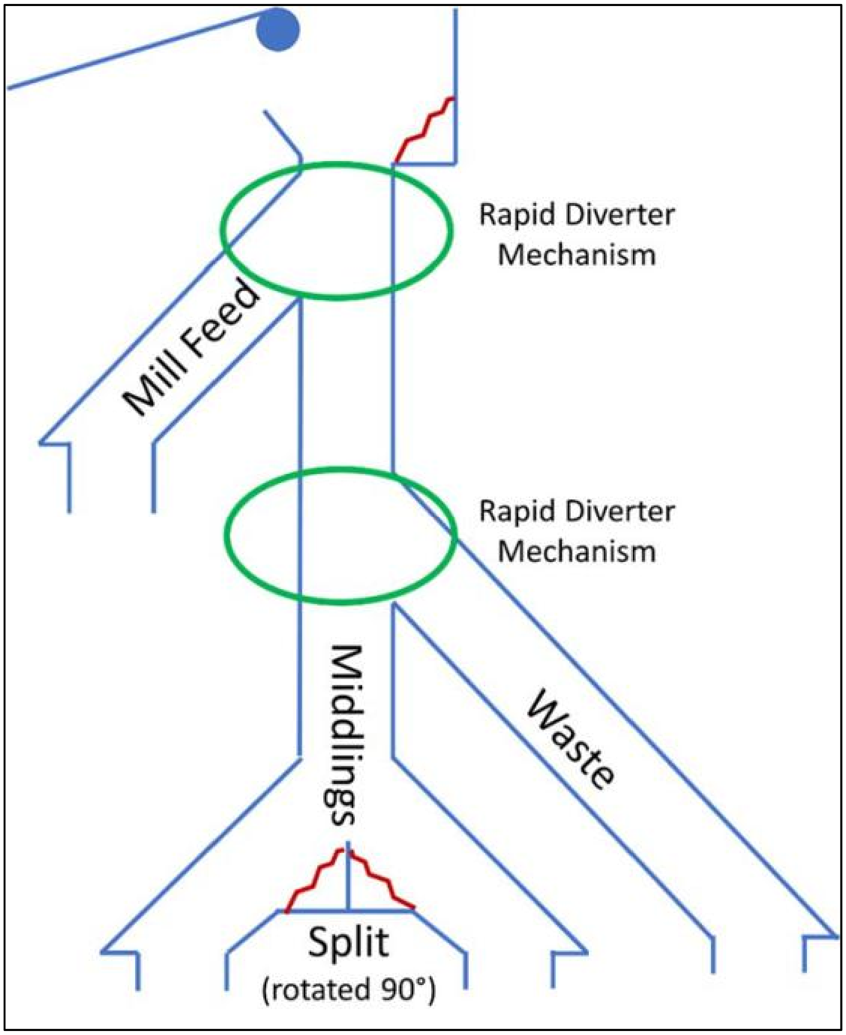


Source: Modified from CWA Engineers Inc., 2019

Figure 17-1: Schematic of three-stage bulk sorting plant with particle sorting

Material feeding the plant (nominally 10,000 tph) is immediately split in two, and two penetrative elemental analyzers, such as a prompt gamma neutron activation analysis analyzer, measure the stream. To make a measurement, such analyzers require a batch of material on the belt to average readings over. For instance, in the case of Scantech's Geoscan analyzers, this is 30 seconds of belt travel time. The length of conveyor from this first analyzer position to the sorting point is dictated by this 30 second interval and the travel speed of the belt. So, a belt traveling at 12 ft per second would require a conveyor length of at least 360 ft between analyzer and sort point. Alternate technologies are being developed to shorten the required measurement interval.

At the sorting point, a signal is received from the analyzer to indicate what the approaching material consists of (mill feed, waste, or middlings). The rapid diversion mechanism then diverts the stream to receiving chutes and conveyors accordingly. Figure 17-2 illustrates a viable diversion system to facilitate the re-direction of a stream. As the intellectual property is not presently protected, details of the rapid diversion mechanism are omitted.



Source: Modified from CWA Engineers Inc., 2019

Figure 17-2: Schematic of bulk sorting diversion system

The CuMo sorting plant would consist of three stages of sorting. Each stage will produce mill feed, waste, and middlings products. The mill feed from each stage will be sent directly to the coarse mill feed stockpile in front of the mill, while the waste will be conveyed to a truck load out bin for delivery by haul truck to the TSF or WRF in Clear Creek.

The middlings portions become feed for subsequent sorting. To take advantage of the increased heterogeneity that comes with smaller scale (Section 13.2.2), the middling streams of the first and second sort are split in two to reduce the 30 second batch size (Figure 17-2). The third stage however will not have the middlings stream as this will next become feed for particle sorting.

17.3 Particle Sorting

The following is original text for the current PEA.

In order to ensure maximum mill feed recovery, particle sorting using XRF based sorting machines would be done taking feed from stockpiles or bins containing the middlings from the third stage of the bulk sort (Figure 17-1). Values so recovered would be added to the mill feed conveyors shown.

Up to eight lines would feed 350 to 400 short tons per hour into particle sorting modules. Based on current XRF particle sorting technology, each module would consist of multiple sorters to handle different size fractions. These sorters are available from a number of vendors, and capacity per unit, per size fraction, ranges from about 50 to 200 short tons per hour depending on the particle size fed.

This study assumes four 100 short ton per hour units are required per line based on the current technological limitations on throughput. The sizing and selection of these units need to be confirmed later as part of a more advanced level of study.

17.4 Mill Design Criteria Summary

The remaining sub-sections were modified from the report, “Summary Report on the CUMO Molybdenum Property, Boise County, Idaho” (Giroux, Dykes, Place, 2015).

The overall approach was to design a robust process plant that could be scaled up and deliver good value for capital. The key project and specific criteria for the plant design and operating costs are provided in Table 17-1.

Table 17-1: Summary of the process plant design criteria (150 ktpd).

Criteria		Units	Value
Sort Feed Capacity		ktpd (short tons)	250
		Mt/y (short tons)	90
Primary Crusher Availability		%	65%
Primary Crusher Throughput/ Feed		t/h (short tons)	16,000
Primary Crusher Selection	Size		60 x 110
	No.		3
Mill Throughput/feed		Mt/y (short tons)	54.75
Mill/Grinding and Flotation Availability		%	92%
Mill Throughput/feed		t/h (short tons)	6,793
Total Power requirement		MW	186
Physical Characteristics	BWI	kWh/t (tonne)	15.8
	SPI®	Mins	84.5
Grind Size	P80	microns	63
Head Grade (Design)		%Cu	0.1
		%MoS_2	0.11
		ppm Ag	2.87
Flotation Recovery (Cu-Ag Zone)	Copper	%	68%
	Silver	%	75%
	Molybdenum	%	86%
Flotation Recovery (Cu-Mo Zone)	Copper	%	85%
	Silver	%	78%
	Molybdenum	%	92%
Flotation Recovery (Mo Zone)	Copper	%	72%
	Silver	%	55%
	Molybdenum	%	95%
Cu Circuit Residence Time	Roughers	Mins	27.5
	Cleaner 1	Mins	10
	Cleaner Scav.	Mins	2.5
	Cleaner 2	Mins	10
	Cleaner 3	Mins	5
Mo Circuit Residence Time	Roughers	Mins	35
	Cleaner 1	Mins	25
	Cleaner Scav.	Mins	25
	Cleaner 2	Mins	25
	Cleaner 3	Mins	25
Cu Concentrate Filtration Rate		kg/m2/h	262
Concentrates Thickening Flux		t/m2/h	0.1
Mo Concentrate Filtration Rate		kg/m2/h	356
Tailings Thickening Flux		kg/m2/h	800
Tailings Thickener Underflow Density		% w/w	65
Collector Consumption (SIBX)		g/t (short ton)	66
Collector Consumption (Aero 3302)		g/t (short ton)	59
Activator Consumption (Moly Oil)		g/t (short ton)	51
Frother Consumption (X-133)		g/t (short ton)	67
Lime Consumption		kg/t (short ton)	0.18
Flocculant Consumption		g/t (short ton)	15
SAG Mill Media Consumption		kg/t (short ton)	0.25
Ball Mill Media Consumption		kg/t (short ton)	0.55
Regrind Mill Media Consumption		kg/t (short ton)	0.04

Detailed process design criteria incorporating the process mass balance, engineering design criteria and key sizing criteria, derived from the results of the metallurgical test-work program were determined and are summarized below.

17.5 Plant Design Basis

The key criteria selected for the plant design are:

- Treatment of 150,000 stpd or 136,000 metric tonnes/day
- Design availability of 92% (at full capacity), being 8,059 operating hours per year, with standby equipment in critical areas, such as cyclone feed pumps and tailing pumps.
- Sufficient plant design flexibility for treatment of all mineralized zones at design tonnage
- The selection of these parameters is discussed in detail below

17.6 Throughput/Mill Feed and Availability

One main throughput/mill feed scenario was nominated by CuMoCo to evaluate different corporate investment hurdles. The authors have nominated overall plant availability at 92% or 8,059 hours per year. This is an industry standard for a large, multi-train, SAG mill grinding and flotation plant with moderately abrasive mineralized material operating in a well serviced geographic region. Benchmarking indicates that similar plants have consistently achieved this level.

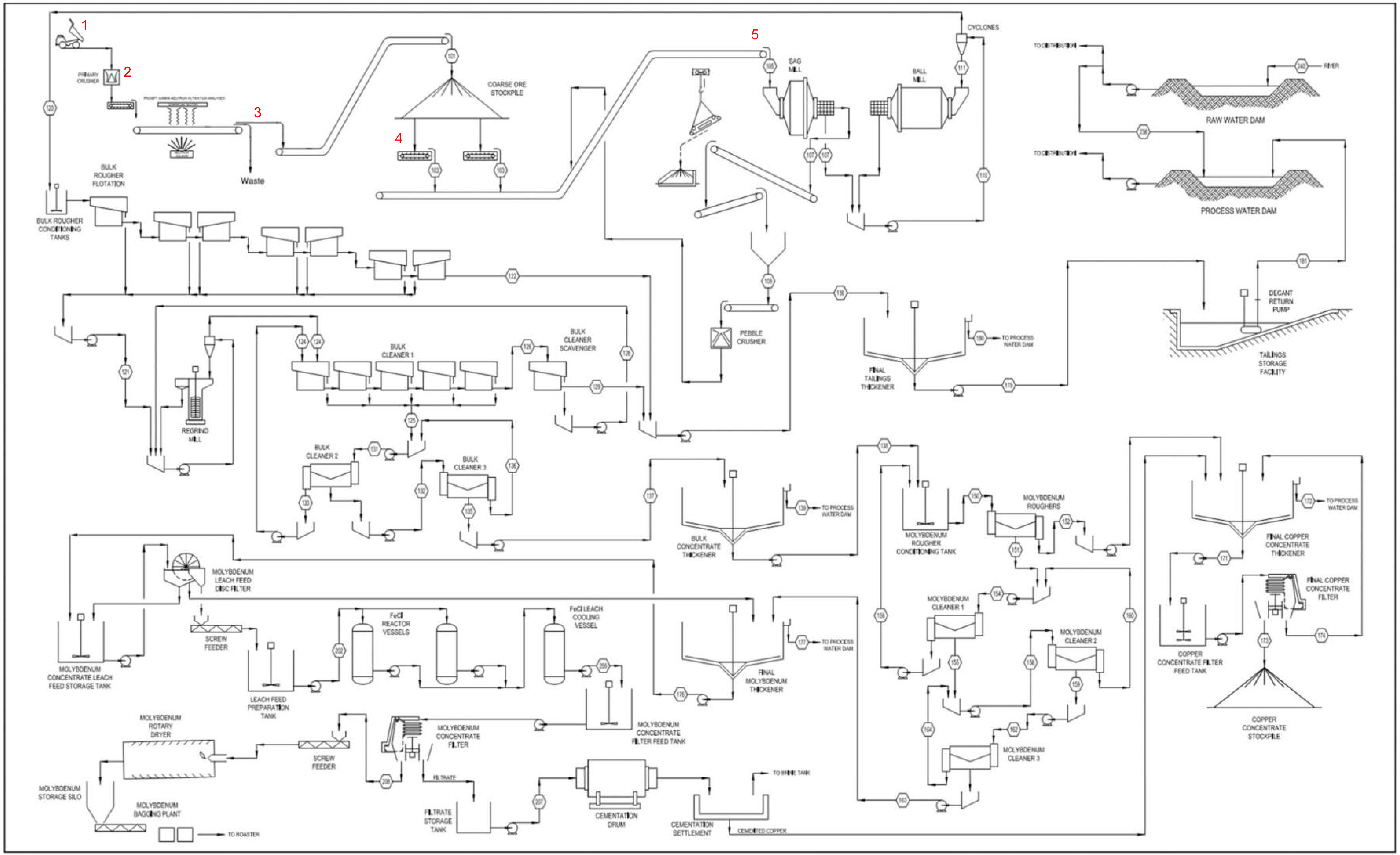
17.7 Processing Strategy

The overall processing strategy is to mine at high tonnage and send all mined material through a multi-stage sensor-based mineral sorting plant (including crushing, screening equipment, bulk and particle sorting). The sorting plant recovers the high-grade high profit rock and rejects marginal and waste rock. This allows the mill and tailings facilities to be significantly smaller while still producing high quantities of concentrate or similarly, can allow a mill of equal size to produce more total concentrate.

The mill process design is based on treating the different mineral types tested individually at the nominated design throughput/mill feed rates. Typically, the range in variability of material parameters such as SAG mill grinding hardness and head grade during process design are considered. However, due to the preliminary nature of the mining schedule, and the grinding and metallurgical test-work, the most competent and hardest of the three mineral types, identified by CuMoCo to date, have been used as the process design criteria.

17.8 Flow Sheet Development and Equipment Sizing

The process plant flow sheet design for the CuMo circuit was conceptually based on those of comparable large flotation plants. Figure 17-3 shows a process schematic for the CuMo plant. Details of the flow sheet design and selection of major equipment for the process are discussed in the sections below.



Source: Ausenco 2009 and modified by Sacré Davey 2018

Figure 17-3: CuMo process schematic

The flowsheet shown in Figure 17-3 was originally authored by Ausenco 2009 and modified by Sacré Davey to add a sorting system (upper left). The section to the left of the coarse material stockpile (4) is compressed, showing a single stage sorting plant instead of two stages, with storage between the bulk and particle sorting stages.

The schematic shows the process starting at the upper left corner with mining trucks (1) delivering the 250,000 stpd (thousand tons per day) sorting feed to a primary gyratory crusher (2) at the edge of pit, the output is then delivered to the bulk sorting plant which generates both a mill feed product that is conveyed to a coarse material stockpile and a middling product for particle sorting. The product from particle sorting combines with the bulk sort mill feed product in the coarse material stockpile. . The SAG mill feed from the coarse material stockpile (4) is conveyed into the SAG mills (5).

17.9 Unit Process Selection

The process plant design is based on a flow sheet with unit operations that are well proven in the sulfide flotation industry, incorporating the following unit process operations. Where considered practical, unit operations are sized to maximize the economies of scale possible with large equipment. The sorting system consists of the following unit processes. Mine sort feed (250,000 stpd) from the open pit is crushed using a primary gyratory crusher to a crushed product size of nominally 80% passing (P80) 120 mm and fed onto the bulk sort feed conveyor. A bulk sorting plant with PGNAA sensors divides the material into mill feed, middlings and waste piles. The middlings are further particle sorted, producing a waste stream and a mill feed stream which combines with the bulk sort mill feed in a course material stockpile. Waste from both sorting processes is loaded into trucks for disposal in the mine. The mill feed (150,000 stpd) is conveyed from the coarse material stockpile to the mill.

The general mill design consists of three 50,000 stpd modules. Each module typically consists of the following unit processes:

- Conical stockpile of crushed mill feed with a live capacity of 18 hours, with two apron feeders per grinding train, each capable of feeding 120% of the full mill throughput/mill feed
- A 22 MW SAG mill, 11.58 m (38 ft.) diameter with 7.60 m (25 ft.) EGL, in closed circuit with pebble crushing
- Pebble crushing will be comprised of two MP800s per grinding train, crushing to a product size of nominally 80% passing (P80) 12 mm
- Three 13 MW ball mills per grinding train, 7.32 m (24 ft.) diameter with 12.19 m (40 ft.) EGL, in closed circuit with hydrocyclones, grinding to a product size of about 80% passing (P80) 63 µm
- Bulk rougher flotation consisting of 200 m^3 forced air tank flotation cells to provide a total of 28 minutes of retention time
- Rougher concentrate regrinding in three 1.0 MW vertical stirred mills per grinding train to a P80 of 10 µm
- Bulk cleaner 1 and cleaner scavenger flotation consisting of 20 m^3 forced air tank flotation cells to provide a total of 13 minutes of retention time

- Bulk cleaner 2 flotation cells consisting of 8 m^3 trough shaped flotation cells to provide a total of 10 minutes of retention time
- Bulk cleaner 3 flotation cells consisting of 8 m^3 trough shaped flotation cells to provide a total of five minutes retention time
- Bulk concentrate thickening in 11 m diameter high rate thickeners
- Molybdenum rougher flotation consisting of 8 m^3 trough shaped flotation cells to provide a total of 35 minutes of retention time
- Molybdenum cleaner 1 consisting of 1.5 m^3 trough shaped flotation cells to provide a total of 25 minutes of retention time
- Molybdenum cleaner 2 flotation cells consisting of 1.5 m^3 trough shaped flotation cells to provide a total of 25 minutes of retention time
- Molybdenum cleaner 3 flotation cells consisting of 1.5 m^3 trough shaped flotation cells to provide a total of 25 minutes retention time
- Copper concentrate thickening in a high rate thickener and filtration in a horizontal plate and frame pressure filter
- Molybdenum concentrate thickening in a high rate thickener
- Molybdenum ferric chloride leach in 4,000 U.S. gallon, glass lined steel leach reactors followed by drying and storage in bulk one-ton bags
- Tailings thickening in a high rate thickener to an underflow density of 65% solids
- TSF for process tailings in a conventional dam
- Raw process plant water supply from site water storage facility reticulated throughout the plant as required. (Harvesting and storage of raw water sufficient to allow continued water supply throughout the year is excluded from the study scope)
- Total water requirement estimated at an initial 190 acre-feet then 10% replacement rate per year due to losses in evaporation and concentrate etc.
- Process water dam and distribution system for reticulation of process water throughout the plant as required. Process water is supplied from water reclaimed from the TSF, from process operations and site run-off with raw water used as make-up water as required
- Potable water is generated by treatment of raw water in a reverse osmosis (RO) unit at the process plant. Potable water is distributed to the plant, and for miscellaneous purposes around the site
- Plant, instrument and flotation air services and associated infrastructure.

18 Project Infrastructure

18.1 General Layout

Refer to the conceptual mine layout presented in Figure 16-8.

18.2 Road Access

Two options for road access routing exist.

- Option 1 is to upgrade the existing gravel roads that come from Highway 55 near Horseshoe Bend via Placerville and Centreville. These roads are currently serviceable paved and gravel roads suitable for light-duty travel. An upgrade is required if concentrate haulage is to be undertaken. An extension to this route would have to be constructed to access the proposed plant site (Figure 16-8). The extension to the road would have to rise in elevation from RL 4,900 ft to RL 6,100 ft. At a maximum grade of 10%, this road is estimated to be approximately four miles long. No specific route has been identified, although the terrain through which the road would travel is rugged, and switchbacks are likely to be required in areas.
  - A subset of this option is to use the existing road from Idaho city to Centreville, but this does not appear to offer any significant benefits over the base assumption.
- Option 2 is to travel via Highway 55 and the Bank-Lowman road that is to the north of the project site. An existing bridge approximately 1.4 miles southeast of Garden Valley would be used to cross the Payette River and gain access to the South Fork Road. This road then follows the river to the east for approximately 6 miles. It is relatively level, generally sloping up to the east at 2% to 3%, following the river valley. Upgrades to this road are likely to be straightforward and relatively low cost. From there, the existing Grimes Pass Road leads south from the South Fork road (~ four miles) to a location near the plant site. This road has consistent, but reasonably steep gradients of approximately 10%. Whilst not ideal, the gradients are potentially manageable for mine traffic including concentrate trucking with upgrades such as safety berms and run-away ramps. A similar extension of new road of four miles would have to be constructed as per Option 1. This route has the advantage over Option 1 of requiring much shorter haulage on non-sealed roads. A significant disadvantage of this route is that the haulage on sealed roads would both be visible to, and potentially affect recreational traffic on these roads.

Regarding Option 2, socio-political opposition to industrial use of these roads is likely. Until this can be further studied, this option is not preferred, leaving Option 1 as the access for the purpose of this PEA.

18.3 Rail Access

A rail line connecting to ports in Oregon runs north-south in the valley along-side Highway 55. Sidings are available at various locations.

The most suitable location for a concentrate loading facility for Option 1 is likely to be in or around the town of Horseshoe Bend. However, Horseshoe Bend is a residential town and community opposition may limit options with respect to the existing small rail yard in the town center, necessitating construction of a new siding and facility.

In the case of road access Option 2, building a loading concentrate loading facility near the junction of Highway 55 and Bank-Lowman road may be possible. An existing siding may be available for use. It is understood from discussions with CuMoCo that this area is heavily used for tourist activities including rafting.

For a single project using a concentrate loading facility, bulk concentrate handling may not be optimal. The use of “Rotainers” (sealed containers specifically designed for transport of concentrate) for truck, storage and rail transport may be an effective solution, particularly in terms of managing environmental effects. The author recommends that this option be included as an option in a PFS-level logistics study.

18.4 Electrical Power

The overall availability of sufficient generating capacity is unlikely to be an issue as the project is proximate to significant power reticulation capacity. The project area is serviced by Idaho Power. No suitable power lines currently run near to the project, but Idaho Power have indicated an intention to install transmission lines to the vicinity of Placerville to the Southwest (ten miles), and to the vicinity of Garden Valley to the Northeast (nine miles). Consideration should be given to the provision of back-up power for critical systems. For example, back-up generation to allow the clearing of pipelines, flotation cells, thickeners and tailings management systems to prevent costly blockages and delays is generally able to be justified.

18.5 Water Supply

Water is likely to be available (subject to licenses) from the Payette River two miles north of the project. The intervening terrain is rugged, and the pipeline route is likely to be significantly longer than the direct distance. An assumption of five miles of supply pipeline was made for the purposes of costing. The river can potentially supply water year-round, and accordingly a surge tank, rather than extensive water storage has been assumed at the project site. A water supply trade-off study is assumed to be undertaken as part of the PFS.

18.6 Tailings Storage Facility

The tailings storage facility will be located at the headwaters of the Clear Creek watershed, in a natural basin formed by the surrounding ridgeline. The TSF will have capacity to store the 1,582 Mt (~900M m^3) of tailings produced, over the 30-year mine life, with an ultimate crest height of 6,950 ft. A starter dam will be constructed to elevation 6,300 ft to facilitate early mine production, followed by an additional five raises spread out over the life of the mine.

Tailings will be piped to the TSF and deposited as conventional slurry from the dam crest. The settled tailings density is assumed to be 1.6 tonnes/m3 and beach slope angles are assumed to be 1-2% for sub-aerially deposited tailings. The water reclaim pond will form against the natural terrain upstream of the dam. The tailings have not tested positive for potential acid generation; however, there is potential for metal leaching.

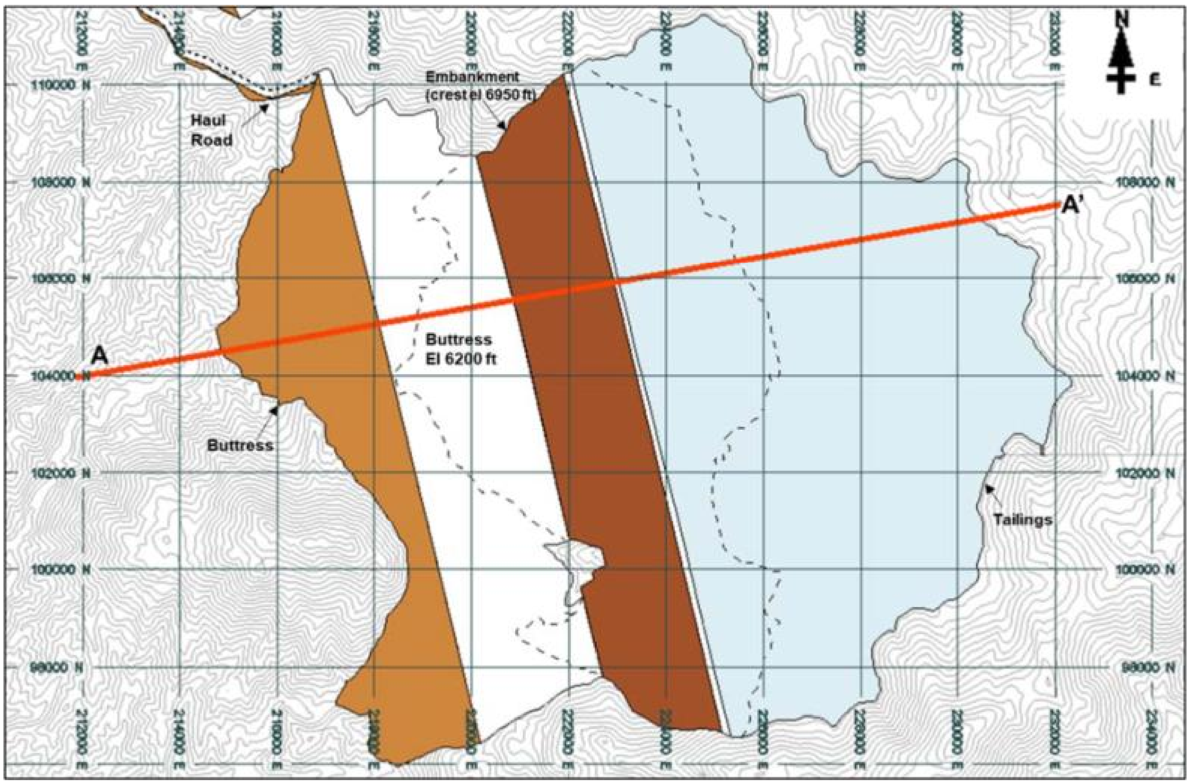
18.6.1 Tailings Embankment

The embankment is designed as downstream construction for geotechnical stability, with the starter dam placed on bedrock. Slopes will be 2.5H:1V on the upstream and 3H:1V on the downstream. The crest of the embankment will be 170 ft wide to accommodate vehicles and equipment. Sort waste and run of mine waste constitute the construction material, transported by haul truck, and then compacted in three-foot lifts

A starter dam is designed to a crest height of 6,300 ft to facilitate the first two years of tailings. The foundation for the started dam will be cleared and overburden stripped to bedrock. The overburden will be stockpiled for use in future reclamation of the waste facilities. Five additional lifts will be constructed to an ultimate crest height of 6,950 ft. A freeboard of 25 ft will be maintained throughout the mine life.

A waste storage facility will buttress the downstream of the embankment up to 6,200 ft elevation providing additional geotechnical stability.

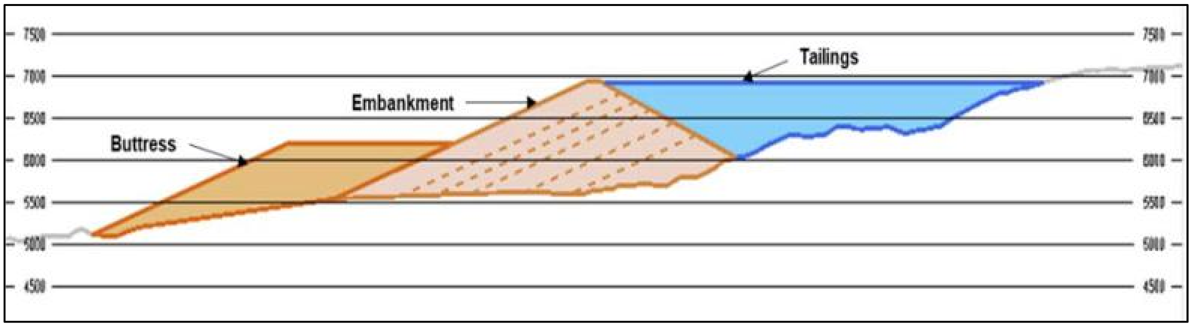
Figure 18-1 and Figure 18-2 show the TSF and WRF concepts.



Source: SRK, 2019

Note: Tailings site, which is located on federal land, is shown in Figure 16-7 relative to property boundary

Figure 18-1: CuMo Clear Creek TSF and WRF buttress



Source: SRK, 2019

Figure 18-2: Cross-section A-A' through Clear Creek TSF and WRF buttress

18.6.2 Tailings Impoundment

The tailings impoundment will facilitate an ultimate capacity of 950M m^3 (900M m^3 required for LoM) of tailings assuming a density of 1.6 tonnes/m^3. Construction of the impoundment area will include the removing of topsoil and vegetation and compacting the exposed fine-grained soils. Tailings will be discharged from the crest of the dam, limiting seepage through the dam. Due to the tailings deposition plan and overall configuration of the TSF it is not expected that a synthetic geomembrane will be required for containment of tailings.

19 Market Studies and Contracts

19.1 Market Analysis

For the purpose of this study, it has been assumed that two concentrates (copper and molybdenum) will be produced with the copper concentrate grading >23% copper sold and shipped to a smelter within the Pacific region, Japan, China, Korea or India for example. The molybdenum concentrate (grading >50% Mo) will be shipped to a roaster controlled by the project where additional credits may be achieved through the production of rhenium and sulfuric acid. Readers should note that no penalty elements have been identified to date. At the current time, no contracts exist for delivery of final product so the report assumes that products will be sold on the open market.

19.1.1 Treatment and refining costs

Treatment and refining charges, metal payability and settlement terms are assumed based on recent published values from current contracts with Asian smelters for the copper concentrate (Freeport-McMoran, First Quantum), while the costs associated with molybdenum are based on published toll milling charges which are higher than for the project's own roaster and therefore considered conservative. Details of these charges were previously reported in Table 16-3.

19.1.2 Metal Prices

Prices used are based on historical averages and reasonable future price projections published. Copper and silver are openly traded on a daily basis on terminal markets. Molybdenum pricing requires additional research and analysis, as the often-quoted London Metal exchange pricing does not reflect current pricing accurately. Roskill's and Platts show the trading price of molybdenum. The authors have identified that London Metals Exchange pricing can be many months out of date as metal buyers and sellers of molybdenum tend to avoid this relatively new market and associated fees.

CPM Group Molybdenum Market Outlook 2017 and 2018 shows the price of molybdenum is controlled by the largest producer which is China and their average cost to produce is between $12 and $13 per pound molybdenum. Their professional estimate of the price of molybdenum moving forward in the next five years is in the range of $12 to $20 per pound. The authors, for the purposes of this updated PEA, have assumed pricing of $15 per pound of molybdenum metal for project economics.

A significant proportion of world-wide molybdenum is produced as a byproduct of base-metals production. This can lead to a "disconnect" of supply and demand in the market, thus causing significant price volatility.

20 Environmental Studies, Permitting, and Social or Community Impact

20.1 Environmental and Permitting

20.1.1 Past and Present Permitting for Exploration Project

ICMC's predecessors submitted an exploration plan of operations in 2007 to the USFS for exploration activities resulting in about 20 miles of drill road of which 4.7 miles were existing unauthorized drill roads from previous operators and 13.3 miles of new temporary roads. An environmental assessment was prepared by the USFS. ICMC was initially issued the Decision Notice /Finding of No Significant Impact (2011 DN/FONSI) by the USFS in February 2011. A lawsuit was filed against the project by the Idaho Conservation League in July 2011. On August 29, 2012, the United States District Court of the District of Idaho (Court) ordered, "that the Defendant Forest Service's decisions regarding groundwater made in the 2011 Environmental Assessment [be] vacated and the matter …remanded to the Forest Service for further proceedings consistent with this opinion…" (USFS 2018). The USFS moved forward with the preparation of a Supplemental Environmental Assessment to undertake further analysis of groundwater and, as needed, address new information/changed circumstances since the 2011 DN/FONSI was issued (e.g., change in status of the wolverine from a regional sensitive species to an Endangered Species Act proposed listed species) (USFS 2018).

The supplemental DN/FONSI (SDN/FONSI) addressing the 2011 Court order and other changes summarized above was signed on September 30, 2015. Plaintiffs from the 2011 lawsuit again filed a lawsuit challenging the 2015 supplemental decision in January 2016. The lawsuit challenged the analysis of potential effects of exploration activities to groundwater and Sacajawea's bitterroot, a sensitive plant species. The Court issued the memorandum decision and order in this lawsuit on July 11, 2016. The Court upheld the SDN/FONSI as to the NEPA challenges related to groundwater, so no further analysis was required. The Court found that the Forest Service's analysis and conclusions concerning Sacajawea's bitterroot to be arbitrary and capricious because it failed to re-examine the baseline Sacajawea's bitterroot population[7] in the project area following the 2014 Grimes Fire and subsequent 2016 Pioneer Fire (USFS 2018).

As occurred in response to the 2014 Grimes Fire, each resource area addressed in the 2015 *CuMo Exploratory Project Supplemental Environmental Assessment* were affected differently. Similar to the updates made in response to the 2014 Grimes Fire, updates were made in the 2018 *Supplemental Redline Environmental Assessment CuMo Exploration Project* (Supplemental Redline Environmental Assessment) to address the change in baseline conditions caused by the 2016 Pioneer Fire. The Supplemental Redline Environmental Assessment focused on the re-evaluation of the Sacajawea's bitterroot baseline, as well as other resources addressed in the 2015 Supplemental Environmental Assessment that were affected by the 2016 Pioneer Fire, to determine whether effects conclusions reached in the 2015 SEA that supported the 2015 SDN/FONSI were different or changed. The Supplemental Redline Environmental Assessment focused only on these topics because the Court determined that other concerns raised during the 2012 and 2015 lawsuits were properly addressed and the evidence and analysis in the 2015 Supplemental Environmental Assessment and supporting

[7] The 2017 Sacajawea's bitterroot survey was occurred within a ten-mile radius and located previously unknown populations totaling about 17,000 plants in six locations (USFS 2018).

project record supported the determination that no significant impacts would occur to other resources from proposed management activities (USFS 2018).

A number of environmental studies were undertaken for the 2011 *Environmental Assessment CuMo Exploration Project* and subsequently revised in 2015 and 2018. The Supplemental Redline Environmental Assessment incorporated habitat changes and resulting impacts related to the 2014 Grimes Fire and the 2016 Pioneer Fire. The following reports supported the preparation of the 2018 EA. Some of these updated reports and some earlier reports can be accessed on https://www.fs.usda.gov/project/?project=52875:

Stantec. Pollinator Habitat Assessment Report CuMo Exploration Project. February 2018

______, Pollinator Habitat Assessment Report CuMo Exploration Project. October 2018

______. Tetra Tech. *Sacajawea's Bitterroot and Other Sensitive Plant Survey Report*. Prepared for Forsgren Associates Inc on Behalf of American CuMo Mining Corporation in support of the CuMo Exploration Project. July 2015

______. *Sacajawea's Bitterroot Baseline Survey Report*. Prepared for Idaho CuMo Mining Corporation in support of the CuMo Exploration Project. July 2016

______. *Sacajawea's Bitterroot Baseline Survey Report*. Prepared for Idaho CuMo Mining Corporation in support of the CuMo Exploration Project. September 2017

______. *Sacajawea's Bitterroot Known Occurrence Survey Report*. Prepared for Boise National Forest in support of the CuMo Exploration Project. September 2017

USFS. Grimes Creek and Mohawk Gulch surface water sampling results, October 2017

______. *CuMo Exploration Project 2015 Supplemental EA and Decision Notice/FONSI Supplemental Information Report*. Prepared by the USDA FS, November 15, 2017

______. *Geologic Hazards, Soils, and Water Resources Technical Report for the CuMo Project*. February 2011, revised November 2018

______. *Fisheries Survey Specialist Report for the CuMo Exploration Project*. April 2011, revised November 2014 and September 2018

______. *Wildlife Specialist Report and Biological Evaluation for Threatened, Endangered, and Sensitive Terrestrial and Avian Species for the CuMo Exploration Project*. February 2011, revised February 2015 and September 2018

______. *Wolverine Addendum to the Wildlife Specialist Report and Biological Evaluation for Threatened, Endangered, and Sensitive Terrestrial and Avian Species for the CuMo Exploration Project*. August 2013, revised February 2015 and September 2018

U.S. Fish and Wildlife Service. CuMo Exploration Project, Updated list of threatened and endangered species, Consultation Code: 01EIFW00-2015-SLI-0236. January 28, 2015, updated March 21, 2018, and updated November 9, 2018.

Vizgirdas, E.R. 016. CuMo Site Visit – 10/27/16: Pioneer Fire Effects in *Lewisia acajaweana* (LESA) Plant Conservation Area (PCA). Field notes prepared for and available through the Boise National Forest Supervisor's Office.

The USFS is currently in process of preparing the final decision which is expected in early 2020.

In June of 2017, the Boise National Forest issued ICMC a Road Use Permit to perform road maintenance on National Forest Service roads 382C, 397, and 397B using best management practices. The road maintenance work was completed in June and July of 2017 (USFS 2018).

20.2 Permitting for Mining Operations

Environmental permitting for mines in Idaho is predicated on land status. Because the mine will be located on public land administered by the U.S. Department of Agriculture – Forest Service, Boise National Forest, Idaho City Ranger District and patented claims (private land owned and controlled by ICMC), the permitting path will involve multiple state and federal agencies as shown in Table 20-1. A more complete list can only be prepared after the mining plan of operations is complete.

20.2.1 Federal Authorizations and Permits

Exploration and mining on lands administered by a federal agency, in this case the USFS, requires authorization to conduct surface-disturbing activities. Mining for locatable minerals on lands administered by the USFS are guided by 36 Code of Federal Regulations Part 228. These regulations require that a mining plan of operations (Plan) be prepared for any operation likely to cause significant disturbance of surface resources. The Plan must provide a detailed description of construction, operations, closure, and reclamation of the proposed mining operation as well as a reclamation cost estimate. Detailed technical documents to support the Plan can include but not be limited to engineering designs for the open pits, processing plants, waste rock dumps, tailings storage facilities, access roads, power supplies, and water supplies.

The "complete" Plan has to provide sufficient detail in order to identify and disclose potential environmental impacts during the mandatory NEPA review process, under which the potential impacts associated with project development are analyzed. The most likely level of NEPA analysis for this project will be an EIS which is a public disclosure document, not a permit or approval document. An EIS is intended to disclose any environmental impacts that may occur from the project and guide the decisions of the public land managers. The USFS will most likely require that an EIS be prepared for the project due to:

- Size of the operation
- If the proposed project is expected to have significant impacts to a critical elements or resources
- If a large potential for use of or impacts to surface water and/or groundwater exists
- If non-governmental organizations or public opposition is expected to be significant

Table 20-1: Major permits and authorizations that may be required[1]

Name	Authorizing Agency
Federal Permits and Authorizations	
Mining Plan of Operations	USFS
EIS Review and Approval	USFS, U.S. Environmental Protection Agency, and U.S. Army Corps of Engineers
Approved Mining Plan of Operations/Record of Decision	USFS
Rights-of-Way for water/power/access corridors outside of Mining Plan of Operations boundary	USFS and/or other federal and state agencies
Clean Water Act Section 404 Wetland Permit	U.S. Army Corps of Engineers
Threatened and Endangered Species Consultation and Compliance with the Endangered Species Act	U. S. Fish and Wildlife Service
Compliance with the Bald Eagle Protection Act	U. S. Fish and Wildlife Service
Permit for Purchasing Explosives	Department of Homeland Security
Mine Safety	Mine Safety and Health Administration
Idaho State Permits and Authorizations	
Stream Channel Alteration Permit	Idaho Department of Water Resources
Water Right Appropriation	
Dam Safety Permit	
Reclamation Plan Approval	Idaho Department of Lands
Title V Operating Permit	Idaho Department of Environmental Quality – Air Quality Division
Approval of Plans for a New Sewage Treatment Facility	Idaho Department of Environmental Quality – Water Quality Division
Compliance with the Safe Drinking Water Act	
Clean Water Act 401 Certification	
Idaho Point Discharge Elimination Permit	
Solid Waste Management	Idaho Department of Environmental Quality – Waste Management & Remediation Division
Transportation and Storage of Hazardous Materials, Chemicals and Fuel Permits	Idaho Department of Transportation
Consultation with State Historic Preservation Officer	Idaho State Historic Preservation Office
Local	
Building Permits	Boise County
Road Maintenance Agreement	
Conditional Use Permit	

[1] No permit applications in relation to mining have been filed to date.

An EIS must consider possible impacts to the following critical elements and resources:

- **Critical elements** – Air quality, aquatics, floodplains, cultural resources, environmental justice, migratory birds, Native American religious concerns, non-native invasive species, threatened and endangered species, solid and hazardous wastes, hydrology including geochemistry, wetlands, and wilderness.
- **Resources** – Soils, geohazards, roadless areas, vegetation, forestry, geology/mineralogy, paleontology, hazardous materials, lands and access, livestock/grazing, recreation, scenic values and noise, socioeconomics, and transportation.

The USFS will require that baseline environmental surveys be conducted which will likely be above and beyond those conducted for the exploration activities. On-the-ground surveys will typically include: cultural resources; vegetation and animal biological resources including threatened, endangered, and sensitive species and migratory birds; soils resources; noxious and invasive species; jurisdictional waters; and hydrology, including geochemistry. These surveys, prepared in accordance with federal and state protocols, will identify the presence or absence of a particular resource and be used as the baseline to assess potential impacts. The same level of study will be required for any rights-of-way for new/improved access roads and water/power line corridors outside of the Plan boundary.

Other resources that will likely have to be addressed via desktop studies and stakeholder consultation include but are not limited to: Native American religious concerns, environmental justice, paleontology, livestock grazing, recreation, wilderness, and lands with wilderness characteristics.

The requirements of the Plan document are fairly well-defined. However, virtually all of the baseline data collection necessary for the impact assessment phase of the project will need to be collected, analyzed and interpreted in conjunction with the USFS in order to ensure that the information collected meet the data quality objectives of the program. A listing of the types of studies that should be undertaken during the mine planning phase and in advance of the NEPA process and in support of the acquisition of various other permits, could include:

- Biological resources
- Cultural resources of all areas proposed for disturbance unless the area has been surveyed within the past ten years
- Hydrogeological assessment (may include impact modeling including potential for pit lakes)
- Jurisdictional waters and wetlands
- Geochemical characterization of mill feed, waste rock, spent mill feed)
- Air quality/meteorological parameters
- Traffic study
- Environmental justice/socioeconomics

The length of time to prepare an EIS varies with the complexity of the project. The USFS is in the process of revising its NEPA procedures to reduce the time and cost of project analysis and decision making, increasing the scale of analysis, accomplishing more work on the ground, and creatively designing new ways to care for the land. The project proponent is also expected to enter into a cost recovery agreement with the USFS for the development of the EIS for specialist time as well as pay a third-party contractor to prepare the EIS.

ICMC will have to provide adequate operational and baseline environmental information for the USFS to analyze potential environmental impacts as required by the NEPA and to determine if the mining plan of operations will prevent significant impacts to the environment. Insufficient baseline data will slow down the EIS process. The same types of baseline information and level of detail collected for the proposed mine will also have to be collected for the alternatives analyzed in the EIS. Baseline information will also have to be developed for rights-of-way for power and water line corridors, and access roads where applicable.

During the EIS process, applicant-committed environmental protection measures and mitigation measures will be identified for the various resources and become part of the mining plan of operations and record of decision. These measures will be used to monitor and mitigate potential impacts.

Other federal agencies, namely the U.S. Army Corps of Engineers, Environmental Protection Agency, and the U. S. Fish and Wildlife Service, may be involved in the EIS process as cooperating agencies; state agencies can also be cooperating agencies. The U.S. Army Corps of Engineers may require permitting under Section 404 of the Clean Water Act if jurisdictional waterways are affected by the mine development. The U.S. Fish and Wildlife Service will become involved if the mine has the potential to affect threatened and endangered species.

20.2.2 Idaho State Authorizations and Permits

As shown in Table 20-1, a number of Idaho state authorizations and permits will also be required from at least five different Idaho state departments and divisions. Much of the information developed for the federal permitting process can be used to obtain the state permits. Idaho agencies typically process complete applications within the EIS process time frame.

20.2.3 Boise County Permits

The Boise County Zone and Development Ordinance is applicable, and a Conditional Use Permit is required for mining activities on federal land located in Boise County.

20.3 Monitoring

Environmental resources within the project area will be monitored prior to mine construction to develop baseline conditions, and during mining operations, reclamation, closure, and post-closure. Resources typically monitored include: climate and air quality; surface and ground water quality and quantity; geochemistry and management of ore, waste rock, and tailings; fisheries, wildlife, noxious weeds and invasive species; effectiveness of stormwater controls, and reclamation success.

During the federal and state permitting processes, ICMC will develop specific monitoring plans that incorporate state and federal monitoring requirements. The monitoring plans must meet the following objectives:

- Demonstrate compliance with the approved plan of operations and other federal or state environmental laws and regulations.
- Provide early detection of potential problems, and to supply information that will assist in directing corrective actions should they become necessary.

- Provide details on type and location of monitoring devices, sampling parameters and frequency, analytical methods, reporting procedures, and procedures to respond to adverse monitoring results.

The TSF will typically be monitored during construction, operation, closure, and post-closure to verify compliance with design specifications, operating conditions, water management, water quality, and reclamation success as required by Idaho regulations and USFS authorizations. Geochemical characterization of waste rock, ore, and tailings will also be undertaken prior to and during mining to guide dump and stockpile designs, stormwater controls, and monitoring. Post-closure monitoring of the waste rock dumps and TSF will be performed in compliance with federal and state permits.

Mine tailings impoundment structure designs in Idaho are regulated under IDAPA 37.03.05 by the Idaho Department of Water Resources. ICMC will have to post a bond to provide a means by which the TSF can be placed in a safe maintenance-free condition if abandoned by the owner without conforming to the approved abandonment.

20.4 Reclamation

20.4.1 Federal Reclamation Performance Bond

The USFS will require a reclamation performance bond under 36 Code of Federal Regulations 228A that calculates costs based on the assumption that the operator defaults, and the USFS must complete reclamation activities. Idaho has a memorandum of understanding which allows the state to recognize valid bonds held by the USFS as long as such bonds are in an amount as great as or greater than the required state bond. The USFS will accept the following bond instruments: negotiable Treasury bills and notes which are unconditionally guaranteed as to both principle and interest in an amount equal at their par value to the penal sum of the bond; or certified or cashier's check, bank draft, post office money order, cash, assigned certificate of deposit, assigned savings account, blanket bond, or an irrevocable letter of credit equal to the penal sum of the bond. The bond will have to be posted prior to surface disturbance occurring.

20.4.2 State Reclamation Performance Bond

A reclamation plan and reclamation cost estimate will also have to be prepared for the project in accordance with Idaho Administrative Procedures Act 20.03.02. Prior to beginning any surface mining on a mine panel covered by a Plan, an operator must submit to the director, on a surface mining reclamation bond form, a performance bond meeting the requirements of this rule. The amount must be the amount necessary to pay the estimated reasonable costs of reclamation required under the reclamation plan for each acre of land to be affected during the first year of operation, plus ten percent. The actual cost of reclamation must not exceed $15,000 per acre of land to be affected. The reclamation bond may be in the following forms: corporate surety bond, collateral bond, or a letter of credit. The bond will have to be posted prior to surface disturbance occurring.

If ponds or lakes are created during the mining process and will remain after reclamation is completed, the Idaho Department of Water Resources requires the operator or landowner to obtain a water right. If a water right cannot be obtained prior to a plan being submitted, then the reclamation plan must include backfilling to an elevation above the local ground water table. Bond calculations must include those backfilling costs.

20.5 Social and Community Impact

ICMC has initiated consultation with various stakeholders namely: government officials at all levels and local communities in regard to the potential social and community impacts or improvements that may occur as the project progresses. All groups are provided regular updates as the project is proceeding (Hilscher et al, 2018).

The project is active in all local communities and for example has been in discussion and committed, subject to proceeding to mine development, to the restoration and reclamation work of the contaminated placer gold dredge tailing that currently are present in the Grimes Creek. Local communities and officials have come out in strong support of the project and are actively working with the project on both the Grimes Creek project and future planning (Hilscher et al, 2018). The contaminated dredge tailings are not located on the CuMo property that is the subject of this technical report. There are no negotiations or agreements with the local communities at this time.

Federal and state planning and permitting processes mandate that the public have an opportunity to provide input. ICMC, in coordination with federal and state agencies, will engage with the public during these mandated public scoping and comment periods. Furthermore, ICMC will have the opportunity to engage with stakeholders and local communities outside of the permitting processes in order to define potential infrastructure and community support needs. Until ICMC presents an actual mining plan of operations for community feedback, there is no additional reasonably available information to disclose.

Typically, small communities have competing social concerns when a mine is planned in the vicinity, i.e., the need for jobs versus changes to the fabric of the community resulting from an influx mining and contractor employees. Potential social issues that could arise from the CuMo project could generally include:

- A shortage of temporary and permanent housing
- Insufficient of capacity of schools, health care, law enforcement, solid waste disposal, and municipal infrastructure
- Insufficient road network capacity leading to traffic slowdowns and degradation of road surfaces
- Increases in crime, drug abuse, and alcoholism

The public will have multiple opportunities to provide comments during the federal and state scoping and comments periods. In the past, ICMC has engaged with the nearby communities concerning the exploration project. This practice is expected to continue during mine development which will allow ICMC and the communities to identify salient issues and work towards resolution.

20.6 Potential Issues

The 2011 Environmental Assessment, 2015 Supplement Environmental Assessment, and the 2018 Supplemental Redline Environmental Assessment identified resource values that occurred or had the potential to occur in the CuMo project area that may affect mine permitting by changing the habitat and/or affecting individuals. These resource values included:

- The presence of Sacajawea's bitterroot, a sensitive plant species. Just over two dozen populations of Sacajawea's bitterroot are known to exist, roughly three-fourths of them on the Boise National Forest (USFS 2019).
- The potential for a number of rare plant habitat for other sensitive and watch plant species exists.
- The Canada lynx (*Lynx canadensis)* is listed under the Endangered Species Act with potential habitat in the project area.
- The wolverine (*Gulo gulo*) was proposed for listing as a threatened species under the Endangered Species Act in 2016 with potential habitat in the project area.
- Other USFS sensitive species have potential habitat within the project area: boreal owl (*Aegolius funereus*), flammulated owl (*Psiloscops flammeolus*), great gray owl (*Strix nebulosi*), mountain quail (*Oreortyx pictus*), northern goshawk (*Accipiter gentilis*), white-headed woodpecker (*Picoides albolarvatus*), grey wolf (*Canis lupus*), bull trout (*Salvelinus confluentus),* and wolverine.

Fresh water supply from surface or ground water will likely be one of the most difficult hurdles to overcome. An estimated 30,000 gpm of fresh water could be required. All water in Idaho is owned by the public; holding a water right does not give the water user ownership of the water. A water right simply gives the user the right to divert water. All water rights in Idaho exist for beneficial uses. The project will be located in Basin 65 which includes the entire Boise River Drainage (IDWR, 2018).

At this time and based on the undertaken studies, no issues could be identified that would materially impact the ability to eventually extract mineral resources at the project; however, ICMC should be prepared to address potential issues associated with but not limited to:

- Water including supply, water rights, and delivery system and potential impacts
- Water management (stormwater, contact/non-contact water, water quality)
- Geochemistry of ore, waste rock, tailings solids and solution, and post-mining pit lake
- Management of ore stockpiles, waste rock dumps, and tailings during operations, closure, and post-closure
- Threatened, endangered, and special status plant and animal species
- Jurisdictional waters
- Transportation and access
- Reclamation and closure

Any issues identified during the permitting process will have to be analyzed, disclosed, and potentially mitigated.

The mine would be located in an area used for weekend summer dispersed recreation and fall big-game hunting and is well-known in the Boise area. A majority of the previous public scoping comments to the environmental assessments were against mining activities (although the commenters were directed to address the proposed action, which was the exploration project). Organized environmental groups such as the Idaho Conservation League and Sierra Club are keeping their constituents informed citing issues of potential pollution of the Boise river which supplies drinking water to the city of Boise. As such, well-funded, organized opposition to mining activities should be anticipated.

However, under the 1872 Mining Law as amended, ICMC has the legal right to develop the mineral resources on their mining claims. The USFS has a requirement to manage ICMC's activities in accordance with its mining regulations at 36 CFR 228A and must ensure compliance with the requirements of the National Environmental Policy Act (. As defined in law and regulations, the USFS is limited in that it may not deny ICMC's mining plan of operations provided that the activities proposed are reasonably incident to mining, not needlessly destructive, and comply with applicable federal, state, and local laws and regulations. The USFS does not have the authority to impose unreasonable requirements that would have the effect of denying the statutory right to explore and develop the mineral resource, provided the mining plan of operations otherwise meets the intent of applicable laws and regulations (USFS 2018).

At this time, a detailed discussion on mine closure and reclamation cannot be completed. However, ICMC will be required to post reclamation bonds to cover direct and indirect costs related to site stabilization, water treatment as needed, post-reclamation and post-mining monitoring, and public safety.

20.7 Mine Closure – General Discussion

There are comprehensive Idaho and USFS closure and reclamation requirements that the project proponent plan for closure and reclamation of mining disturbances on all affected land. Regulatory authorities will require that a surety or bond be posted sufficient to cover third-party costs to physically and chemically stabilize the site prior to the onset of mining. A reclamation cost estimate will have to be prepared that will be approved by state and federal agencies prior to any mining surface disturbance; the bond amount will have to be posted using an approved financial instrument. The financing costs associated with such a surety have not been modelled. The initial submissions will require a detailed discussion on how the mining disturbance will be physically and chemically stabilized and the duration of the closure process as reclamation and closure will be analyzed in the EIS.

The plans for final closure must address the long-term potential for surface and ground water contamination from the closed facility as well as stabilization of slopes, soils, and vegetation on mining disturbances. Typically, the closure permitting process involves a decommissioning plan or a permanent capping plan along with a post-closure monitoring commitment. Permit applicants should consider ways of closing a facility which will eliminate the possibility of future surface and ground water contamination and thereby eliminate the need for long-term water treatment and monitoring.

After mining operations cease, all buildings, infrastructure, and facilities from the CuMo Mine that have not been identified for a specific post-mining use, must be removed from the site during the reclamation, salvage, and site demolition phase. These activities will generally include, but not be limited to the:

- Regrading to a stable configuration, placement of growth media, and seeding of all disturbed surfaces without a postmining use

- Removal of surface pipelines and power lines, and the secure and stable abandonment of underground pipelines (including removal if required)
- Demolition of process facilities and salvage/removal of equipment and residual reagents for proper disposal
- Managing the drain-down solution to reduce the volume which may include the construction and operation of an evapotranspiration cell. Depending on site conditions, a water treatment plant and discharge of treated water may be necessary to prevent unauthorized discharges of mine water not meeting water quality standards
- Ongoing monitoring of closure compliance for surface and ground water quality, soil stabilization, and revegetation success
- Maintaining public safety features such as warning signs, pit berms, and other barriers

To the extent practicable, reclamation and closure activities will be conducted concurrently with mining and disturbance to: reduce the overall final reclamation and closure costs, minimize environmental liabilities, and limit exposure to surety or bonding costs. At the current phase of the CuMo project, a site-specific closure cost estimate has not yet been developed. An approximate closure cost of $150M has been assumed for preliminary economic evaluation. This estimate is not based on site-specific considerations and should be considered order-of-magnitude only within the accuracy of a PEA level study.

21 Capital and Operating Costs

21.1 Capital Cost Estimate

A summary of initial capital costs is provided in Table 21-1.

Table 21-1: Summary of initial capital costs

Capital Costs	($M)
Mine - Equipment, etc.	344
Capitalized Mine Operating Costs	330
Sort Plants	160
Mill	1,293
Roaster	208
Tailings	22
Infrastructure	76
Total Initial Capital Directs	**2,433**
Contingency on Initial Capital Directs (excl Mining)	**176**
Indirects	
Mine	15
Plant (incl. Sort)	354
Roaster	81
Infrastructure	14
Total Initial Capital Indirects	**464**
Sustaining Capital	
Mine	428
Sort Plants	42
Mill	349
Roaster	56
Tailings	84
Infrastructure	10
Total Sustaining Capital	**970**
Closure and Reclamation	150
Total Capital Costs	**4,193**
Initial Capex	**3,071**
Sustaining and Expansion Capex	**972**
Closure	**150**

21.1.1 Mining Capital Costs

The author developed the LOM schedule for the CuMo project, and based on this, derived equipment fleet requirements (Table 16-5). A breakdown of capital costs by equipment type of the primary mine equipment is provided in Table 21-2. These are presented here exclusive of contingency for clarity.

In addition to the primary mine equipment, ancillary equipment costs (light vehicles, maintenance vehicles, etc.) are factored at 5% of the primary equipment cost. This totaled $16.8M. Other capital costs for haul roads, earthworks, and technical equipment totaled $21.5M. Note all costs here are before contingency.

Table 21-2: Mine primary equipment capital costs

Equipment Type	Units	Initial Capital Cost
Rotary Blast Hole Drill	$M	28.6
Electric Cable Shovel	$M	97.7
Autonomous Trucks	$M	114.4
Track Dozer	$M	11.0
Rubber Tire Dozer	$M	5.8
Grader	$M	6.5
Water Truck	$M	8.7
Backhoe	$M	2.2
Total	**$M**	**274.9**

The total initial mine equipment direct capital cost including the above costs and contingency is estimated at $345M as shown in Table 21-1.

Mine indirects were estimated at $15M.

The mining capitalized pre-production pre-stripping costs of $329M are incurred in the two years of mining activity prior to processing facility commissioning.

21.1.2 Processing Capital Costs

A summary of the estimated capital cost for the processing and on-site ancillary facilities is provided in Table 21-3 and for the roaster in Table 21-4.

The CuMo circuit capital cost estimate for the process plant, roaster and related ancillary infrastructures was derived by factoring the mechanical equipment costs, which are defined in the concept study mechanical equipment list (Ausenco, 2009). Equipment costs were based on recent equipment quotations, or from previous projects. The cost estimates for all other disciplines were factored from the mechanical equipment list.

Indirect costs, including project contingency have been provided for in the capital cost estimates. Indirect costs have been estimated based on a factor of the total direct costs established from previous projects.

Table 21-3: Summary of plant initial capital cost estimate

Category	Units	150 ktpd
Direct Costs		
Site Development	$M	29
Sorting Plant	$M	160
Concentrator	$M	1005
Concentrator Services	$M	50
Concentrator Infrastructure	$M	87
Molybdenum Plant	$M	62
Tailings Line	$M	21
Spares and First Fill	$M	38
Total Direct Costs	**$M**	**1,453**
Indirect Costs		
Temporary Construction Facilities	$M	30
EPCM	$M	222
Pre-production Owner's Costs	$M	60
Project Fee	$M	42
Contingency	$M	139
Total Indirect Costs	**$M**	**492**
Total	**$M**	**1,944**

Table 21-4: Summary of roaster initial capital cost estimate

Category	Units	150 kt/d
Direct Costs		
Site Works	$M	13
Concentrate Feed Handling	$M	21
Molybdenum Roaster	$M	75
Rhenium Recovery	$M	52
Acid Plant	$M	46
Gas Scrubbing	$M	-
Total Direct Costs	**$M**	**208**
Indirect Costs		
Temporary Construction Facilities	$M	21
EPCM	$M	42
Pre-production Owner's Costs	$M	12
Project Fee	$M	6
Contingency	$M	21
Total Indirect Costs	**$M**	**102**
Total	**$M**	**309**

Assumptions

Geotechnical

- A detailed geotechnical and drainage assessment of the proposed site is not yet available. For the purpose of the study, no allowance for special ground preparation has been made.

Base Date and Exchange Rates

- The authors have reviewed, verified and confirmed all information is valid at the date of the report that cost estimate is current. The estimate and all costs are expressed in 2019 United States dollars. In the verification process, comparative quotes were solicited from appropriate vendors for updated equipment costs from the initial 2009 estimate produced by Ausenco. These were adjusted by exchange factors of 1.25 CAD to 1 USD or 1.10 CAD to 1 Euro when necessary.

Electricity Supply

- It is assumed that power is available to satisfy demand requirements for the proposed plant. Costs associated with power distribution to the site have been included within this estimate as detailed below. All other costs of power supply, including reticulation to the assumed take-off point on Highway 21, all land access, and licensing and permitting are excluded. It should be noted Idaho Power is currently in the final stages permitting a brand new power line extension from Horseshoe Bend to Garden Valley. This power line comes within 10 miles of the property and should reduce the costs associated with power.

- High and medium voltage switch gear and distribution within the battery limits have been included in the estimate. Individual drive switchgear and cabling have been included as part of the area factors.

Water Supply

- A water supply capable of supplying the required demand of the processing plant is assumed to be available. For this reason, costs associated with any increase in water supply have not been included within this estimate. The costs associated with water (and air) reticulation within the scope have been estimated based on the area piping factors.

Contingency

The estimate currently includes an amount of 10% of the total cost of the fixed plant as an estimate recommended for contingency.

Owner's Costs

Owner's costs have been excluded from this estimate.

Project Fee

A project fee of 3% of the direct costs has been included.

Escalation

Escalation provision for currency inflation past Q1 2020 has not been included in the estimate.

21.1.3 Tailings Storage Facilities Capital Costs

The capital cost estimate for the TSF makes provision for constructing the initial starter dam of the TSF to an elevation of 6,300 ft, which is sufficient to store the first two years of tailings production. The tailings dam would be constructed using run of mine waste and sort waste and compacted in one-meter lifts. As the waste is already being delivered to the footprint for disposal, the only cost included for placement in the estimate is to cover the incremental compaction costs. No allowance was provided for spreading the material as it is assumed that the dozers already on the waste disposal area will handle that activity. The cost estimates are for an unlined TSF and it is estimated that lining the TSF would cost an additional 20 to 30 percent of the unlined construction cost.

An allowance has also been made for excavating the overburden encountered beneath the starter dam footprint to ensure a good foundation for the dam. The presence of unsuitable foundation soils and the soils areal extent and depth will be evaluated in future studies by geotechnical site investigations. The cost estimates will be adjusted based on the results of the investigations. This material would be stockpiled for use in reclamation activities later on in the mine life. Costs were also estimated for the general foundation clearing within the footprint of the tailings impoundment in advance of waste placement.

The storage capacity of the TSF will be increased through five additional raises of the dam in years 2, 5, 10, 15 and 20 to an ultimate elevation of 6,950 ft. Sustaining capital has been estimated for each of these raises to accommodate compaction of the waste rock in the compacted dam zone as foundation preparation in years 2 and 5 when the footprint is undergoing expansion to the south.

21.1.4 Capital Cost Estimate Exclusions

No specific allowance or estimate was made for items such as foreign currency fluctuations, escalation, etc., which will be reviewed in greater detail in the pre-feasibility study and subsequent feasibility study. The following items are excluded from this study:

- Power generation (power is assumed to be purchased)
- Project acquisition costs
- Pre-feasibility study costs
- Feasibility study costs
- Legal fees
- Corporate costs
- Exploration, geotechnical and sterilization costs
- Water compensation
- Bore field or raw water dam
- Construction camp
- Plant or infrastructure outside of the battery limits
- All Owner payable taxes, government and other charges (operating cost not capital)
- License and royalty fees
- No allowances are made for special incentives (schedule, safety or others)
- Sustaining or deferred capital costs (operating cost not capital)
- Cost changes due to currency fluctuation
- Force Majeure issues
- Owners cost prior to project approval
- Sunk cost
- Future scope changes
- Project interest / financing costs
- Project insurances
- Permits / cost of permits
- Mine / plant closure and rehabilitation costs (included in financial model)
- Training of operations personnel
- Working capital
- Land acquisition
- Environmental consultants, studies, permitting and mitigation
- Any operational insurance such as business interruption insurance & machinery breakdown, etc
- Costs for community relations and services
- Any bridges or tunnels, permanent or temporary
- Maintenance of all roads and bridges and facilities mentioned above

- Additional test-work
- Provision of hardstand for the construction site area
- Rubbish disposal
- Dust suppression
- Excavation of rock
- Site drainage

21.2 Operating Cost Estimate

The total LOM operating costs for the CuMo project are summarized in Table 21-5.

Table 21-5: Summary of LOM operating costs

Operating Costs	LOM ($M)	Unit Rates ($/t)	Unit Rates ($/lb Mo.Eq.)
Mining	5,797	$3.66	$2.99
Bulk Sort	778	$0.49	$0.40
Middling Sort	192	$0.12	$0.10
Processing	7,042	$4.45	$3.63
Sort Waste Delivery	395	$0.25	$0.20
G&A	805	$0.51	$0.42
Less Capitalized Operating Costs	-329	-$0.21	-$0.17
Total Operating Costs	**14,680**	**$9.28**	**$7.57**

The estimate was prepared with a base date of July 2019 to an accuracy level of ±40%. Various parties contributed to the estimates as detailed below. These estimates exclude sustaining capital expenditure requirements but include realization costs associated with sale of final products.

21.2.1 Mine Operating Costs

The author estimated the mine operating costs based on comparison to similar projects. Site-specific haulage profiles were considered to ensure that short haul options into Charlotte Gulch in early years are reflected as well as the longer hauls to Clear Creek for TSF construction and WRF disposal.

The non-haulage operating costs are estimated at approximately $0.70/t. Adding haulage gives an average mine operating cost of $1.28/t, ranging from $0.91 to $1.87/t of material moved. Mine operating costs per ton of material processed is $3.66. The total LOM operating cost is estimated at $5,797M. Note that $329M of these mine operating costs in the pre-production period were capitalized.

21.2.2 Sort Plant Operating Costs

For the bulk sorting system, a unit cost of $0.10/t was assumed for each stage of sorting. To this is added $0.20/t for primary crushing, giving a LOM total operating cost of $778.1M.

For the particle sorting system, a unit cost of $0.30/t of material fed was assumed, giving a LOM operating cost of $192.1M.

21.2.3 Mill Operating Costs

The total process operating costs have been developed on an annual basis throughout the life of the mine. Cost estimates were generated the selected throughput/mill feed scenario based on the metallurgical samples tested by SGS Canada Inc. These have been combined, using the CuMo mine plan to produce LOM and annual operating estimates. A summary of the average operating costs per ton of mill feed treated for the project is outlined in Table 21-6. The costs have been divided into the key cost centers.

Table 21-6: Estimated plant average operating costs

Category	Units	150 kt/d
Labor	$/ton	0.19
Power	$/ton	1.84
Maintenance Materials and Services	$/ton	0.68
Reagents and Consumables	$/ton	1.74
Total	**$/ton**	**4.45**

a) Labor

Site labor costs from the overall workforce schedule of personnel numbers, positions, salaries and overhead costs based on projects of similar size and location. Total employee costs have been developed by applying on-cost factors to base salaries. The on-costs include the cost of travel, overtime and shift premiums, leave pay, bonuses, pension and superannuation benefits, insurance coverage, educational assistance and supply of uniforms and personal protective equipment.

b) Power

Power is to be supplied to the mine site from the local power grid, provided by Idaho Power. Unit power cost rates have been supplied by CuMoCo at $0.063/kWh, based on information from the Thompson Creek Mine (Thompson Creek Mine Model, MineCost (2009)). This has been confirmed by the authors with large scale commercial rates (2018) in Idaho being as low as $0.055/kWh. Thus using $0.063/kWh can be considered reasonable.

c) Maintenance Consumables and Services

Maintenance consumable costs were estimated as a percentage of the direct installed capital cost (percent factor). The factor is based on actual data from similar projects and takes into consideration an assumed bond abrasion index of 0.25.

d) Reagents and Consumables

Reagent consumptions have been estimated from metallurgical test-work or comparable operations. Although reagent consumptions will vary according to metallurgical and production parameters, the average predicted consumptions, by material type, have been used for this exercise.

Budget quoted costs have been used for major plant reagents. Unit costs include an allowance for delivery to site but do not include duties, brokerage, handling charges or applicable taxes.

21.2.4 General Site and Administrative Costs

The author has assumed a general site and administrative (G&A) cost of $0.50/t mill feed based on comparison to similar size operations. At the modelled throughputs, this amounts to approximately $27.5M per year at full production.

22 Economic Analysis

22.1 Cautionary Statements

22.1.1 Certainty of Preliminary Economic Assessment

The preliminary economic assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

22.1.2 Mineral Resources are Not Reserves

Mineral resources are not mineral reserves and do not have demonstrated economic viability.

22.2 General

Economic analysis was undertaken using a discounted cashflow model that was constructed in MS EXCEL®. The model used constant (real) 2019 United States dollars and modelled the project cashflows in annual periods.

The model assumes a 36-month physical construction period.

The model does not place the project within an estimated calendar timeline and is intended only as an indication of the economic potential of the project to assist in investment decisions. Between the date of this report and the commencement of construction, a period of time sufficient for the pre-feasibility and feasibility study work programs to be executed must be allowed.

Important Note: The economic model considered only cashflows from the beginning of actual construction forward. Schedule and expenditure for the pre-feasibility study, including technical and economic studies, engineering studies, cost estimating, resource delineation and infill drilling, pit slope geotechnical characterization, metallurgical sampling and test-work, associated exploration, strategic optimization, mine, plant and infrastructure design, permitting and other pre-construction activities were NOT modelled.

Attention is drawn to Section 26 where the work plan and costs for the pre-feasibility study period of the project are summarized.

Table 22-1 shows a summary of key project parameters and project economics. LOM project annual cash flow is shown graphically in Figure 22-1.

22.3 Summary

The summary of CuMo project economics is provided in Table 22-1.

Table 22-1: Summary of potential project economics

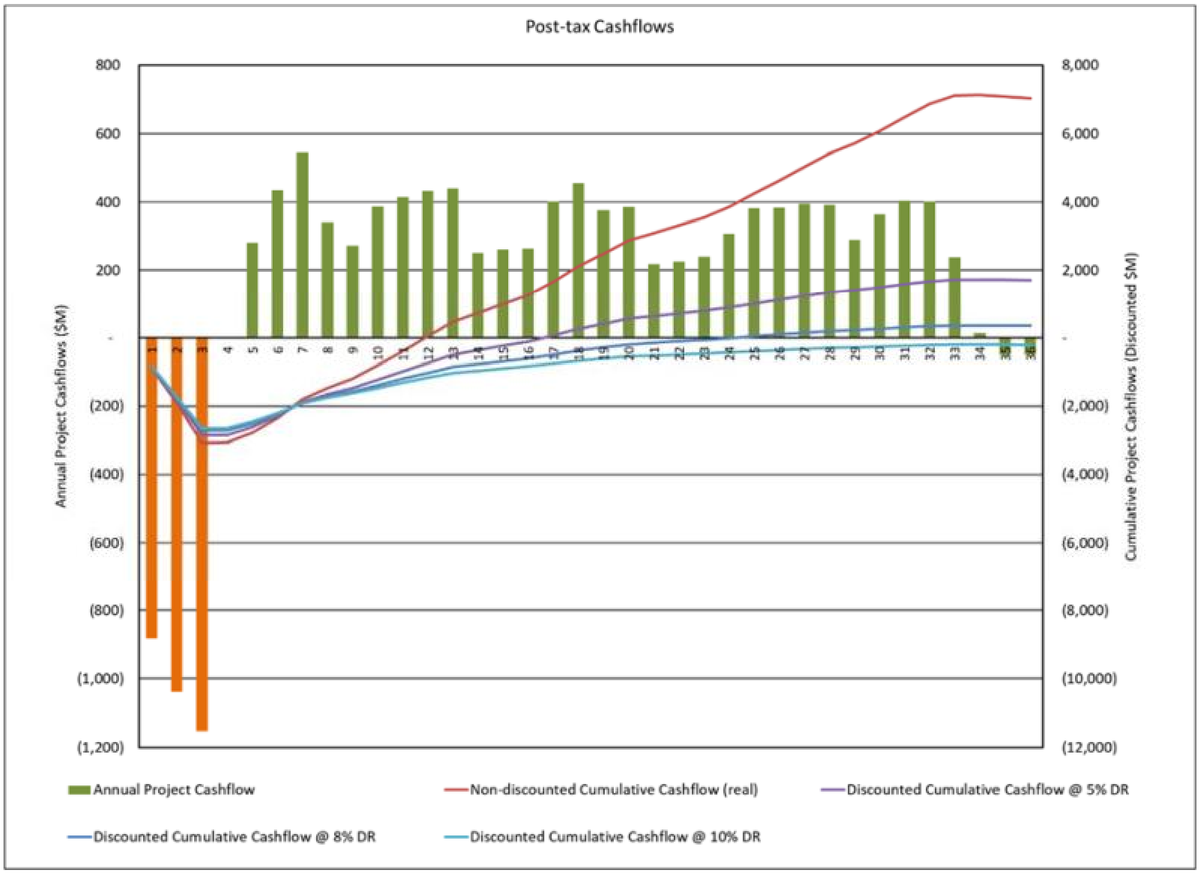
Project Metric	Units	Value
Pre-Tax NPV @ 5%	$M	2,470
Pre-Tax NPV @ 8%	$M	800
Pre-Tax NPV @ 10%	$M	113
Pre-Tax IRR	%	10%
After-Tax NPV @ 5%	$M	1,709
After-Tax NPV @ 8%	**$M**	**356**
After-Tax NPV @ 10%	$M	-205
After-Tax IRR	%	9%
Undiscounted After-Tax Cash Flow (LOM) (no capital)	$M	11,092
Undiscounted After-Tax Cash Flow (LOM) (capital)	$M	7,032
Payback Period from Start of Processing	years	8.0
Initial Capital Expenditure	$M	3,071
LOM Sustaining Capital Expenditure	$M	972
Closure	$M	150
LOM C-1 Cash Costs After By-product Credits	$/lb Mo	4.67
Nominal Flotation Process Capacity	stpd	150,000
Mine Life (years @ > 90% of full production)	years	28
LOM Flotation Mill Feed	kst	1,582,526
LOM Grades		
Molybdenite (MoS_2)	%	0.074%
Molybdenum (elemental Mo)	%	0.044%
Copper	%	0.105%
Silver	grams per tonne	3.00
LOM Waste Volume	kst	2,425,101
LOM Strip Ratio (Waste:Sort Feed)	ratio	1.11
Mass Pull to Mill from Sort Feed	%	72%
LOM Strip Ratio (Waste:Mill Feed)	ratio	1.53
First Five Years Average Annual Metal Production		
Molybdenum	klbs/yr	34,976
Copper	klbs/yr	93,394
Silver	kounces/yr	3,940
LOM Average Annual Metal Production		
Molybdenum (Mo Metal)	klbs/yr	43,072
Copper	klbs/yr	84,229
Silver	kounces/yr	3,575
LOM Average Mill Process Recovery		
Molybdenum (Mo Metal)	% contained metal	91.87%
Copper	% contained metal	76.33%
Silver	% contained metal	70.42%

The project as presented, and under the current assumptions, has the potential to be economic. The after-tax NPV is positive and has been tested across a range of sensitivities with respect to capital costs, operating costs and revenue (price).

Attention is drawn to the cautionary statements in Section 22.1 and the risks and opportunities discussed in Sections 25.2.7 and 25.3.6 respectively.

22.4 Project Cashflows

Project cashflows are summarized in Table 22-2 & Table 22-3, and shown graphically in Figure 22-1. Cumulative cashflows at discount rates (non-escalated) of 0%, 5%, 8% and 10% are also shown.



Source: SRK, 2019

Figure 22-1: Project cashflow summary chart

22.5 Production Schedule

The production schedule evaluated is summarized in Table 22-4. Metal production quantities and mine physicals are shown graphically in Figure 22-1.

Table 22-2: LOM annual project cash flow

PREFINANCE SUMMARY CASH FLOW	Units	LOM Total	Year -3	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
Payable Revenue																			
Payable Revenue from Molybdenum	$M	19,383	0	0	0	244	450	585	789	555	444	564	737	825	863	416	524	701	882
Payable Revenue from Copper	$M	7,581	0	0	0	117	359	341	263	321	358	334	258	241	207	373	326	276	212
Payable Revenue from Silver	$M	1,877	0	0	0	41	82	79	56	86	95	79	61	58	52	94	77	62	45
By-product Revenue	$M	266	0	0	0	3	6	8	11	8	6	8	10	11	12	6	7	10	12
Total Revenue from Payable Metal	**$M**	**29,106**	**0**	**0**	**0**	**406**	**897**	**1,013**	**1,119**	**969**	**902**	**985**	**1,066**	**1,135**	**1,134**	**888**	**934**	**1,048**	**1,151**
Moly Equivalent (MoEq) Payable Pounds	*mmlbs*	*1,908*	*0.0*	*0.0*	*0.0*	*26.6*	*58.8*	*66.4*	*73.4*	*63.6*	*59.2*	*64.6*	*69.9*	*74.4*	*74.3*	*58.2*	*61.3*	*68.7*	*75.5*
Total TCRC Freight & Royalty	**$M**	**1,253**	**0**	**0**	**0**	**19**	**47**	**48**	**46**	**47**	**48**	**47**	**44**	**45**	**43**	**48**	**45**	**45**	**43**
Total Minesite Revenue	**$M**	**27,853**	**0**	**0**	**0**	**387**	**851**	**965**	**1,073**	**923**	**855**	**938**	**1,021**	**1,090**	**1,091**	**841**	**889**	**1,004**	**1,108**
OPERATING COSTS																			
Mining	$M	5,797	0	142	173	181	191	187	196	197	211	191	202	214	205	201	199	190	188
Bulk Sort	$M	778	0	0	0	20	30	26	22	33	30	27	24	27	27	33	27	24	23
Middling Sort	$M	192	0	0	0	8	9	6	3	11	9	6	5	7	7	11	7	5	4
Processing	$M	7,042	0	0	0	122	244	244	244	244	244	244	244	244	244	244	244	244	244
Sort Waste Delivery	$M	395	0	0	0	17	18	12	7	23	18	13	10	15	15	23	13	8	8
G&A	$M	805	0	5	9	14	27	27	27	27	27	27	27	27	27	27	27	27	27
Less Capitalized Operating Costs		-329	0	-147	-182	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Operating Costs	**$M**	**14,680**	**0**	**0**	**0**	**362**	**520**	**503**	**500**	**535**	**539**	**509**	**513**	**533**	**524**	**538**	**516**	**498**	**494**
Operating Cashflow	**$M**	**13,173**	**0**	**0**	**0**	**25**	**331**	**461**	**574**	**388**	**316**	**429**	**508**	**556**	**567**	**302**	**373**	**506**	**614**
Summary Capex by Project Phase																			
Construction Costs	$M	3,071	882	1,038	1,151	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sustaining Capital Costs	$M	972	0	0	0	31	6	15	17	62	51	30	23	19	17	29	30	130	79
Closure Costs	$M	150	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Grand Total Capex (Including Closure)	**$M**	**4,193**	**882**	**1,038**	**1,151**	**31**	**6**	**15**	**17**	**62**	**51**	**30**	**23**	**19**	**17**	**29**	**30**	**130**	**79**
Working Capital	$M	-133	0	0	0	-9	44	12	12	-16	-7	11	11	9	1	-33	8	18	17
Pretax Cash Flow	**$M**	**9,113**	**-882**	**-1,038**	**-1,151**	**3**	**281**	**434**	**544**	**342**	**272**	**387**	**475**	**528**	**549**	**306**	**335**	**358**	**519**
Total Tax	**$M**	**2,081**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**60**	**95**	**109**	**56**	**73**	**93**	**117**
After-tax Net Cash Flow (Undiscounted)	**$M**	**7,032**	**-882**	**-1,038**	**-1,151**	**3**	**281**	**434**	**544**	**342**	**272**	**387**	**415**	**433**	**439**	**251**	**262**	**265**	**401**
After-tax Net Cash Flow (at 5% DR)	**$M**	**1,709**	**-861**	**-965**	**-1,019**	**2**	**226**	**332**	**396**	**237**	**180**	**244**	**249**	**247**	**239**	**130**	**129**	**124**	**179**
After-tax Net Cash Flow (at 8% DR)	**$M**	**356**	**-848**	**-925**	**-950**	**2**	**199**	**284**	**330**	**192**	**142**	**186**	**185**	**179**	**168**	**89**	**86**	**80**	**113**
After-tax Net Cash Flow (at 10% DR)	**$M**	**-205**	**-841**	**-900**	**-907**	**2**	**183**	**257**	**293**	**167**	**121**	**157**	**153**	**145**	**134**	**69**	**66**	**60**	**83**

Note: MoEq lbs = (Revenue from recovered, payable metal plus by-products before deduction of Royalty TCRC and freight)/(Price of Mo per lb)

Table 22-3: LOM annual project cash flow – continued

PREFINANCE SUMMARY CASH FLOW	Units	LOM Total	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Closure
Payable Revenue																			
Payable Revenue from Molybdenum	$M	19,383	932	892	839	372	471	518	639	758	753	806	800	573	679	756	784	233	
Payable Revenue from Copper	$M	7,581	157	156	181	369	345	328	297	249	212	201	180	258	250	192	163	57	
Payable Revenue from Silver	$M	1,877	36	39	50	100	98	86	73	61	49	45	46	65	60	48	40	12	
By-product Revenue	$M	266	13	12	12	5	6	7	9	10	10	11	11	8	9	10	11	3	
Total Revenue from Payable Metal	**$M**	**29,106**	**1,139**	**1,100**	**1,082**	**846**	**920**	**939**	**1,017**	**1,078**	**1,024**	**1,063**	**1,037**	**904**	**998**	**1,006**	**998**	**305**	
Moly Equivalent (MoEq) Payable Pounds	*mmlbs*	*1,908*	*74.7*	*72.1*	*70.9*	*55.5*	*60.3*	*61.5*	*66.7*	*70.7*	*67.2*	*69.7*	*68.0*	*59.3*	*65.4*	*66.0*	*65.5*	*20.0*	
Total TCRC Freight & Royalty	**$M**	**1,253**	**39**	**39**	**40**	**47**	**48**	**46**	**46**	**44**	**40**	**40**	**39**	**41**	**42**	**39**	**36**	**11**	
Total Minesite Revenue	**$M**	**27,853**	**1,099**	**1,061**	**1,041**	**799**	**872**	**892**	**971**	**1,034**	**984**	**1,023**	**998**	**863**	**956**	**967**	**962**	**294**	
OPERATING COSTS																			
Mining	$M	5,797	186	198	197	208	219	206	196	188	183	179	175	183	119	116	125	50	
Bulk Sort	$M	778	24	26	28	35	31	28	24	22	23	24	24	32	26	24	24	9	
Middling Sort	$M	192	5	6	8	13	10	7	5	3	4	5	4	10	6	5	4	2	
Processing	$M	7,042	244	244	244	244	244	244	244	244	244	244	244	244	244	244	244	94	
Sort Waste Delivery	$M	395	10	13	17	26	19	14	9	6	8	9	9	20	11	9	8	3	
G&A	$M	805	27	27	27	27	27	27	27	27	27	27	27	27	27	27	27	11	
Less Capitalized Operating Costs		-329	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Total Operating Costs	**$M**	**14,680**	**497**	**514**	**521**	**553**	**552**	**526**	**504**	**491**	**490**	**488**	**484**	**516**	**435**	**425**	**432**	**168**	
Operating Cashflow	**$M**	**13,173**	**602**	**547**	**520**	**246**	**320**	**367**	**467**	**543**	**494**	**535**	**515**	**348**	**521**	**542**	**530**	**126**	
Summary Capex by Project Phase																			
Construction Costs	$M	3,071	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Sustaining Capital Costs	$M	972	25	70	32	23	17	48	48	35	17	17	17	17	17	17	17	17	
Closure Costs	$M	150	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	150
Grand Total Capex (Including Closure)	**$M**	**4,193**	**25**	**70**	**32**	**23**	**17**	**48**	**48**	**35**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**150**
Working Capital	$M	-133	0	-5	-2	-42	13	6	17	15	-9	10	-4	-31	30	5	0	-151	-64
Pretax Cash Flow	**$M**	**9,113**	**577**	**483**	**491**	**265**	**290**	**312**	**401**	**494**	**486**	**508**	**501**	**361**	**475**	**520**	**513**	**260**	**-86**
Total Tax	**$M**	**2,081**	**121**	**107**	**104**	**47**	**65**	**73**	**93**	**111**	**103**	**113**	**109**	**73**	**111**	**115**	**111**	**22**	**0**
After-tax Net Cash Flow (Undiscounted)	**$M**	**7,032**	**456**	**376**	**386**	**218**	**225**	**240**	**308**	**383**	**383**	**395**	**392**	**289**	**364**	**404**	**402**	**238**	**-86**
After-tax Net Cash Flow (at 5% DR)	**$M**	**1,709**	**194**	**152**	**149**	**80**	**79**	**80**	**98**	**116**	**110**	**108**	**103**	**72**	**86**	**91**	**86**	**49**	**-15**
After-tax Net Cash Flow (at 8% DR)	**$M**	**356**	**119**	**90**	**86**	**45**	**43**	**42**	**50**	**58**	**54**	**51**	**47**	**32**	**38**	**39**	**36**	**20**	**-6**
After-tax Net Cash Flow (at 10% DR)	**$M**	**-205**	**86**	**64**	**60**	**31**	**29**	**28**	**33**	**37**	**34**	**32**	**29**	**19**	**22**	**22**	**20**	**11**	**-3**

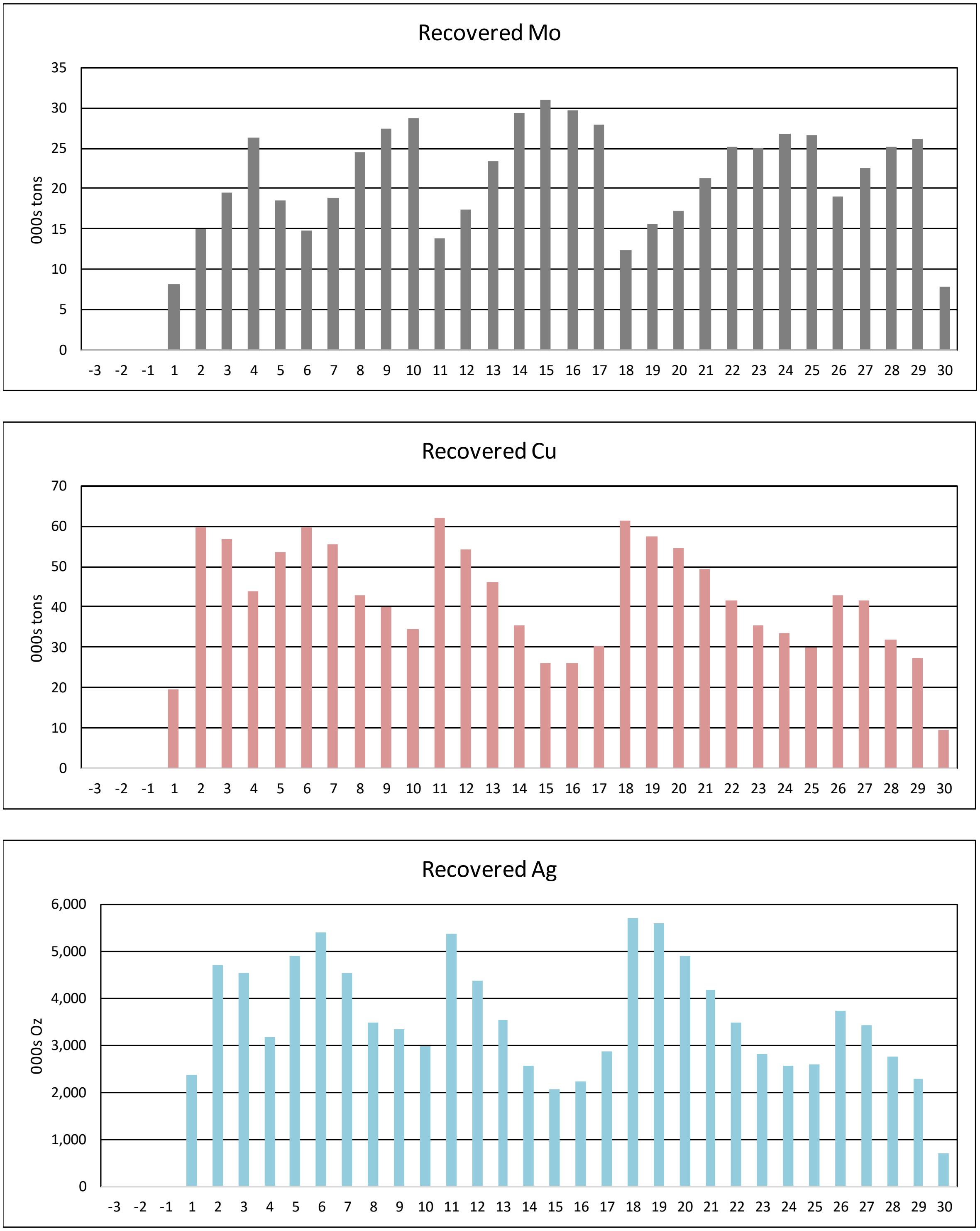
Notes: MoEq lbs = (Revenue from recovered, payable metal plus by-products before deduction of Royalty TCRC and freight)/(Price of Mo per lb)
Closure is costed over 3 years but summarized into a single year in this table for brevity

Table 22-4: Production schedule summary

Item	Units	LOM Totals	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
Mill Feed	kst	1,582,526	0	0	27,375	54,750	54,900	54,750	54,750	54,750	54,900	54,750	54,750	54,750	54,900	54,750	54,750	54,750
MoS_2	*%*	*0.074%*	*0.000%*	*0.000%*	*0.058%*	*0.053%*	*0.063%*	*0.088%*	*0.060%*	*0.052%*	*0.062%*	*0.082%*	*0.087%*	*0.096%*	*0.047%*	*0.063%*	*0.076%*	*0.093%*
Cu	*%*	*0.105%*	*0.000%*	*0.000%*	*0.093%*	*0.138%*	*0.128%*	*0.101%*	*0.133%*	*0.139%*	*0.131%*	*0.108%*	*0.106%*	*0.089%*	*0.150%*	*0.125%*	*0.109%*	*0.088%*
Ag	*gpt (metric)*	*3.00*	*0.00*	*0.00*	*3.63*	*3.51*	*3.36*	*2.64*	*3.83*	*4.06*	*3.56*	*3.04*	*2.90*	*2.46*	*4.03*	*3.39*	*2.95*	*2.32*
Waste	kst	2,425,101	156,915	174,639	132,521	90,212	108,661	114,796	91,153	98,714	84,230	93,481	102,999	89,800	71,062	82,589	78,547	74,561
Strip Ratio (waste:sort feed)	*ratio*	*1.11*	*0.00*	*30.20*	*2.73*	*1.09*	*1.49*	*1.73*	*1.00*	*1.19*	*1.13*	*1.32*	*1.33*	*1.16*	*0.79*	*1.11*	*1.16*	*1.12*
Head Grade (% MoS_2 Eq. recoverable)	*%*	*0.123%*	*0.000%*	*0.000%*	*0.099%*	*0.109%*	*0.123%*	*0.136%*	*0.118%*	*0.110%*	*0.120%*	*0.130%*	*0.138%*	*0.138%*	*0.108%*	*0.114%*	*0.128%*	*0.140%*
Recovered Mo	kst	646	0	0	8	15	19	26	19	15	19	25	27	29	14	17	23	29
Recovered Cu	kst	1,263	0	0	20	60	57	44	53	60	56	43	40	34	62	54	46	35
Recovered Ag	koz	107,239	0	0	2,367	4,710	4,535	3,190	4,898	5,400	4,529	3,485	3,340	2,982	5,379	4,378	3,545	2,581
MoEq lbs	*mmlbs*	*1,908.2*	*0.0*	*0.0*	*26.6*	*58.8*	*66.4*	*73.4*	*63.6*	*59.2*	*64.6*	*69.9*	*74.4*	*74.3*	*58.2*	*61.3*	*68.7*	*75.5*

Item	Units	LOM Totals	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30
Mill Feed	kst	1,582,526	54,900	54,750	54,750	54,750	54,900	54,750	54,750	54,750	54,900	54,750	54,750	54,750	54,900	54,750	54,750	21,101
MoS_2	*%*	*0.074%*	*0.101%*	*0.098%*	*0.093%*	*0.042%*	*0.050%*	*0.058%*	*0.071%*	*0.083%*	*0.086%*	*0.086%*	*0.090%*	*0.064%*	*0.074%*	*0.083%*	*0.086%*	*0.065%*
Cu	*%*	*0.105%*	*0.069%*	*0.068%*	*0.080%*	*0.152%*	*0.139%*	*0.127%*	*0.110%*	*0.096%*	*0.080%*	*0.076%*	*0.076%*	*0.102%*	*0.094%*	*0.076%*	*0.067%*	*0.059%*
Ag	*gpt (metric)*	*3.00*	*1.93*	*2.00*	*2.38*	*4.32*	*4.24*	*3.78*	*3.34*	*2.85*	*2.34*	*2.27*	*2.40*	*3.00*	*2.79*	*2.32*	*2.11*	*1.64*
Waste	kst	2,425,101	65,818	75,310	62,410	62,934	97,313	79,768	73,603	65,620	55,778	47,465	41,473	36,454	6,318	3,744	4,553	1,662
Strip Ratio (waste:sort feed)	*ratio*	*1.11*	*0.93*	*1.00*	*0.77*	*0.66*	*1.15*	*1.05*	*1.08*	*1.02*	*0.83*	*0.68*	*0.60*	*0.42*	*0.09*	*0.05*	*0.07*	*0.07*
Head Grade (% MoS_2 Eq. recoverable)	*%*	*0.123%*	*0.138%*	*0.134%*	*0.132%*	*0.103%*	*0.112%*	*0.114%*	*0.124%*	*0.131%*	*0.124%*	*0.129%*	*0.126%*	*0.110%*	*0.121%*	*0.122%*	*0.122%*	*0.096%*
Recovered Mo	kst	646	31	30	28	12	16	17	21	25	25	27	27	19	23	25	26	8
Recovered Cu	kst	1,263	26	26	30	62	57	55	49	41	35	34	30	43	42	32	27	9
Recovered Ag	koz	107,239	2,074	2,247	2,872	5,698	5,587	4,889	4,166	3,482	2,818	2,560	2,608	3,728	3,416	2,761	2,303	712
MoEq lbs	*mmlbs*	*1,908.2*	*74.7*	*72.1*	*70.9*	*55.5*	*60.3*	*61.5*	*66.7*	*70.7*	*67.2*	*69.7*	*68.0*	*59.3*	*65.4*	*66.0*	*65.5*	*20.0*

Notes: By-product production of rhenium and sulfuric acid is not shown here, but is included in economic analysis
MoEq lbs = (Revenue from recovered, payable metal plus by-products before deduction of Royalty TCRC and freight)/(Price of Mo per lb)



Source: SRK, 2019

Figure 22-2: Metal production schedule graph

Note that by-products rhenium and sulfuric acid are included in revenue calculations but physicals are not reported in this graph-set.

22.6 Pricing Assumptions

Flat non-escalated prices were assumed for the life of the project. Table 22-5 shows the price assumptions used.

Table 22-5: Pricing assumptions for economic analysis

Commodity	Units	Price
Molybdenum metal	$/lb	$15.00
Copper	$/lb	$3.00
Silver	$/oz	$17.50
Rhenium	$/lb	$1,750.00
Sulfuric Acid	$/t	$50.00

22.7 Processing Recovery Assumptions

The estimated processing recoveries were applied to the grades of material delivered to the mill from the different mineralized zones (per Table 14-13). Note that the material has already been upgraded by mineral sorting and particle recovery at this stage and these numbers reflect only recovery of upgraded material.

Table 22-6: Processing recovery assumptions used for economic analysis

Molybdenum Recovery	Copper Recovery	Silver Recovery	Rhenium Recovery	Sulfuric Acid Recovery
91.6%	76.1%	70.7%	90%	95%

Note: Rhenium and sulfuric acid recoveries are based on existing plant operation data at Molymet in Chile and Mexico, Jiangxi Copper in China and Sino Platinum Metals in China – all with actual recoveries higher than those used in the report.

22.8 Capital Costs

Capital costs used for the evaluation are summarized in Table 22-7. Additional detail regarding the estimation of the capital costs is contained in Section 21. Note that the capital costs presented do not include any costs prior to construction commencement. Please refer to Section 26 for an estimate of the pre-feasibility study work program and costs.

Table 22-7: Capital cost summary

Capital Costs	($M)
Mine – Equipment, etc.	344
Capitalized Mine Operating Costs	330
Sort Plants	160
Mill	1,293
Roaster	208
Tailings	22
Infrastructure	76
Total Initial Capital Directs	**2,433**
Contingency on Initial Capital Directs (excl Mining)	**176**
Indirects	
Mine	15
Plant (incl. Sort)	354
Roaster	81
Infrastructure	14
Total Initial Capital Indirects	**464**
Sustaining Capital	
Mine	428
Sort Plants	42
Mill	349
Roaster	56
Tailings	84
Infrastructure	10
Total Sustaining Capital	**970**
Closure and Reclamation	150
Total Capital Costs	**4,193**
Initial Capex	**3,071**
Sustaining and Expansion Capex	**972**
Closure	**150**

22.9 Operating Costs

Operating costs (Opex) are summarized in Table 22-8. The capitalized Opex is pre-stripping, which has been re-allocated and included in the mining capital costs shown in Table 22-8. The unit costs are expressed as total operating costs (before re-allocation) divided by total tonnage.

Table 22-8: Operating costs summary

Operating Costs	LOM ($M)	Unit Rates ($/t)	Unit Rates ($/lb Mo.Eq.)
Mining	5,797	$3.66	$2.99
Bulk Sort	778	$0.49	$0.40
Middling Sort	192	$0.12	$0.10
Processing	7,042	$4.45	$3.63
Sort Waste Delivery	395	$0.25	$0.20
G&A	805	$0.51	$0.42
Less Capitalized Operating Costs	-329	-$0.21	-$0.17
Total Operating Costs	**14,680**	**$9.28**	**$7.57**

Note: MoEq lbs = (Revenue from recovered, payable metal plus by-products before deduction of Royalty TCRC and freight)/(Price of Mo per lb)

The operating cost net of by-product credits (i.e. net revenue from by-products deducted from total opex) is estimated at $4.67 per pound of molybdenum produced, based on the price assumptions for by-products shown in Table 22-5.

22.10 Royalties

No royalties were applied to project for economic analysis.

22.11 Taxation

Corporate taxation in the United States is extremely complex. For this study, the taxation was modeled in a highly simplified manner, as is appropriate for a PEA level of study. Depreciation was also modeled in a simplified fashion, suitable for a PEA evaluation. The project valuation is relatively insensitive to variations in depreciation treatment. A total tax rate of approximately 22% was modeled.

22.12 Off-Site Costs

Off-site costs (concentrate freight, port handling, treatment charges and refining charges) were deducted from payable revenue. The basis for the charges is summarized in Section 19.

22.13 Sensitivity Analysis

The project as currently characterized returns a positive NPV at an 8% discount rate. This indicated the potential of the deposit to support an economic project (note cautionary statements in Section 22.1).

Table 22-9 to Table 22-12 summarize the sensitivity of the project NPV ($B at 8% discount rate) to variations in key input assumptions across a change of +/-20%.

Mineral resources are not reserves and do not have demonstrated economic viability.

Table 22-9: Two-factor sensitivity (NPV(8%) in $M) – Capex and Opex

Post Tax NPV		Opex						
		-30%	-20%	-10%	0%	10%	20%	30%
Capital	-30%	$2,646.6	$2,287.5	$1,926.6	$1,565.2	$1,204.5	$844.3	$478.2
	-15%	$2,254.1	$1,892.8	$1,531.4	$1,169.2	$804.0	$436.8	$67.5
	0%	$1,858.9	$1,497.1	$1,132.7	$764.6	$395.4	$25.4	($346.7)
	15%	$1,461.1	$1,095.1	$726.1	**$355.8**	($16.7)	($389.1)	($764.6)
	30%	$1,057.6	$687.7	$316.9	($56.5)	($431.4)	($806.4)	($1,187.8)
	45%	$649.2	$278.0	($95.4)	($471.4)	($848.6)	($1,228.1)	($1,615.8)
	60%	$239.1	($134.3)	($510.3)	($888.2)	($1,269.0)	($1,654.8)	($2,045.8)

Table 22-10: Two-factor sensitivity (NPV(8%) in $M) – Capex and metal prices

Post Tax NPV		Price (all metals)						
		-30%	-20%	-10%	0%	10%	20%	30%
Capital	-30%	($506.3)	$195.9	$885.6	$1,565.2	$2,245.7	$2,923.3	$3,601.5
	-15%	($933.7)	($217.5)	$479.0	$1,169.2	$1,850.5	$2,531.8	$3,211.5
	0%	($1,365.2)	($634.8)	$68.1	$764.6	$1,454.6	$2,136.6	$2,818.7
	15%	($1,803.9)	($1,058.4)	($346.1)	**$355.8**	$1,051.9	$1,740.7	$2,423.5
	30%	($2,247.3)	($1,486.9)	($763.2)	($56.5)	$643.6	$1,338.9	$2,027.7
	45%	($2,693.1)	($1,917.8)	($1,184.3)	($471.4)	$233.3	$932.1	$1,626.7
	60%	($3,141.5)	($2,352.4)	($1,610.5)	($888.2)	($179.1)	$523.5	$1,221.4

Table 22-11: Two-factor sensitivity (NPV(8%) in $M) – Opex and metal prices

Post Tax NPV		Price (all metals)						
		-30%	-20%	-10%	0%	10%	20%	30%
Opex	-30%	($630.4)	$74.4	$771.0	$1,461.1	$2,144.2	$2,825.4	$3,507.6
	-20%	($1,012.4)	($300.4)	$401.0	$1,095.1	$1,782.5	$2,464.1	$3,146.2
	-10%	($1,402.0)	($676.9)	$27.9	$726.1	$1,418.5	$2,102.7	$2,784.9
	0%	($1,803.9)	($1,058.4)	($346.1)	**$355.8**	$1,051.9	$1,740.7	$2,423.5
	10%	($2,223.5)	($1,448.8)	($721.0)	($16.7)	$682.1	$1,376.5	$2,062.1
	20%	($2,649.3)	($1,851.4)	($1,103.6)	($389.1)	$311.7	$1,009.0	$1,699.8
	30%	($3,062.8)	($2,267.6)	($1,494.6)	($764.6)	($58.5)	$639.2	$1,334.9

Table 22-12: Sensitivity (NPV(8%) in $M) – Individual metal prices

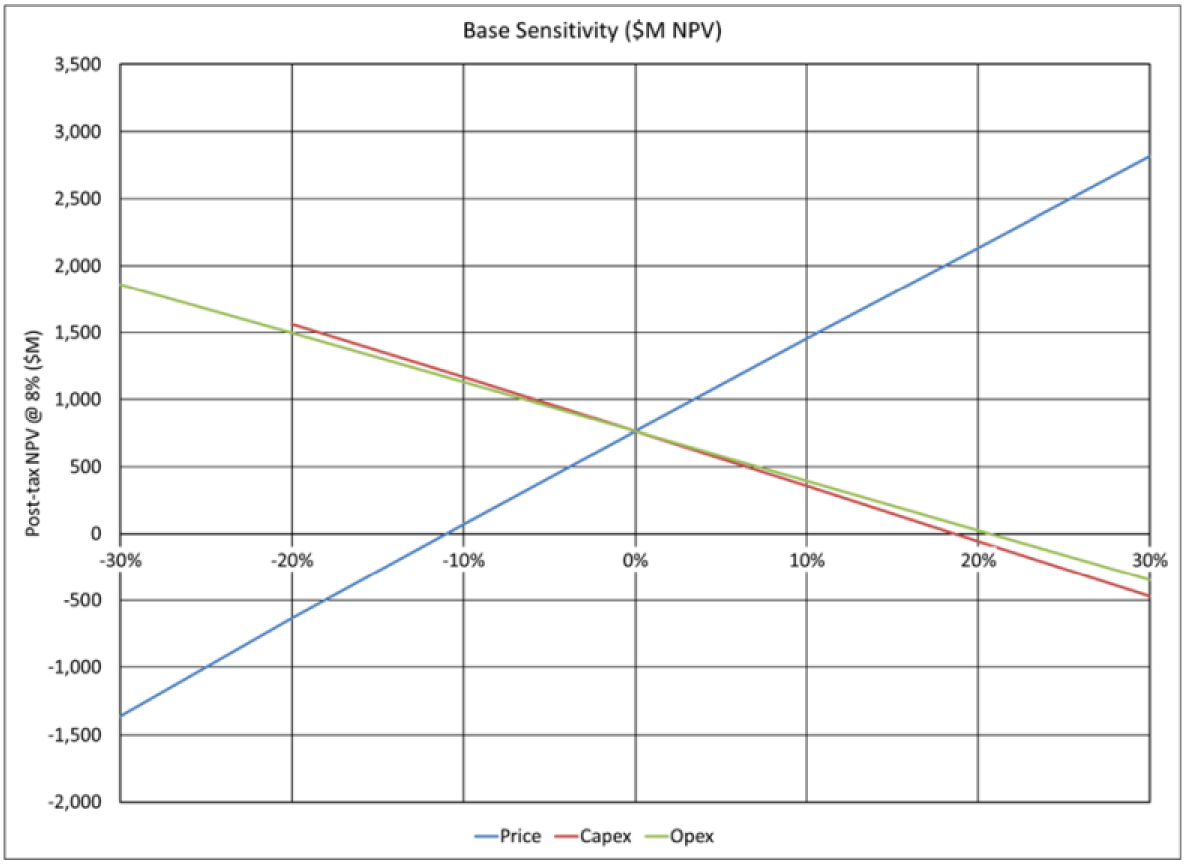
Post Tax NPV	Metal Prices						
	-30%	-20%	-10%	0%	10%	20%	30%
Molybdenum Price	$10.50	$12.00	$13.50	$15.00	$16.50	$18.00	$19.50
Post-tax NPV ($M)	-$1,008	-$548	-$96	**$356**	$804	$1,251	$1,694
Copper Price	**$2.10**	**$2.40**	**$2.70**	**$3.00**	**$3.30**	**$3.60**	**$3.90**
Post-tax NPV ($M)	**-$228**	**-$33**	**$161**	$356	**$549**	**$741**	**$933**
Silver Price	**$12.25**	**$14.00**	**$15.75**	**$17.50**	**$19.25**	**$21.00**	**$22.75**
Post-tax NPV ($M)	**$220**	**$265**	**$311**	$356	**$401**	**$446**	**$490**

Figure 22-4 shows how the project NPV varies as individual commodity prices are varied across a range of +/-30%. Molybdenum, being the main source of revenue, demonstrates greater sensitivity.

Source: SRK, 2019

Figure 22-3: Metals price sensitivity – net present value

Figure 22-4 shows how the project NPV varies as price and operating costs are varied across a range of +/-30%. Capital costs are varied across a range of -20% to 40%. As is common to all minerals industry projects, commodity price is a highly significant driver of value.



Source: SRK 2019

Figure 22-4: Single factor sensitivity – net present value

23 Adjacent Properties

There are no adjacent properties applicable to the CuMo project for disclosure in this report

24 Other Relevant Data and Information

There is no other relevant data available about the CuMo project.

25 Interpretations and Conclusions

25.1 Conclusions

25.1.1 Mineral Resource

The CuMo project hosts a **measured** mineral resource, at a $5.00/t RCV cut-off of 0.3 billion tons at grades of 0.081% MoS_2, 0.076% Cu, 2.09 ppm Ag, and 0.030 ppm Re.

The CuMo project also hosts an **indicated** mineral resource, at a $5.00/t RCV cut-off of 1.97 billion tons at grades of 0.053% MoS_2, 0.085% Cu, 2.57 ppm Ag, and 0.019 ppm Re.

There is a further **inferred** resource of 2.56 billion tons at grades of 0.048% MoS_2, 0.067% Cu, 2.13 ppm Ag, 0.017 ppm Re.

It is noted that the convention for the CuMo project is to assay for elemental molybdenum to report %Mo, but then this is multiplied by 1.6681 to calculate %MoS_2 in resource estimation and mine planning. Thus, the molybdenum grades for %MoS_2 are 1.6681 times that for %Mo.

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

25.1.2 Mining

The CuMo project is to be developed as a large-scale open pit operation, leveraging economies of scale in large mining equipment and optimization of truck hauls to reduce operating costs. It should take full advantage of emerging autonomous machine operation to further improve costs. In this PEA, the author has assumed autonomous operation of both the truck and drill fleets.

The author conducted limited investigation into mass material movement out of the pit (such as Doppelmayr RailCon and Dos Santos sandwich-belt high-angle conveyors). While promising, trade-off studies and further evaluations are required for inclusion in the project development strategy.

25.1.3 Bulk Sorting

The author investigated the application of bulk sorting to the CuMo project and found it an appropriate technology for the mineralization at CuMo. There is sufficient heterogeneity at sub-bench scale (i.e. at the 10 ft interval of exploration hole sampling) to warrant the consideration of bulk sorting.

Current bulk sorting requires consideration of batches of conveyed material, up to 30 seconds, for discretization. To improve sorting at smaller scales, a multi-stage bulk sorting plant has been conceptualized, which provides for three stages of splitting and sorting of the sort feed to achieve adequate segregation of waste, mill feed and middlings material.

25.1.4 Particle Sorting

The author reviewed particle sorting analysis on 400 quarter-core samples across the different CuMo mineralized zones. This demonstrated heterogeneity which would make particle sorting attractive, but not at the scale envisioned for the CuMo project. However, with bulk sorting providing reduced volumes for particle sort feed (i.e. the middlings stream), particle sorting becomes more viable.

25.1.5 Project Economics

The project as currently characterized returns a positive NPV at an 8% discount rate. This indicated the potential of the deposit to support an economic project.

The PEA described herein is preliminary in nature and is partly based on inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary assessment based on these mineral resources will be realized.

25.2 Project Risks

25.2.1 Mineral Resource

The mineral resource is supported by exploration results, test-work and modelling. As with any mineral resource estimate there is uncertainty inherent in the estimation process. There is a risk that the grades and metallurgical recoveries may be lower than currently modelled. There is also a risk that the interpretation of the results is inaccurate and that less mineralized material is present than is currently modelled.

Additional exploration and test-work will reduce this risk as the project is advanced.

25.2.2 Mining

The mining concepts for CuMo are largely proven. The adoption of autonomous equipment does possess some risk in that federal and local regulators may require extensive efforts by proponents to ensure the safety of their operations.

The CuMo open pit is envisioned to be a large, deep pit (up to 3,500 ft deep). With this comes the potential geotechnical risk for wall failures. While the author has assumed a relatively flat overall wall angle for the PEA (37°), there may be risks associated with yet unknown rock mass or structural geology conditions that may require consideration of even flatter slopes in places.

25.2.3 Mineral Sorting

The technology envisioned in this PEA for bulk sorting, PGNAA, has had limited application to molybdenum-copper deposits. While demonstrated for some low-grade copper-molybdenum deposits, testing is required to verify that molybdenum is measurable at the specific grades envisioned for CuMo.

Additional testing is required to obtain the final results expected from both bulk and particle sorting.

25.2.4 Processing

There is a risk that achieved recoveries could be lower than estimated, that throughputs will not be achieved and that costs may be higher than modelled. The process recovery, throughput and cost estimates will be refined as part of the pre-feasibility study.

This project also includes the installation of a large quantity of particle sorting machines which, to the knowledge of the author, is beyond the scale of any currently operating concentrators.

25.2.5 Project Infrastructure

The planned mine will be a green-fields site and requires construction of mine and process-related infrastructure including the TSF. Access roads in and around the project site will be required. There is a risk that the designs, costs and implementation timelines for the provision of this infrastructure may not be as anticipated, increasing costs and schedule.

25.2.6 Permitting

At this time, no issues were identified that would materially impact the ability to eventually extract mineral resources at the project. There is a risk that the mining plan of operations would identify and characterize issues that may lengthen the timeline and increase the costs of permitting the project. Note that the PEA described in this report does not quantify the timeline and costs for the pre-construction and permitting activities.

Previous environmental analyses have identified the presence of a rare plant Sacajawea's bitterroot , and potential habitat for Endangered Species Act wildlife, and USFS sensitive species. These potential issues will need to be analyzed and disclosed in NEPA documents and potentially mitigated.

The mine will be located in an area used for weekend summer dispersed recreation and fall big-game hunting and is well-known in the Boise area. Organized environmental groups such as the Idaho Conservation League and Sierra Club are keeping their constituents informed so as to coordinate opposition to the project. As such, well-funded, organized opposition to mining activities should be anticipated.

25.2.7 Economic Risks

Project Strategy Risk

Overall, the author considers that the likelihood of a major revision to project strategy emerging from the pre-feasibility study to be moderate. Mineral sorting as contemplated in this study is not a mature technology, and there is a risk that the assumptions used may not prove accurate. Elimination of the mineral sorting pre-process from the strategy has the potential to reduce the economic proposition of the project.

Commodity Price Risk

There is a risk that commodity prices may not be consistent with assumptions made in this study. In particular, molybdenum, which contributes the majority of project value is historically subject to significant price volatility.

Capital Cost Risk

There is a risk that the capital required to build and operate the project may be higher than that forecast in this study. The author recommends that the precision of the estimates be refined at pre-feasibility study and feasibility study before commitment to project construction is made.

Operating Cost Risk

There is a risk that the operating costs incurred to operate the project may be higher than that forecast in this study. The author notes that variability in the operating cost drivers (productivity, input costs and labor costs) over time is expected. The analysis assumes constant conditions but is best thought of as reflecting an expectation of average costs. The authors recommend that the precision of the estimates be refined at pre-feasibility and feasibility study stages prior to commitment to project construction.

Schedule Risk

There is a risk that the schedule to build the project may vary from that assumed in the study. This is an asymmetrical risk, with significantly more downside scope than upside. This risk is exacerbated by the seasonality of the location, with somewhat difficult construction conditions occurring in some winter months. Smalls delays have the potential to be more significant than might otherwise be the case if they push critical path activities into winter months, thereby incurring a much longer delay.

Process Recovery Risk

There is a risk that achieved recoveries could be lower than estimated, reducing the revenue and economic returns of the project. The process recovery estimates will be refined as part of the pre-feasibility study and feasibility study.

Permitting and Pre-construction Schedule Risk

This was not explicitly considered for the purposes of this study in the economic analysis as the analysis is conducted only from the commencement of construction. Nevertheless, the risk of longer-than-anticipated permitting timeline will reduce the project value is considered from “today” forward.

25.3 Project Opportunities

25.3.1 Mineral Resource

The exploration drilling and thus mineral resource model for CuMo is constrained on the western extents of the deposit. There is opportunity with increased exploration to expand the resource to the west, thus offering either more process feed within the current envisioned open pit or increasing the size of the open pit to the west. This expansion can be done with only minimal effects on the location of the mill, sort plant or crusher.

25.3.2 Mining

With increased knowledge of the rock mass and structural geology, through additional geotechnical field programs and investigation, there is potential to steepen the wall angles for CuMo.

Further consideration of high angle conveying solutions in combination with semi-mobile crushing and conveying (IPCC) concepts could highlight opportunities for cost savings at CuMo. Applying IPCC to sort feed, which needs to be crushed either way and is up to 50% of the mined material, poses the greatest opportunity.

25.3.3 Mineral Sorting

The bulk sorting analysis was conducted on drill core that was sampled on a standard 10 ft interval. Thus, heterogeneity could only be assessed down to this scale. With multiple stage sorting and splitting, smaller size packets of material could be measured. As heterogeneity increases with reduced scale, there is potential that better segregation of waste, mill feed and middlings is possible.

The opportunity would be for increased waste rejection and ultimately reduced middlings fractions to improve the economics of the project.

Ultimately, the potential for exploitation of the heterogeneity of the deposit may not be firmly quantified by way of studies conducted on exploration-level data. Much higher-resolution sampling and sorting may be possible at an operational scale. This has the potential to enhance project economics, but the quantum of that improvement is difficult to quantify.

The field of mineral-sorting is the subject of significant research and development. There exists an opportunity for this project to exploit improvements in technology.

25.3.4 Processing

Additional metallurgical work to determine optimum grind size (the current assessment is based on the finest grind tested to date), analyze recoveries of the various metals, and analyze the effects of the higher grade coming from the mineral sorters on metal recoveries. This has the potential to improve project economics.

Optimization of reagents to reduce costs and improve metallurgical recoveries has the potential to improve recoveries.

Further testing is required to determine if there may be opportunity to economically recover tungsten from the mineralized material.

25.3.5 Project Infrastructure

Further studies may allow for optimization of infrastructure design, costing and schedule. Whilst optimization is worth pursuing, the author views modification to the infrastructure concepts to be unlikely to materially affect the economic proposition at a strategic level for the project.

25.3.6 Economic Opportunities

Real Option Value

In the case of a large, long-life open-pit mine such as is contemplated for the CuMo project, there exists significant optionality that can be leveraged to improve project cashflows and values. The simple sensitivity analysis conducted in Section 22.13 assumes a constant operating strategy, even as assumptions are varied. In practice, management has the option to alter strategy in response to those variations. Downsides can be mitigated, and upsides can be leveraged for greater returns.

It is also expected that the mine would be run using a dynamic cut-off policy where sorting strategies and cut-offs, mill-feed cut-offs, stockpiling strategies and mining rates will all be varied in real time to maximize returns as prices and costs vary. The benefits of this strategy are not reflected in the central estimate approach to valuation summarized in this report.

Project Strategy Opportunity

While the probability of a major revision to project strategy can be considered moderate, careful consideration and revision of the strategic decisions should be a feature of studies going forward. In particular, effort should be made to enhance the optionality of the project, particularly where this is low cost.

Commodity Price Opportunity

There is a risk that commodity prices may not be consistent with assumptions made in this study. Higher prices, both realized and forecast would lead to re-optimization of the mine and processing plans with a potential to create additional value beyond that shown by the sensitivity analysis summarized in Section 22.11.

Capital Cost Opportunity

Opportunities to reduce or defer capital expenditure may be realized in future studies. Care should be taken when considering the relationship between lower capital opportunities and technical risk to the project.

Operating Cost Opportunity

Operating costs may be lower than forecast for the purposes of this study. Lower costs should feed into both strategic and short-term mine planning, to allow optimization of stockpiling, sorting and mill feed strategies.

Schedule Opportunity

This risk is highly asymmetric. The authors consider that the opportunity to execute a significantly shorter construction program is low. The authors caution that optimized schedules with multiple critical or near-critical path activities will contain additional embedded risks.

Process Recovery Opportunity

Further metallurgical test-work will allow for optimization of the process flow sheet and plant design in the pre-feasibility and feasibility studies. Better than planned recoveries are possible.

Pit Slope Angle Opportunity

This is not considered to be a significant opportunity from an economic perspective. Strip ratios are relatively low, and incremental change in waste-movement volumes do not impact the overall project economics significantly.

26 Recommendations

26.1 Mineral Resources

Exploration work consisting mainly of drilling is required to reach pre-feasibility. It is estimated that a total of 33 additional holes for 71,000 ft plus an additional five geotechnical holes for 12,000 ft on the deposit plus additional 74,800 ft allocated to condemnation drilling of waste dump, mill site and tailings pond areas, making a total of 157,800 ft of drilling budgeted. This drilling is broken into the following categories.

- In-fill drilling
- Delineation drilling
- Orientated geotechnical drilling – requires oriented core recovery system
- Drilling for metallurgical sample – large diameter hole (PQ size) recommended
- Condemnation drilling waste dump, mill and tailings site

The shortest time to complete this work will be two seasons using four drill rigs each season.

26.2 Pit Geotechnical

The author provides these recommendations for the next steps of geotechnical assessment:

- Geotechnical database QA/QC assessment (to address the inconsistencies and potentially poor data observed in the existing data set)
  - Select a sub-set (~10%) of resource drill holes that give good spatial coverage of the proposed pit walls, and from multiple drilling campaigns
  - Undertake quantitative basic geotechnical logging using the full core photographs of these drill holes (TCR, SCR, RQD and FF/m)
  - FF/m vs RQD plots for both data-sets
  - Comparison of the values in the database with the photo-logged values
  - Assessment of differences in order to determine whether variance is systematic or random, and consequently decide on the respective approach to address e.g. apply correction factor, re-logging more of the drill holes
  - Qualitative assessment of the rock susceptibility to deterioration by comparing core in the photos (fresh), to the current condition of the stored core (aged)
- Major structures assessment
  - Log the photos of the core for major structures
  - Develop conceptual integrated litho-structural 3-D model
- Geotechnical-specific diamond-cored drill holes targeted to provide coverage of the proposed interim and ultimate pit walls, and compatible with the pit depth

- Geotechnical logging to RMR_B89 system (historical logging to RMR_L90 which is typically for underground mine applications)
- Field (empirical and point load) and laboratory (uniaxial and triaxial compressive strength and direct shear) testing of fresh core to determine intact rock strength
- Calculate RMR values and conduct comparison with lithology, alteration and mineralogy zones of the 3-D geology model to establish broad geotechnical domains
- Establish pit sectors and domain-representative sections to conduct pit slope stability analyses and select pit design angles

26.3 Mining

The author recommends further study of the application of high angle conveying of sort feed at CuMo.

The author further recommends the continued consideration of autonomous haulage for CuMo, with commensurate refinement of performance parameters and costs.

26.4 Mineral Sorting

The author recommends that CuMoCo engage with bulk and particle sorting technology providers to advance testing of penetrative technologies (e.g. PGNAA) and other mineral sensing techniques for the measurement of molybdenum in lower grade applications.

26.5 Processing

Metallurgical aspects to be studied were highlighted in the preliminary metallurgical analysis, some of which require larger samples to finalize the detailed flow sheet and determine how many cleaning stages will be required for flotation, as well as to confirm the total number and configuration of particle sorting units required. One important part of the analysis is a grinding versus recoverability study, as in the previous study only two grinding sizes were studied: coarse and fine. The fine grind proved to be more profitable despite the increase in costs. Further study with multiple grinding size options is required to determine an optimum grinding system.

Work will consist of collecting and analyzing a large, 2+ ton bulk sample to determine the optimum flow sheet for the deposit; and a variability study to analyze variations within the deposit. A total of 100 to 150 twenty-kilogram samples will be used for the variability study.

It is recommended that single stage SAG milling to 71 microns be evaluated as a possible way to reduce capital costs of the multiple grinding circuits. If a slightly coarser primary grind can be shown to be feasible, this will allow smaller grinding line modules to be selected. Production from a single line SAG plant can then be doubled with the addition of a ball mill, without changing the feed end storage or feed conveying equipment. This would lead to starting up the plant at much lower tonnage and create options to increase production as needed.

It is also recommended that the potential recovery of tungsten as an economic mineral be considered in future test-work planning.

26.6 Tailings Management

Engineering studies, including TSF design and potential water management and treatment design, including:

- Updating the TSF and Clear Creek waste facility designs based on field investigation results
- Developing tailings deposition plan and waste placement sequence to match pit development and mill output
- Detailed analysis of the water and load balance to predict the accumulation of mill reagents in the process water circuit from the tailings

26.7 Permitting

A mining plan of operations and reclamation cost estimate must be prepared to identify locations of the mine, waste rock dumps, roads (haul and access), power and water line corridors from the source to the point of use, mill, tailings storage facility, and other support facilities. Operating plans must be developed in conjunction with the mining plan of operations. ICMC should develop robust reclamation and closure plans for the facilities. ICMC should also begin acquiring any necessary water rights. Stakeholder outreach should continue.

Once the facility locations have been determined, ICMC should coordinate with state and federal agencies to identify the baseline studies that will need to be completed to support the development of an environmental impact statement and initiate those studies.

26.8 Plan and Budget for Additional Work

Table 26-1 sets out a summary of work expected to be completed prior to final permitting being completed. The estimated time frame for this work program is three years.

Table 26-1: Budget for additional work

Item	Additional Information	Budget (000s $)
Diamond Drilling		
Delineation, infill, metallurgy	48,097 m (157,800 ft) @ $100/ft	15,780
Road Construction	2 km @ $50,000/km	100
Sample Preparation and Analysis	8,800 @ $60 each	528
Metallurgical Testing	Sample Collection, etc.	125
	Batch Round of Testing	1,000
	Variability Test-work	1,200
Land Acquisition and Staking Costs		8,000
Environmental Studies	Environmental Assessment	713
	Baseline Studies Startup	12,500
	Environmental Plan of Operations	800
	Environmental Impact Statement	23,500
	Permitting	3,000
Engineering Studies Scoping	Mill Site, Tailings Site Analysis	550
	Intergoverment Task Force Creation	500
	Mining Plan of Operations	1,200
	Pre-feasibility Study	5,500
Mobilization-Demobilization		427
Road Maintenance		325
Supervision and Project Management	Supervision	225
	Corporate Manager	360
	Project Manager	240
	Assistant Geologist(2)	364
	Technicians (12)	1,174
Vehicles	5 Vehicles	150
Accommodation and Food	30 Personnel	760
Travel		42
Project Office and Warehouse		1,225
Land Filing Fees	Current BLM: $155/claim/year	87
Land Filing Fees	Projected Additional Filing Fees	256
Consultants	(Mining, Metallurgical and Marketing)	575
Resource Modeling		1,650
Public Relations and Project	Public Relations and Legal, etc.	2,550
Presentation	Liaison County and State Officials	1,250
Subtotal		**86,655**
Contingency		13,345
Total		**100,000**

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28 Signature Page and Certificates

This technical report was written by the following “Qualified Persons”. The effective date of this technical report is 11 March 2020.

Original signed

Gilles Arseneau, P.Geo.
SRK Consulting (Canada) Inc.

Original signed

Neil Winkelmann, FAusIMM
SRK Consulting (Canada) Inc.

Original signed

Bob McCarthy, P.Eng.
SRK Consulting (Canada) Inc.

Original signed

Valerie Sawyer, SME
SRK Consulting (USA), Inc.

Original signed

Andy Thomas, P.Eng.
SRK Consulting (Canada) Inc.

Original signed

Gary Giroux, P.Eng.
Giroux Consultants Ltd.

Original signed

Calvin Boese, P.Eng.
SRK Consulting (Canada) Inc.

Original signed

John Starkey, P.Eng.
Sacré-Davey Engineering

Reviewed by

Original signed

Bob McCarthy, P.Eng.

Certificates of QPs are included on the following pages.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "Preliminary Economic Assessment & NI 43-101 Technical Report for the CuMo Project, USA", Boise County, Idaho", prepared for American CuMo Mining Corp. with an effective date March 11, 2020 (the "Technical Report").

I, Andy Thomas do hereby certify that:

1. I am a Senior Geotechnical Engineer with SRK Consulting (Canada) with an office at 22nd Floor, 1066 West Hastings Street, Vancouver, BC, V6E 3X2, Canada.
2. I am a graduate of the University of The University of Adelaide in 2004 where I obtained a Bachelor of Engineering (Civil & Environmental) and a Bachelor of Science (Geology). I am also a graduate of The University of British Columbia in 2014 where I obtained a Master of Engineering (Geological). Aside from the time spent studying at The University of British Columbia, I have practiced my profession continuously since 2005. My relevant experience includes geotechnical and hydrogeological investigations and geotechnical design of open pits in Australia, North America and South America.
3. I am a Professional Engineer registered with the Engineers and Geoscientists British Columbia, license #44961.
4. I have visited the property on 30 and 31 October 2018.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I accept professional responsibility for sections 1.12.2, 16.2.1, and 26.2 of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 25th day of May 2020 in Vancouver, B.C., Canada.

"original signed and sealed"

Andy Thomas, P.Eng.
Senior Consultant
SRK Consulting (Canada)

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: "Preliminary Economic Assessment & NI 43-101 Technical Report for the CuMo Project, USA", Boise County, Idaho", prepared for American CuMo Mining Corp. with an effective date March 11, 2020, (the "Technical Report").

I, John Starkey, P. Eng. do hereby certify that:

1. I am the Principal Consulting Engineer with Starkey & Associates Inc., with an office at Suite 212, 151 Randall Street, Oakville, ON, L6J 1P5, Canada. Starting on Jan 6th, 2020, I was sub-contracted by Sacre-Davey Engineering (SDE) 315 Mountain Highway, North Vancouver, BC, to review and sign off as QP for the relevant mineral processing sections in the above-mentioned report as listed below.
2. I am a graduate of the University of Toronto in 1961 with a B.A.Sc. in Mining Engineering. I have practiced my profession continuously since 1961. I have 59 years of relevant experience in mining and mineral processing operations and process plant engineering design. I have worked on a variety of projects for recovering molybdenum, copper, nickel, gold, silver, lead, zinc, tin and iron deposits throughout the world. My more recent work includes feasibility level studies for: Barrick (Pueblo Viejo, 2019), Xstrata, (Alumbrera, Antapaccay, Las Bambas, and Freida River, 2008). Between 1989 and the present, I worked independently on many projects for Applied Ore Testing Inc., Minnovex, Davy, Aker Kvaerner, SRK, and SDE, at the same time developing the SPI and SAGDesign tests to accurately measure ore hardness, and properly design grinding mills for over 250 projects.
3. I have not visited the subject property, nor have I previously worked on the CuMo Project.
4. I am a Professional Engineer registered with Professional Engineers Ontario, license No. 44133015, and I am a Designated Consulting Engineer in Ontario. I have been a member of CIM since 1981.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the issuer as defined in Section 1.5 of NI 43-101.
7. I accept professional responsibility for sections 1.10.4, 1.11.4, 1.12.5, 13.1, 13.2.1, 17.1, 17.3, 17.4, 17.5, 17.6, 17.7, 17.8, 17.9, 21.1.2 (Processing), 21.1.4, 21.2.3, 25.1.4, 25.2.4, 25.3.4, and 26.5 of the Technical Report.
8. As of the effective date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible, not misleading.
9. I have read NI 43-101 and Form 43-101F1 and confirm that this Technical Report has been prepared in compliance therewith.

Dated this 25th day of May 2020 in Oakville, Ontario, Canada.

"*original signed and sealed*"

John Starkey, P.Eng., BASc., FCIM
Principal Consulting Engineer
Starkey & Associates Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: “Preliminary Economic Assessment & NI 43-101 Technical Report for the CuMo Project, USA”, Boise County, Idaho”, prepared for American CuMo Mining Corp. with an effective date March 11, 2020 (the “Technical Report”).

I, Calvin Boese, P.Eng, do hereby certify that:

1. I am a Senior Consultant - Mining with SRK Consulting (Canada) Inc. with an office Suite 600, 350 3rd Ave North, Saskatoon, SK, S7K 6G7, Canada;
2. I am a graduate of the University of Saskatchewan with a B.Sc. in Civil Engineering (1999) and a M.Sc. in Geo-Environmental Engineering (2004). I have worked as a Geotechnical Engineer for 19 years. Most of my professional practice has focused on the geotechnical aspects of mining, including the site selection, design, permitting, operation and closure of mine waste facilities in Canada, the US, Indonesia and Turkey;
3. I am a Professional Engineer (P.Eng. #29478) registered with the Association of Professional Engineers, Geologists of British Columbia. I am also a registered Professional Engineer in Alberta and Saskatchewan;
4. I have visited the property during October 30, 2018;
5. I have read the definition of “qualified person” set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101;
6. As a qualified person, I am independent of the issuer as defined in Section 1.5 of NI 43-101;
7. I am the co-author of the Technical Report, responsible for sections 1.12.6, 18.6, 21.1.3, 26.6 and accept professional responsibility for those sections of this technical report;
8. I have had no prior involvement with the subject property;
9. I have read NI 43-101 and Form 43-101F1 and confirm that this Technical Report has been prepared in compliance therewith;
10. As of the effective date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading;

Dated this 25th day of May 2020 in Vancouver, British Columbia, Canada.

“original signed and sealed”

Calvin Boese, P.Eng., M.Sc.
Senior Consultant
SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: "Preliminary Economic Assessment & NI 43-101 Technical Report for the CuMo Project, USA", Boise County, Idaho", prepared for American CuMo Mining Corp. with an effective date March 11, 2020 (the "Technical Report").

I, G. H. Giroux, of 982 Broadview Drive, North Vancouver, British Columbia, do hereby certify that:

1. I am a consulting geological engineer with an office at 982 Broadview Drive, North Vancouver, British Columbia.

2. I am a graduate of the University of British Columbia in 1970 with a B.A. Sc. and in 1984 with a M.A. Sc. both in Geological Engineering.

3. I have practiced my profession continuously since 1970. I have had over 40 years of experience estimating mineral resources. I have previously completed resource estimations on a wide variety of molybdenum deposits including the Ajax, Redbird, Davidson, Sphinx and Chu Deposits.

4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.

6. I accept professional responsibility for section 1.5, 1.10.1, 1.11.1, 1.12.1, 10.6, 11, 12, 14, 25.1.1, 25.2.1, 25.3.1, 26.1, and Appendices 2, 3 and 4 on data verification and resource estimations completed in Vancouver during 2015. I have visited the property on June 1-3, 2015.

7. I have previously completed a statistical review of this property in 2005 and a resource estimation in 2008.

8. As of the date of this certificate, 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.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 25th day of May 2020 in Vancouver, British Columbia.

"original signed and sealed"

GIROUX CONSULTANTS LTD.
G. H. Giroux, P.Eng. MASc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: “Preliminary Economic Assessment & NI 43-101 Technical Report for the CuMo Project, USA”, Boise County, Idaho”, prepared for American CuMo Mining Corp. with an effective date March 11, 2020 (the “Technical Report”).

I, Dr. Gilles Arseneau, P. Geo., residing in North Vancouver, B.C. do hereby certify that:

1. I am an Associate Consultant with the firm of SRK Consulting (Canada) Inc. (“SRK”) with an office at Suite 2200-1066 West Hastings Street, Vancouver, BC, Canada.
2. I graduated with a B.Sc. in Geology from the University of New Brunswick in 1979; an M.Sc. in Geology from the University of Western Ontario in 1984 and a Ph.D. in Geology from the Colorado School of Mines in 1995. I have practiced my profession continuously since 1995. I have worked in exploration in North and South America and have extensive experience with Archean gold deposits and porphyry hosted precious metal mineralization.
3. I am a Professional Geoscientist registered with the Association of Professional Engineers and Geoscientists of British Columbia, registration number 23474.
4. I have not visited the property.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
6. I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
7. I am the co-author of this report and responsible for sections 1.3, 1.4, 6, 7, 8, 9, 10 and accept professional responsibility for those sections of this Technical Report.
8. I have had no prior involvement with the subject property.
9. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith.
10. As of the effective date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading;

Dated this 25th day of May 2020 in Vancouver, British Columbia, Canada.

“*original signed and sealed*”

Dr. Gilles Arseneau, P. Geo.
Associate Consultant

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: “Preliminary Economic Assessment & NI 43-101 Technical Report for the CuMo Project, USA”, Boise County, Idaho”, prepared for American CuMo Mining Corp. with an effective date March 11, 2020 (the “Technical Report”).

I, Neil M. Winkelmann, FAusIMM, do hereby certify that:

1. I am a Principal Consultant Mining with the SRK Consulting (Canada) Inc with an office at Suite 2200 – 1066 West Hastings Street, Vancouver, British Columbia, Canada;
2. I am a graduate of the University of New South Wales, Australia with a B.Eng. in Mining (1984). I am a graduate of the University of Oxford with an MBA in 2005. I have practiced my profession continuously since 1984 and I have 32 years’ experience in mining. I have significant experience in the valuation of minerals-industry projects accrued over the past 10 years;
3. I am registered as a Fellow of The Australasian Institute of Mining and Metallurgy, AusIMM #323673;
4. I have not visited the property;
5. I have read the definition of “qualified person” set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101;
6. As a qualified person, I am independent of the issuer as defined in Section 1.5 of NI 43-101;
7. I am the co-author of the Technical Report and accept professional responsibility for sections 1.9, 1.10.5, 1.10.7, 1.11.5, 1.11.6, 4.2, 4.3, 18.2, 18.3, 18.4, 18.5, 19, 21.1.2 (Infrastructure), 22, 25.1.5, 25.2.5, 25.2.7, 25.3.5, 25.3.6 and Appendix 1 of this Technical Report;
8. I have had no prior involvement with the subject property;
9. I have read NI 43-101 and Form 43-101F1 and confirm that this Technical Report has been prepared in compliance therewith;
10. As of the effective date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading;

Dated this 25th day of May 2020 in Vancouver, British Columbia, Canada.

“original signed and sealed”

Neil M. Winkelmann, FAusIMM
Principal Consultant (Mining)
SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: “Preliminary Economic Assessment & NI 43-101 Technical Report for the CuMo Project, USA”, Boise County, Idaho”, prepared for American CuMo Mining Corp. with an effective date March 11, 2020 (the “Technical Report”).

I, Robert McCarthy, P.Eng, do hereby certify that:

1. I am a Principal Consultant - Mining with SRK Consulting (Canada) Inc. with an office Suite 2200 – 1066 West Hastings Street, Vancouver, British Columbia, Canada;
2. I am a graduate of the University of British Columbia with a Bachelor in Applied Sciences degree in Mining and Mineral Process Engineering in 1984. I have practiced my profession for over 30 years. I have been directly involved in open pit mining operations and the design of open pit mining operations in Canada, Brazil, Peru, Mozambique, Russia, Argentina, and the United States;
3. I am a Professional Engineer registered with the Association of Professional Engineers & Geoscientists of British Columbia, license # 27309;
4. I have visited the property on 30 and 31 October 2018;
5. I have read the definition of “qualified person” set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101;
6. As a qualified person, I am independent of the issuer as defined in Section 1.5 of NI 43-101;
7. I am the co-author of the Technical Report, responsible for sections 1.1, 1.2, 1.6, 1.8, 1.10.2, 1.10.3, 1.11.2, 1.11.3, 1.12.3, 1.12.4, 1.12.8, 2, 3, 4.1, 5, 13.2.2, 15, 16 (except 16.2.1), 17.2, 18.1, 21.1.1, 21.2.1, 21.2.2, 21.2.4, 23, 24, 25.1.2, 25.1.3, 25.2.2, 25.2.3, 25.3.2, 25.3.3, 26.3, 26.4, 26.8, 27, and 28 of this Technical Report;
8. I have had no prior involvement with the subject property;
9. I have read NI 43-101 and Form 43-101F1 and confirm that this Technical Report has been prepared in compliance therewith;
10. As of the effective date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.

Dated this 25th day of May 2020 in Vancouver, British Columbia, Canada.

“original signed and sealed”

Robert McCarthy, P.Eng
Principal Consultant (Mining)
SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: "Preliminary Economic Assessment & NI 43-101 Technical Report for the CuMo Project, USA", Boise County, Idaho", prepared for American CuMo Mining Corp. with an effective date March 11, 2020 (the "Technical Report").

I, Valerie Jean Sawyer, SME do hereby certify that:

1. I am a Principal Consultant with SRK Consulting (U.S.), Inc. with an office at 1250 Lamoille Highway, Suite 520, Elko, Nevada 89801.
2. I am a graduate of the Michigan Technological University in 1981 where I obtained a Bachelor of Science in Metallurgical Engineering. I have practiced my profession continuously since 1981. My relevant experience includes over 35 years of experience in federal, state, and local mine environmental permitting and compliance and metallurgical engineering in the western United States.
3. I am a Registered Member with the Society for Mining, Metallurgy & Exploration, member number 4192564.
4. I have not visited the property.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
6. As a qualified person, I am independent of the issuer as defined in Section 1.5 of NI 43-101;
7. I am the co-author of the Technical Report and take professional responsibility for sections 1.7, 1.10.6, 1.12.7, 4.4, 4.5, 4.6, 4.7, 20, 25.2.6, and 26.7;
8. I have not had prior involvement with the property;
9. I have read NI 43-101 and Form 43-101F1 and confirm that this Technical Report has been prepared in compliance therewith;
10. As of the effective date of this certificate, to the best of my knowledge, information, and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.

Dated this 25th day of May 2020 in Elgin, Arizona, USA.

"*original signed and sealed*"

Valerie Jean Sawyer, SME
Principal Consultant, SRK Consulting (U.S.), Inc.

Appendix 1: Claims List

Unpatented CuMo Claim List 2018

Item	Claim Name/Number	BLM Serial No.	County Instrument Number	Loc Dt
1	CUMO #1	188031	201255	Mar-05
2	CUMO #2	188032	201256	Mar-05
3	CUMO #3	188033	201257	Mar-05
4	CUMO #4	188034	201258	Mar-05
5	CUMO #5	188035	201259	Mar-05
6	CUMO #6	188036	201260	Mar-05
7	CUMO #7	188037	201261	Mar-05
8	CUMO #8	188038	201262	Mar-05
9	NEW CUMO #9	187938	199561	Nov-04
10	NEW CUMO #10	187939	199562	Nov-04
11	NEW CUMO #11	187940	199563	Nov-04
12	NEW CUMO #12	187941	199564	Nov-04
13	NEW CUMO #13	187942	199565	Oct-04
14	NEW CUMO #14	187943	199566	Oct-04
15	NEW CUMO #15	187944	199567	Oct-04
16	NEW CUMO #16	187945	199568	Oct-04
17	NEW CUMO #17	187946	199569	Oct-04
18	NEW CUMO #18	187947	199570	Oct-04
19	NEW CUMO #19	187948	199571	Oct-04
20	NEW CUMO #20	187949	199572	Oct-04
21	NEW CUMO #21	187950	199573	Oct-04
22	NEW CUMO #22	187951	199574	Nov-04
23	NEW CUMO #23	187952	199774	Nov-04
24	NEW CUMO #24	187953	199775	Nov-04
25	NEW CUMO #25	187954	199575	Nov-04
26	NEW CUMO #26	187955	199576	Nov-04
27	NEW CUMO #27	187956	199577	Nov-04
28	NEW CUMO #28	187957	199578	Nov-04
29	NEW CUMO #29	187958	199579	Nov-04
30	NEW CUMO #30	187959	199580	Nov-04
31	NEW CUMO #31	187960	199581	Nov-04
32	NEW CUMO #32	187961	199582	Nov-04
33	NEW CUMO #33	187962	199583	Nov-04
34	NEW CUMO #34	187963	199584	Nov-04
35	NEW CUMO #35	187964	199585	Nov-04
36	NEW CUMO #36	187965	199586	Nov-04
37	NEW CUMO #37	187966	199587	Nov-04
38	NEW CUMO #38	187967	199588	Nov-04
39	NEW CUMO #39	187968	199589	Nov-04
40	NEW CUMO #40	187969	199590	Nov-04
41	NEW CUMO #41	187970	199591	Nov-04
42	NEW CUMO #42	187971	199592	Nov-04
43	NEW CUMO #43	187972	199593	Nov-04
44	NEW CUMO #44	187973	199594	Nov-04
45	NEW CUMO #45	187974	199595	Nov-04

Unpatented Cumo Claim List 2018 - Page 2

Item	Claim Name/Number	BLM Serial No.	County Instrument Number	Loc Dt
46	NEW CUMO #46	187975	199596	Nov-04
47	NEW CUMO #47	187976	199597	Nov-04
48	NEW CUMO #48	187977	199598	Nov-04
49	NEW CUMO #49	187978	199599	Nov-04
50	NEW CUMO #50	187979	199600	Nov-04
51	NEW CUMO #51	187980	199601	Nov-04
52	NEW CUMO #52	187981	199602	Nov-04
53	NEW CUMO #53	187982	199603	Nov-04
54	NEW CUMO #54	187983	199604	Nov-04
55	NEW CUMO #55	187984	199605	Nov-04
56	NEW CUMO #56	187985	199606	Nov-04
57	NEW CUMO #57	187986	199607	Nov-04
58	NEW CUMO #58	187987	199608	Nov-04
59	NEW CUMO #59	187988	199609	Nov-04
60	NEW CUMO #60	187989	199776	Nov-04
61	NEW CUMO #61	187990	199777	Nov-04
62	CUMO #62	188205	202147	May-05
63	CUMO #63	188206	202148	May-05
64	CUMO #64	188207	202149	May-05
65	CUMO #65 FRACT.	188208	202150	May-05
66	CUMO #66	188209	202151	May-05
67	CUMO #67 FRACTION	188210	202152	May-05
68	CUMO #68 FRACT.	188211	202153	May-05
69	CUMO #69 FR.	188212	202154	May-05
70	CUMO #70 FRACT.	188213	202155	May-05
71	CUMO #71	188214	202156	May-05
72	CUMO #72	188215	202157	May-05
73	CUMO #73	188216	202158	May-05
74	CUMO #74	188217	202159	May-05
75	CUMO #75	188218	202160	May-05
76	CUMO #76	188219	202161	May-05
77	CUMO #77	188220	202162	May-05
78	CUMO #78	188221	202163	May-05
79	CUMO #79	188222	202164	May-05
80	CUMO #80	188223	202165	May-05
81	CUMO #81	188224	202166	May-05
82	CUMO #82	188225	202167	May-05
83	CUMO #83	188226	202168	May-05
84	CUMO #84	188227	202169	May-05
85	CUMO #85	188228	202271	May-05
86	CUMO #86	188229	202272	May-05
87	CUMO #87	188230	202273	May-05
88	CUMO #88	188231	202274	May-05
89	CUMO #89	188232	202275	May-05
90	CUMO #90	188233	202276	May-05

Unpatented Cumo Claim List 2018 - Page 3

Item	Claim Name/Number	BLM Serial No.	County Instrument Number	Loc Dt
91	CUMO #91	188234	202277	May-05
92	CUMO #92	188235	202278	May-05
93	CUMO #93	188236	202279	May-05
94	CUMO #94	188237	202281	May-05
95	CUMO #95	188238	202282	May-05
96	CUMO #98	188239	202366	May-05
97	CUMO #99	188240	202367	May-05
98	CUMO #100	188241	202368	May-05
99	CUMO #101	188242	202369	May-05
100	CUMO #107 FRACTION	188244	202371	May-05
101	CUMO #109	188246	202373	May-05
102	CUMO #121	188258	202283	May-05
103	CUMO #122	188259	202284	May-05
104	CUMO #123	188260	202285	May-05
105	CUMO #124	188283	202286	May-05
106	CUMO #125	188261	202287	May-05
107	CUMO #126	188262	202288	May-05
108	CUMO #127	188263	202289	May-05
109	CUMO #128	188264	202290	May-05
110	CUMO #132	188268	202294	May-05
111	CUMO #133	188269	202295	May-05
112	CUMO #134	188270	202296	May-05
113	CUMO #135	188271	202297	May-05
114	CUMO #136	188272	202298	May-05
115	CUMO #137	188273	202299	May-05
116	CUMO #138	188274	202300	May-05
117	CUMO #139	188275	202301	May-05
118	CUMO #140	188276	202302	May-05
119	CUMO #141	188277	202303	May-05
120	CUMO #142	188278	202304	May-05
121	CUMO #143	188279	202305	May-05
122	CUMO #144	188280	202306	May-05
123	CUMO #145	188281	202307	May-05
124	CUMO #146	188282	202308	May-05
125	CUMO #147	188284	202309	May-05
126	CUMO #148	188285	202310	May-05
127	CUMO #149 FRACT.	188286	202311	May-05
128	CUMO #150	188257	202312	May-05
129	CUMO #151 FRACT.	188287	202313	May-05
130	CUMO #152	188288	202314	May-05
131	CUMO #153	188289	202315	May-05
132	CUMO #154	188290	202316	May-05
133	CUMO #155	188291	202317	May-05
134	CUMO #156	188292	202318	May-05
135	CUMO #157	188293	202319	May-05

Unpatented Cumo Claim List 2018 - Page 4

Item	Claim Name/Number	BLM Serial No.	County Instrument Number	Loc Dt
136	CUMO #158	188294	202320	May-05
137	CUMO #159	188295	202323	May-05
138	CUMO #160	188486	202321	May-05
139	CUMO #161	188491	202322	May-05
140	CUMO #176 FRACT.	188306	202324	May-05
141	CUMO #177 FRACT.	188307	202325	May-05
142	CUMO #178	188308	202326	May-05
143	CUMO #179	188309	202327	May-05
144	CUMO #180	188310	202328	May-05
145	CUMO #181	188311	202329	May-05
146	CUMO #182 FRACT.	188312	202330	May-05
147	CUMO #183 FRACT.	188313	202331	May-05
148	CUMO #184	188314	202332	May-05
149	CUMO #185	188315	202333	May-05
150	CUMO #186	188316	202334	May-05
151	CUMO #187	188317	202335	May-05
152	CUMO #188 FRACT.	188318	202336	May-05
153	New Cumo 190 Fraction	203192	230231	Oct-10
154	New Cumo 191 Fraction	203193	230232	Oct-10
155	New Cumo 192 Fraction	203194	230233	Oct-10
156	New Cumo 193 Fraction	203195	230234	Oct-10
157	Cumo 194	203196	230229	Oct-10
158	Cumo 195 Fraction	203197	230230	Oct-10
159	Cumo 196 Fraction	203198	230228	Oct-10
160	Cumo 197 Fraction	203199	230235	Oct-10
161	Cumo 198 Fraction	203200	230236	Oct-10
162	Cumo 199 Fraction	203201	230237	Oct-10
163	Cumo 200 Fraction	203202	230238	Oct-10
164	Cumo 201 Fraction	203203	230239	Oct-10
165	Sharon #1	177221	159054	Oct-94
166	Sharon #2	177222	159055	Oct-94
167	Sharon #3	177223	159056	Oct-94
168	Sharon#4	177224	159057	Oct-94
169	Sharon#5	177225	159058	Oct-94
170	Sharon#6	177226	159059	Oct-94
171	Sharon#7	177227	159060	Oct-94
172	Sharon#8	177228	159061	Oct-94

Unpatented CuMo Claim List 2018 - Page 5

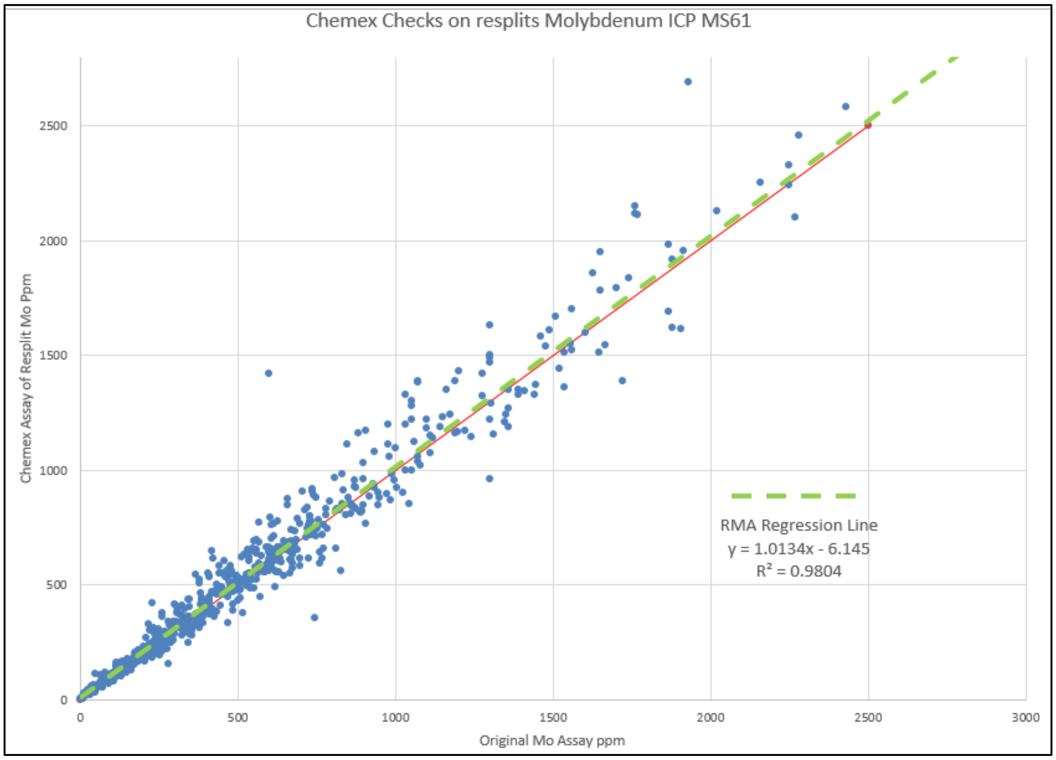
Item	Claim Name/Number	BLM Serial No.	County Instrument Number	Loc Dt
173	Sharon#8	177228	159061	Oct-94
174	Sharon#9	177229	159062	Oct-94
175	Sharon#10	177230	159063	Oct-94
176	BlackJack#1	177236	159064	Oct-94
177	BlackJack#2	177237	159065	Oct-94
178	JRA No. 16	106515	76851	Sep-82
179	JRA No. 18	106517	76853	Sep-82
180	JRA No. 20	106519	76855	Sep-82
181	JRA No. 20	106520	76856	Sep-82
182	JRA No. 29	106528	76864	Sep-82
183	JRA No. 30	106529	76865	Sep-82
184	JRA No. 31	106530	76866	Sep-82
185	JRA No. 45	106544	76880	Sep-82

Patented Cumo Claim List 2018 - Page 1

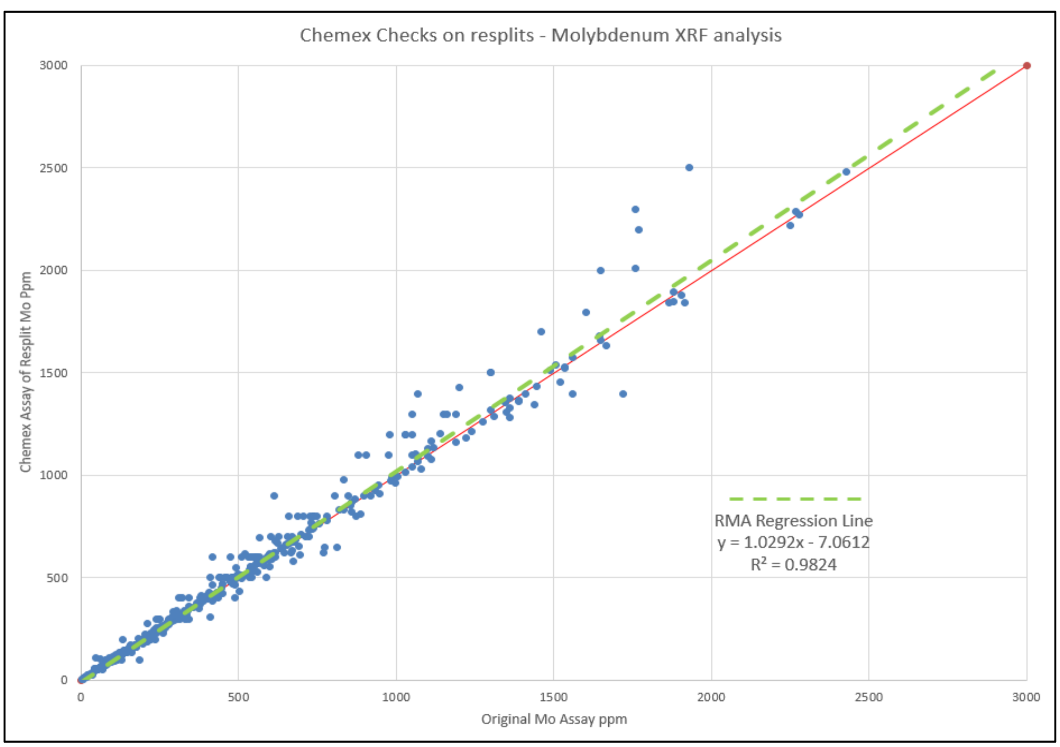
Item	Claim Name/Number	Patent Number	Year Granted	Survey Number
1	Blackbird	11830026	1902+1983	3636
2	Red Flag	11830026	1902+1983	3636
3	Enterprise	39183	1902	1706
4	Enterprise Fraction	39183	1902	1706
5	Commonwealth	39183	1902	1706
6	Baby Mine	39183	1902	1706
7	Duane #6	39183	1945	3455
8	German American	1155808	1945	3455
9	Homestake #6	1155808	1945	3455
10	Coon Dog #1	1155808	1945	3455
11	Coon Dog #3	1155808	1945	3455
12	Coon Dog #4	1155808	1945	3455
13	Coon Dog #5	1155808	1945	3455
14	Coon Dog #10	1155808	1945	3455
15	Grey Eagle #2	1155808	1945	3455
16	Grey Eagle #3	1155808	1945	3455
17	Missing Link #1	1155808	1945	3455
18	Missing Link #4	1155808	1945	3455
19	Ida	1155808	1945	3455
20	Daily	1155808	1945	3455
21	Jumbo	645180	1918	2830
22	Jumbo #2	645180	1918	2830
23	Snowstorm	645180	1918	2830
24	Sunset #1	119757	1909	2269
25	Last Dollar	119757	1909	2269
26	Sunset #2	119757	1909	2269
27	Gold Dollar #1	119757	1909	2269
28	Gold Dollar #2	119757	1909	2269
29	Gold Dollar #3	119757	1909	2269
30	Pheasant Lode	564946	1917	2679
31	Golden Age Placer	535188	1916	2680
32	Wills Placer	951698	1925	3052
33	Gerdo	645179	1918	2831
34	Harper #1	1144749	1944	3456
35	Harper #2	1144749	1944	3456
36	Florence	546017	1916	2681
37	Charlotte	546017	1916	2681
38	Francis	546017	1916	2681
39	Theron Fraction	546017	1916	2681
40	Theron	546017	1916	2681
41	Idaho	546017	1916	2681

Appendix 2: Re-Splits of Rejects

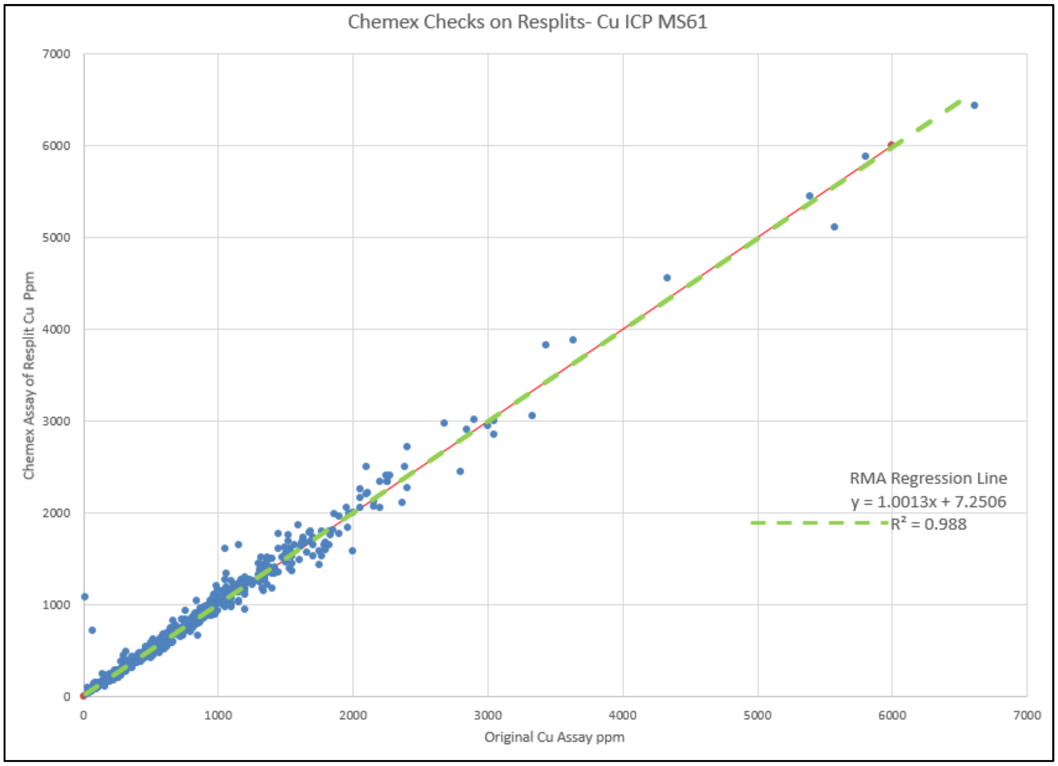
Results for Mo - Chemex - original vs. ICP check



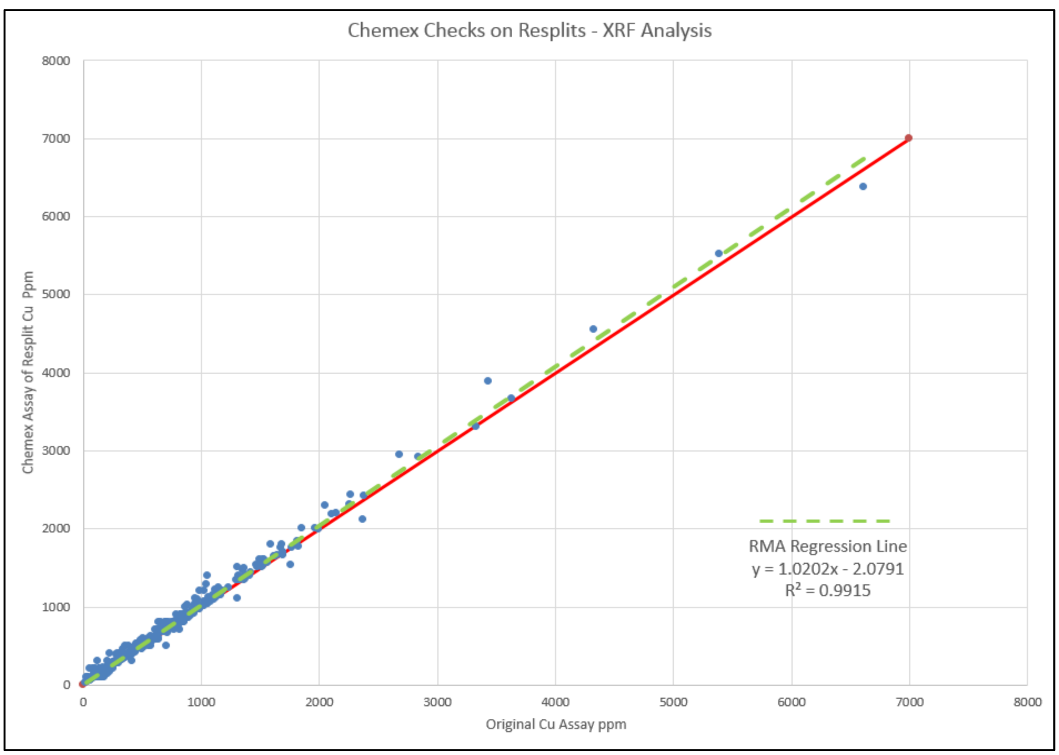
Results for Mo - Chemex - original vs. XRF check



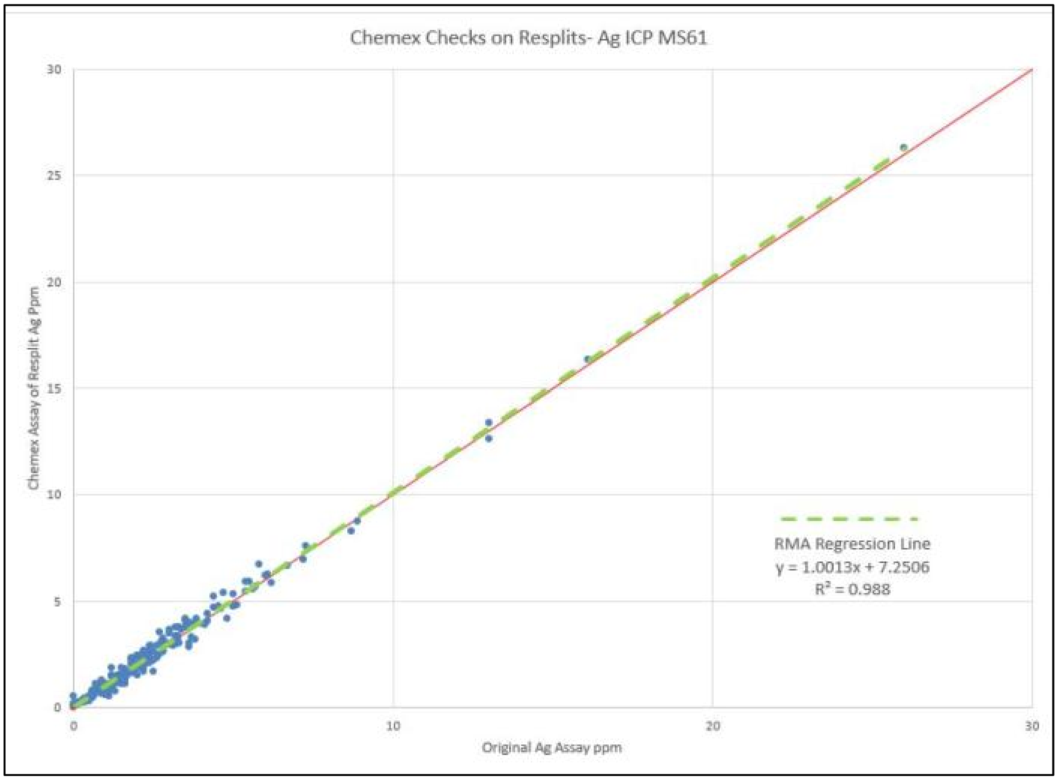
Results for Cu - Chemex - original vs. ICP check



Results for Cu - Chemex - original vs. XRF check



Results for Ag - Chemex original vs. ICP check



Appendix 3: Drill Holes used in Resource Estimate

Hole	Northing	Easting	Elevation	Dip	Azimuth	Length (feet)
71-01	120989.9	219904.5	6026.5	-90	0	1884
71-02	120575.0	219820.0	6060.0	-70	0	405
71-03	120250.0	219905.0	6165.0	-90	0	70
C71-04	120785.0	219940.0	6045.0	-90	0	113
C72-05	120524.8	220569.9	6201.7	-90	0	1416
C72-06	121749.0	219919.0	5902.0	-90	0	663
C72-07	121491.0	219823.0	5962.0	-90	0	275
C72-08	118890.0	220025.0	6467.0	-90	0	379
C74-09	121438.0	220687.0	5890.0	-60	168	804.6
C75-10	119755.7	221220.4	6341.0	-90	0	2381
C76-11	120455.8	221250.2	5996.0	-90	0	3003
C76-12	120955.0	221432.0	5742.0	-43	190	1340
C77-13	119471.9	219902.9	6426.3	-90	0	1804
C77-14	119085.4	221271.3	6613.3	-90	0	2123.8
C77-15	119772.1	221950.9	6339.0	-90	0	1933.2
C78-16	119209.7	219147.5	6247.9	-90	0	2131.7
C78-17	118711.9	219886.6	6544.3	-90	0	2281.5
C78-18	119823.5	222649.1	6168.3	-90	0	2361
C79-19	120178.0	219887.0	6170.0	-90	0	2280
C79-20	120878.0	220787.0	6105.0	-90	0	2543
C81-24	120671.1	222009.5	6069.8	-90	0	1000
C81-25	119890.0	219289.7	6019.0	-90	0	1011
C81-26	121338.1	221432.9	5767.5	-90	0	1193

Hole	Northing	Easting	Elevation	Dip	Azimuth	Length (feet)
27-06	120031.9	220207.9	6351.4	-90	0	1849
28-06	119539.8	220816.8	6321.1	-90	0	1711
29-07	119778.9	221246.7	6343.7	-70	140	2281.7
30-07	119732.2	219616.8	6213.1	-90	0	2416.5
31-07	119792.5	221243.3	6342.3	-70	45	2104
32-07	119558.4	220822.6	6323.6	-70	190	2044
33-07	118476.7	221227.0	6796.8	-90	0	2095
34-07	118658.3	220487.4	6534.2	-70	95	1769
35-08	118655.2	220480.4	6533.2	-90	0	2817
36-08	119335.3	219448.7	6274.6	-90	0	2488
37-08	119780.4	221246.8	6341.5	-70	335	2195
38-08	118655.2	220480.4	6533.2	-70	180	2441
39-08	118917.9	220813.2	6575.1	-90	0	2688
40-08	119530.1	220791.4	6321.4	-70	225	2252
41-08	119630.2	218962.5	6219.9	-90	0	3018
42-08	118748.9	219911.0	6549.2	-70	270	2707
43-08	120612.8	220052.8	6173.8	-80	40	1313
44-08	118085.1	221515.9	6739.4	-65	75	3047
45-08	119802.3	218821.4	6183.7	-80	330	1796
46-09	118913.9	220811.3	6575.1	-75	110	959
47-09	120686.7	219421.7	5832.6	-90	0	2530
48-09	120690.0	219425.0	5825.5	-70	305	2576
49-09	119094.6	221745.7	6645.3	-90	0	2847
50-09	121548.0	219843.5	5832.6	-75	270	1826
51-09	121534.9	219859.8	5828.5	-90	0	1593.5
52-09	118499.5	221251.3	6791.2	-75	20	2772
53-09	119803.9	218830.5	6183.4	-75	15	2461
54-09	119534.9	219005.1	6195.9	-75	15	1096
55-10	117559.6	218422.5	6724.2	-65	0	2479
56-10	117559.9	218421.9	6724.2	-65	305	1294
57-10	117559.3	218422.2	6724.2	-90	0	534
58-11	119142.8	219970.3	6451.3	-90	0	1885
59-11	119095.6	221745.9	6645.3	-75	0	1910
60-12	117559.9	218421.9	6724.2	-50	180	1455
61-12	118748.9	219911.0	6549.2	-75	335	1318
62-12	116866.1	218040.5	6628.7	-50	135	1484
63-12	116866.8	218041.5	6628.7	-60	330	807
64-12	118913.9	220811.3	6575.1	-75	25	2139
65-12	118148.8	221117.5	6785.7	-80	315	1908
66-12	118674.0	221687.8	6689.7	-90	0	2241
67-12	118913.9	220811.3	6575.1	-70	340	1978
68-12	119095.6	221745.9	6645.3	-70	310	2133.5

Appendix 4: Semi-variograms

A4.1- MoS_2 in Cu-Mo and Mo Zones

A4.2-MoS_2 in Cu-Ag Zone

A4.3-Cu in Cu-Ag and Cu-Mo Zones

A4.4-Cu in Mo Zone

A4.5-Ag in Cu-Ag and Cu-Mo Zones

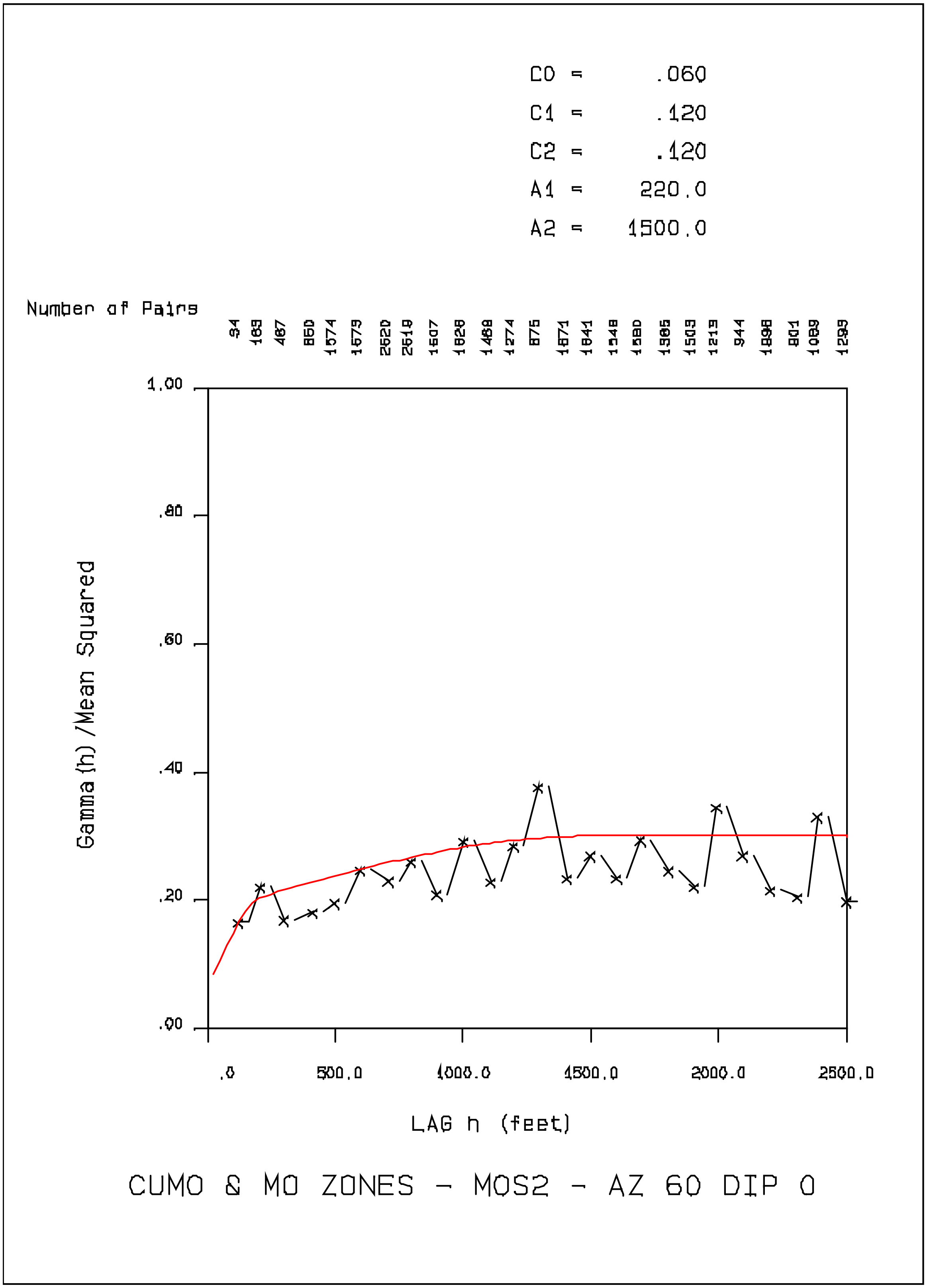
A4.6-Ag in Mo Zone

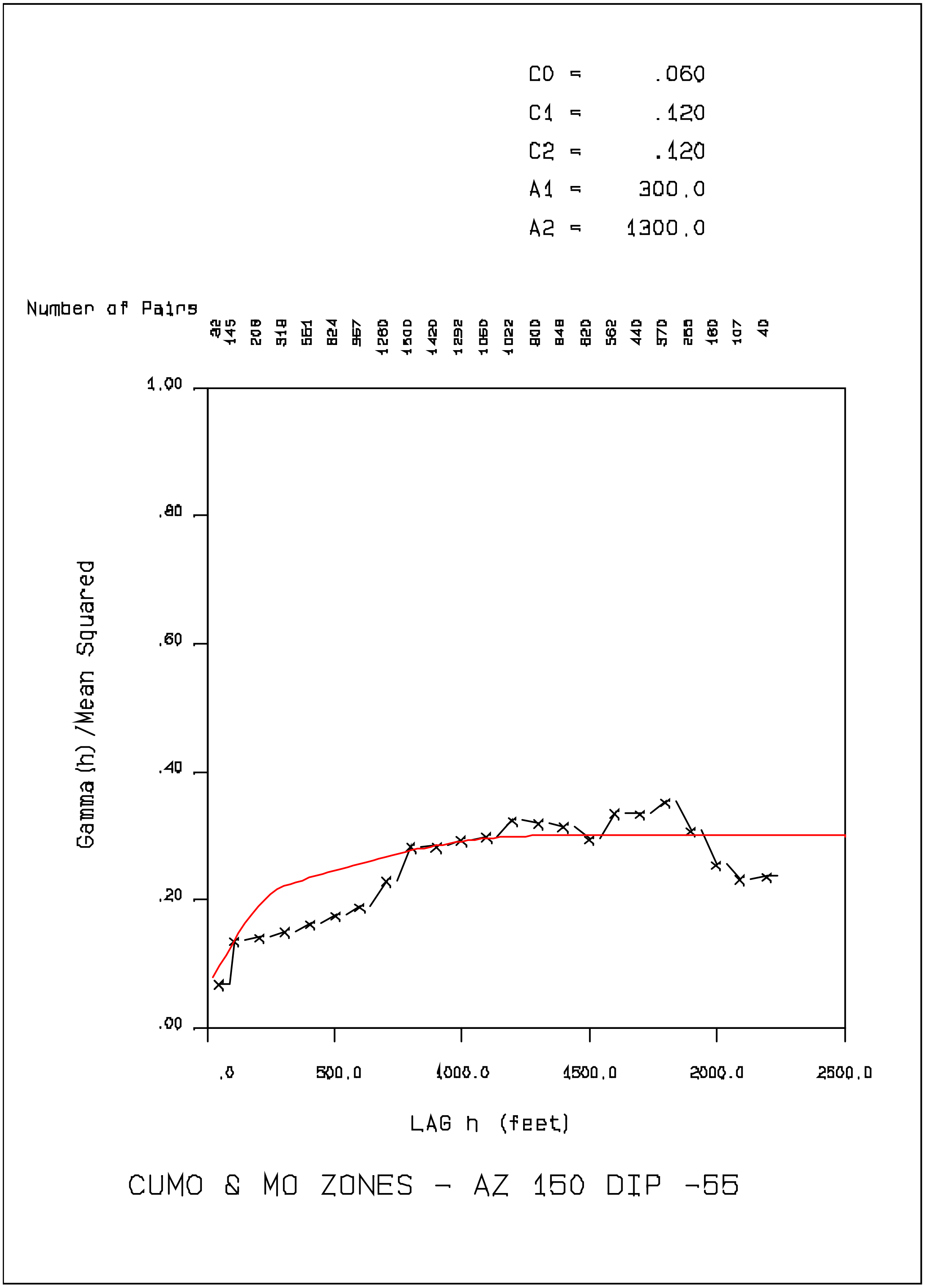
A4.7-W in Cu-Ag Zone

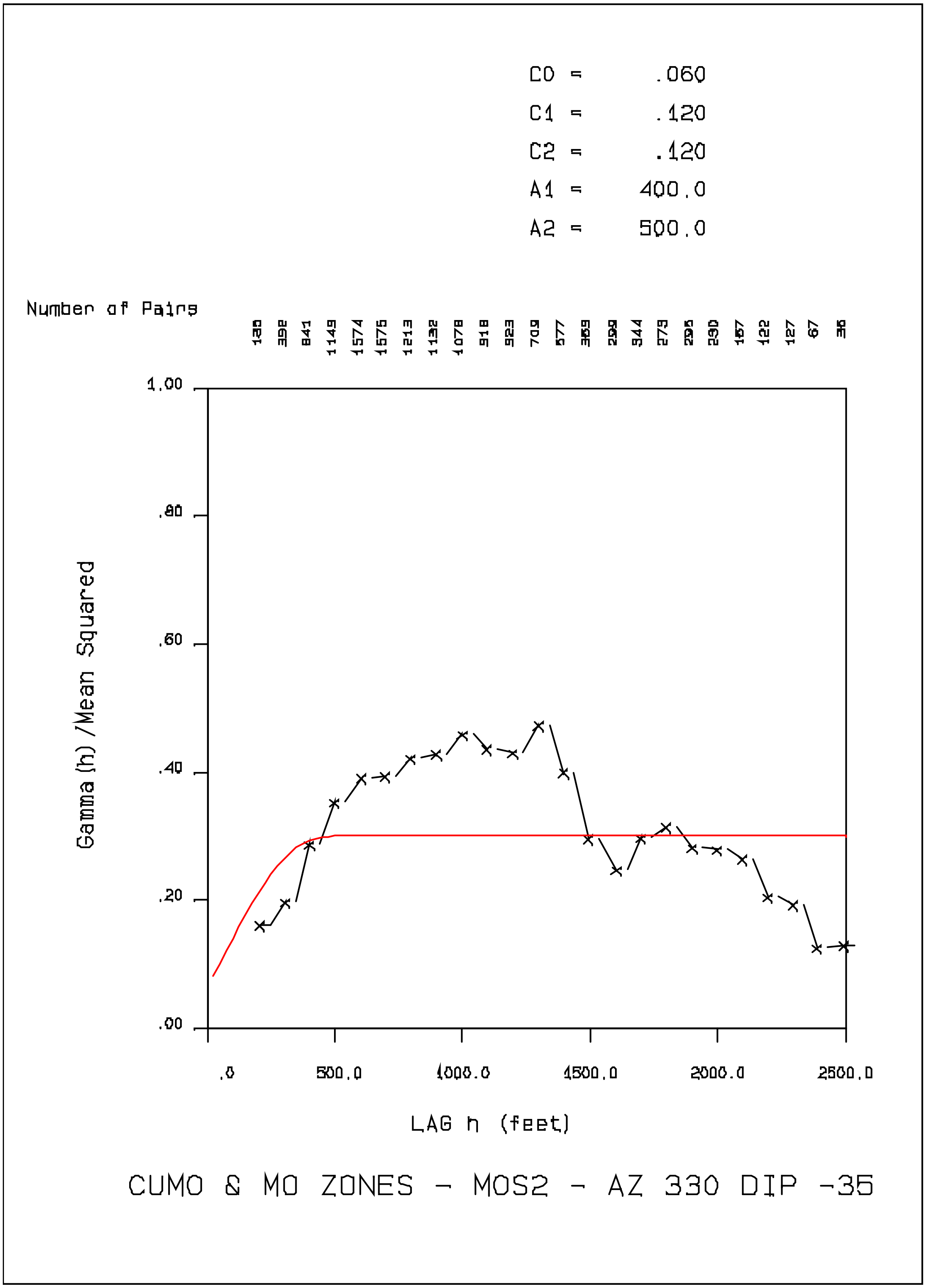
A4.8-W in Cu-Mo and Mo Zones

*Tungsten is included for reference only, as it was not used in resource estimation

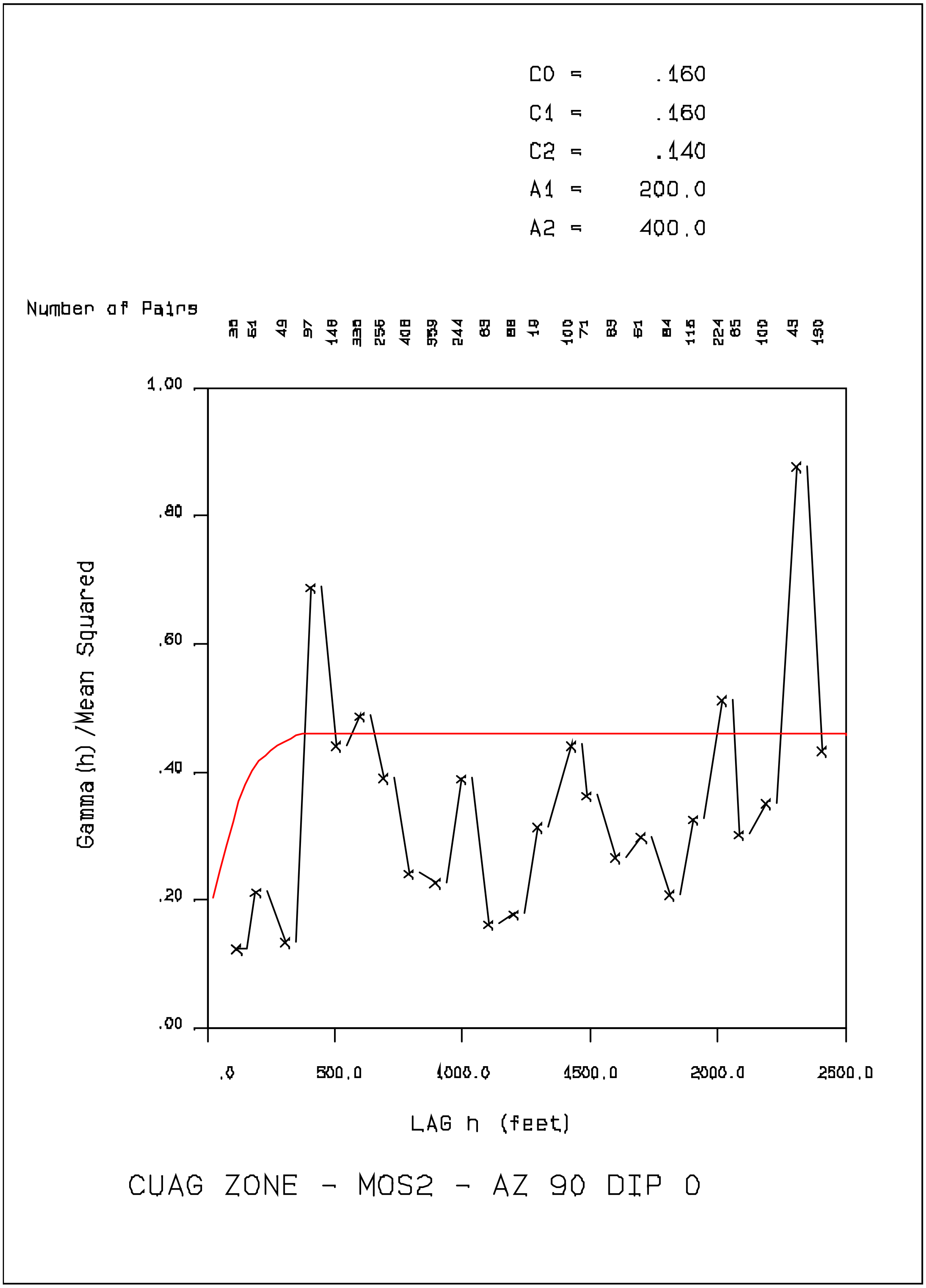
A4.1 – MoS_2 in CuMo and Mo Zones

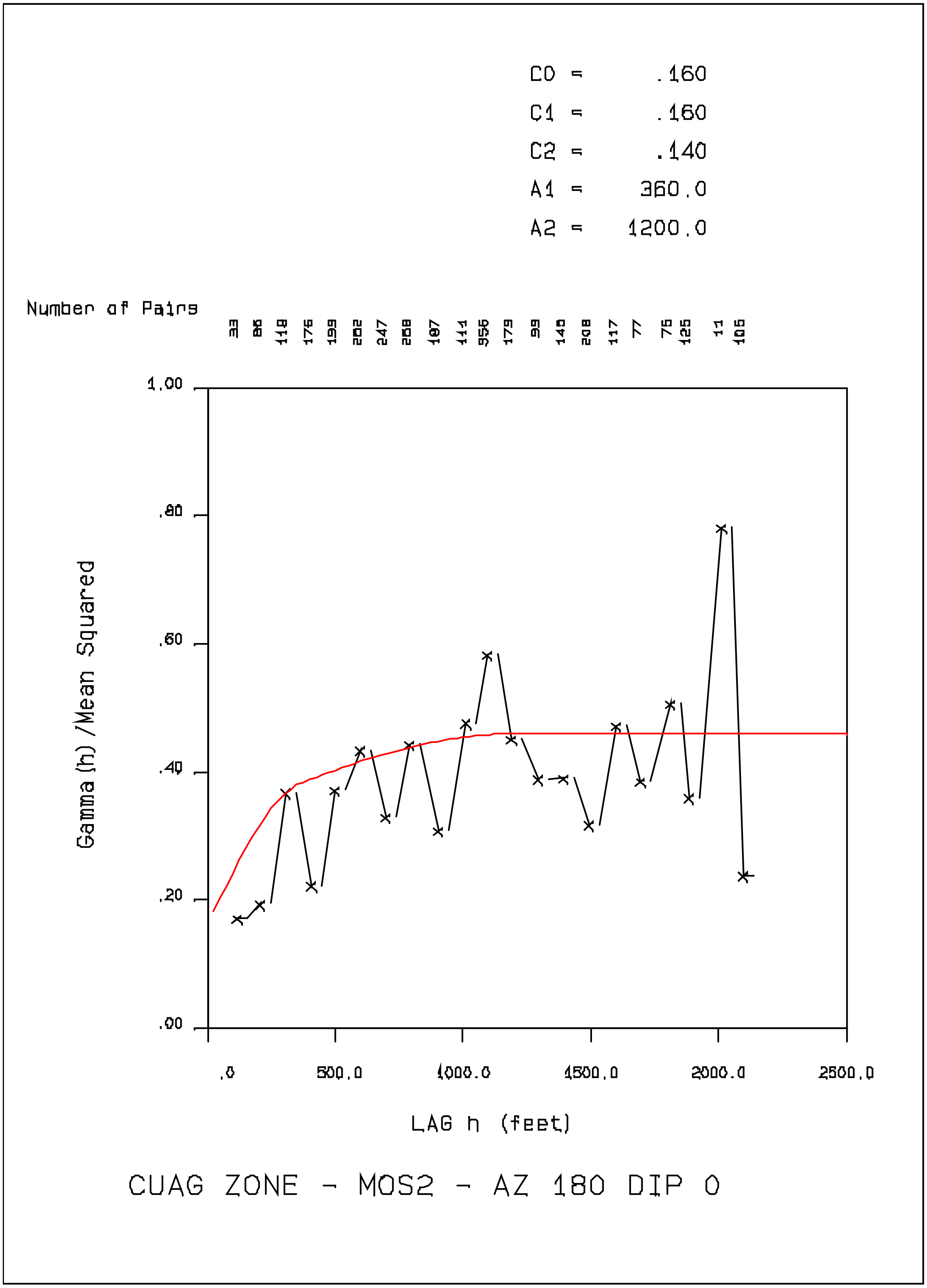


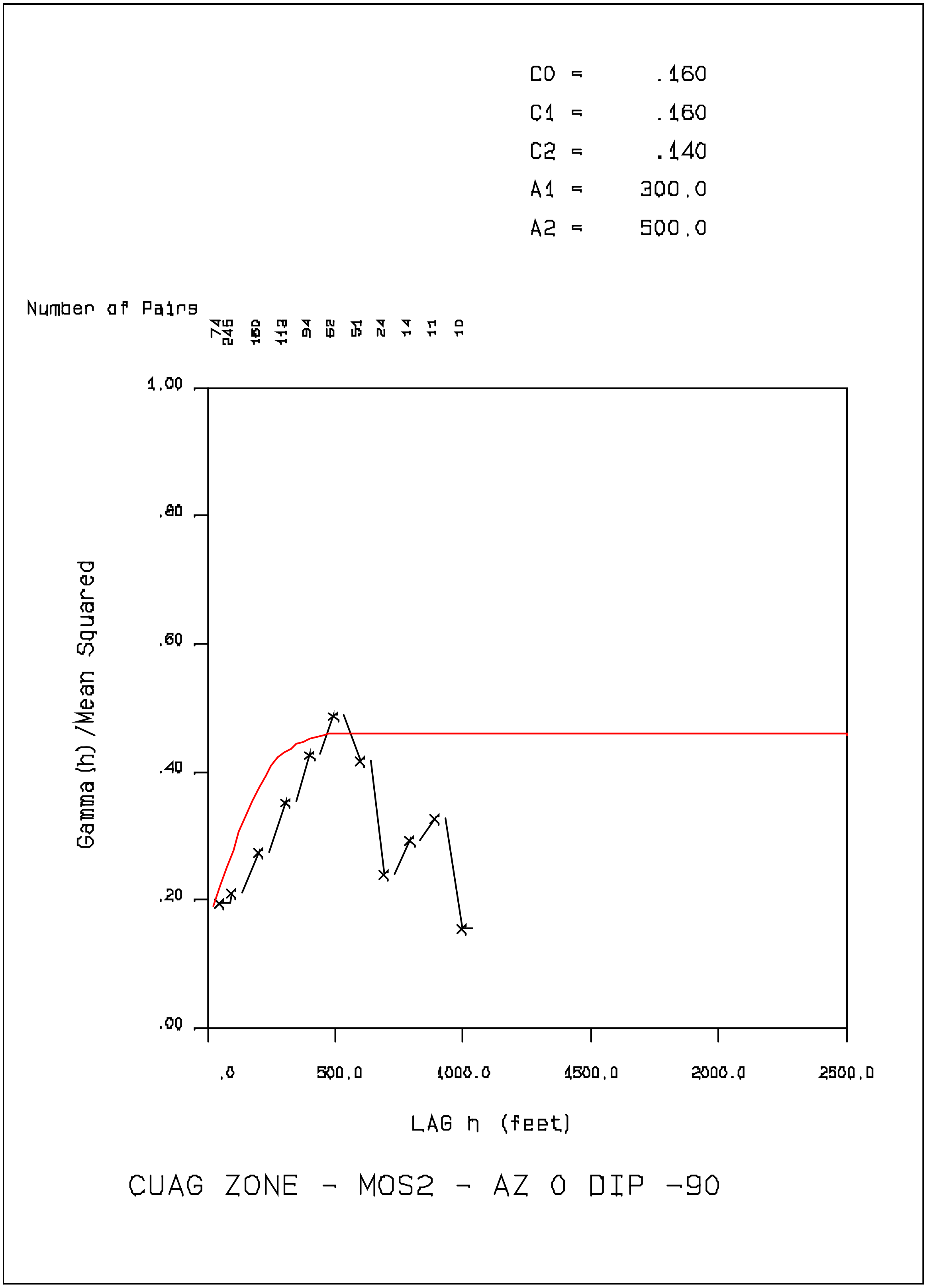

C0 = .060
C1 = .120
C2 = .120
A1 = 300.0
A2 = 1300.0
Number of Pairs
32 145 208 318 561 824 967 1280 1500 1420 1292 1060 1022 800 848 820 562 440 370 255 160 107 40
Gamma (h) /Mean Squared
1.00
.80
.60
.40
.20
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUMO & MO ZONES - AZ 150 DIP -55


C0 = .060
C1 = .120
C2 = .120
A1 = 400.0
A2 = 500.0
Number of Pairs
Gamma (h) /Mean Squared
1.00
.80
.60
.40
.20
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUMO & MO ZONES - MOS2 - AZ 330 DIP -35

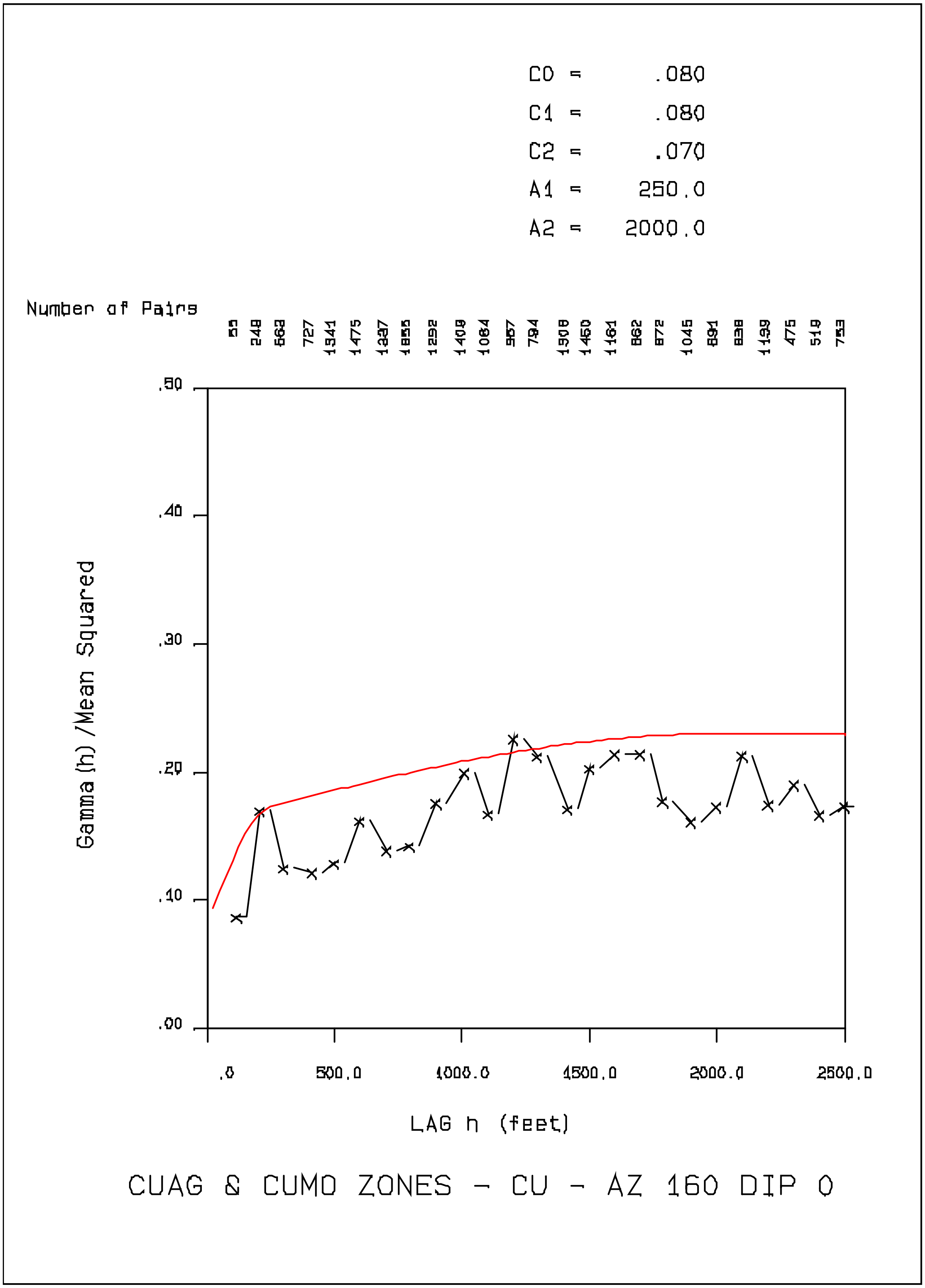
A4.2 – MoS_2 in Cu-Ag Zone:

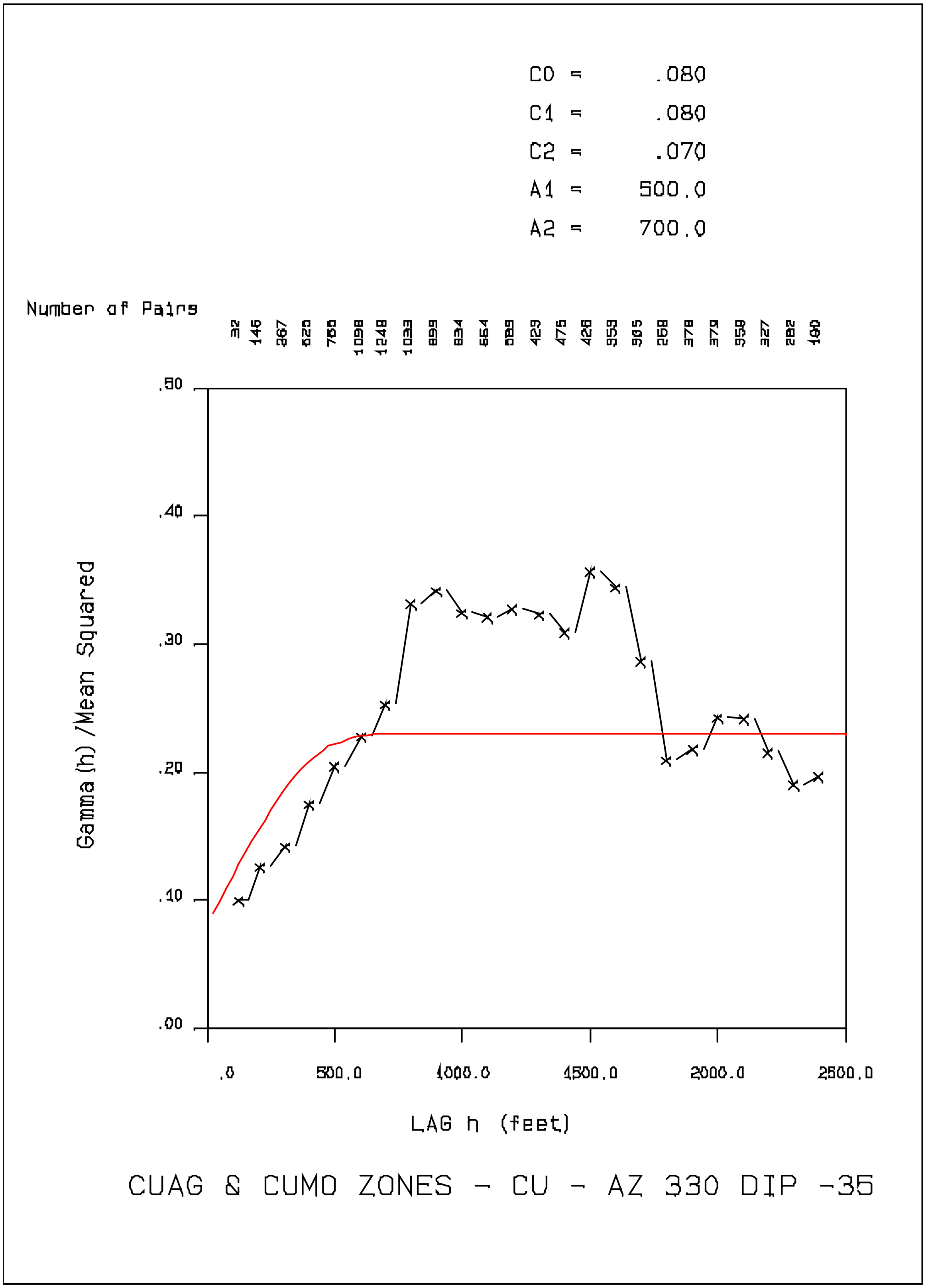


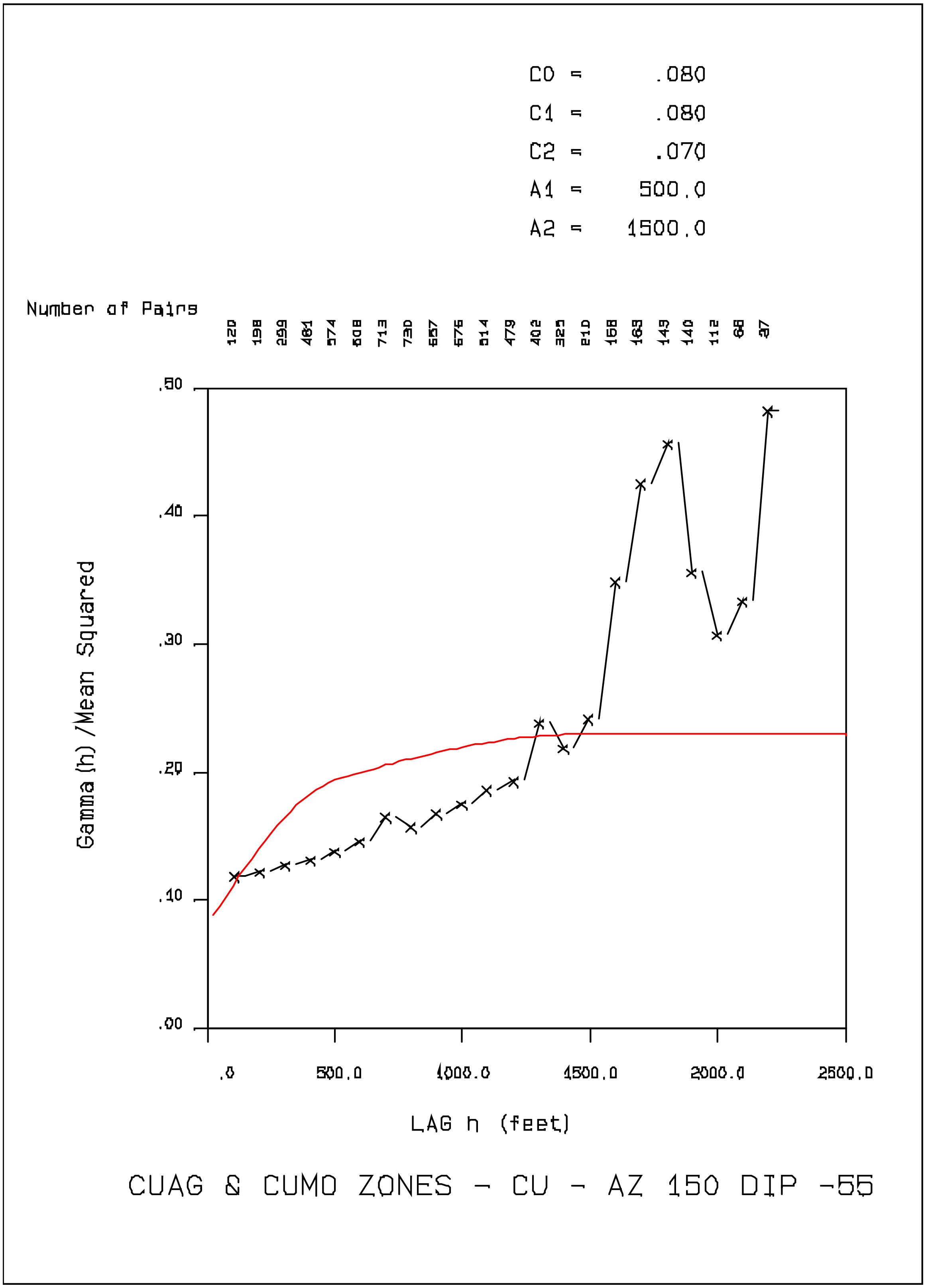

C0 = .160
C1 = .160
C2 = .140
A1 = 360.0
A2 = 1200.0
Number of Pairs
33
86
118
176
193
202
247
258
187
411
356
179
93
145
208
117
77
75
125
11
105
Gamma (h) /Mean Squared
1.00
.80
.60
.40
.20
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG ZONE - MOS2 - AZ 180 DIP 0


C0 = .160
C1 = .160
C2 = .140
A1 = 300.0
A2 = 500.0
Number of Pairs
74
245
150
113
94
62
51
24
14
11
10
1.00
.80
.60
.40
.20
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG ZONE - MOS2 - AZ 0 DIP -90

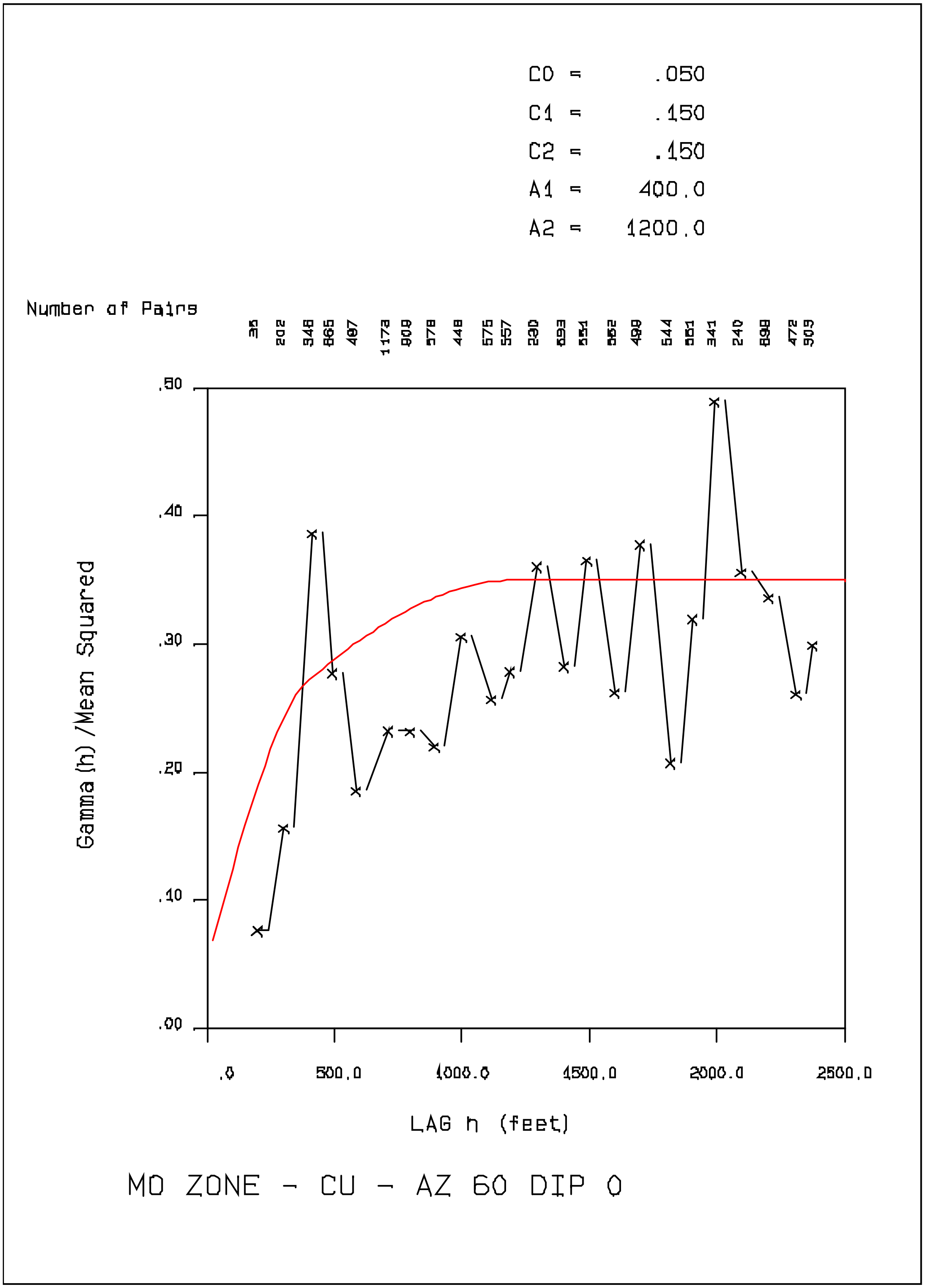
A4.3 – Cu in Cu-Ag and Cu-Mo Zones:

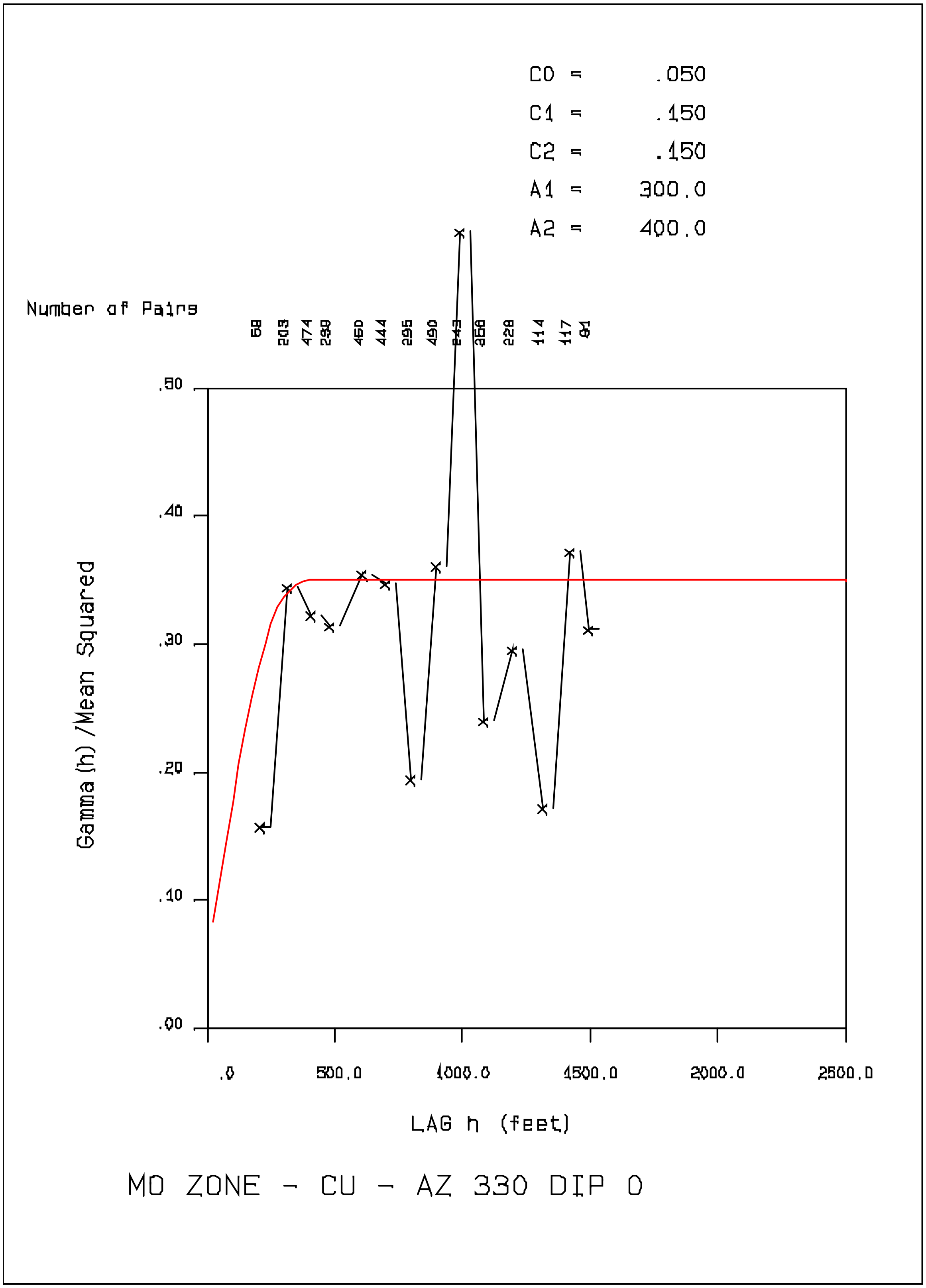


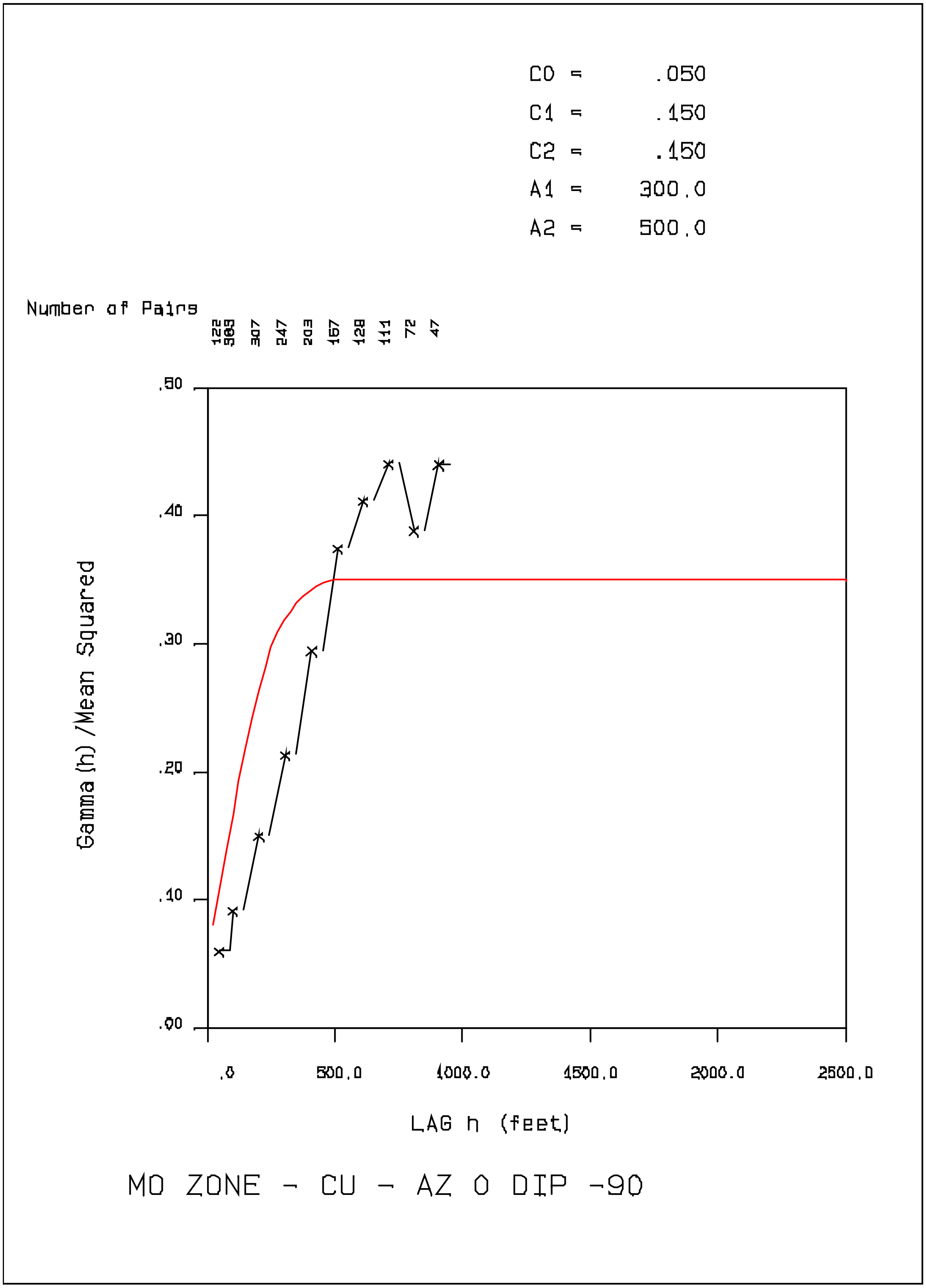

C0 = .080
C1 = .080
C2 = .070
A1 = 500.0
A2 = 700.0
Number of Pairs
32 146 367 525 765 1098 1248 1033 899 834 654 589 429 475 426 353 305 268 378 379 359 327 282 190
.50
.40
.30
.20
.10
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG & CUMO ZONES - CU - AZ 330 DIP -35


C0 = .080
C1 = .080
C2 = .070
A1 = 500.0
A2 = 1500.0
Number of Pairs
120 198 299 461 574 608 713 730 657 676 614 479 402 325 210 168 163 143 140 112 68 37
Gamma (h) /Mean Squared
.00
.10
.20
.30
.40
.50
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG & CUMO ZONES - CU - AZ 150 DIP -55

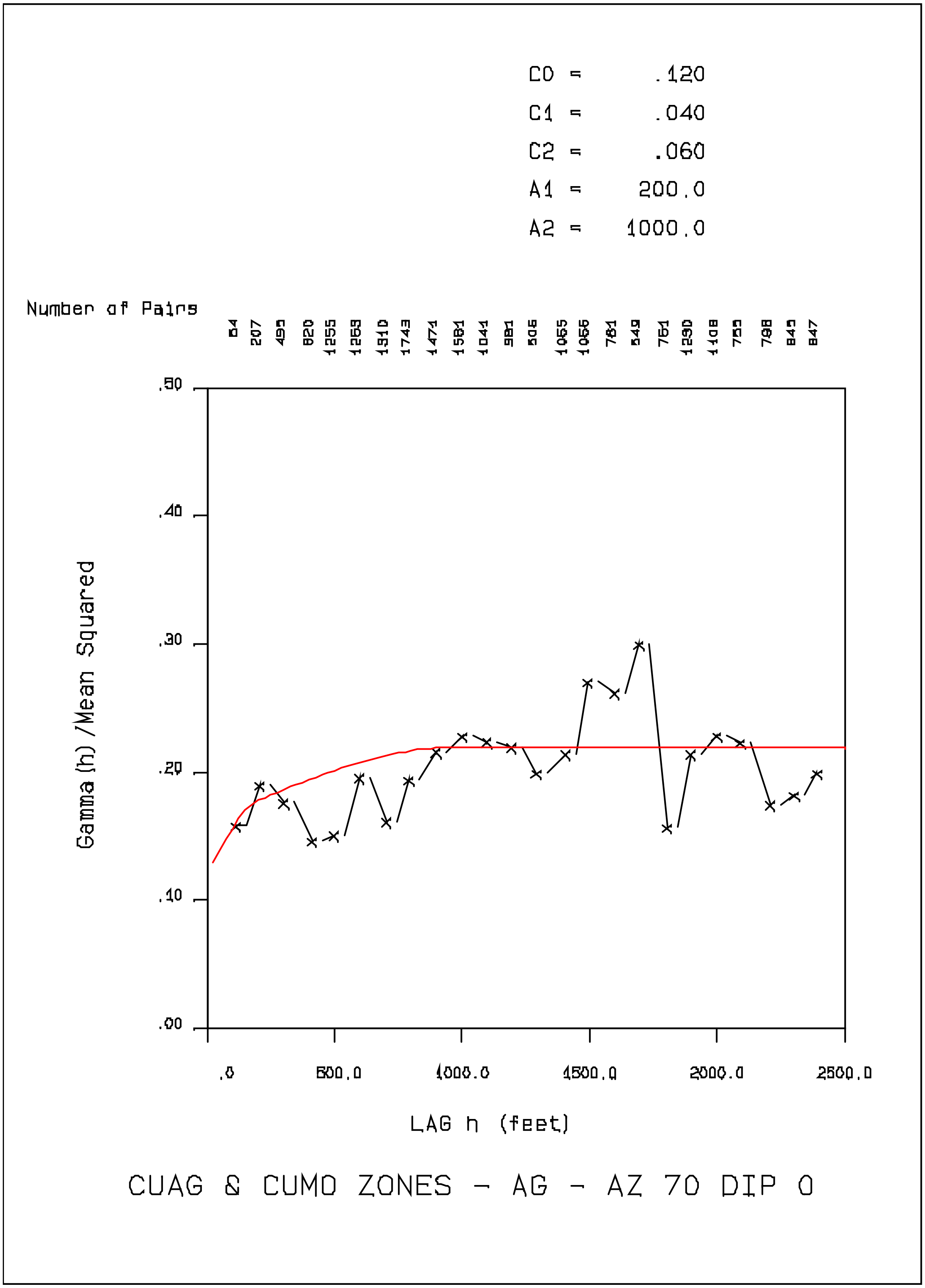
A4.4 – Cu in Mo Zone:

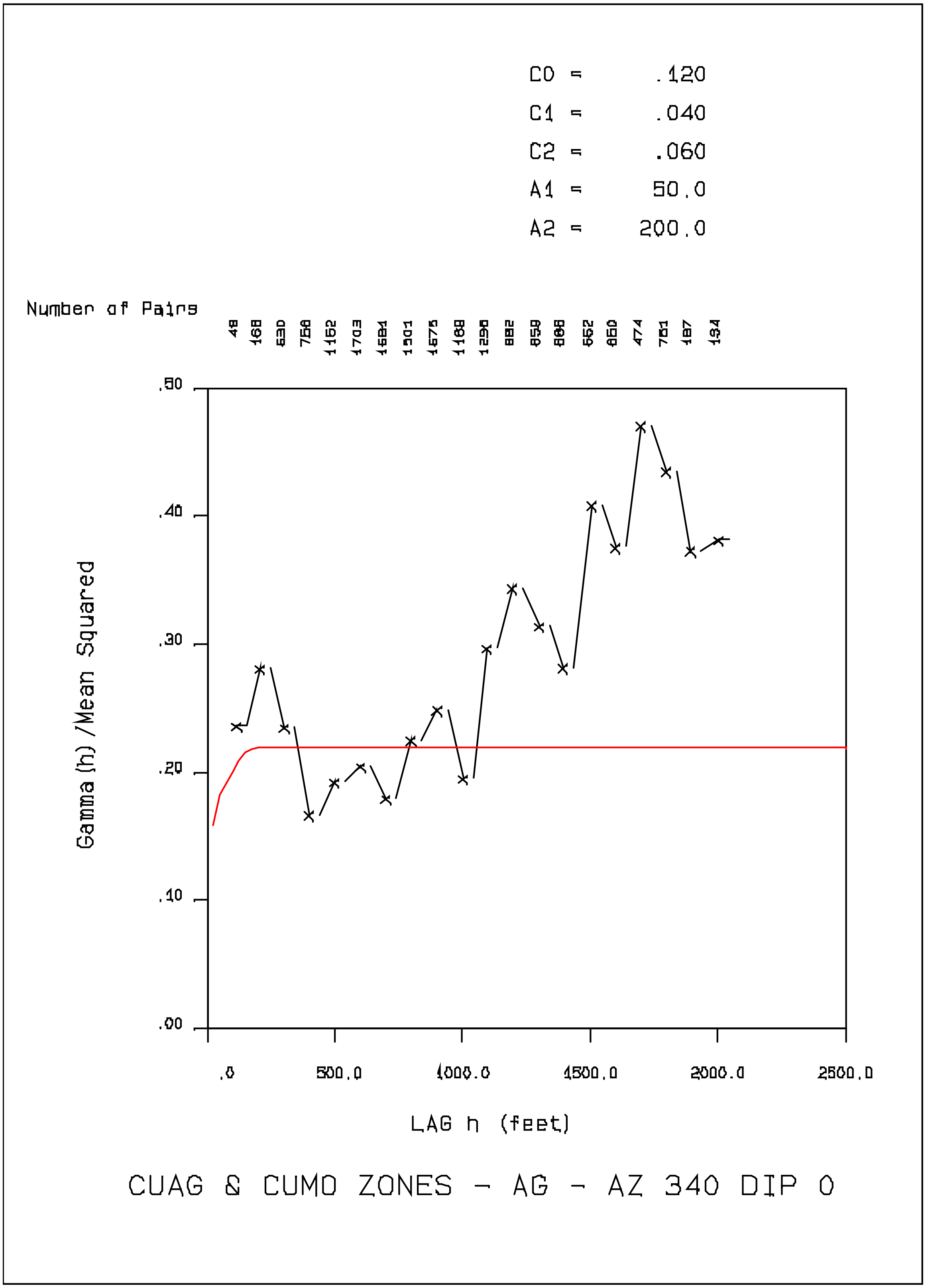


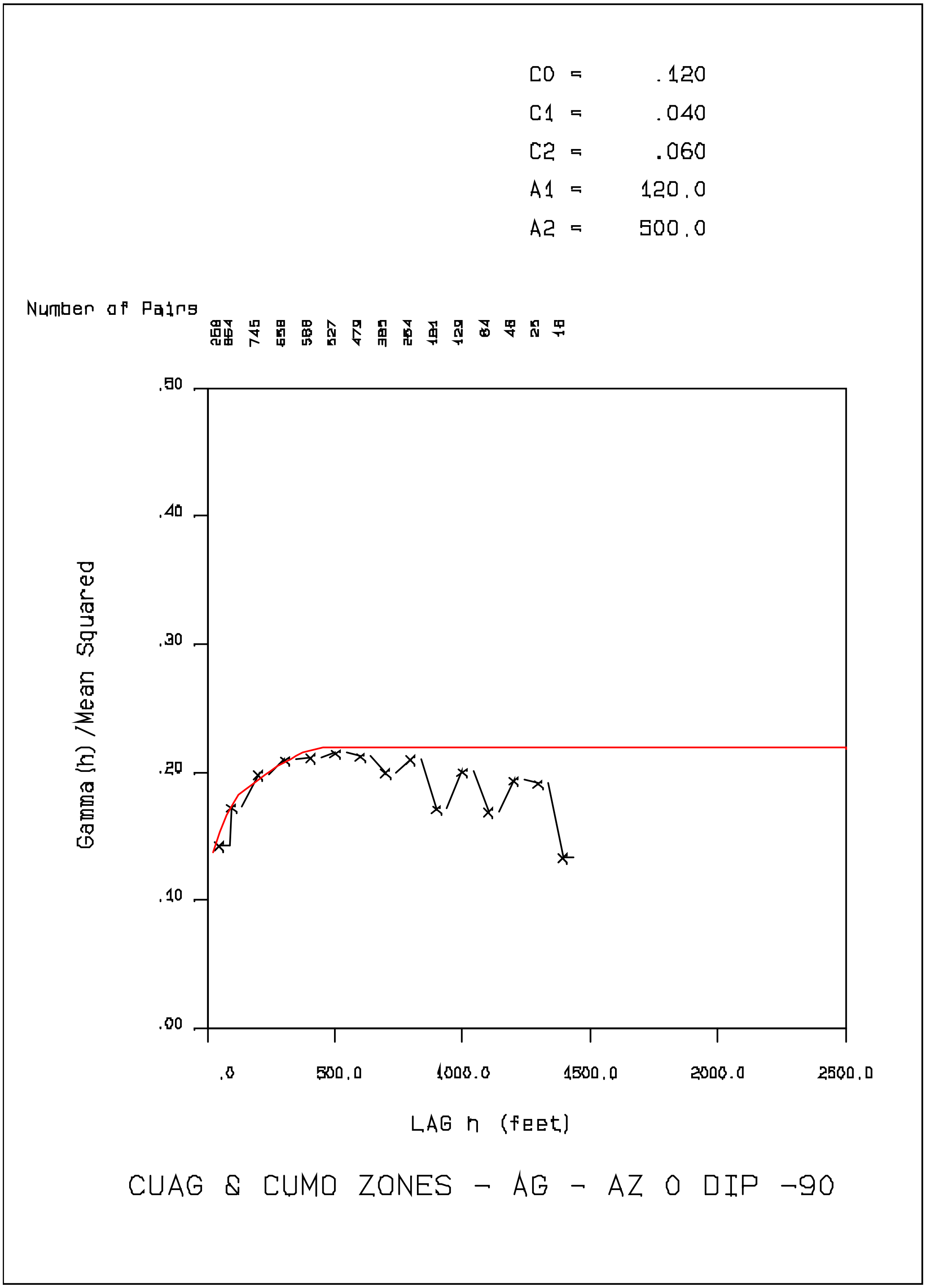

C0 = .050
C1 = .150
C2 = .150
A1 = 300.0
A2 = 400.0
Number of Pairs
68
203
474
238
460
444
295
490
243
358
228
114
117
91
Gamma (h) /Mean Squared
.00
.10
.20
.30
.40
.50
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
MO ZONE - CU - AZ 330 DIP 0


C0 = .050
C1 = .150
C2 = .150
A1 = 300.0
A2 = 500.0
Number of Pairs
122
303
307
247
203
167
128
111
72
47
Gamma (h) /Mean Squared
.50
.40
.30
.20
.10
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
MO ZONE - CU - AZ 0 DIP -90

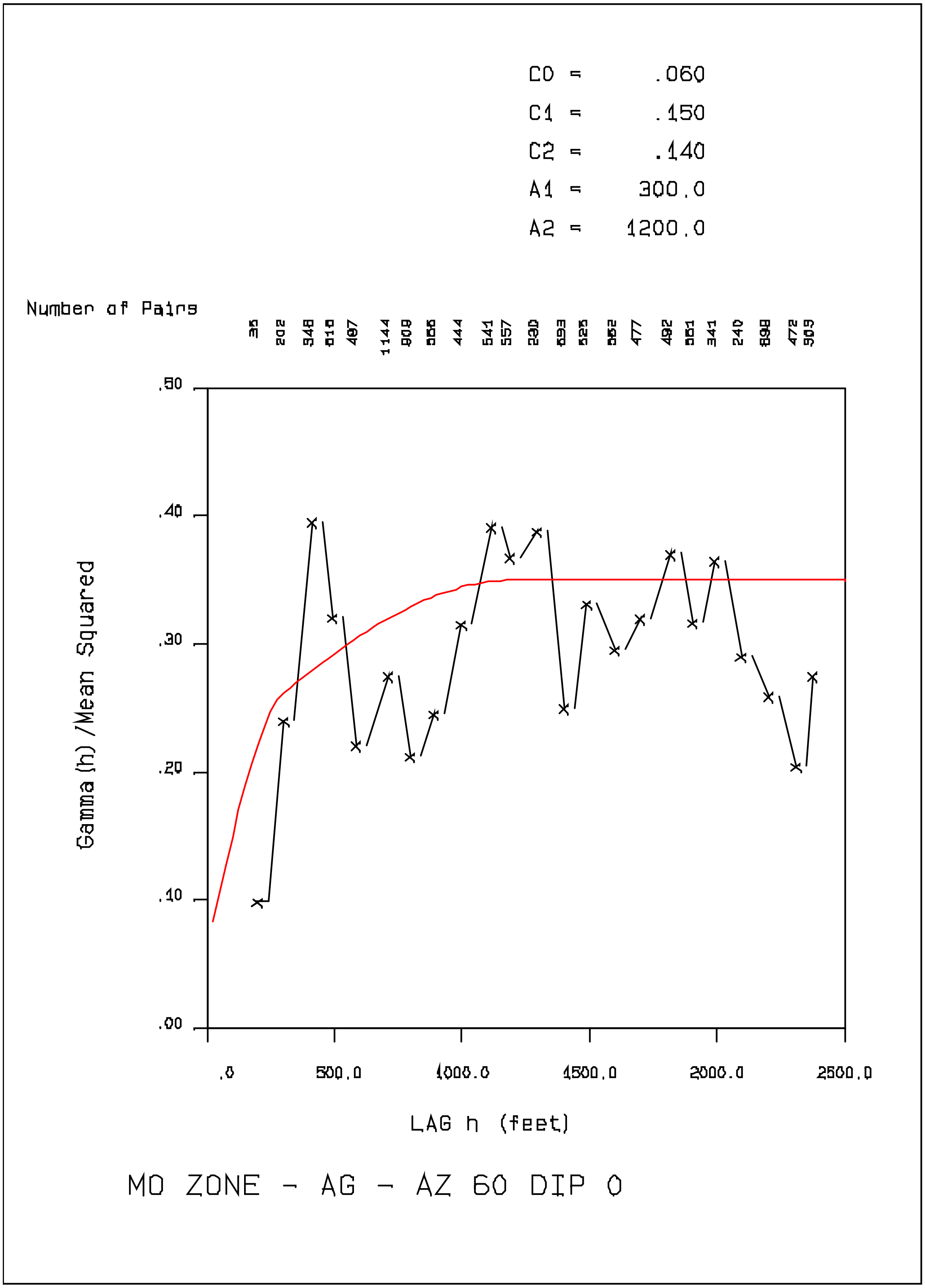
A4.5 – Ag in Cu-Ag and Cu-Mo Zones:

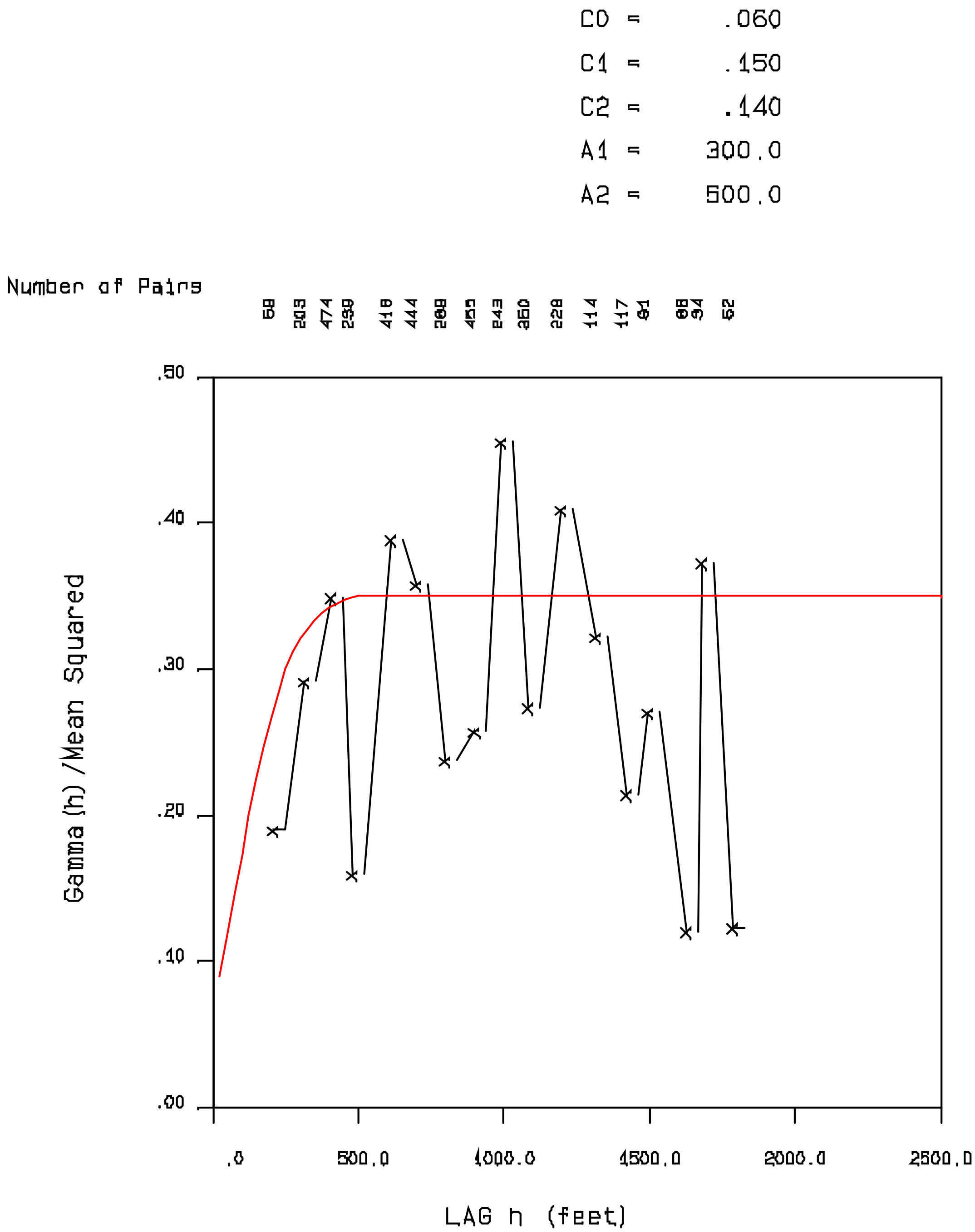



C0 = .120
C1 = .040
C2 = .060
A1 = 50.0
A2 = 200.0
Number of Pairs
48 168 630 756 1162 1703 1684 1501 1675 1168 1296 882 859 886 562 660 474 761 187 134
Gamma (h) /Mean Squared
.50
.40
.30
.20
.10
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG & CUMO ZONES - AG - AZ 340 DIP 0

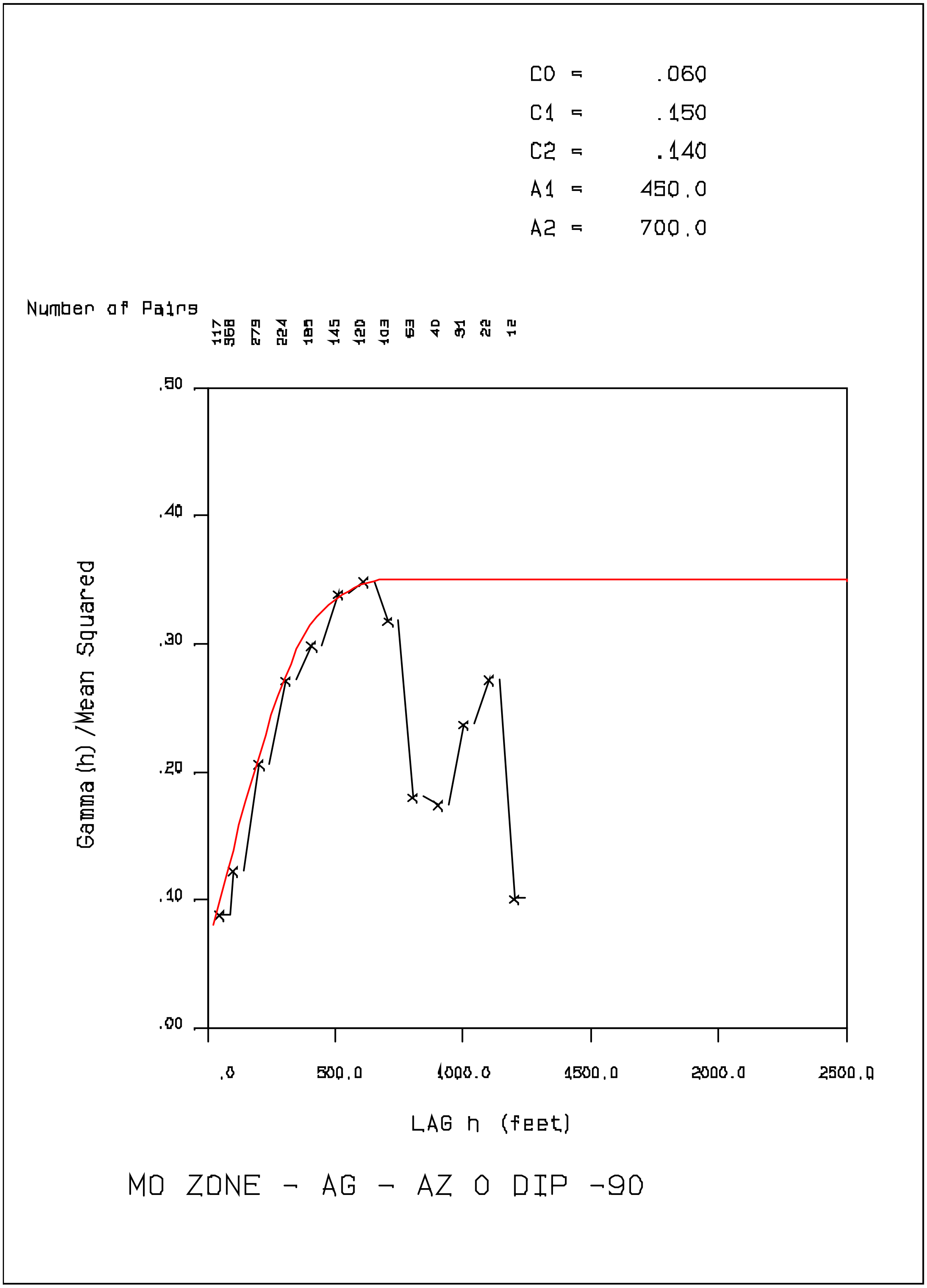

C0 = .120
C1 = .040
C2 = .060
A1 = 120.0
A2 = 500.0
Number of Pairs
268 864 745 658 588 527 479 385 264 184 129 84 48 25 18
Gamma (h) /Mean Squared
.50
.40
.30
.20
.10
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG & CUMO ZONES - AG - AZ 0 DIP -90

A4.6 – Ag in Mo Zone:

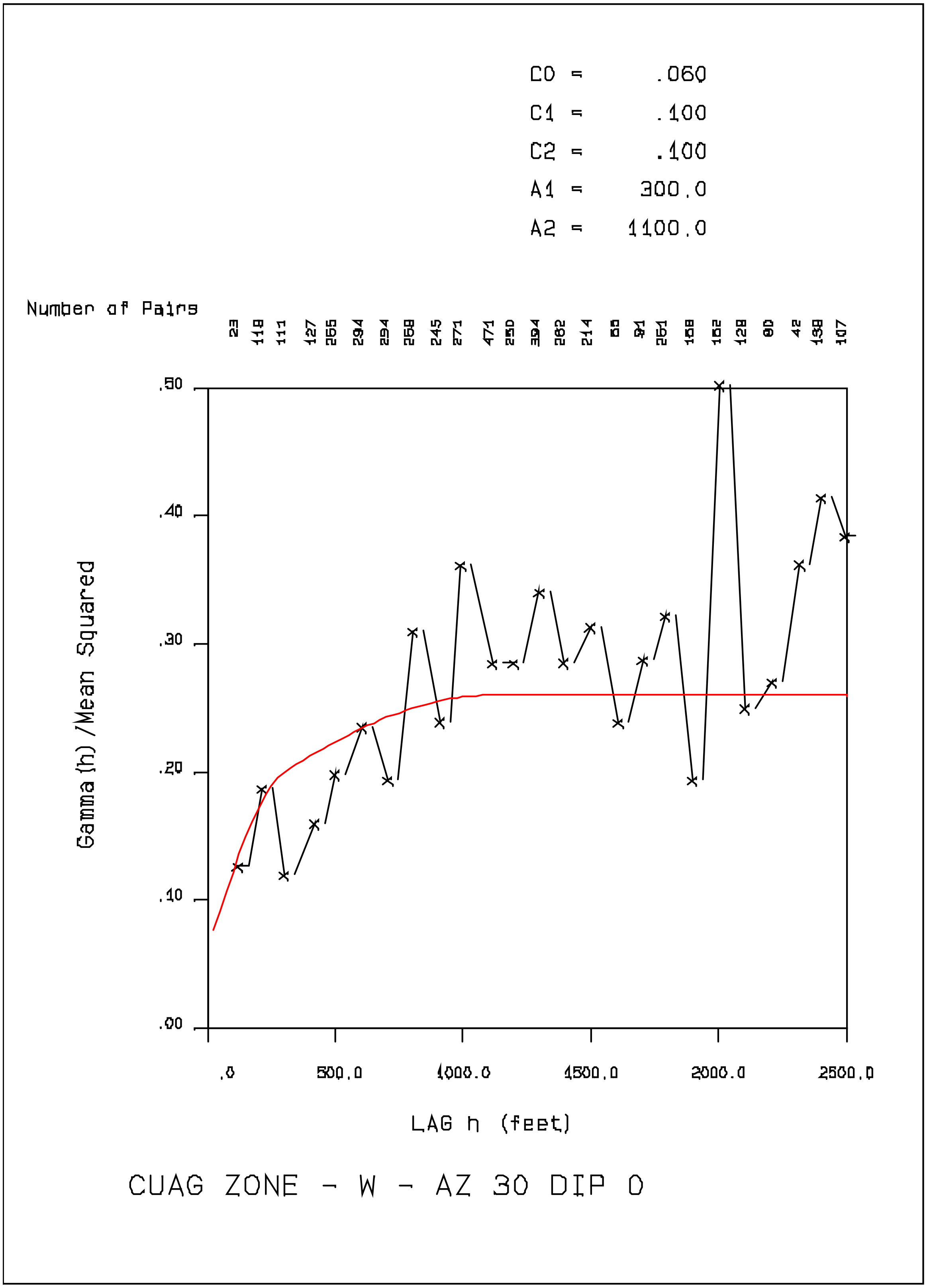


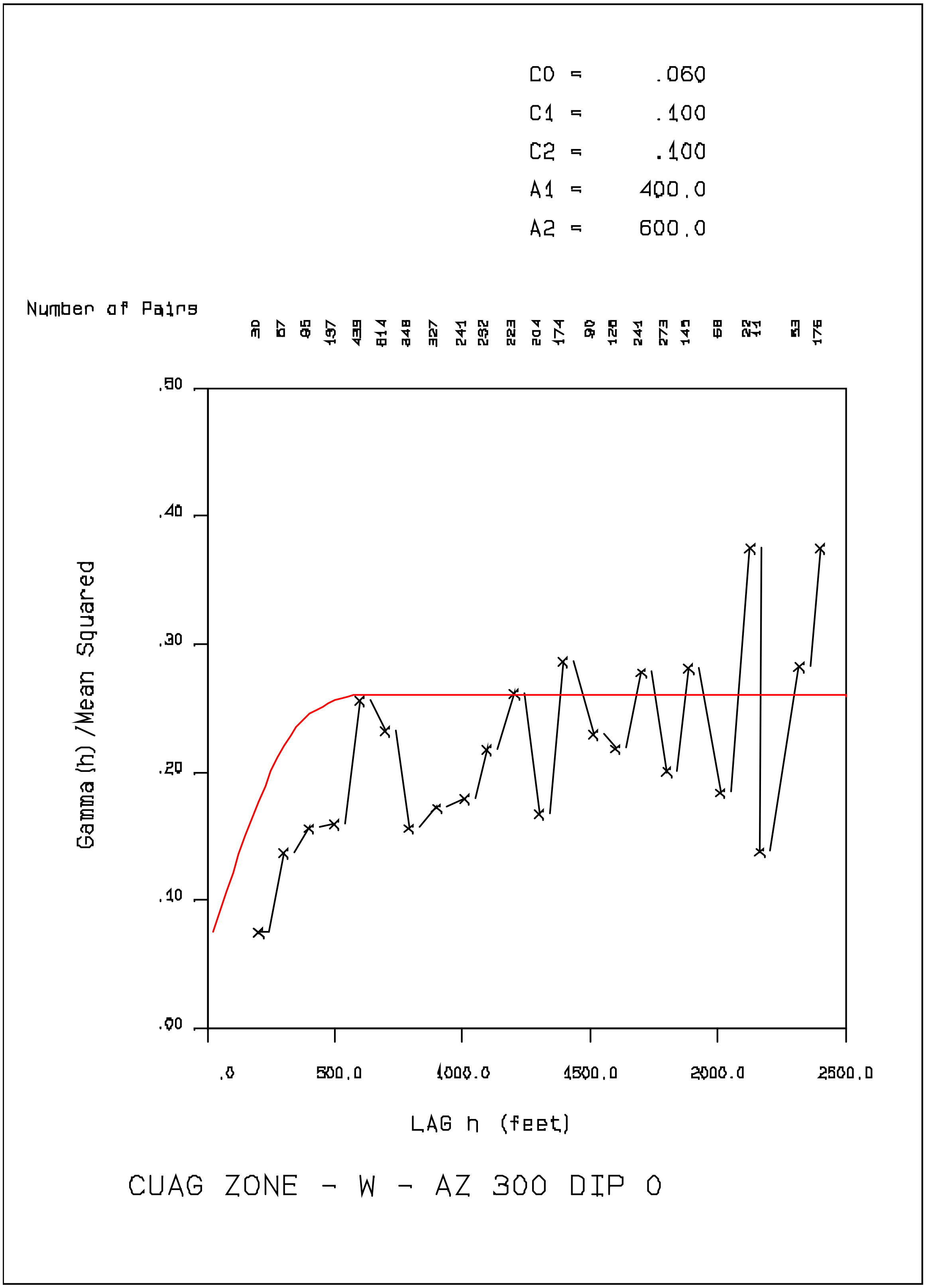


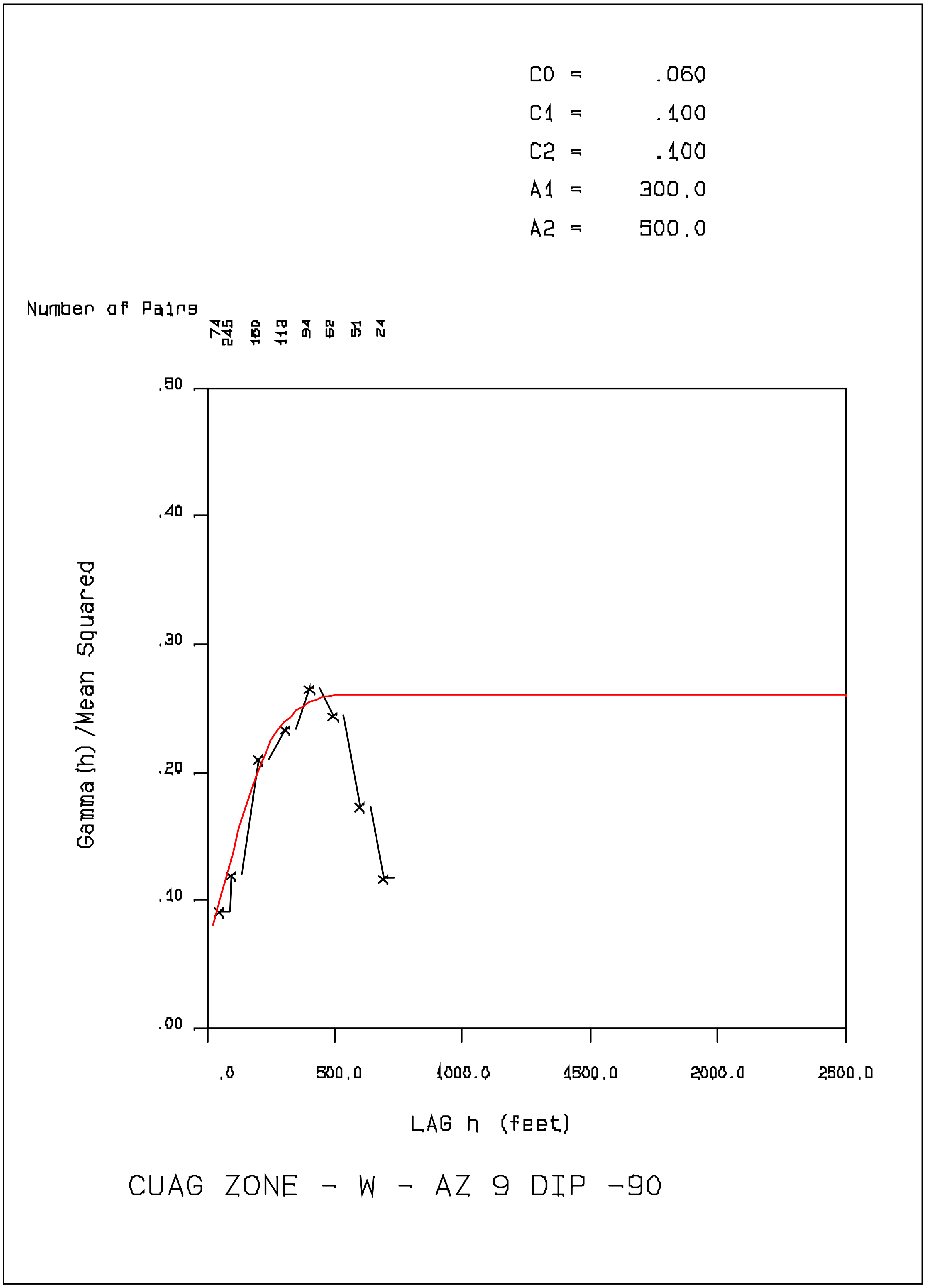
MO ZONE - AG - AZ 330 DIP 0


C0 = .060
C1 = .150
C2 = .140
A1 = 450.0
A2 = 700.0
Number of Pairs
117 368 279 224 185 145 120 103 63 40 31 22 12
Gamma (h) /Mean Squared
.50
.40
.30
.20
.10
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
MO ZONE - AG - AZ 0 DIP -90

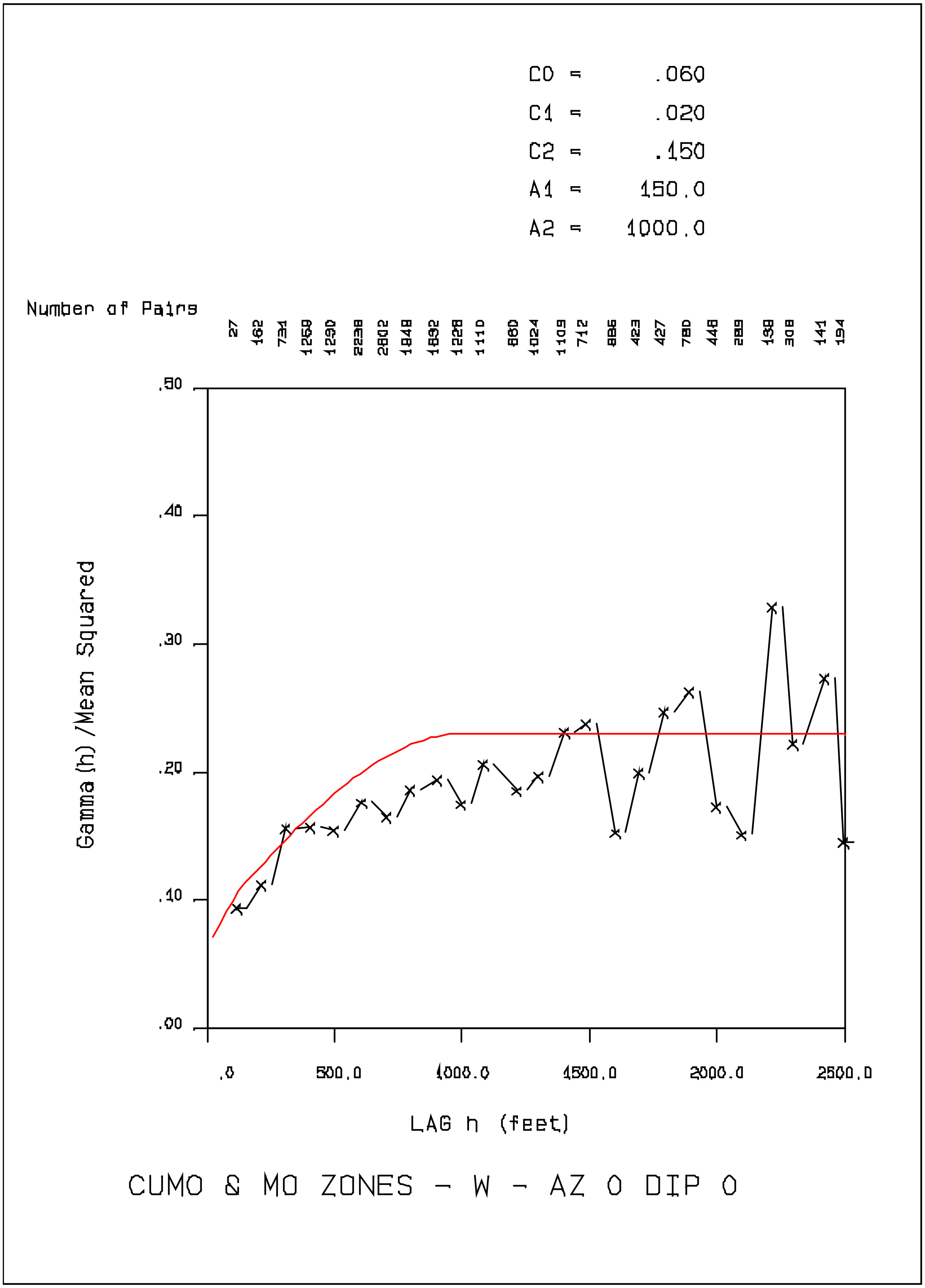
A4.7 – W in Cu-Ag Zone:

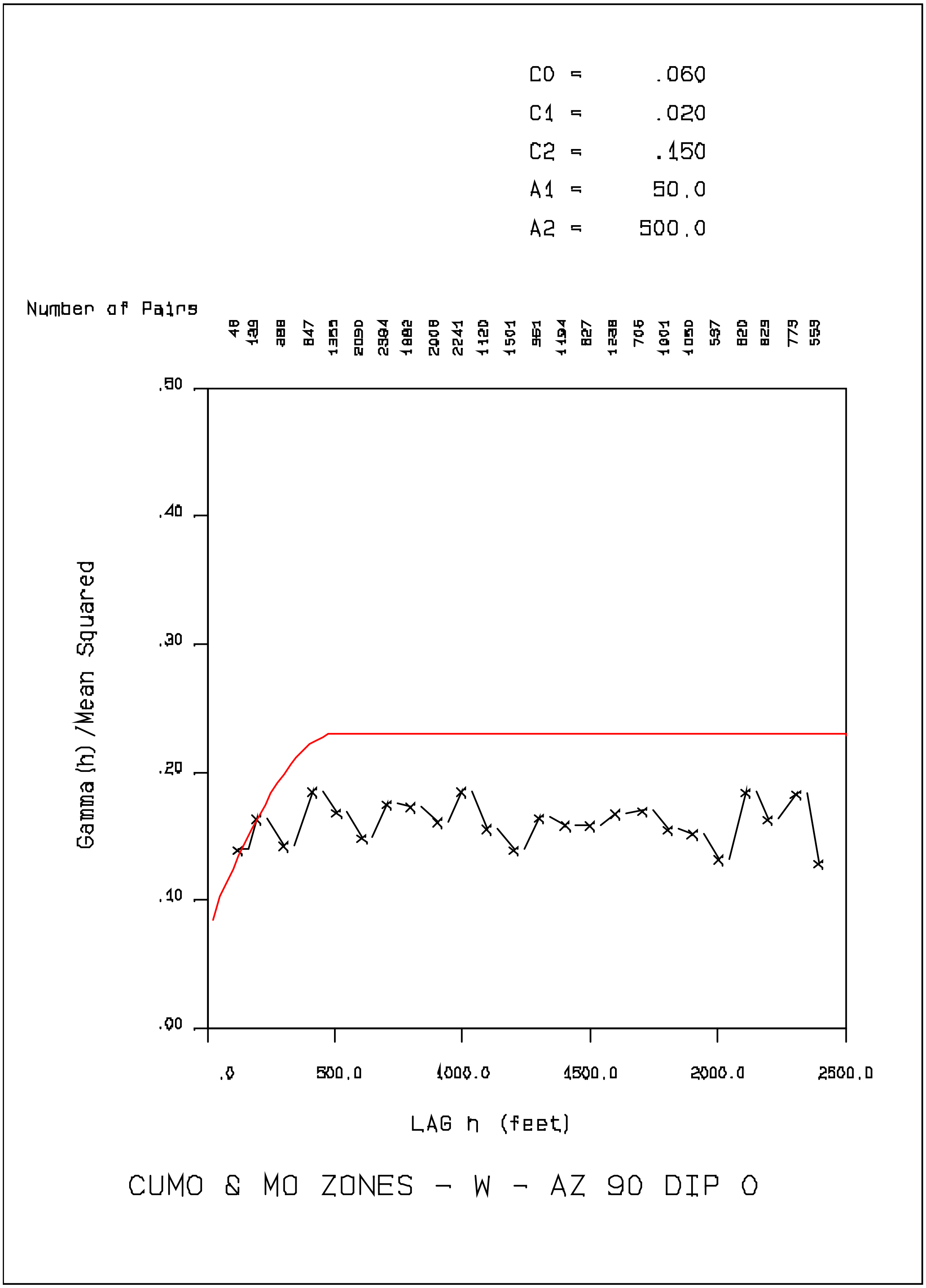


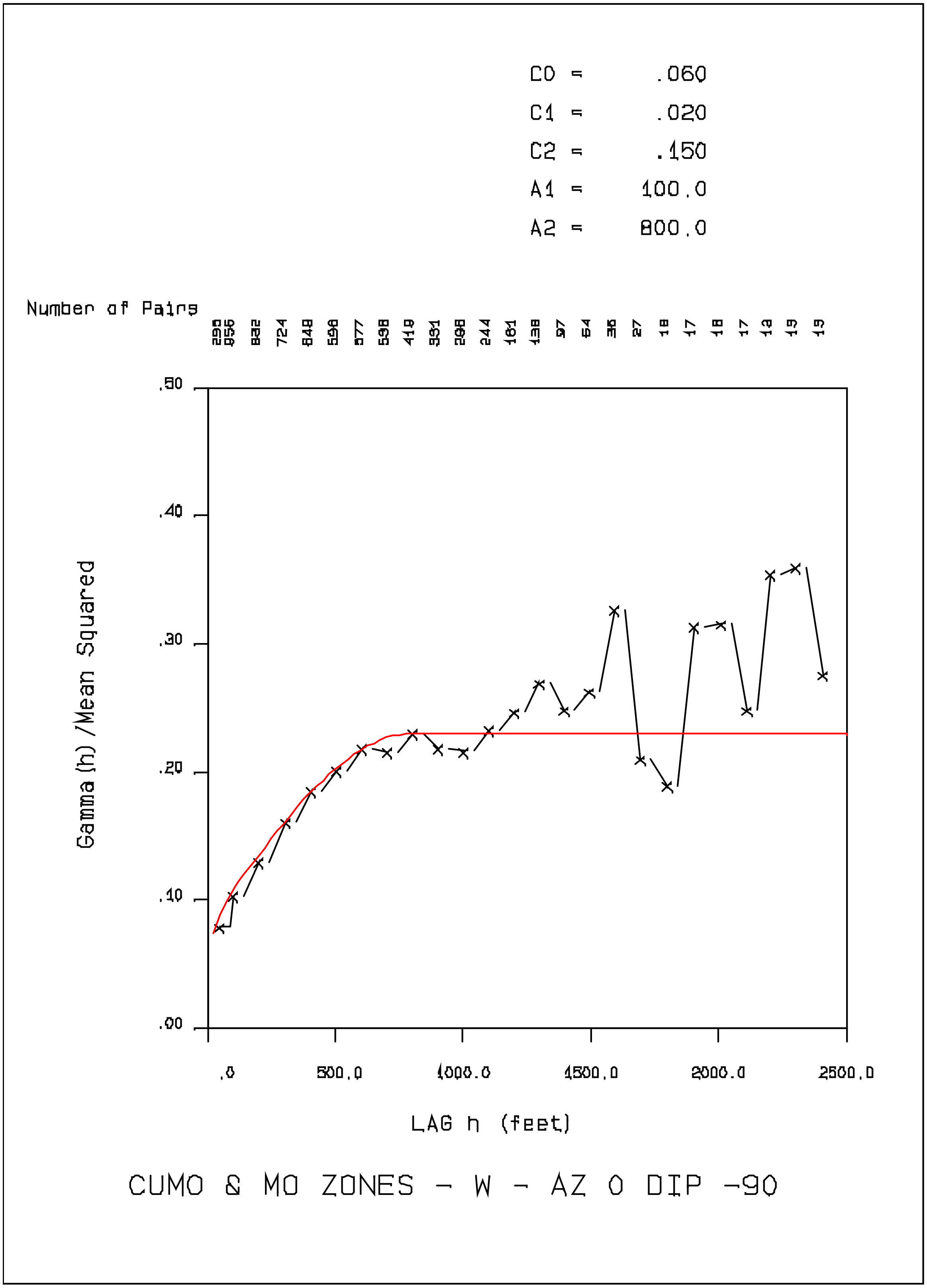

C0 = .060
C1 = .100
C2 = .100
A1 = 400.0
A2 = 600.0
Number of Pairs
30 57 95 137 439 614 348 327 241 232 223 204 174 90 126 241 273 145 58 22 11 53 176
.50
.40
.30
.20
.10
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG ZONE - W - AZ 300 DIP 0


C0 = .060
C1 = .100
C2 = .100
A1 = 300.0
A2 = 500.0
Number of Pairs
74
245
150
113
94
62
51
24
.50
.40
.30
.20
.10
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG ZONE - W - AZ 9 DIP -90

A4.8 – W in CuMo and Mo Zones:




C0 = .060
C1 = .020
C2 = .150
A1 = 50.0
A2 = 500.0
Number of Pairs
48 139 388 847 1355 2050 2384 1882 2008 2241 1120 1501 961 1194 827 1238 706 1001 1050 537 820 829 779 559
.50
.40
.30
.20
.10
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUMO & MO ZONES - W - AZ 90 DIP 0


C0 = .060
C1 = .020
C2 = .150
A1 = 100.0
A2 = 800.0
Number of Pairs
295 956 832 724 648 596 577 538 419 331 298 244 161 138 97 64 36 27 18 17 18 17 13 13 13
.50
.40
.30
.20
.10
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUMO & MO ZONES - W - AZ 0 DIP -90

Appendix 5: Scatter Plots showing Results from Historic Data Verification

A5.1 – Skyline original MoS_2 versus Skyline duplicate MoS_2 from duplicate drill core

A5.2 – Skyline original MoS_2 versus Skyline duplicate MoS_2 from rejects

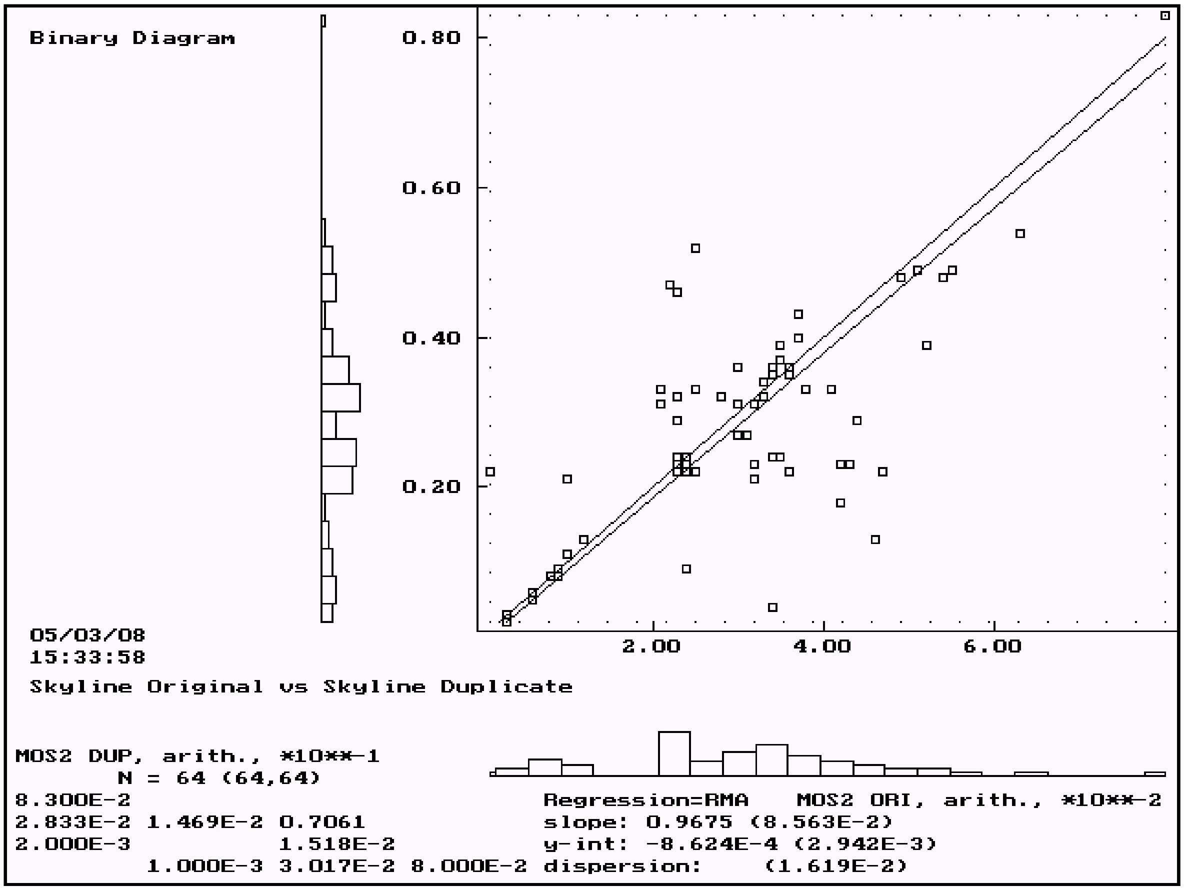
A5.3 – Skyline original MoS_2 versus Skyline duplicate MoS_2 from pulps

A5.4 – Skyline original MoS_2 versus Amax check MoS_2 on pulps from drill core

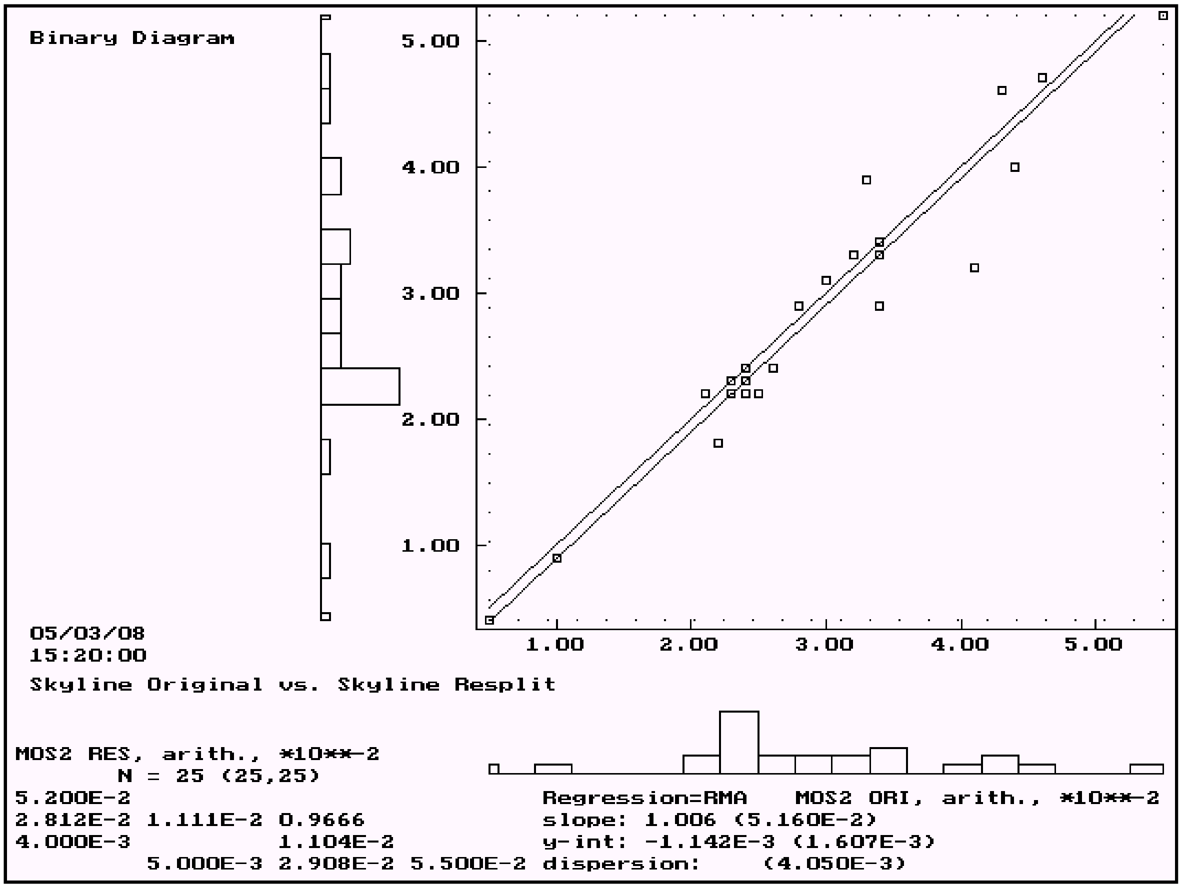
A5.5 – Skyline original MoS_2 versus Amax check MoS_2 on RC cuttings

A5.6 – Skyline original MoS_2 versus Hazen check MoS_2 on pulps

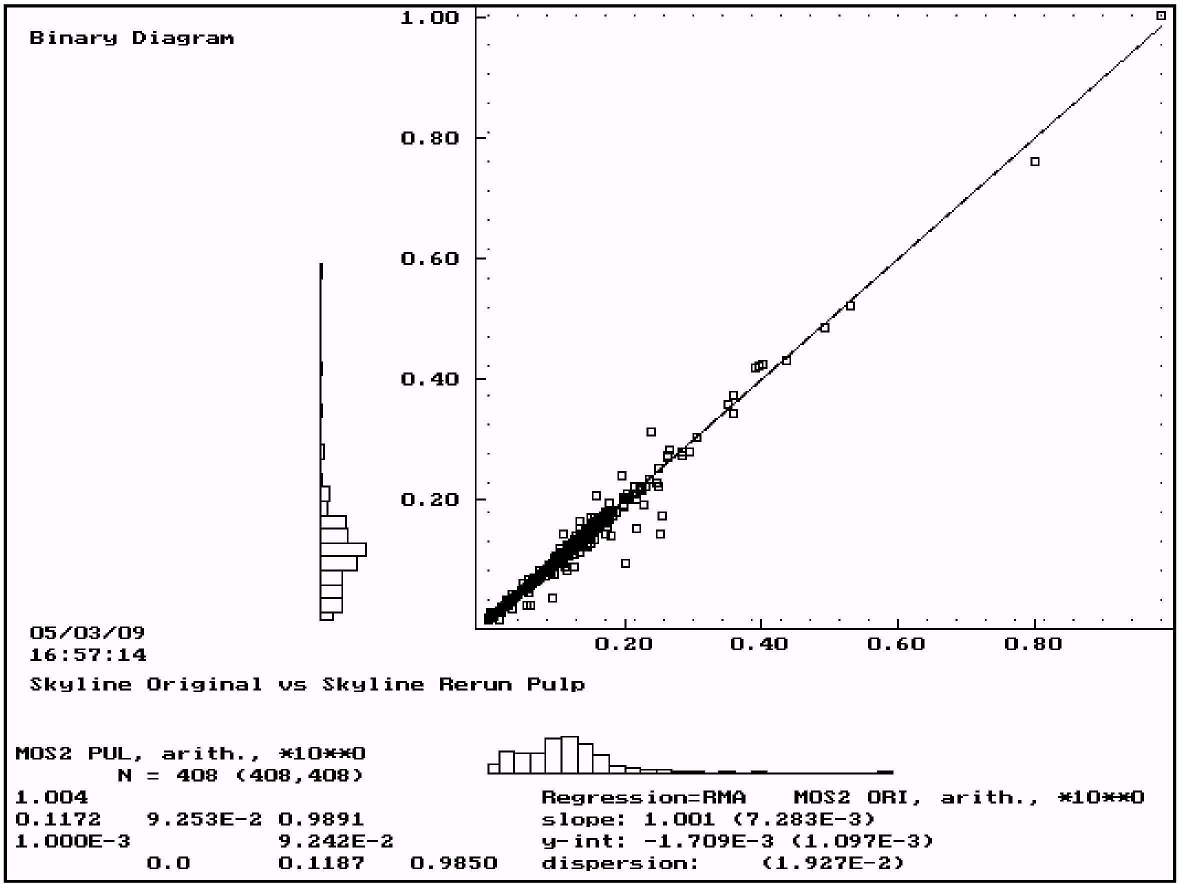
A5.1 – Skyline Original MoS_2 versus Skyline Duplicate MoS_2 from duplicate drill core



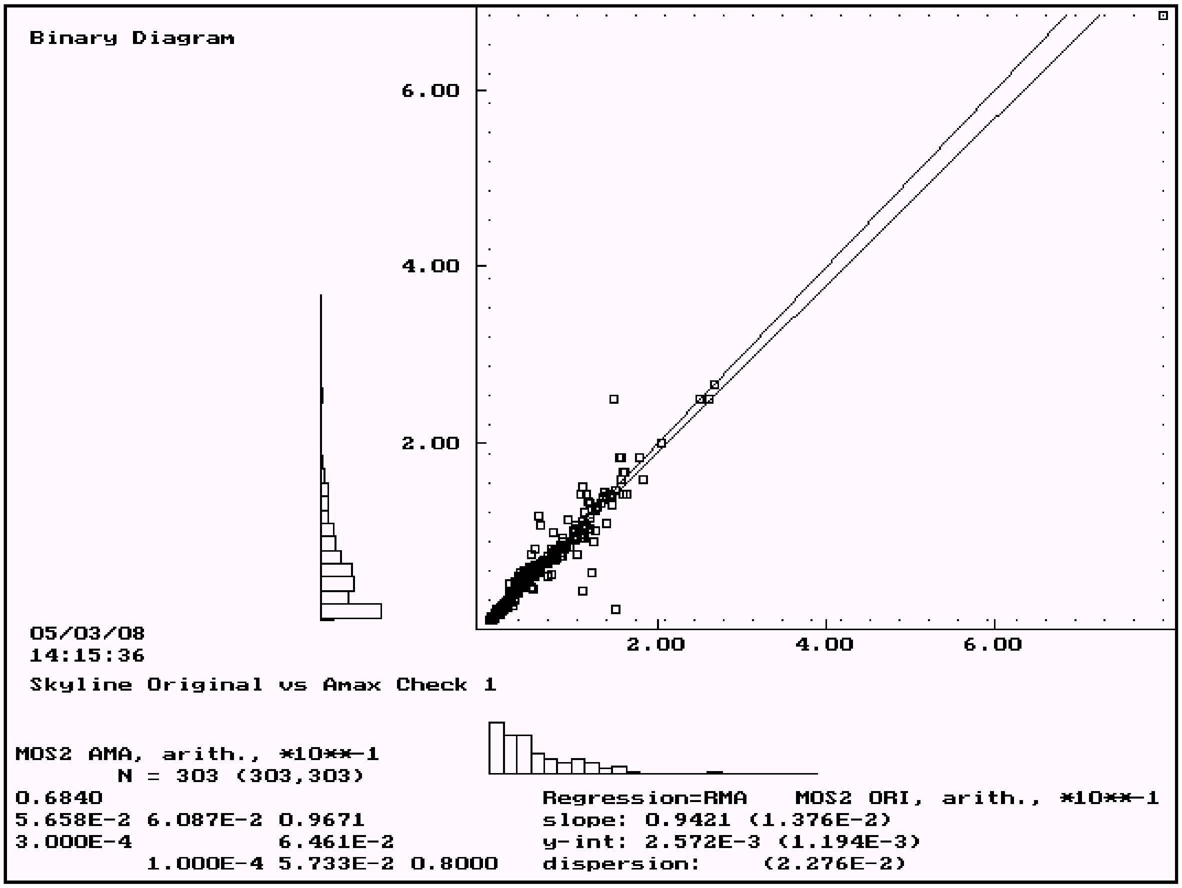
A5.2 – Skyline Original MoS_2 versus Skyline Duplicate MoS_2 from rejects



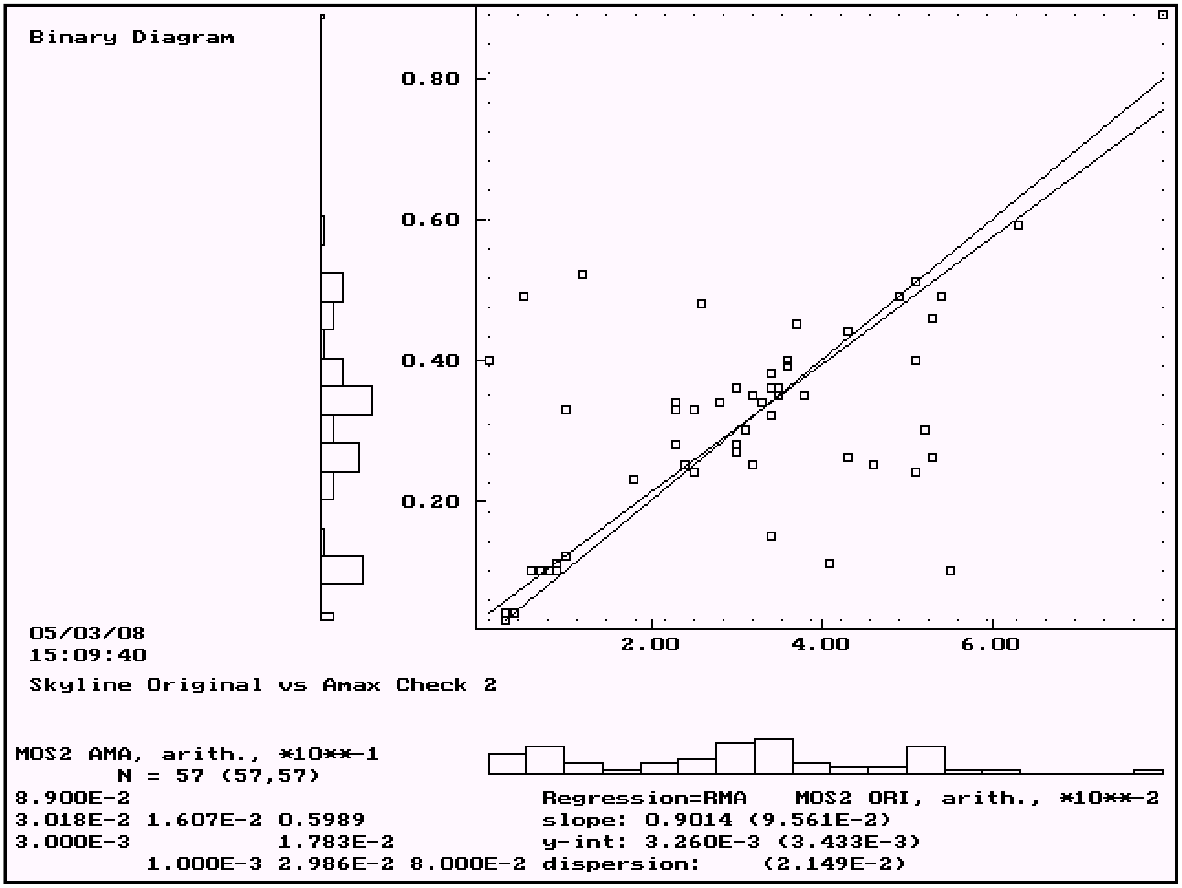
A5.3 – Skyline Original MoS_2 versus Skyline Duplicate MoS_2 from pulps



A5.4 – Skyline Original MoS_2 versus Amax Check MoS_2 on pulps from drill core



A5.5 – Skyline Original MoS_2 versus Amax Check MoS_2 on RC Cuttings



A5.6 – Skyline Original MoS_2 versus Hazen Check MoS_2 on pulps