American CuMo Mining Corporation

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RESOURCE ESTIMATE UPDATE JUNE 2011, UPDATED JUNE 2012

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Mosquito Consolidated Gold Mines Limited: CUMO Project
Project No. AU3568 / V1002

Resource Estimate Update
June 2011

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This report has been prepared by Snowden Mining Industry Consultants (‘Snowden’) on behalf of Mosquito Consolidated Gold Mines Limited.

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1 Summary

This Technical Report refers to the Cumo Project, a copper-molybdenum mineral exploration project located in 37 miles (60 km) northeast of Boise, Idaho, USA. Situated in a historic lode gold camp with recorded production of 2.8 million ounces, molybdenite mineralization was not discovered in this area until 1963 by Amax Exploration. After conducting surface sampling in 1964, Amax dropped the property.

It was subsequently explored by Curwood Mining Company, Midwest Oil Corporation (later Amoco Minerals Company), Amax and then Climax Molybdenum Company, a subsidiary of Amax Inc. Drilling was done between 1969 and 1982 for a total of 10,980.7 metres (36,025.8 feet) in 22 diamond drillholes. A grade-tonnage estimate of 1.5 billion tons (1.36 billion tonnes) at 0.092% MoS_2 (Not compliant with NI43-101) was estimated using a 0.05% MoS_2 cutoff by block modelling in 1983 by Climax.

The property was re-staked in 1998 by Cumo Molybdenum Mining Inc. and optioned to Mosquito Consolidated Gold Mines Ltd in 2004. Kobex Resources Ltd optioned the property from Mosquito in 2005 and commenced drilling in 2006. In late 2006, Mosquito resumed control and has since completed the 2006, 2007 and 2008 exploration drilling program and has completed 14,729 metres (44,188 feet) of drilling in 19 diamond drillholes. During 2009 and 2010 Mosquito drilled 12 more drillholes (22,968 ft), for improving the resource categorization and better understand the 3D extend of the deposits.

The CUMO deposit is located at the south-western 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. In terms of potential total contained molybdenum, based on the historical data, CUMO ranks fourth among all porphyry Cu-Mo deposits when included in the 2005 USGS list of world porphyry copper deposits.

Mosquito's work has revealed the presence of four distinct metal zones within the deposit (called cuag, cumo, mo, msi) covered by a weathering zone which has been traditionally called the oxide zone. Three of these zones were previously interpreted by Amax as distinct ore shells that were produced by separate intrusions. Re-interpretation of geology, alteration and the down-hole histograms for Cu, Ag and Mo have shown the metal zones are part of a single, large, concentrically zoned system with an upper copper-silver zone, called "cuag" zone in this report. This is underlain by a transitional copper-molybdenum"cumo" zone, which is in turn underlain by a lower molybdenum-rich "mo" zone, and a low grade copper molybdenum "msi" zone .

Three-dimensional modelling, of the above zonations as provided by Mosquito indicates the current area being drilled is located on the north side of a much larger system of which only a small part (1 km or 3000 feet) has been drilled.

A resource estimate update was completed at the request of Mosquito based on a total of 54 diamond drillholes totalling 99,404 ft. Of these, 12 diamond drillholes were completed in 2009 and 2010.

A geological model separating the CUMO Deposit into five geological domains was produced by Mosquito geologists. In addition major fault blocks were identified both by assay data and by marker beds. Assays were tagged by one of five geologic domains:

- a weathered zone called the “oxide” domain
- a near surface copper-silver zone, called the “cuag” domain
- a deeper copper-molybdenum zone, called the “cumo” domain
- and a still deeper molybdenum zone, called the “mo” domain
- all of which are underlain by a potassic-silica zone with lower grade molybdenum and copper, referred to as the “msi” domain.

Statistics on each variable in each domain led to the capping of assays based on the grade distribution within that domain. Review of the grades near the boundaries led to the use of soft and hard boundaries between the five geological domains.

Uniform downhole 20 ft. composites were produced for each domain. For variography the major post-mineralization fault blocks were transferred back to their original position using marker beds. Semi-variograms were produced for each variable within each domain based on the samples original pre-fault locations and the soft boundaries.

A block model with block dimensions of 100 ft x 100 ft x 50 ft. was superimposed on the mineralized domains. Grade was interpolated into blocks by ordinary kriging. A tonnage, short ton, factor was determined for each domain based on multiple assigned specific gravity determinations.

Blocks have been classified as an Indicated Resource or Inferred Resource based on a number of criteria including the data quality and distribution, geological confidence and continuity, spatial continuity and estimation quality.

To take into account the four main economic metals, Mo, Cu, Ag, W a Recoverable Value (RCV) was calculated for each block based on reasonable metal prices, estimated grades and estimated recoveries in each of five domains. The resource is summarized for RCV cut-offs. Metal recoveries and metal prices used in the calculation of RCV are presented in Table 1.1. The RCV formula used for each domain is presented in Table 1.2.

Table 1.1 Metal price used in RCV definition and Pit shell

Element	Metal price used in the model for RCV	Metal price used for pit shell optimization
Mo	$16/lb	$25/lb
Cu	$2/lb	$3/lb
Ag	$12/oz	$20/oz
W	$7/lb	$10/lb

Table 1.2 RCV formula for each geolgical domain

Zone	Density	RCV
oxide	12.24	(230.202*MOCAP)+(25.2*CUCAP)+(0.2275*AGCAP)
cumo	12.3	(264.73*MOCAP)+(35.7*CUCAP)+(0.273*AGCAP)+(WCAP*0.0049)
cuag	12.18	(247.47*MOCAP)+(28.56*CUCAP)+(0.2625*AGCAP)+(WCAP*0.0049)
mo	12.32	(273.365*MOCAP)+(30.24*CUCAP)+(0.192*AGCAP)+(WCAP*0.0049)
msi	12.44	(269*MOCAP)+(30.24*CUCAP)+(0.192*AGCAP)+(WCAP*0.0049)
		MOCAP = estimated Mo grade (top-cut)
		CUCAP = estimated Cu grade (top-cut)
		AGCAP = estimated Ag grade (top-cut)
		WCAP = estimated W grade (top-cut)

In order to determine the quantities of material offering reasonable prospects for economic extraction from an open pit, Snowden used a Whittle pit optimizer to evaluate the profitability of each resource block based on selected optimization parametres from the Thompson Creek mine (i.e. a comparable an open pit molybdenum project located in Idaho).

The optimization parametres included: ore mining and processing costs of $7.52 per processed ton, overall pit slope angles of 45 degrees, metallurgical recoveries Table 1.3, and appropriate dilution and offsite costs and royalties. The metals prices for the pit shell for different metals are presented in Table 1.1. The reader is cautioned that the results from the conceptual pit optimization work are used solely for the purpose of reporting Mineral Resources that have “reasonable prospects” for economic extraction by an open pit and do not represent an attempt to estimate Mineral Reserves.

Table 1.3 Metallurgy recoveries for each gelogical domain

Zone	Cu%	MoS_2%	Ag %	W %
oxide	60%	80%	70%	35%
cuag	68%	85%	73%	35%
cumo	87%	92%	78%	35%
mo	80%	95%	55%	35%
msi	80%	95%	55%	35%

A summary of the global Mineral Resources at a range of cut-off grades by classification level are presented in Table 1.4 and Table 1.5. The selected cut-off grade for reporting of Mineral Resources of $2.50 RCV is highlighted in bold. Grade-Tonnage curves by resource classification level are shown in Figure 1.1 and Figure 1.2.

Table 1.4 Indicated Resource in CUMO deposit

RCV Cutoffs($)	Million TONS	MoS_2 (%)	Cu (%)	Ag (gpt)	W (gpt)	RCV	Ag (M.Oz)	Cu (M.lb)	MoO_3 (M.lb)	W (M.IB)
0.50	2,052	0.058	0.080	2.37	38.63	18.94	142	3291	2144	159
1.00	2,052	0.058	0.080	2.37	38.63	18.94	142	3291	2144	159
1.50	2,050	0.058	0.080	2.37	38.67	18.96	142	3290	2144	159
2.00	2,037	0.059	0.081	2.38	38.89	19.07	141	3281	2144	158
2.50	**2,026**	**0.059**	**0.081**	**2.38**	**39.06**	**19.16**	**141**	**3272**	**2143**	**158**
3.00	2,006	0.059	0.081	2.39	39.29	19.32	140	3254	2142	158
3.50	1,978	0.060	0.082	2.40	39.67	19.55	138	3225	2139	157
4.00	1,952	0.061	0.082	2.41	39.97	19.76	137	3205	2136	156
4.50	1,923	0.062	0.083	2.42	40.28	19.99	136	3179	2131	155
5.00	1,894	0.062	0.083	2.42	40.59	20.23	134	3147	2126	154
5.50	1,868	0.063	0.084	2.43	40.83	20.43	132	3122	2121	153
6.00	1,841	0.064	0.084	2.44	41.07	20.65	131	3089	2114	151

Table 1.5 Inferred Resource in CUMO deposit

RCV Cutoffs($)	Million TONS	MoS_2 (%)	Cu (%)	Ag (gpt)	W (gpt)	RCV ($)	Ag (M.Oz)	Cu (M.lb)	MoO_3 (M.lb)	W (M.lb)
0.50	5,024	0.026	0.063	1.94	9.12	9.43	285	6331	2352	92
1.00	4,719	0.028	0.066	1.99	9.71	9.99	273	6235	2352	92
1.50	4,448	0.029	0.068	2.03	10.28	10.52	263	6059	2351	91
2.00	4,232	0.031	0.069	2.06	10.77	10.97	255	5868	2349	91
2.50	**3,947**	**0.033**	**0.070**	**2.11**	**11.51**	**11.60**	**243**	**5553**	**2343**	**91**
3.00	3,754	0.035	0.071	2.13	12.02	12.05	233	5305	2337	90
3.50	3,525	0.037	0.071	2.13	12.66	12.62	219	4971	2328	89
4.00	3,258	0.039	0.070	2.11	13.46	13.35	201	4547	2312	88
4.50	3,067	0.042	0.069	2.10	14.03	13.92	188	4231	2296	86
5.00	2,915	0.043	0.068	2.10	14.43	14.40	178	3974	2279	84
5.50	2,768	0.045	0.067	2.09	14.79	14.88	169	3717	2259	82
6.00	2,623	0.047	0.065	2.10	15.08	15.39	160	3424	2237	79

Figure 1.1 Indicated Resource Grade-Tonnage Curve

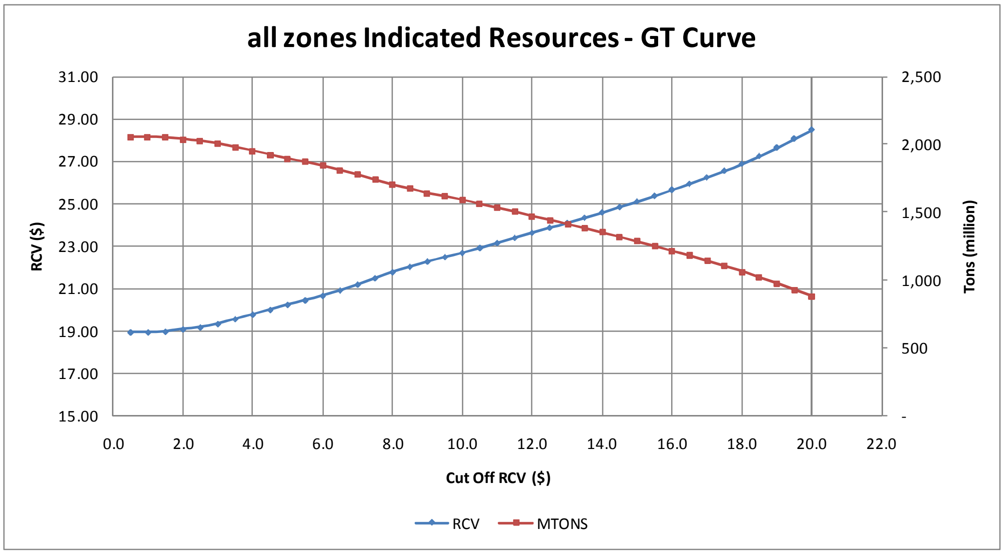
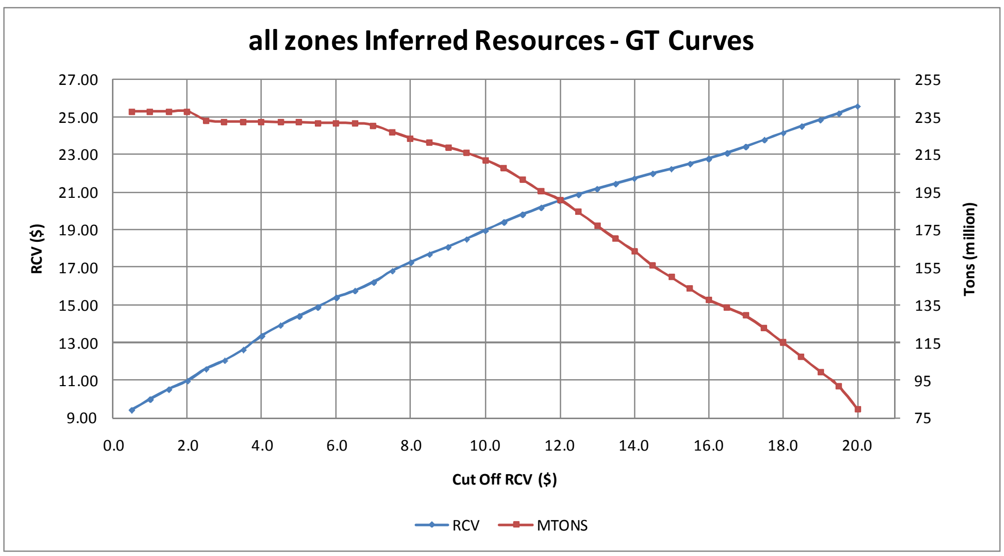


Figure 1.2 Inferred Resource Grade-Tonnage Curve



Based on the Mineral Resources defined to date, it is recommended that the CUMO project be advanced to feasibility stage. The drilling program recommended by Ausenco in November 2009 is in progress and is proposed to be carried out over a minimum time frame of two years at an estimated cost of $72.5 million.

2 Introduction

This Technical Report has been prepared by Snowden Mining Industry Consultants (Snowden) for Mosquito Consolidated Gold Mines Ltd. (Mosquito), in compliance with the disclosure requirements of Canadian National Instrument 43-101 (NI43-101), to disclose relevant information about the CUMO Property (CUMO), Boise County, Idaho, USA.

Snowden was asked by Mosquito to provide an update to the 2009 Resource calculation prepared by Holmgren and Giroux in 2009; This report can be found on SEDAR.

In 2009 Mosquito Consolidated published a preliminary economic assessment (PEA) and a NI 43-101 technical report. Since the completion of the 2009 PEA Mosquito has conducted an additional 22,968 ft of drilling from 12 drillholes. Mosquito requested Snowden to prepare a new resource estimate for CUMO to incorporate the results from the 12 additional holes into the resource estimate.

Unless otherwise stated, information and data contained in this report or used in its preparation has been provided by Mosquito Consolidated Gold Mines Ltd. This Technical Report has been compiled from sources cited in the text by Mr. Ivor Jones, MSc (Geo), FAusIMM (CP), Group General Manager Geosciences at Snowden.

This report reproduces several sections from the 2009 preliminary economic assessment as the information contained has not changed significantly (with the exception of the size of the Mineral Resource) since the time of filing. These sections are identified and reproduced here for clarity and completeness.

Mr. Jones, Mr. Scott, Mr. Kehmeier and Mr. Khoury are Qualified Persons as defined by NI 43-101. Mr. Jones and Mr. Chapman of Snowden visited the CUMO Project site on 23 November 2010. Mr. Scott, Mr. Kehmeier and Mr. Khoury have not visited site. The responsibilities of each author are provided in Table 2.1.

Table 2.1 Responsibilities of each co-author

Author	Responsible for section/s
Ivor Jones (Snowden)	1: Summary; 2: Introduction; 3: Reliance on other experts; 4: Property description and location, 5: Accessibility, climate, local resources, infrastructure, and physiography; 6: History; 7: Geological setting; 8: Deposit types; 9: Mineralisation; 10: Exploration; 11: Drilling; 12: Sampling method and approach; 13: Sample preparation, analyses, and security; 14: Data verification; 15: Adjacent properties; 16.1.2 Metallurgical testing - Sample selection; 17: Mineral Resource and Mineral Reserve estimates; 18.5: Environmental Considerations; 19: Interpretation and conclusions; 20: Recommendations
Reproduced sections (Ausenco/Vector, 2009) Author: Richard Kehmeier	18.1: Mining Operation Design; 18.8: Mining Capital Costs; and 18.12.1: Mining Operating Costs;
Reproduced sections (Ausenco/Vector, 2009) Author: Charles Khoury	18.2: TSF Design; 18.3: Waste Dump Design; 18.4: Low-Grade Ore Stockpile Design; and 18.10: Tailings Capital Costs;
Reproduced section (Ausenco/Vector, 2009) Author: Kevin Scott	16 (except sub-section 16.1.2): Mineral processing and metallurgical testing; 18.6: Taxed and Royalties, 18.7: Capital Cost Estimate ; 18.9: Process Plant Capital Costs; 18.11: Capital Cost Exclusions;, 18.12.2: Mining Operation Cost Comparison; 18.12.3: Process Plant Operating Costs; and 18.13: Economic Analysis;

This report is intended to be used by Mosquito Consolidated Gold Mines Ltd. to update the previously filed National Instrument 43-101 Report (CUMO Property Preliminary Economic Assessment, 2009) and to report an updated Mineral Resource estimate on the CUMO Property in Boise County, Idaho.

3 Reliance on other experts

The preparation of this report has been based upon public and private information provided by Mosquito regarding the property.

This report and the information contained within are based on work conducted and managed by Ivor Jones, the Qualified Person responsible for the updated resource estimate.

The authors believe that the information provided and relied upon for preparation of this report is accurate at the time of compiling this report and that the interpretations and opinions expressed in it are reasonable and based on current understanding of mining and processing techniques and costs, economics, mineralization processes and the host geologic setting. The authors have made reasonable efforts to verify the accuracy of the data relied upon in this report.

The results and opinions expressed in this report are conditional upon the aforementioned information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions made herein the Authors reserve the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to the Authors subsequent to the date of this report.

4 Property description and location

Information in this section has been excerpted from "Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report" dated May 13, 2009 and filed on SEDAR on May 14, 2009 and updated to reflect changes to the status of the project.

4.1 General

The CUMO property is located approximately 59 kilometres (37 miles) northeast of the city of Boise, Idaho, USA (Figure 7.1) 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 7.2). 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, 876,000N (NAD 27 CONUS).

4.2 Mineral tenure

The property consists of 345 unpatented and un-surveyed contiguous mining lode claims covering an area of approximately 7,100 acres. Most of the claims consist of full-sized, 600 ft by 1,500 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.

The mining lode claims are named the CUMO #1-8 claims, New CUMO #9-61 claims, CUMO #62-188 claims, and SF 1-167 claims. The original claim blocks, CUMO 1 to 8 were recorded December 11, 1998, and later abandoned and re-staked as New CUMO 1-8. However, a title search revealed that a significant portion of the New CUMO 1-8 claims may not be valid since they were staked over existing claims that have since been dropped.

As a result, to ensure clear title, the New CUMO 1-8 claims were abandoned and re-staked as CUMO 1-8 with a recording date of March 28, 2005. The New CUMO 9-55 and 57-61 claims were staked by Western Geoscience Inc. and recorded December 1, 2004. The New CUMO 62-188 claims were staked by CUMO Molybdenum Inc. and recorded between May 16 and 24, 2005. The SF 1-167 were staked by CUMO Molybdenum Inc. and recorded between May 24 and June 24, 2005.

In Idaho, staked claims expire annually on September 1. Therefore, the annual fee of $135/claim must be paid to the BLM prior to Aug 31, 2011 or all claims will expire on Sept 1, 2011. At $135/claim, the company must make annual payments to the BLM of US$46,575 to keep all claims in good standing.

Figure 4.1 CUMO Property Location Map (Supplied by Mosquito)

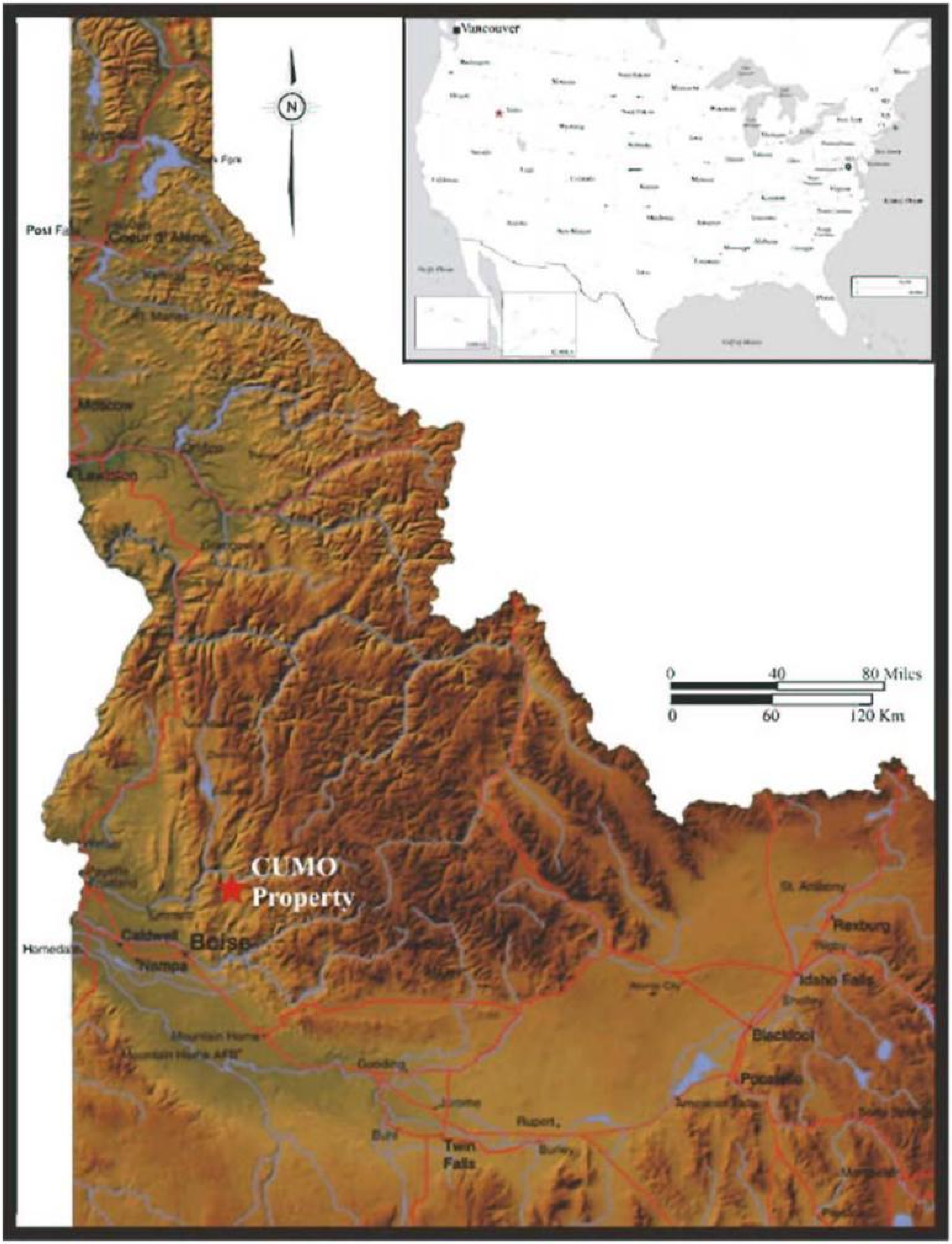
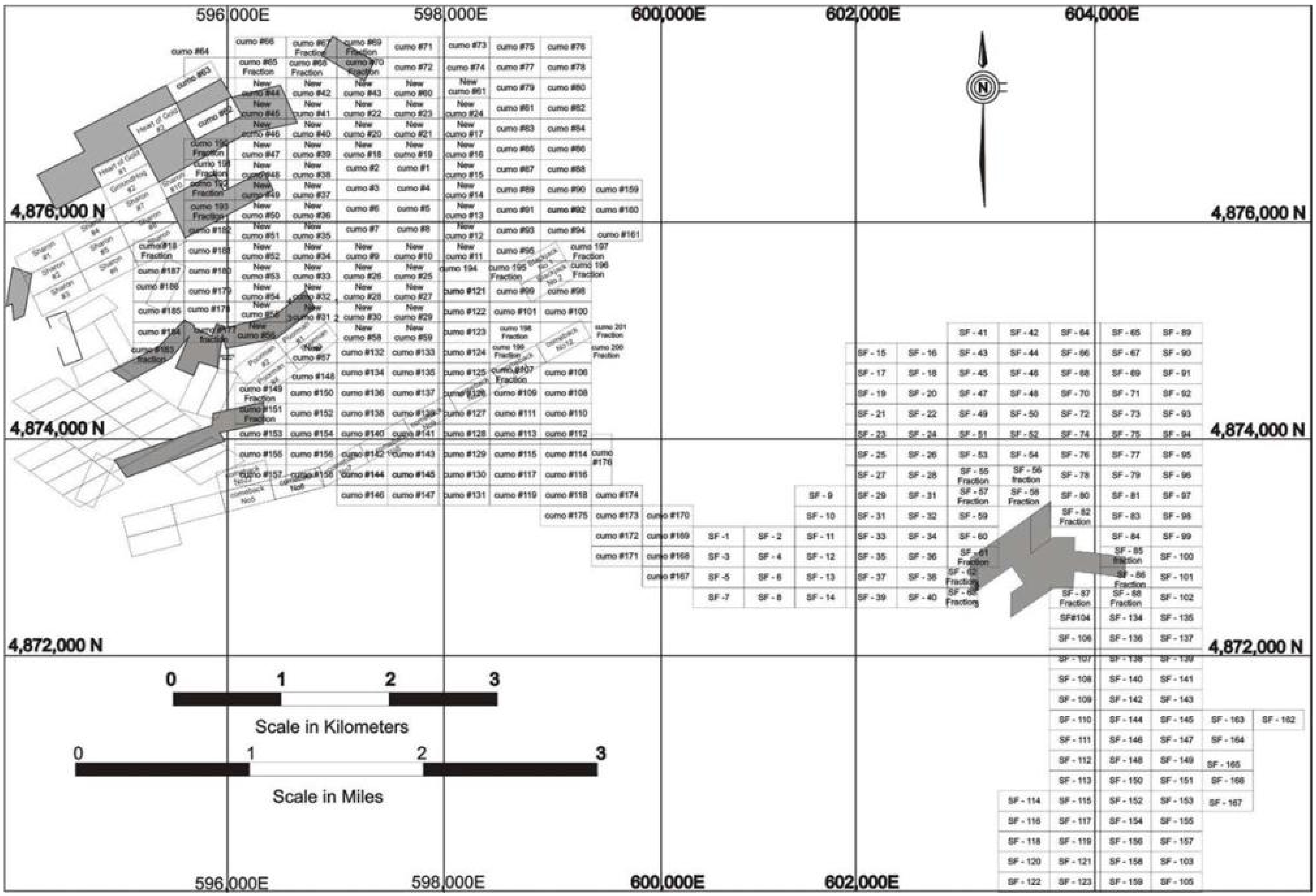


Figure 4.2 Claim location map for the CUMO property (Supplied by Mosquito)



4.3 Permits

Exploration on Federal lands requires a permit 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 permit is Categorical Exclusion (CE). This is the least intense disturbance and requires some public notification. Snowden understands that track mounted auger drilling and no new road clearing would fit in this category according to United States Forest Service (USFS) personnel.
- Environmental assessment (EA) requires an in-depth study with 30 days for public comment, plus additional time for appeal. Snowden understands that drilling with an RC rig using water, new road construction, etc., would require this level of permit. USFS personnel suggest that one year may be required to receive a permit. 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. Several aspects should be factored into timing of exploration plans.

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 is currently permitted under an existing Categorical Exclusion (CE) permit. An application for a Water Use Permit for 2008 has also been filed with the Idaho Department of Water Resources.

In January 2007, a plan of operations was submitted for an expanded program involving construction of new roads for drill access, and the US Forest service has given notice that an Environmental Assessment (EA) will be required for that program.

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 (FONSI) was delivered by the United States Forest Service (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 Mosquito 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 5 years.

Snowden understands that the status at the time of this report is that the Idaho conservation league filed a challenge in the “United States District Court for the District of Idaho” on December 15th 2011: “Plaintiffs Idaho Conservation League, Idaho Rivers United, and Golden Eagle Audubon Society seek summary judgment reversing and remanding the Forest Service’s February 2011 approval of the CuMo Mine Exploration Project, in the upper Grimes Creek watershed of the Boise National Forest.” The US Forest Service was named as defendant while Mosquito Mining Corp was named as Intervener Defendant. Mosquito has worked through the litigation process and filed a response brief and reply brief. The US Forest service has also filed response and reply briefs. The Idaho Conservation League also filed a reply brief. The case is now with a Judge who will decide whether or not to overturn the FONSI and Plan of Operations based on the merits of the briefs.

4.4 Environmental aspects

Snowden understands that Mosquito has completed (February 14, 2011) an exhaustive and extensive Environmental Assessment of the project area and that the results indicate the lack of any endangered species or plant life in the CuMo deposit area. Environmentally, Cumo sits near the head of Grimes creek which is a tributary of the Boise River.

The area has been heavily disturbed over the past 100 years through logging and mining activities. Grimes Creek is considered a contaminated creek by the Idaho State Department of Environmental Quality (DEQ) due to the presence of placer gold tailings, arsenic, lead, and zinc.

Starting about 8 kilometres (5 miles) downstream and covering approximately 37 km (23 miles), there are extensive dredge tailings from historic (1900's) gold placer dredges. Several occurrences of metallic mercury have been reported in these old placer tailings.

In addition to the extensive Placer tailings, there are numerous old gold mines with waste dumps containing copper, lead, zinc and arsenic that are currently and have been leaching into the environment for over 75 years. These are relatively small and thus have had no major effects, but are significant.

Snowden understands that testing by SGS shows that the tailings from the ¾ tonne metallurgical bulk sample are not acid generating and contain no harmful quantities of toxic elements.

Overall, the project is not located in a wilderness or pristine area, but a heavily disturbed and mildly contaminated area of historic mining and logging.

5 Accessibility, climate, local resources, infrastructure and physiography

This section is excerpted and updated from "Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report" dated May 13, 2009 and filed on SEDAR on May 14, 2009.

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 65 kilometres (40 miles) to the town of Banks, Idaho, and then east on the Banks Lowman Road towards the town of Garden Valley for approximately 16 kilometres (10 miles). One mile east of Garden Valley is a secondary road heading south across the Payette River. The western most edge of the CUMO claim block is approximately 16 kilometres (10 miles) from Garden Valley.

Alternatively, access can be gained by traveling northeast from Boise along Highway 21 to 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 1,700 metres (5,100 feet) to 2,400 metres (7,200 feet).

The climate is defined by summer temperatures to a maximum of 38° C (100° F) and cold, windy winters with lows to of -23° C (-10° F). Precipitation is moderately light with an average rainfall of 0.76 metres (30 inches) and an average snowfall of approximately 3.6 metres (140 inches).

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 32 kilometres (20 miles) by road 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 can be conducted year-round, due to the established road and its proximity to infrastructure. The property is large enough to support all future exploration or mining operations including facilities and potential waste disposal areas.

6 History

This section is excerpted and updated from “Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report” dated May 13, 2009 and filed on SEDAR on May 14, 2009.

6.1 Ownership history

On October 13, 2004, Mosquito Consolidated Gold Mines Ltd completed an “Option to Purchase Agreement” with CUMO Molybdenum Mining Inc. to purchase 8 unpatented mineral claims located in Boise County, Idaho, USA known as “CUMO Molybdenum Property”. As part of the original CUMO and Mosquito agreement, all claims acquired within 8 kilometres (5 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.

On January 21, 2005, Mosquito Consolidated Gold Mines Ltd 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 Mosquito Consolidated Gold Mines Ltd.

6.2 Exploration history and evaluation

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 three kilometres of the CUMO property, some of which have recorded minor past production of base and precious metals.

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 4 kilometre (2.5 mile) rough access road was constructed in 1964 to facilitate collection of rock samples and geologic mapping. The property was subsequently dropped due to 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 (4 rotary holes initially, followed by 6 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 with AMOCO as operator.

During the period 1973 to 1981, the AMAX-AMOCO JV completed 9,395 metres (30,822 feet) of drilling (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, 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

Company	Year	Holes	Footage	Metres	Comments
Midwest	1969	4	378	115.2	rotary holes shallow due to water
	1970	0	653	199	2 rotary holes deepened with core 400 depth
	1971	1	2,251	686.1	one core hole deepened further to 1884 ft
	1972	3	1,892	576.7	one core hole deepened from 810-1416 ft
Amax	1974	1	805	245.4	hole 9-9A
	1975	1	2,382	726	hole 10
	1976	2	4,343	1,323.70	one vertical, other 1340ft @-45
	1977	3	5,861	1,786.40	3 vertical DDH 1804-2124 feet deep
	1978	3	6,774	2,064.70	3 vertical DDH 2132-2361 feet deep
	1979	2	4,823	1,470.00	vertical DDH to 2543 foot depth
	1980	3	2,630	801.6	RC holes
	1981	3	3,204	976.6	vertical DDH 1,000 to 1,193 foot depths
	Total	**26**	**35,996**	**10,919.4**	

In 1983, Climax Molybdenum transferred its interest in the property to AMAX Exploration Inc. and no further work appears to have been done on the property until 2006 when diamond drilling was undertaken on behalf of Kobex Resources Ltd and Mosquito Resources Corp. Refer to section 10 for details.

A summary of significant intersections from the historical CUMO drilling is given in Table 6.2. The description of the calculation and formulas used for producing the metal equivalents and the recovered metal value for the intersections is covered in section 10.2.

Table 6.2 Significant Intersections from Historical CUMO Drilling

Hole Name	From feet	To feet	Length feet	From metres	To metres	Length metres	Zone	recv Cu Equiv.	recv MoS_2 equiv.	MoO_3 Equiv lbs	MoS_2 %	Cu %	Ag Gms/T	Re ppm	W ppm	Recovered Metal value US$
C71-01	231.00	1884.00	1653.00	70.40	574.20	503.80	main	0.50	0.07	1.30	0.06	0.12	2.59	0.00	45.62	20.86
C71-01	390.00	470.00	80.00	118.90	143.30	24.40	sub	0.77	0.11	2.01	0.10	0.14	2.56	0.00	43.75	32.15
C71-01	1700.00	1884.00	184.00	518.20	574.20	56.10	sub	0.71	0.10	1.87	0.10	0.08	1.21	0.00	53.57	29.98
C72-05	450.00	1416.00	966.00	137.20	431.60	294.40	main	0.53	0.08	1.38	0.06	0.13	4.46	0.00	74.87	22.16
C74-09	460.00	804.60	344.60	140.20	245.20	105.00	main	0.64	0.09	1.69	0.08	0.12	7.16	0.00	71.40	26.99
C75-10	220.00	2160.00	1940.00	67.10	658.40	591.30	main	0.68	0.10	1.79	0.10	0.05	1.43	0.00	48.25	28.67
C76-11	140.00	2428.30	2288.30	42.70	740.10	697.50	main	0.52	0.08	1.38	0.07	0.05	1.55	0.00	36.25	22.01
C76-11	1300.00	1960.00	660.00	396.20	597.40	201.20	sub	0.84	0.12	2.20	0.13	0.03	0.77	0.00	57.58	35.25
C76-12	98.30	1430.00	1331.70	29.90	435.90	405.90	main	0.33	0.05	0.86	0.04	0.06	1.66	0.00	44.77	13.71
C77-13	680.00	1804.00	1124.00	207.30	549.90	342.60	main	0.76	0.11	2.00	0.11	0.05	1.98	0.00	49.33	31.99
C77-14	780.00	2123.80	1343.80	237.70	647.30	409.60	main	0.79	0.12	2.08	0.11	0.06	1.84	0.00	65.38	33.21
C77-14	1200.00	1960.00	760.00	365.80	597.40	231.60	sub	1.03	0.15	2.69	0.15	0.06	1.91	0.00	73.83	43.07
C77-15	600.00	1933.20	1333.20	182.90	589.20	406.40	main	0.78	0.11	2.05	0.11	0.06	1.73	0.00	56.89	32.86
C77-15	1260.00	1880.00	620.00	384.00	573.00	189.00	sub	1.00	0.15	2.62	0.15	0.02	0.75	0.00	69.10	41.84
C78-16	1000.00	2131.70	1131.70	304.80	649.70	344.90	main	0.64	0.09	1.67	0.09	0.04	1.86	0.00	32.20	26.73
C78-17	1160.00	2281.50	1121.50	353.60	695.40	341.80	main	0.49	0.07	1.29	0.06	0.08	2.55	0.00	39.92	20.71
C78-18	1400.00	2361.00	961.00	426.70	719.60	292.90	main	0.90	0.13	2.37	0.13	0.08	2.71	0.00	40.86	37.97
C79-19	120.00	2280.00	2160.00	36.60	694.90	658.40	main	0.73	0.11	1.91	0.10	0.08	2.27	0.00	48.94	30.49
C79-20	165.00	1800.00	1635.00	50.30	548.60	498.30	main	0.56	0.08	1.47	0.07	0.11	3.83	0.00	52.04	23.52
C81-25	190.00	1011.00	821.00	57.90	308.20	250.20	main	0.58	0.08	1.51	0.07	0.13	2.42	0.00	58.22	24.17
C81-25	740.00	1011.00	271.00	225.60	308.20	82.60	sub	0.72	0.10	1.88	0.09	0.14	2.98	0.00	84.32	30.13
C81-26	30.00	750.00	720.00	9.10	228.60	219.50	main	0.42	0.06	1.10	0.03	0.18	7.58	0.00	28.14	17.59
C06-27	120.00	1849.00	1729.00	36.60	563.60	527.00	main	0.60	0.09	1.57	0.08	0.06	1.60	0.02	49.48	25.10
C06-27	1080.00	1849.00	769.00	329.20	563.60	234.40	sub	0.89	0.13	2.33	0.13	0.04	0.99	0.04	58.84	37.26
C06-28	50.00	1690.00	1640.00	15.20	515.10	499.90	main	0.69	0.10	1.81	0.10	0.07	1.92	0.05	54.29	29.02
C06-28	840.00	1240.00	400.00	256.00	378.00	121.90	sub	1.06	0.16	2.79	0.16	0.03	0.98	0.09	67.82	44.63
C07-29	190.00	2230.00	2040.00	57.90	679.70	621.80	main	0.74	0.11	1.94	0.10	0.08	2.13	0.05	53.46	31.01
C07-29	1180.00	1790.00	610.00	359.70	545.60	185.90	sub	1.11	0.16	2.92	0.17	0.04	1.20	0.08	36.91	46.71
C07-30	40	2,386.00	2,346.00	12.2	727.3	715.1	main	0.75	0.110	1.97	0.108	0.06	2.05	0.04	41.28	$31.53
C07-30	1,180.00	1,988.00	808	359.7	605.9	246.3	sub	1.21	0.177	3.19	0.185	0.04	1.46	0.07	37.03	$51.01
C07-31	22	2,104.00	2,082.00	6.7	641.3	634.6	main	0.48	0.070	1.26	0.064	0.07	1.76	0.02	43.25	$20.17

Hole Name	From feet	To feet	Length feet	From metres	To metres	Length metres	Zone	recv Cu Equiv.	recv MoS_2 equiv.	MoO_3 Equiv lbs	MoS_2 %	Cu %	Ag Gms/T	Re ppm	W ppm	Recovered Metal value US$
C07-31	780	1,540.00	760	237.7	469.4	231.6	sub	0.57	0.083	1.49	0.081	0.05	1.45	0.03	45.31	$23.89
C07-32	22	2,104.00	2,082.00	6.7	641.3	634.6	main	0.79	0.115	2.06	0.109	0.09	2.26	0.04	61.14	$33.04
C07-32	780	1,540.00	760	237.7	469.4	231.6	sub	0.93	0.135	2.43	0.129	0.1	2.62	0.05	77.08	$38.88
C07-33	721.8	2,094.00	1,372.20	220	638.3	418.2	main	0.24	0.035	0.64	0.026	0.07	2.01	0.01	47.68	$10.20
C07-33	1,980.00	2,094.00	114	603.5	638.3	34.7	sub	0.64	0.094	1.68	0.084	0.1	2.68	0.03	67.05	$26.92
C07-34	140	1,769.00	1,629.00	42.7	539.2	496.5	main	0.30	0.044	0.80	0.034	0.08	2.3	0.01	53.46	$12.79
C07-34	1,550.00	1,769.00	219	472.4	539.2	66.8	sub	0.57	0.083	1.49	0.074	0.09	2.36	0.02	67.14	$23.84
C08-35	120	2,640.00	2,520.00	36.6	804.7	768.1	main	0.43	0.062	1.12	0.057	0.06	1.73	0.02	36.79	$17.91
C08-35	420	2,640.00	2,220.00	128	804.7	676.7	sub	0.47	0.068	1.22	0.062	0.07	1.69	0.02	38.98	$19.60
C08-35	1,730.00	2,640.00	910	527.3	804.7	277.4	sub	0.62	0.090	1.62	0.089	0.05	1.37	0.03	35.4	$25.92
C08-36	560	2,488.00	1,928.00	170.7	758.3	587.7	main	0.61	0.089	1.60	0.088	0.05	1.42	0.03	34.09	$25.66
C08-36	920	2,488.00	1,568.00	280.4	758.3	477.9	sub	0.69	0.101	1.82	0.103	0.04	1.04	0.03	33.42	$29.17
C08-37	60	2,195.00	2,135.00	18.3	669	650.7	main	0.59	0.086	1.55	0.084	0.05	1.67	0.03	42.18	$24.72
C08-37	780	2,130.00	1,350.00	237.7	649.2	411.5	sub	0.69	0.100	1.80	0.104	0.02	1.17	0.04	41.01	$28.81
C08-38	170	2,441.00	2,271.00	51.8	744	692.2	main	0.27	0.039	0.70	0.029	0.06	4.4	0	31.51	$11.17
C08-39	310	2,688.00	2,378.00	94.5	819.3	724.8	main	0.69	0.101	1.81	0.099	0.06	1.38	0.03	51.71	$29.03
C08-39	900	2,390.00	1,490.00	274.3	728.5	454.2	sub	0.82	0.119	2.15	0.122	0.04	1.09	0.04	57.03	$34.36
C08-40	60	2,252.00	2,192.00	18.3	686.4	668.1	main	0.81	0.118	2.12	0.115	0.06	3.79	0.04	46.45	$33.87
C08-40	390	2,080.00	1,690.00	118.9	634	515.1	sub	0.90	0.131	2.36	0.129	0.06	4.27	0.05	45.45	$37.69
C08-40	1,110.00	1,820.00	710	338.3	554.7	216.4	sub	0.98	0.144	2.58	0.142	0.04	7.78	0.06	44.76	$41.34
C08-41	850	2,830.00	1,980.00	259.1	862.6	603.5	main	0.51	0.075	1.34	0.067	0.08	2.23	0.02	42.87	$21.44
C08-41	1,490.00	2,030.00	540	454.2	618.7	164.6	sub	0.77	0.112	2.01	0.107	0.08	2.99	0.03	38.02	$32.20
C08-41	2,490.00	2,830.00	340	759	862.6	103.6	sub	0.55	0.080	1.45	0.077	0.06	1.53	0.03	33.55	$23.13
C08-42	550	2,707.00	2,157.00	167.6	825.1	657.5	main	0.37	0.054	0.97	0.044	0.06	5.81	0.01	25.02	$15.47
C08-42	950	2,707.00	1,757.00	289.6	825.1	535.5	sub	0.40	0.059	1.06	0.047	0.07	6.78	0.01	26.75	$16.89
C08-42	1,970.00	2,707.00	737	600.5	825.1	224.6	sub	0.45	0.066	1.19	0.063	0.05	1.61	0.01	21.22	$19.02
C08-43	165	1,303.00	1,138.00	50.3	397.2	346.9	main	0.39	0.057	1.02	0.044	0.09	4.23	0.02	52.17	$16.29
C08-43	660	820	160	201.2	249.9	48.8	sub	0.57	0.083	1.49	0.071	0.11	3.14	0.03	44.74	$23.82
C08-44	1125	2840	1715	342.9	865.6	522.7	main	0.20	0.030	0.53	0.028	0.02	0.89	0.01	28.71	$8.54
C08-44	2560	2690	130	780.3	819.9	39.6	sub	0.38	0.055	0.99	0.055	0.02	1.47	0.01	20.09	$15.78
C08-45	170	1796	1626	51.8	547.4	495.6	main	0.29	0.042	0.75	0.021	0.15	3.08	0.00	41.75	$11.98
C08-45	1010	1796	786	307.8	547.4	239.6	sub	0.38	0.055	1.00	0.032	0.18	3.05	0.00	39.74	$15.94

6.3 Historical resources and reserves

Based on the 26 drillholes drilled from 1969 to 1981, a block model was constructed in 1983 by Amax, extending between local grid coordinates 17,000 to 25,000 East and 16,000 to 23,000 North. The individual blocks were 100 feet in both the North-South and East-West directions and were 50 feet in height. Blocks were located from 7000 feet down to 3050 feet above sea level. Grades were estimated using 50 foot drillhole assay composites and grade zone boundaries. Kriging was performed within a 1500 foot horizontal search limited to 300 feet vertically (Table 6.3).

Table 6.3 CUMO historical resource, 1983 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
0.1	470	0.131
0.12	280	0.145
0.14	140	0.170

** Note that MoS_2 contains 60% Molybdenum by weight*

The resource estimate by Climax was done prior to the inception of NI 43-101 and does not follow the categories outlined in NI 43-101. There is no distinction between Measured, Indicated and Inferred resources, although Climax classified the tonnage as “well-tested” (24%), “possible” (50%) and “not quantitatively measured” (26%) based on individual block errors (kriging standard deviation).

With respect to the historical estimate:

(i) a qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves; and

(ii) the issuer is not treating the historical estimate as current mineral resources or mineral reserves.

6.4 Production

There is no production from this deposit.

7 Geological setting

This section is excerpted and updated from “Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report” dated May 13, 2009 and filed on SEDAR on May 14, 2009.

7.1 Regional geology

The regional tectonic setting consists of a basement of amalgamated Archean and Paleoproterozoic crystalline terranes 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 superposed 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 a 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 be controlled by 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 foredeep assemblages overlying an Archean basement that is juxtaposed with accreted conjoined terranes. 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.

Figure 7.1 Tectonic map of the western United States (Hildenbrand et al, 2000)

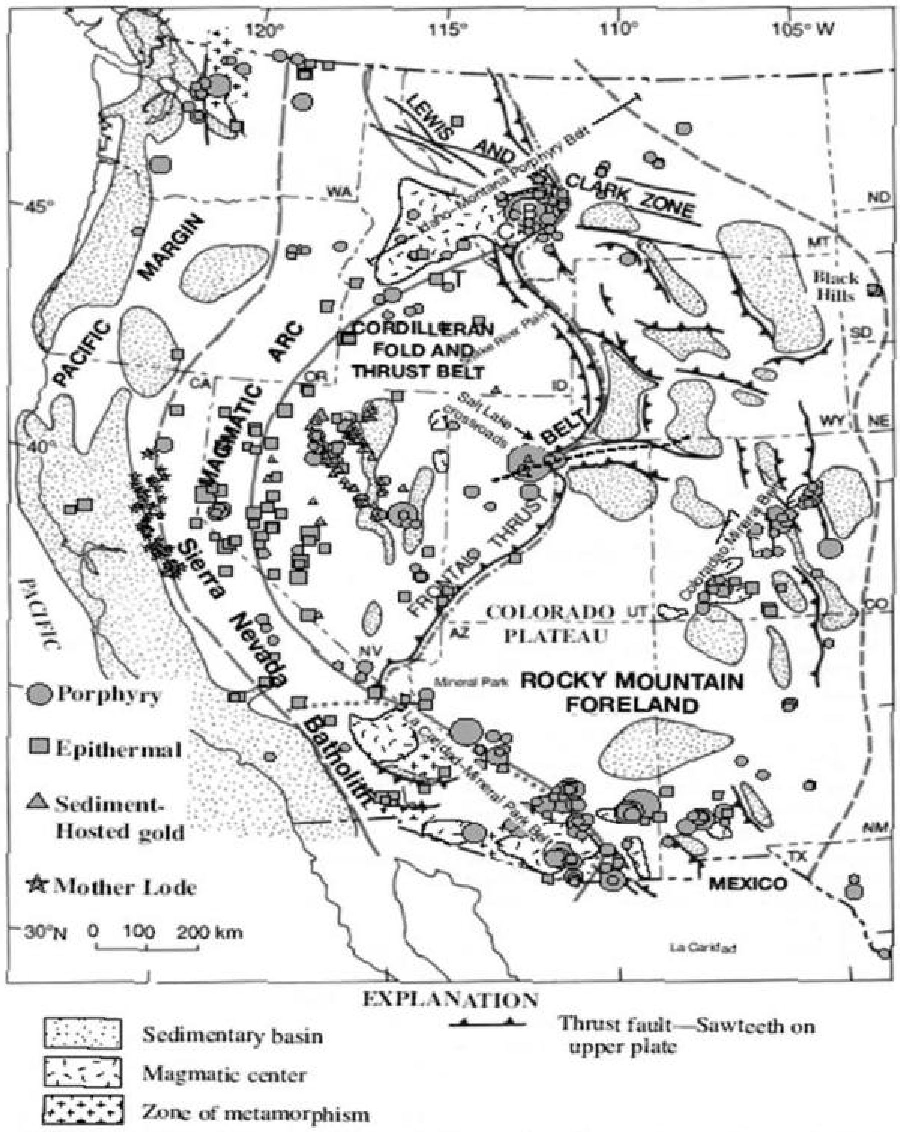
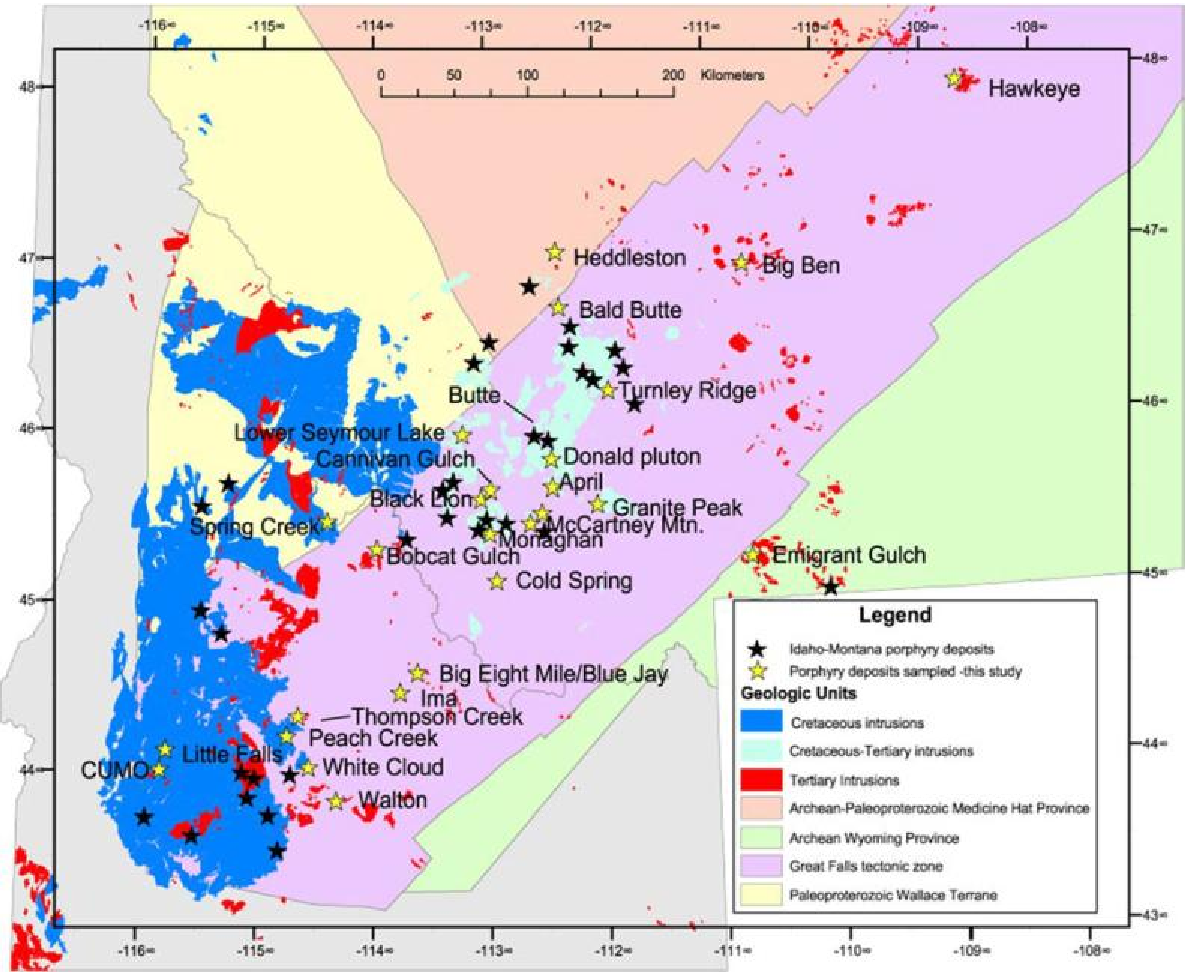


FIG. 1. Map of the western United States cordillera showing ore deposits superimposed on major tectonic elements and Laramide igneous zones, sedimentary basins, and metamorphic belts. The western United States is divided into four generalized geologic provinces (boundaries shown as heavy solid and dashed 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).

Figure 7.2 Distribution of Idaho-Montana Porphyry deposits in relation to the Great Falls Tectonic Zone. (From Lund et al, 2005)



The Trans-Rocky Mountain fault system is a major, deep-seated, northwest trending, intra-continental 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.

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 45-52 Ma 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).

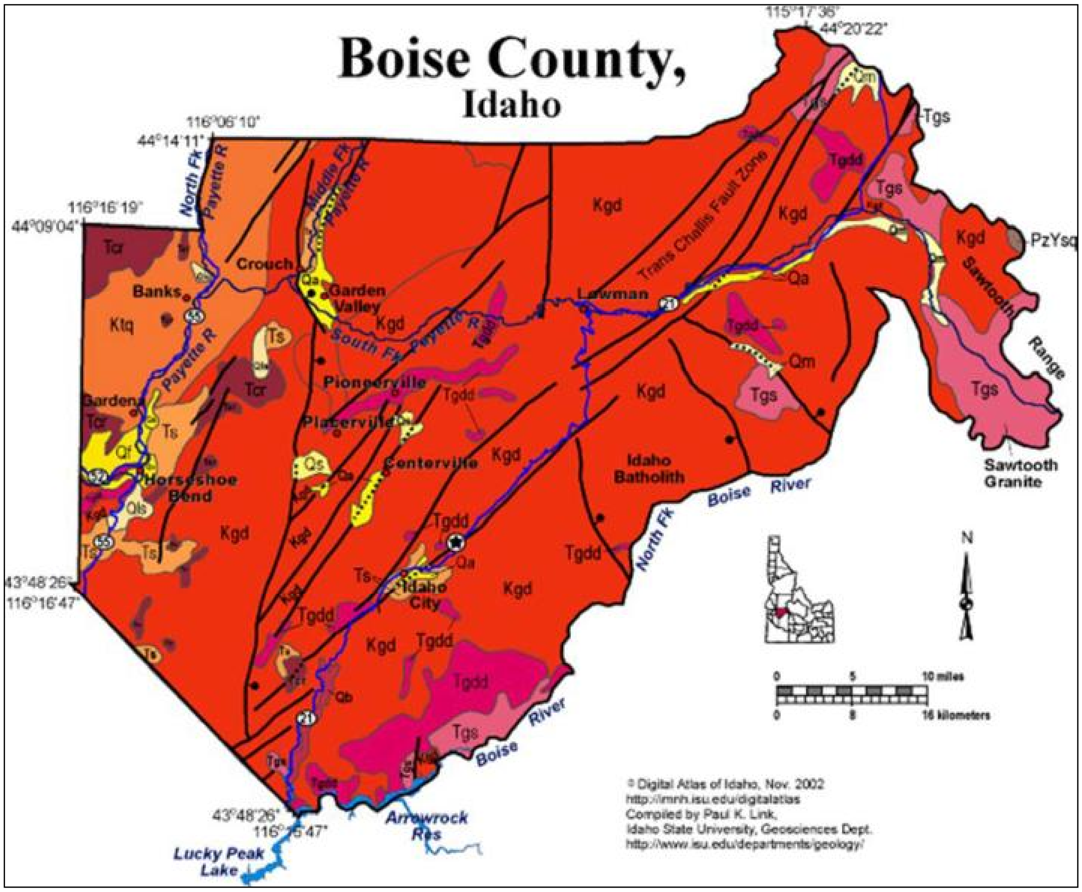
The CUMO deposit is situated within the Atlanta Lobe of the Idaho batholith. The western margin of the Atlanta lobe is strongly folded and metamorphosed into gneissic rocks, which are well exposed near McCall. The western side is composed of tonalite, 95 to 85 million years old. The batholith core is biotite granodiorite; and the eastern side of the lobe is muscovite-biotite granite approximately 76 to 72 million years old. (Digital Atlas of Idaho http://imnh.isu.edu/digitalatlas/geo/bathlith/bathdex.htm)

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 of 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%. Sericite alteration of feldspar is common. Killsgaard and others (1985) report the age of this unit as 82-69 Ma based on potassium-argon dating.

Figure 7.3 Geology of Boise County, Idaho, showing geologic setting of CUMO deposit



Description of Units for Boise County, Idaho

Qa	Quaternary alluvial deposits
Qs	Quaternary surficial cover, including colluvium, fluvial, alluvial fan, lake, and windblown deposits. Included fluveolian cover on Snake River Plain, (Snake River Group).
Qf	Pleistocene silicic volcanic rocks; rhyolite lava and ash-flow tuff (includes Yellowstone Group).
Qls	Quaternary landslide deposits (only Weiser Area).
Ts	Tertiary sedimentary rocks, undifferentiated. Includes Oligocene and Eocene sedimentary rocks in east-central Idaho (Paleogene basins of Janecke). In northern and western Idaho this unit contains Miocene lake and stream deposits formed adjacent to and above the Columbia River and Weiser basalts, which formed dams in stream canyons.
Tcr	Miocene basalt (Columbia River Basalt Group); flood basalt, extensively exposed in western Idaho; fed by fissures, many of which are near the Idaho-Oregon border. Flowed eastward up valleys cut into the Idaho mountains.
Tgs	Eocene granite, pink granite, syenite, rhyolite dikes, and rhyolitic shallow intrusive; last phase of the Challis magmatic event (46 to 44 Ma). Forms craggy scenic mountain landscape in central and northern Idaho.
Tgdd	Eocene granodiorite and dacite porphyry intrusive, also includes diorite and, in northern Idaho, minor granitic rock; intermediate phase of Challis magmatic event (50 to 46 Ma). Summit Creek stock.
Kgd	Cretaceous granitic rocks of the 2 mica suite. Idaho batholith and related plutons; granite and granodiorite that contains both muscovite and biotite. Sodium (Na) rich. Intruded between 80 and 65 Ma.
Ktg	Cretaceous tonalite and quartz diorite; hornblende and biotite bearing early phases of the Idaho batholith. Intruded about 90 to 95 Ma.
PzYsq	Paleozoic/Mesoproterozoic schist and quartzite; age uncertain.

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

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 of 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).

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 feet thick and may exhibit flow banding, whereas rhyolite porphyry dikes can reach 200 feet 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 5, a north-tending 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 to 1982 resulted in subdivision of those five units into 17 separate units as follows in Table 7.1.

Table 7.1 Summary of rock units at CUMO

UNIT	AGE	ROCK TYPE	TEXTURE	Grain Size (groundmass)
Tl	Tertiary	lamprophyre	porphyritic	fine
Td	Tertiary	diabse	massive, amydaloidal	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	Equigraniular - diabasic	fine
Kqm	Cretaceous	biotite-quartz monzonite	Equigranular to porphyritic	coarse-medium

Baker (1983) noted that the "ranges of textures in the various dyke 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".

In 2006, three main intrusive types were observed in the holes drilled, 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 mineralization. Examples of the main rock types are shown in Table 7.1.

The quartz monzonite porphyry (unit Tbqmp) varies considerably in proportion and size of phenocrysts, with at least four varieties recognized (Figure 6). The first and possibly earliest phase (Tbqmp Type I) is dark to medium grey, with 10-15%, <7mm feldspar phenocrysts, 1-2% fine-grained biotite, and <5% quartz set in a fine-grained groundmass. The second phase (Tbqmp Type II) is medium to light grey, with 30% feldspar phenocrysts and minor biotite set in a medium-grained groundmass. The third phase (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 2006, which appear to offset the interpreted mineral zones. The full significance of these fault structures to the deposit geometry remains to be determined.

Figure 7.4 Core photographs of felsic porphyry types recognized in the 2008 program

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)

8 Deposit types

This section is reproduced and updated from "Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report" dated May 13, 2009 and filed on SEDAR on May 14, 2009.

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). A description of porphyry molybdenum-copper deposits and their associated alteration halos was discussed in the Kobex 2004 Technical Report and is not included herein. See Summary Report on the CUMO Molybdenum Property, Boise County, Idaho, dated April 25, 2005.

The main difference between these porphyry types is that molybdenite is the principal ore mineral in the porphyry molybdenum (low F) type, whereas chalcopyrite, molybdenite, and lesser bornite are the ore minerals on porphyry Cu-Mo deposits. More significantly, the typical size of porphyry Mo (low F) deposit is relatively small (most deposits are around 94 MT at 0.085% MoS2 and very few deposits exceed 500 MT) compared to the average porphyry Cu-Mo (500 Mt with 0.41 % Cu, 0.016 % Mo, 0.012 g/t Au and 1.2 g/t Ag) in which tonnages can range up to over 2 billion tonnes.

The CUMO deposit is primarily of economic interest for its Mo content but contains significant values of Cu and Ag. According to Carten et al (1993), low-grade zones of copper enrichment typically form above and partially overlap with molybdenum ore shells in porphyry molybdenum deposits. The CUMO deposit is classified as a porphyry Mo-Cu deposit (Mo greater than 0.04% and Cu being 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 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).

The mineral deposit profile for porphyry Cu-Mo listed below is from the British Columbia Geological Survey website:

(http://www.empr.gov.bc.ca/Mining/Geolsurv/MetallicMinerals/MineralDepositProfiles/PROFILES/L04.htm).

PORPHYRY Cu+/-Mo+/-Au

Per Panteleyev, A. (1995): Porphyry Cu+/-Mo+/-Au, in Selected British Columbia Mineral Deposit Profiles, Volume 1 - Metallics and Coal, Lefebure, D.V. and Ray, G.E., Editors, British Columbia Ministry of Energy of Employment and Investment, Open File 1995-20, pages 87-92.

IDENTIFICATION

SYNONYM: Calcalkaline porphyry Cu, Cu-Mo, Cu-Au.

COMMODITIES (BYPRODUCTS): Cu, Mo and Au are generally present but quantities range from insufficient for economic recovery to major ore constituents. Minor Ag in most deposits; rare recovery of Re from Island Copper mine.

EXAMPLES (British Columbia - Canada/International):

Volcanic type deposits (Cu + Au * Mo) - Fish Lake (092O041), Kemess (094E021,094), Hushamu (EXPO, 092L240), Red Dog (092L200), Poison Mountain (092O046), Bell (093M001), Morrison (093M007), Island Copper (092L158); Dos Pobres (USA); Far Southeast (Lepanto/Mankayan), Dizon, Guianaong, Taysan and Santo Thomas II (Philippines), Frieda River and Panguna (Papua New Guinea).

Classic deposits (Cu + Mo * Au) - Brenda (092HNE047), Berg (093E046), Huckleberrry (093E037), Schaft Creek (104G015); Casino (Yukon, Canada), Inspiration, Morenci, Ray, Sierrita-Experanza, Twin Buttes, Kalamazoo and Santa Rita (Arizona, USA), Bingham (Utah, USA),El Salvador, (Chile), Bajo de la Alumbrera (Argentina).

Plutonic deposits (Cu * Mo) - Highland Valley Copper (092ISE001,011,012,045), Gibraltar (093B012,007), Catface (092F120); Chuquicamata, La Escondida and Quebrada Blanca (Chile).

Of particular note is the Plutonic form of deposit, which occurs in batholithic settings. This may be a close geometric model for the CUMO deposit, as mineralization occurs within rocks of the Idaho batholith as well as later dikes and breccias, and the alteration is diffuse, with relatively low overall sulphide content.

8.1 Geological considerations

8.1.1 Capsule description

Stockworks of quartz veinlets, quartz veins, closely spaced fractures and breccias containing pyrite and chalcopyrite with lesser molybdenite, bornite and magnetite occur in large zones of economically bulk-mineable mineralization in or adjoining porphyritic intrusions and related breccia bodies. Disseminated sulphide minerals are present, generally in subordinate amounts. The mineralization is spatially, temporally and genetically associated with hydrothermal alteration of the hostrock intrusions and wallrocks.

8.1.2 Tectonic setting

In orogenic belts at convergent plate boundaries, commonly linked to subduction-related magmatism. Also in association with emplacement of high-level stocks during extensional tectonism related to strike-slip faulting and back-arc spreading following continent margin accretion.

8.1.3 Depositional environment / geological setting

High-level (epizonal) stock emplacement levels in volcano-plutonic arcs, commonly oceanic volcanic island and continent-margin arcs. Virtually any type of country rock can be mineralized, but commonly the high-level stocks and related dikes intrude their coeval and cogenetic volcanic piles.

8.1.4 Age of mineralization

Two main periods in the Canadian Cordillera: the Triassic/Jurassic (210-180 Ma) and Cretaceous/Tertiary (85-45 Ma). Elsewhere deposits are mainly Tertiary, but range from Archean to Quaternary.

8.1.5 Host/associated rock types

Intrusions range from coarse-grained phaneritic to porphyritic stocks, batholiths and dike swarms; rarely pegmatitic. Compositions range from calcalkaline quartz diorite to granodiorite and quartz monzonite. Commonly there is multiple emplacement of successive intrusive phases and a wide variety of breccias. Alkalic porphyry Cu-Au deposits are associated with syenitic and other alkalic rocks and are considered to be a a distinct deposit type (see model L03).

8.1.6 Deposit form

Large zones of hydrothermally altered rock contain quartz veins and stockworks, sulphide-bearing veinlets; fractures and lesser disseminations in areas up to 10 km2 in size, commonly coincident wholly or in part with hydrothermal or intrusion breccias and dike swarms. Deposit boundaries are determined by economic factors that outline ore zones within larger areas of low-grade, concentrically zoned mineralization. Cordilleran deposits are commonly subdivided according to their morphology into three classes - classic, volcanic and plutonic (see Sutherland Brown, 1976; McMillan and Panteleyev, 1988):

<u>Volcanic type deposits (e.g. Island Copper)</u>

are associated with multiple intrusions in subvolcanic settings of small stocks, sills, dikes and diverse types of intrusive breccias. Reconstruction of volcanic landforms, structures, vent-proximal extrusive deposits and subvolcanic intrusive centres is possible in many cases, or can be inferred. Mineralization at depths of 1 km, or less, is mainly associated with breccia development or as lithologically controlled preferential replacement in hostrocks with high primary permeability. Propylitic alteration is widespread and generally flanks early, centrally located potassic alteration; the latter is commonly well mineralized. Younger mineralized phyllic alteration commonly overprints the early mineralization. Barren advanced argillic alteration is rarely present as a late, high-level hydrothermal carapace.

<u>Classic deposits (e.g., Berg)</u>

are stock related with multiple emplacements at shallow depth (1 to 2 km) of generally equant, cylindrical porphyritic intrusions. Numerous dikes and breccias of pre, intra, and post-mineralization age modify the stock geometry. Orebodies occur along margins and adjacent to intrusions as annular ore shells. Lateral outward zoning of alteration and sulphide minerals from a weakly mineralized potassic/propylitic core is usual. Surrounding ore zones with potassic (commonly biotite-rich) or phyllic alteration contain molybdenite * chalcopyrite, then chalcopyrite and a generally widespread propylitic, barren pyritic aureole or 'halo'.

Plutonic deposits (e.g., the Highland Valley deposits)

are found in large plutonic to batholithic intrusions immobilized at relatively deep levels, say 2 to 4 km. Related dikes and intrusive breccia bodies can be emplaced at shallower levels. Hostrocks are phaneritic coarse grained to porphyritic. The intrusions can display internal compositional differences as a result of differentiation with gradational to sharp boundaries between the different phases of magma emplacement. Local swarms of dikes, many with associated breccias, and fault zones are sites of mineralization. Orebodies around silicified alteration zones tend to occur as diffuse vein stockworks carrying chalcopyrite, bornite and minor pyrite in intensely fractured rocks but, overall, sulphide minerals are sparse. Much of the early potassic and phyllic alteration in central parts of orebodies is restricted to the margins of mineralized fractures as selvages. Later phyllic-argillic alteration forms envelopes on the veins and fractures and is more pervasive and widespread. Propylitic alteration is widespread but unobtrusive and is indicated by the presence of rare pyrite with chloritized mafic minerals, saussuritized plagioclase and small amounts of epidote.

8.1.7 Texture/structure

Quartz, quartz-sulphide and sulphide veinlets and stockworks; sulphide grains in fractures and fracture selvages. Minor disseminated sulphides commonly replacing primary mafic minerals. Quartz phenocrysts can be partially resorbed and overgrown by silica.

8.1.8 Ore mineralogy (principal and subordinate):

Pyrite is the predominant sulphide mineral; in some deposits the Fe oxide minerals magnetite, and rarely hematite, are abundant. Ore minerals are chalcopyrite; molybdenite, lesser bornite and rare (primary) chalcocite. Subordinate minerals are tetrahedrite/tennantite, enargite and minor gold , electrum and arsenopyrite. In many deposits late veins commonly contain galena and sphalerite in a gangue of quartz, calcite and barite.

8.1.9 Gangue mineralogy (principal and subordinate)

Gangue minerals in mineralized veins are mainly quartz with lesser biotite, sericite, K-feldspar, magnetite, chlorite, calcite, epidote, anhydrite and tourmaline. Many of these minerals are also pervasive alteration products of primary igneous mineral grains.

8.1.10 Alteration mineralogy

Quartz, sericite, biotite, K-feldspar, albite, anhydrite/gypsum, magnetite, actinolite, chlorite, epidote, calcite, clay minerals, tourmaline. Early formed alteration can be overprinted by younger assemblages. Central and early formed potassic zones (K-feldspar and biotite) commonly coincide with ore. This alteration can be flanked in volcanic hostrocks by biotite-rich rocks that grade outward into propylitic rocks. The biotite is a fine-grained, 'shreddy' looking secondary mineral that is commonly referred to as an early developed biotite (EDB) or a 'biotite hornfels'. These older alteration assemblages in cupriferous zones can be partially to completely overprinted by later biotite and K-feldspar and then phyllic (quartz-sericite-pyrite) alteration, less commonly argillic, and rarely, in the uppermost parts of some ore deposits, advanced argillic alteration (kaolinite-pyrophyllite) .

8.1.11 Weathering

Secondary (supergene) zones carry chalcocite, covellite and other Cu*2S minerals (digenite, djurleite, etc.), chrysocolla, native copper and copper oxide, carbonate and sulphate minerals. Oxidized and leached zones at surface are marked by ferruginous 'cappings' with supergene clay minerals, limonite (goethite, hematite and jarosite) and residual quartz.

8.1.12 Ore controls

Igneous contacts, both internal between intrusive phases and external with wallrocks; cupolas and the uppermost, bifurcating parts of stocks, dike swarms. Breccias, mainly early formed intrusive and hydrothermal types. Zones of most intensely developed fracturing give rise to ore-grade vein stockworks, notably where there are coincident or intersecting multiple mineralized fracture sets.

8.1.13 Associated deposit types

Skarn Cu (K01), porphyry Au (K02), epithermal Au-Ag in low sulphidation type (H05) or epithermal Cu-Au-Ag as high-sulphidation type enargite-bearing veins (L01), replacements and stockworks; auriferous and polymetallic base metal quartz and quartz-carbonate veins (I01, I05), Au-Ag and base metal sulphide mantos and replacements in carbonate and non- carbonate rocks (M01, M04), placer Au (C01, C02).

8.1.14 Comments

Subdivision of porphyry copper deposits can be made on the basis of metal content, mainly ratios between Cu, Mo and Au. This is a purely arbitrary, economically based criterion, an artifact of mainly metal prices and metallurgy. There are few differences in the style of mineralization between deposits although the morphology of calcalkaline deposits does provide a basis for subdivision into three distinct subtypes - the 'volcanic, classic, and plutonic' types. A fundamental contrast can be made on the compositional differences between calcalkaline quartz-bearing porphyry copper deposits and the alkalic (silica undersaturated) class. The alkalic porphyry copper deposits are described in a separate model - L03.

8.1.15 Exploration Guides

<u>Geochemical signature</u>

Calcalkalic systems can be zoned with a cupriferous (* Mo) ore zone having a ‘barren’, low-grade pyritic core and surrounded by a pyritic halo with peripheral base and precious metal-bearing veins. Central zones with Cu commonly have coincident Mo, Au and Ag with possibly Bi, W, B and Sr. Peripheral enrichment in Pb, Zn, Mn, V, Sb, As, Se, Te, Co, Ba, Rb and possibly Hg is documented. Overall the deposits are large-scale repositories of sulphur, mainly in the form of metal sulphides, chiefly pyrite.

8.1.16 Geophysical signature

Ore zones, particularly those with higher Au content, can be associated with magnetite-rich rocks and are indicated by magnetic surveys. Alternatively the more intensely hydrothermally altered rocks, particularly those with quartz-pyrite-sericite (phyllic) alteration produce magnetic and resistivity lows. Pyritic haloes surrounding cupriferous rocks respond well to induced polarization (I.P.) surveys but in sulphide-poor systems the ore itself provides the only significant IP response.

8.1.17 Other exploration guides

Porphyry deposits are marked by large-scale, zoned metal and alteration assemblages. Ore zones can form within certain intrusive phases and breccias or are present as vertical 'shells' or mineralized cupolas around particular intrusive bodies. Weathering can produce a pronounced vertical zonation with an oxidized, limonitic leached zone at surface (leached capping), an underlying zone with copper enrichment (supergene zone with secondary copper minerals) and at depth a zone of primary mineralization (the hypogene zone).

8.1.18 Economic factors

<u>Typical grade and tonnage</u>

Worldwide according Cox and Singer (1988) based on their subdivision of 55 deposits into subtypes according to metal ratios, typical porphyry Cu deposits contain (median values): Porphyry Cu-Mo: 500 Mt with 0.41 % Cu, 0.016 % Mo, 0.012 g/t Au and 1.22 g/t Ag.

9 Mineralization

This section is excerpted and updated from “Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report” dated May 13, 2009 and filed on SEDAR on May 14, 2009.

9.1 Description of mineralized zones

The CUMO deposit is located in a famous 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 to 2.8 million ounces of which 74% was from placer operations (Anderson, 1947). According to Killsgaard and others (1989) more gold has been produced from the Boise Basin than any other mining locality in Idaho. 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 groups that he referred to as early Tertiary and early Miocene. The first group consists of gold-quartz veins containing minor sulphides that occur within the Idaho batholith and are associated with weak wall rock alteration. Associated sulphides include pyrite, arsenopyrite, sphalerite, tetrahedrite, chalcopyrite, galena and stibnite. The second group of deposits occurs within porphyry dikes and stocks as well as in the batholith, and is characterized by relatively abundant sulphides, 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 CUMO (within 4 to 6 miles/6 to 10 kilometres), 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. The Blackjack deposit on Grimes Creek is described by Anderson (1947) as distinct, being characterized by a 15 foot (5 metre) wide sulphide matrix breccia developed in quartz monzonite porphyry, with no conspicuous fault control.

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. Little Falls was extensively drilled between 1978 and 1981, where mineralization occurs within a rhyolite dike that is part of a swarm of dikes that extends northeast from CUMO. An age of 29±3 Ma was obtained by fission-track dating of a zircon from one of the mineralized dikes (Killsgaard et al, 1989).

To the northeast of CUMO, along the Idaho trans-Challis fault system, are several molybdenum and molybdenum-copper occurrences that are thought to be related to Tertiary intrusive rocks (Killsgaard et al, 1989). These include Molybdenum Lode, the Bobcat Gulch porphyry system, molybdenite-bearing quartz veins at Spring Creek, and anomalous Mo in soils northwest of Leesburg (Killsgaard et al, 1989).

9.2 Property mineralization

Mineralization at the CUMO 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 centimetres 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 9.1 and Figure 9.2 show examples of mineralization at CUMO from the recent drillholes.

Figure 9.1 Photographs of mineralized core from the CUMO 2006 program, hole C06-28

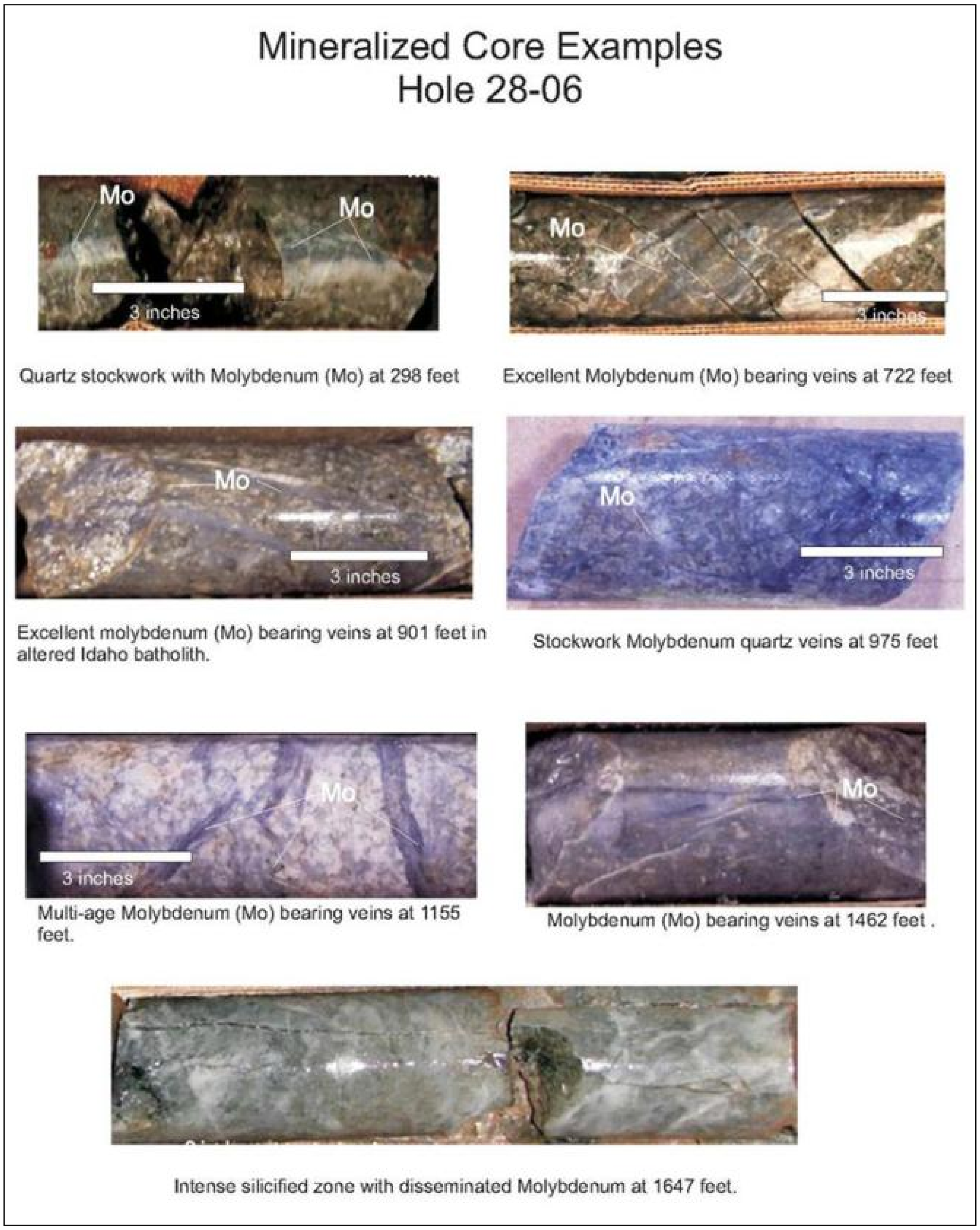


Figure 9.2 Photographs of molybdenite mineralization in 2008 drill core

a) Quartz - MoS2 veinlets in porphyry unit Tbqmp3 C35-08 (2291 ft)

b) Stockwork Quartz - MoS2 veinlets in Quartz Monzonite unit Kqm C35-08 (2496 ft)

c) Quartz Mos2 veinlet in intrusive breccia unit Tbx C08-37 1896.5 ft

d) Coarse MoS2 in white quartz veinlet. C36-08 (1566.5 ft)

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 in Figure 9.3. A concentric pattern is clearly evident, which is also shown by the distribution of anomalous Mo and Cu rock geochemical results (Figure 9.4a and Figure 9.4b). 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.

Figure 9.3 Surface distribution of quartz and epidote veinlets and metal zonation

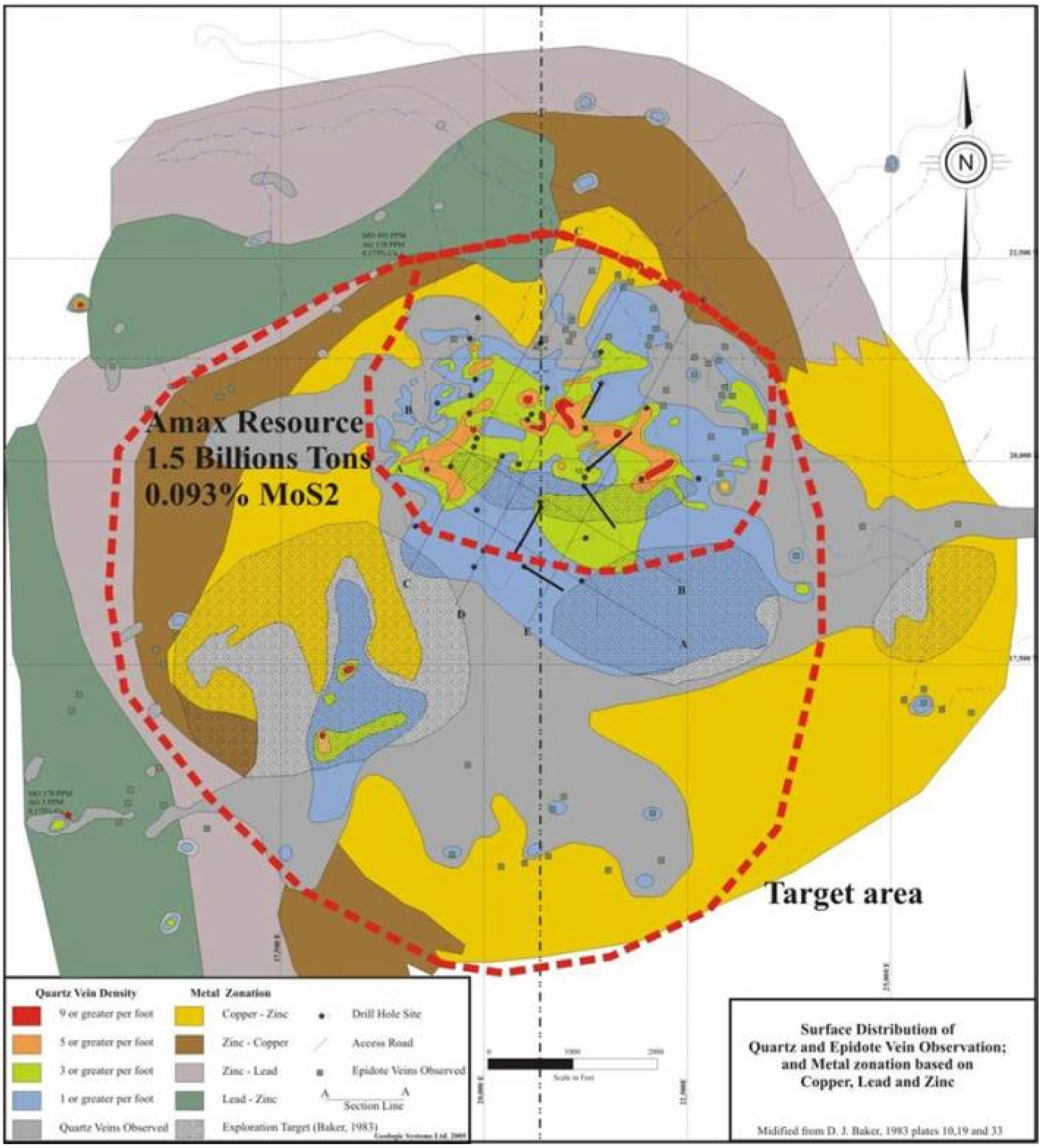
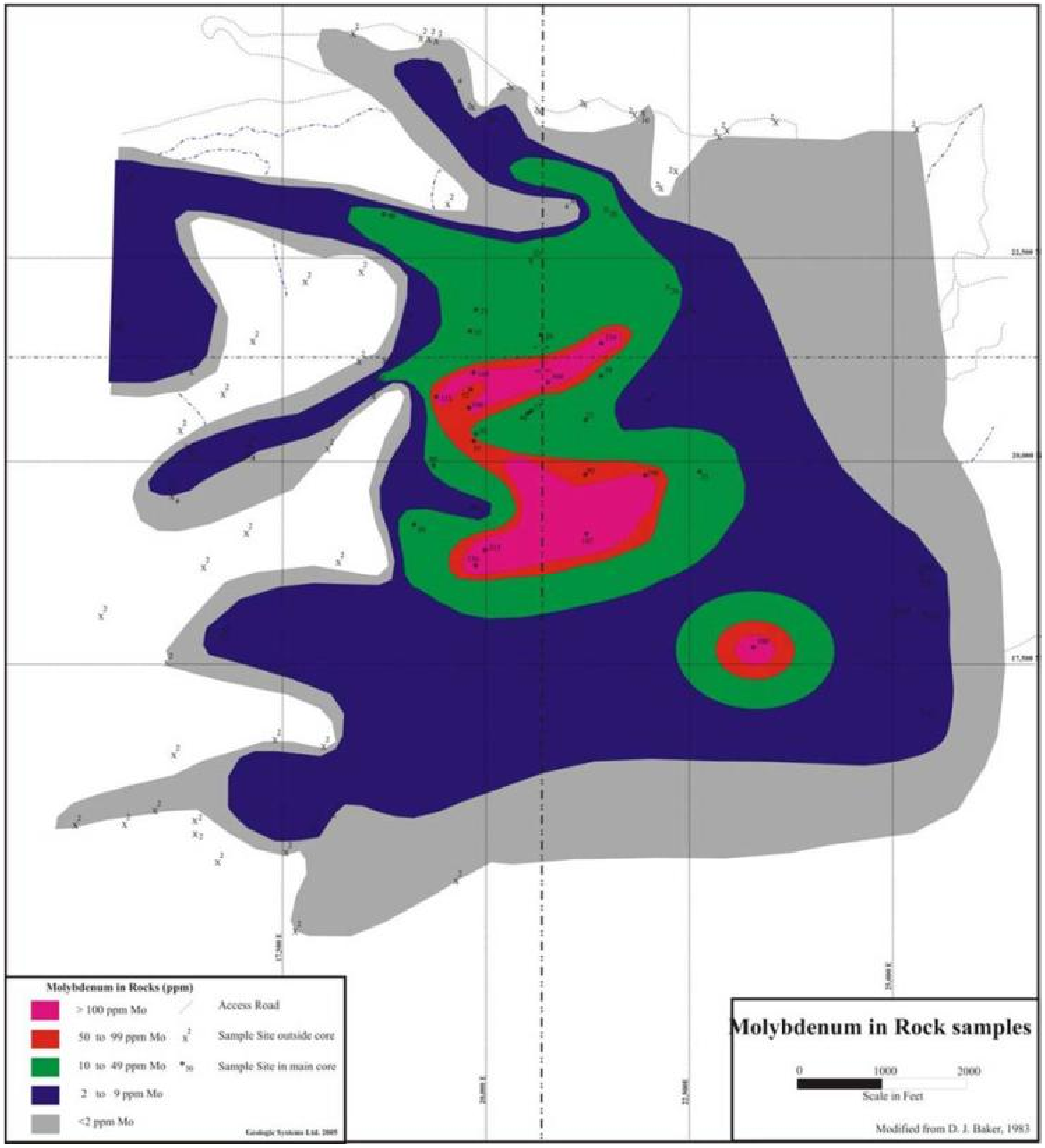
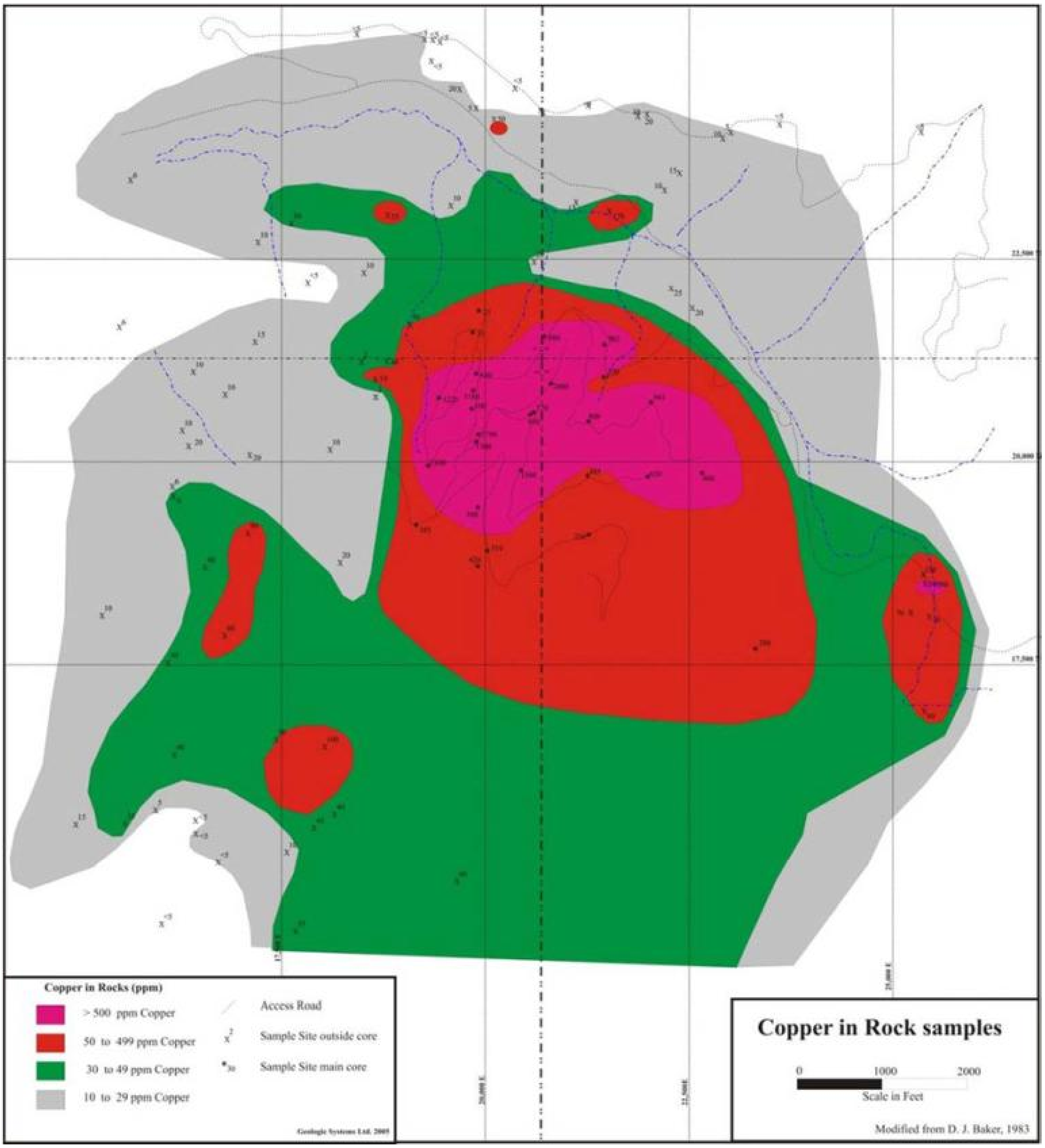


Figure 9.4 Geochemical distribution of Mo (a) and Cu (b) in surface rock chip samples



a)



b)

In terms of rock types, Amax suggested a textural/chemical evolution of Tertiary igneous rocks from older, phenocryst-rich quartz monzonite/quartz latite to younger, phenocryst-poor siliceous post-mineral rhyolite. Amax proposed a conceptual model of a central quartz-rich core (with magnetite) that grades into a quartz molybdenite + pyrite veins which progresses into a quartz-chalcopyrite + pyrite and quartz vein shell which are covered by a shell of epidote +quartz **+** pyrite veins. They found the alteration assemblages weakly developed and difficult to map (Baker, 1985).

In detail, Amax interpreted two shells of molybdenite mineralization, with the upper shell being richer in copper and silver, but of lower molybdenite grade, and the lower shell being molybdenite-rich and depleted in copper and silver (Baker 1983). They 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).

Mosquito Consolidated Gold Mining Ltd. acquired the CUMO property with the intention of exploring for a large scale, low cost, open pit accessible molybdenum deposit. The 2006 results confirmed the thickness and grade of mineralization on the property as indicated by previous drilling (Amax), and demonstrated continuity of mineralization between the original wide-spaced holes (Kobex/Mosquito).

The 2006 drilling revealed the presence of three distinct metal zones within the deposit: an upper copper-silver zone, underlain by a transitional copper-molybdenum zone, in turn underlain by a lower molybdenum-rich zone.

Three-dimensional modeling of results was conducted by Mr. Shaun Dykes (P.Geo.) formerly of Mosquito and indicates the currently drilled area is located on the north side of a potentially large mineralized system, of which only a small part has been drilled to date.

In 2007 and 2008 Mosquito reconfirmed the conceptual model in terms of the distribution of the quartz core and vein zones, but the current interpretation is that these features are part of a single large porphyry system underlain by a single source intrusion. The vein paragenesis/metal zones are interpreted as concentric zones formed above and/or within a one-source intrusion. The various porphyry dikes are interpreted as inter-mineral intrusions that emanated from the source intrusive body.

10 Exploration

Aside from diamond drilling no other exploration has been completed since the last 43-101 Report being "Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report" dated May 13, 2009 and filed on SEDAR on May 14, 2009. Additional information regarding the 2009 and 2010 drilling programs has been added.

11 Drilling

This section is summarised and updated from "Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report" dated May 13, 2009 and filed on SEDAR on May 14, 2009. Additional information regarding the 2009 and 2010 drilling programs has been added.

11.1 General

In 2006, diamond drilling was done by Kettle Drilling Inc. of Cour d'Alene on behalf of Kobex Resources Ltd. and Mosquito Resources Corp. Kobex commenced drilling in August, 2006 and completed one hole. On October 6, 2006, the Kobex Resources Ltd 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 183 metres (600 feet), and the action was taken before any assays were received. Mosquito Mining Corp. (wholly owned US subsidiary of Mosquito Consolidated Gold Mines Ltd.) assumed control of the project on October 10, 2006 and completed this hole to a depth of 521 metres (1710 feet) before the program was halted due to the onset of winter conditions.

In 2007 and 2008, diamond drilling was done by Kirkness Drilling of Carson City, Nevada. Kirkness drilled eleven (11) +600 metre (+2000 foot) diamond drillholes. Table 11.1 provides details of the drilling undertaken from 2006 to 2008.

Table 11.1 Summary of 2006, 2007 and 2008 Diamond Drilling at CUMO.

Hole	Northing	Easting	Elevation	Dip	Azimuth	Length
Number	feet	feet	feet	degrees	degrees	feet
27-06	120,016.7	220,160.3	7105	-90	000	1849
28-06	119,531.6	120,796.4	7170	-90	000	1711
29-07	120,016.7	220,160.3	6305	-70	140	2281.7
30-07	119,531.6	220,796.4	6206	-90	000	2416.5
31-07	120,016.7	220,160.3	6305	-70	045	2104
32-07	119,480.0	220,720.3	6316	-70	190	2044
33-07	118585.3	221,268.9	6798	-90	000	2095 stopped
34-07	118530.5	220,343.8	6512	-70	095	1769 stopped
35-08	118658.3	220487.4	6534.	-90	000	2817 completed
36-08	119266.8	219322.9	6457	-90	000	2488 completed
37-08	119755.7	221220.4	6341	-70	335	2195 completed
38-08	118658.3	220487.4	6534	-70	180	2441 completed
39-08	118872.7	220777.6	6466	-90	000	2688 completed
40-08	119539.8	220816.8	6321	-70	225	2252 completed
41-08	119545.7	219005.8	6247	-90	000	3018 completed
42-08	118711.9	219886.6	6544	-70	270	2707 stopped (winter)
43-08	120515.6	220178.6	6198	-80	040	1308 stopped by fault
44-08	118068.1	221448.9	6733	-65	075	3047 completed
45-08	119802.3	218821.4	6183	-80	330	1796 stopped (winter)

All three Mosquito drilling programs were supervised by Senior Geologist, Matt Ball, Ph.D., P.Geo., CUMO Property, Garden Valley, Idaho.

In 2009 a total of 6,235 metres (20,456 feet) in 9 holes were completed whilst in 2010 a total of 1,313 metres (4,307 feet) from 3 holes were completed. The 2009 and 2010 holes were sited to infill gaps in the existing drilling coverage and were drilled along existing tracks and roads.

Figure 11.1shows the locations of all holes drilled to date in the deposit. Mr. Shaun M. Dykes, M.Sc. (Eng), P.Geo., former Exploration Manager and former Director of Mosquito Consolidated Gold Mines Ltd., was the designated qualified person for the CUMO Project at the time of drilling, and prepared the technical information on the 2006, 2007, 2008, 2009, 2010 results.

All holes were surveyed down the hole at regular intervals using a Reflex survey instrument. Table 11.2 provides details of the drilling undertaken in 2009 and 2010. Table 11.3 summarizes the drilling undertaken to date by Mosquito on the CUMO project. Figure 11.1 shows the locations of all holes drilled to date in the deposit.

Table 11.2 Summary of 2009 and 2010 Diamond Drilling at CUMO

Hole	Northing	Easting	Elevation	Dip	Azimuth	Length (feet)
48-09	120,741.3	219,432.5	5827	-70	305	2576 completed
49-09	118,881.6	221,719.8	6668	-90	n/a	2847 completed
50-09	121,752.9	219,929.4	5885	-75	270	1826 completed
51-09	121,752.9	219,929.4	5885	-90	n/a	1583.5 completed
52-09	118,585.3	221,268.9	6798	-75	020	2772 completed
53-09	119,802.3	218,821.4	6183	-75	015	2461 completed
54-09	119,802.3	218,821.4	6183	-75	015	2471 completed
55-10	117,559.6	218422.4	6724.2	-65	0	2479 completed
56-10	117,559.9	218421.8	6724.2	-65	305	1294 completed
57-10	117,559.3	218422.2	6724.2	-90	n/a	534 stopped (winter)

Table 11.3 Summary of drilling undertaken by Mosquito Consolidated Gold Mines Ltd

Company	Year	Holes	Feet	Metres	Comments
Kobex	2006	2	3,560	1,085.10	Kobex drilled 1.5 holes only, completed by mosquito
Mosquito	2007	6	12,710	3,874.20	vertical and angle holes
	2008	11	26,770	8,159.70	vertical and angle holes
	2009	9	18,661	5,687.80	vertical and angle holes
	2010	3	4,307	1,312.80	vertical and angle holes
	Total	**31**	**66,008**	**20,119.6**	

Mr. Shaun M. Dykes, M.Sc. (Eng), P.Geo., former Exploration Manager and former Director of Mosquito Consolidated Gold Mines Ltd., was the designated qualified person for the CUMO Project at the time of this drilling, and prepared the technical information on the 2006, 2007, 2008, 2009, 2010 results.

A summary of significant intersections for all the CUMO drilling undertaken by Mosquito is given in Table 11.4. Potential economic metals include copper, molybdenum, silver, tungsten, rhenium and gallium. The presence of the by-product elements silver, tungsten, rhenium and gallium is significant in terms of the economic development of the property.

The 2006 - 2010 results confirmed the thickness 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 - 2010 drilling data supports the presence of three distinct metal zones within the deposit. Amax previously interpreted these zones as distinct ore 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 metal zones are a part of a single, large, concentrically zoned system with an upper copper-silver zone (cuag), underlain by a transitional copper-molybdenum zone (cumo), in turn underlain by a lower molybdenum-rich zone (mo) (Figure 11.2).

Three-dimensional modeling of the above zonation conducted by Mosquito indicates the current area being drilled is located on the north side of a larger system (Figure 11.3).

A summary of significant intersections for all the CUMO drilling are given in Table 11.4. Potential economic metals include copper, molybdenum, silver, tungsten, rhenium and gallium. The presence of the by-product elements silver, tungsten, rhenium and gallium is very significant in terms of the economic development of the property.

As a result of the multi-element nature of the mineralization, Mosquito decided to calculate both a copper and molybdenum equivalent for the intercepts (refer to Section 11.1.1).

Figure 11.1 Map showing the location of completed and proposed drillholes

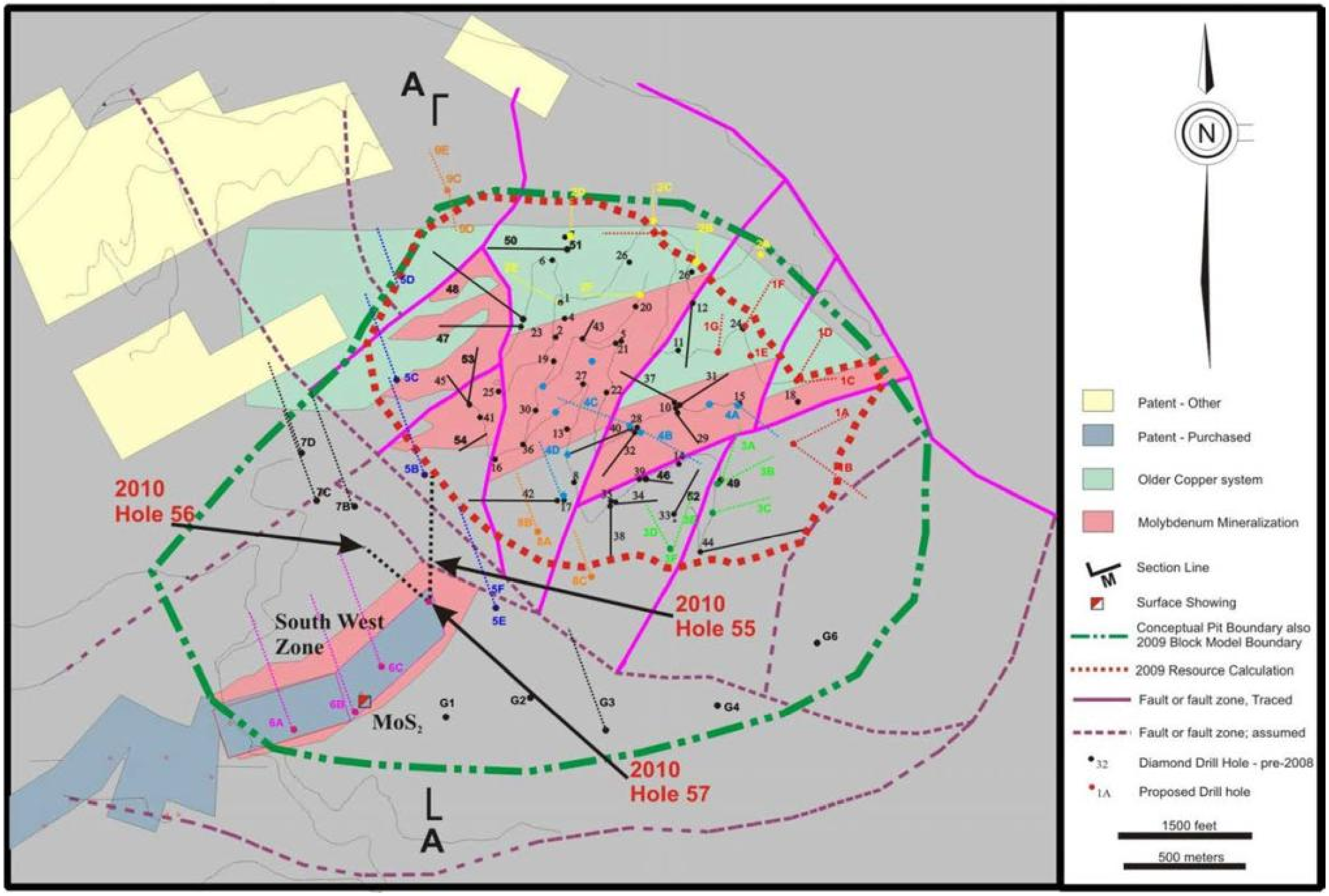


Table 11.4 Significant Intersections from 2009 AND 2010 CUMO Drilling

Hole (name)	From (feet)	To (feet)	Length (feet)	From (metres)	To (metres)	Length (metres)	Zone	Recv Cu Equiv.	Recv MoS_2 equiv.	MoO_3 Equiv lbs	MoS_2 %	Cu %	Ag Gms/T	Re ppm	W ppm
C09-47	840	1963.5	1123.5	256	598.5	342.4	main	0.63	0.092	1.65	0.071	0.18	4.29	0.02	19.84
C09-48	290	1736.5	1446.5	88.4	529.3	440.9	main	0.51	0.074	1.34	0.051	0.18	5.03	0.02	19.53
C09-49	960	2840	1880	292.6	865.6	573	main	0.55	0.081	1.45	0.079	0.05	1.7	0.03	17.42
C09-49	1520	2420	900	463.3	737.6	274.3	sub	0.73	0.106	1.91	0.105	0.06	1.91	0.04	17.33
C09-50	810	1524.5	714.5	246.9	464.7	217.8	main	0.33	0.048	0.86	0.026	0.15	5.29	0.01	20.38
C09-51	520	1570	1050	158.5	478.5	320	main	0.39	0.057	1.03	0.037	0.15	4.86	0.02	19.31
C09-52	890	2700	1810	271.3	823	551.7	main	0.61	0.089	1.60	0.085	0.07	1.69	0.03	17.58
C09-52	1790	2640	850	545.6	804.7	259.1	sub	0.94	0.137	2.47	0.141	0.05	1.29	0.06	16.63
C09-53	800	2471	1671	243.8	753.2	509.3	main	0.75	0.110	1.97	0.089	0.19	4.07	0.02	18.19
C09-53	1510	2471	961	460.2	753.2	292.9	sub	0.88	0.128	2.31	0.115	0.15	3.68	0.03	18.97
C10-55	230	420	190	70.1	128	57.9	main	0.73	0.107	1.92	0.107	0.05	1.69	0.03	16.77
C10-55*	1190	1200	10	362.7	365.8	3	sub	3.93	0.574	10.32	0.025	0.07	35.44	0	21.08
C10-56	220	500	280	67.1	152.4	85.3	main	0.28	0.041	0.73	0.042	0.01	0.42	0.01	21.47
C10-57	300	490	190	91.4	149.4	57.9	main	0.49	0.071	1.27	0.07	0.02	3.8	0.02	21.3

* *Sample also averaged 2.15 g/t Au*

The description of the calculation and formulas used for producing the metal equivalents and the recovered metal value for the intersections is covered in section 11.1.1 below.

Figure 11.2 Geology bench plan at 5000 ft elevation

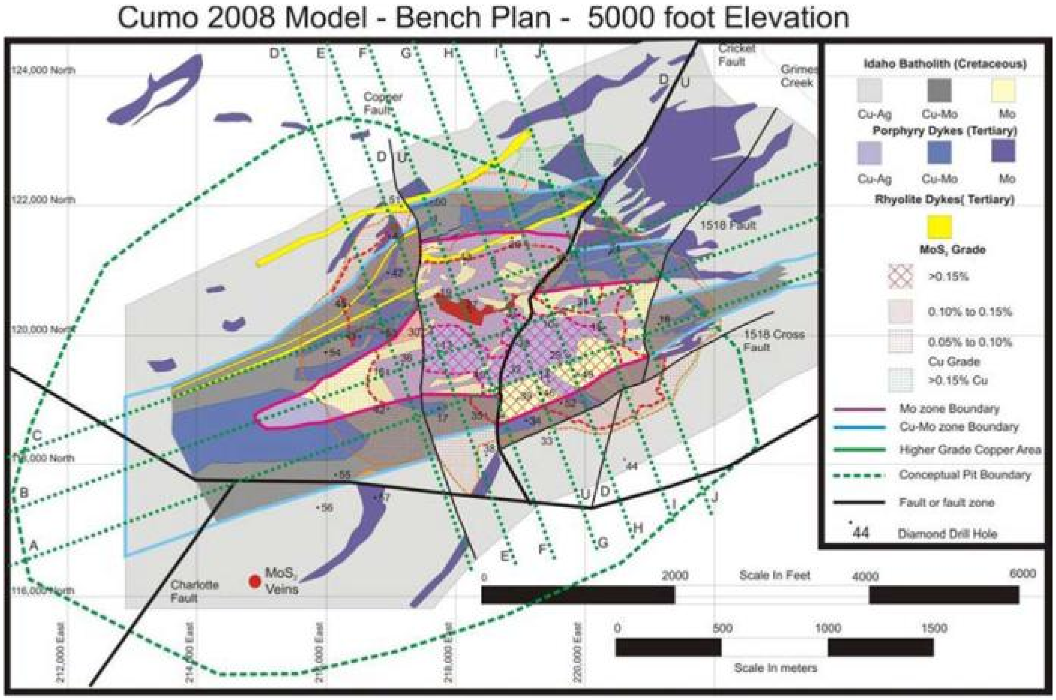
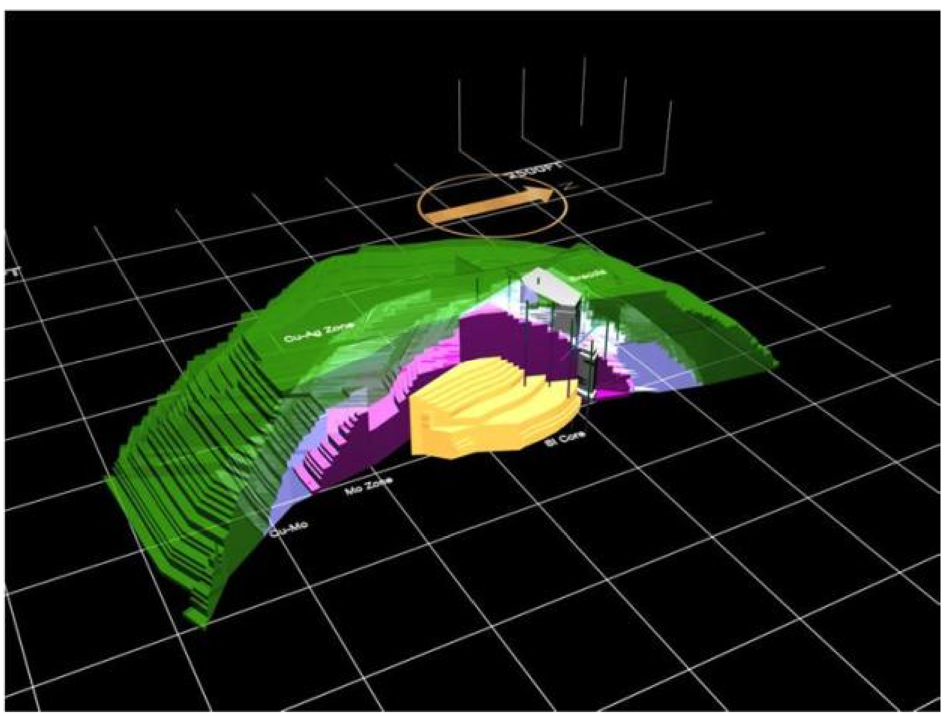


Figure 11.3 Snapshot of 3D model of CUMO deposit showing concentric pattern of metal zones



Yellow is barren silica core, purple is Mo zone, blue Cu-Mo zone, green is Cu-Ag zone.

Table 11.5 Summary of 2006 to 2010 diamond drilling

Hole Number	Northing feet	Easting feet	Elevation feet	dip degrees	azimuth degrees	Length feet
27-06	120,016.7	220,160.3	6351.4	-90	na	1849
28-06	119,531.6	220,796.4	6321.1	-90	na	1711
29-07	120,016.7	220,160.3	6343.3	-70	140	2281.7
30-07	119,585.8	219750.7	6213.1	-90	000	2416
31-07	120,016.7	220,160.3	6326.4	-70	045	2104
32-07	119,480.0	220,720.3	6316	-70	190	2044
33-07	118585.3	221,268.9	6798	-90	na	2095 suspended
34-07	118530.5	220,343.8	6512	-70	095	1769 suspended
35-08	118658.3	220487.4	6534.	-90	na	2817 completed
36-08	119266.8	219322.9	6457	-90	na	2488 completed
37-08	119755.7	221220.4	6341	-70	335	2195 completed
38-08	118658.3	220487.4	6534	-70	180	2445 completed
39-08	118872.7	220777.6	6466	-90	na	2688 completed
40-08	119539.8	220816.8	6321	-70	225	2252 completed
41-08	119545.7	219005.8	6247	-90	na	3018 completed
42-08	118711.9	219886.6	6544	-70	270	2707 stopped (winter)
43-08	120515.6	220178.6	6198	-80	040	1313 stopped by fault
44-08	118068.1	221448.9	6733	-65	075	3047 completed
45-08	119802.3	218821.4	6183	-80	330	1796 stopped (winter)
46-09	118,917.9	220,813.2	6575.1	-70	110	959 abandoned
47-09	120,741.3	219,432.5	5827	-70	270	2530 completed
48-09	120,741.3	219,432.5	5827	-70	305	2576 completed
49-09	118,881.6	221,719.8	6668	-90	na	2847 completed
50-09	121,752.9	219,929.4	5885	-75	270	1826 completed
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55-10	117,559.6	218422.4	6724.2	-65	0	2479 completed
56-10	117,559.9	218421.8	6724.2	-65	305	1294 completed
57-10	117,559.3	218422.2	6724.2	-90	na	534 stopped (winter)

11.1.1 Metal equivalent estimation

Copper equivalent (Cu. Equiv.) and Molybdenite equivalent (MoS_2 Equiv.) are based on the following metal prices (all in US$): Copper $2.10/lb, Molybdenum Oxide ($16/lb), Silver $0.35/gram and Tungsten $0.22/gram.($7.00 per lb). Other factors include 1% = 20 pounds; 1 ppm = 1 gm/T; 1000 ppb =1 ppm = 1 gm/T.

Molybdenum is based on either ferro-molybdenite or molybdenum oxide.

The price used in these estimates is $16 per pound Molybdenum oxide.

To obtain the amount of Molybdenum oxide that can be produced from MoS_2, the following is required: convert MoS_2 to Mo by dividing MoS_2 by 1.6681 then convert to MoO_3 (Molybdenum Oxide) by multiplying by 1.5. Therefore the amount of molybdenum oxide is pounds MoS_2 times 1.5 / 1.6681.

Estimated metallurgical recoveries used in the calculations are as follows for each metal zone. Recoveries are slightly lower than those currently reported by SGS in their recent metallurgical study, as they have been adjusted by Ausenco to reflect losses during the cleaning and roasting stages. However, it should be noted that there is no testwork as yet for the msi zone, and therefore the msi zone has only assumed recoveries.

Table 11.6 Metallurgical recoveries for equivalency calculations

Zone	Cu%	MoS_2%	Ag %	W %
oxide	60%	80%	70%	35%
cuag	68%	85%	73%	35%
cumo	87%	92%	78%	35%
mo	80%	95%	55%	35%
msi	80%	95%	55%	35%

Recovery (recv) for a metal is taken from the above table for each assay/block in a particular zone and is applied in the following formula:

RCV = Cu x 20 x $(Cu) x recv(Cu) + MoS_2 x 20 x (1.5/1.6681) x $($MoO_3$) x recv($MoS_2$) + Ag x $(Ag) x recv(Ag)+ W x $(W) x recv(W)

Then,

Cu Equiv = RCV / ($(Cu) x recv(Cu) x 20)

Mo Equiv = RCV / ($(MoO3) x recv(MoS2) x 20 x 1.5/1.6681)

12 Sampling method and approach

12.1 General sampling

This section is reproduced in total for completeness from “Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report” dated May 13, 2009 and filed on SEDAR on May 14, 2009. The sampling method and approach described below were also used in the 2009 and 2010 drilling programs.

Sampling was restricted to Diamond Drillhole (DDH) core and metallurgical sampling of previously drilled DDH core. Standard core sampling methods were employed for both drill core and metallurgical samples.

DDH drill core was placed in wooden core boxes during the 2006, 2007 and 2008 drilling seasons. In 2008, Mosquito’s staff, overseen by a geologist, transferred the remaining core stored in cardboard boxes to wooden core boxes for better preservation.

At the time of drilling, each core box is clearly labeled by the driller’s helper with the DDH hole number, core box number, and “to” and “from” drill core footages. Full core boxes are sealed with a lid. The driller(s) and/or geologist(s) then deliver the core boxes to the secure core storage warehouse[1] located in Garden Valley, Idaho. The core boxes are laid out in sequence upon long tables specifically made for core logging purposes. A geologist then logs the core for lithology, structure, alteration and mineralization. Geotechnical measurements for Rock Quality Designation (RQD) are recorded. Each core box is additionally labelled using a metal Dymo labelling tool for long-term preservation of identification. The core is photographed, two boxes at a time, using a mounted Nikon digital camera. It is then delivered to the core-cutting technician. The photographs are downloaded onto computer files specific to each drillhole.

Half-core is collected at regular 10-foot intervals for analysis. Sample lengths are adjusted to lithological contacts in cases where barren dikes are intersected.

Half core sample intervals are placed in ether cloth or heavy plastic sample bags with the sample number placed on the outside of the bag in black magic marker. Individual sample interval tags are included in each sample bag. The bag is 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 is 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 is given a second clear water bath prior to being bagged or stored and the recirculation tank is voided and refilled based upon clarity.

The half core is sent for analysis and the other half retained and stored at the core storage warehouse in Garden Valley, Idaho. The remaining core is stacked upon a standard pallet and sealed with a plywood cover. Each plywood cover is clearly labelled with the cores information. The pallet is then strapped with a metal banding tool and stored within the archive section of the core storage warehouse in Garden Valley, Idaho.

[1] *The core storage warehouse in Garden Valley, Idaho, is secure in the sense that it is a steel building, well insulated, with secure doors that contain security locks. The project manager, and Senior Geologist Matt Ball, lives in an apartment attached to the building. The area is well-lit and is seldom without occupancy by Mosquito staff. The doors are locked when the building is unoccupied.*

Blanks and standards are inserted into the sample stream at a frequency of one every 20 samples. The core-cutting technician selects the exact intervals and notes them on his sample log. The core technician inserts 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 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 12.1).

Table 12.1 Certified standards prepared for CUMO project

Standard	Element	Certified Mean	2 Standard Deviation (between lab)
CUMO1	Tot. Cu	1155 ppm	65 ppm
CUMO1	Tot. Mo	354 ppm	17 ppm
CUMO2	Tot. Cu	151 ppm	12 ppm
CUMO2	Tot. Mo	970 ppm	66 ppm
CUMO3	Tot. Cu	856 ppm	30 ppm
CUMO3	Tot. Mo	51.7 ppm	7.8 ppm

The bagged core samples are string or wire tied and then stored temporarily in holding pallets at the core storage warehouse in Garden Valley. When enough samples are accumulated, the samples are delivered to ALS-Chemex in Elko, Nevada for preparation and analysis. Kobex shipped their samples whereas Mosquito personnel deliver the samples.

12.2 Density determinations

Specific gravity determinations were made by Amax for Cumo for each grade Domain. The measurements were made using the weight in air/weight in water procedure by Skyline Laboratories of Colorado.

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.

13 Sample preparation, analyses, and security

This section is an excerpt and updated from "CUMO 2009 PEA Technical Report, November 18, 2009". Sample preparation, analytical procedure and security procedures for the 2009 and 2010 drill programs were also undertaken as described below.

13.1 Sample preparation and Analysis

Samples submitted to ALS-Chemex were logged in the ALS-Chemex tracking system, weighed, dried and finely crushed to better than 70 % passing a 2 mm (Tyler 9 mesh, US Std. No.10) screen. A split of up to 250 g is taken and pulverized to better than 85 % passing a 75 micron (Tyler 200 mesh, US Std. No. 200) screen.

Samples submitted by Kobex were routinely analyzed by the ALS-Chemex ME-ICP61 procedure code for 39 elements using a four (4) acid digestion with analysis by Plasma Emission Spectroscopy (ICP-AES).

http://www.alschemex.com/learnmore/learnmore-techinfo-principles-analyticalmethodologies.htm#Inductively%20Coupled%20Plasma%20Emission%20Spectroscopy%20(ICP-AES)

Samples submitted by Mosquito were routinely analyzed by the ALS-Chemex ME-MS ICP61 procedure code for 47 elements using a four (4) acid digestion with analysis by Inductively Coupled Plasma Mass Spectrometry (ICP-MS).

http://www.alschemex.com/learnmore/learnmore-techinfo-principles-analyticalmethodologies.htm#Inductively%20Coupled%20Plasma%20Mass%20Spectroscopy%20(ICP-MS)

Samples submitted by Mosquito for inter-laboratory check analysis were analyzed by SGS Minerals Services by the SGS ICM40B for 50 elements using a four (4) acid digestion/ICP-AES and ICP-MS. http: //www.sgs.com/geochem.

13.2 Security

A contemporary, well-kept, large steel building is used to warehouse Mosquito'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 cement and the walls are steel. There are few windows. Mosquito 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.

13.3 Statement on the adequacy of sample preparation, security and analytical procedures

Giroux (2009) reviewed sample procedures, security, duplicates, analytical procedures for all samples taken prior to 2009. Findings from Giroux (2009) report have been reproduced in sections 12 and 13 of this report. Snowden has examined the information presented and sees no reason to doubt the accuracy of this work.

Snowden has reviewed the sample preparation, security and analytical procedures used during 2009 and 2010 drilling programs and considers them to be reasonable for the evaluation of the CUMO deposit.

14 Data verification

This section is reproduced in total for completeness from "Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report" dated May 13, 2009 and filed on SEDAR on May 14, 2009.

14.1 General

During the site visits, Holmgren conducted data verification consisting of inspecting the drill collars in the field, a detailed inspection of the core logging facilities and sample handling procedures, random cross checks of the assay certificates, database and samplers records and verification of the standard and blank handling and inserting procedures.

During Snowden's site visit in November 2010, Snowden conducted similar data verification checks to those previously undertaken by Holmgren, except Snowden's checks were limited to those able to be done within Mosquito's warehouse/storage facility.

14.2 Historic database (summarized from Holmgren & Giroux, 2009)

As reported in the May 2009 report, ("Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report"), there were six data sets available to verify the original Skyline MoS_2 assay data base. 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 laboratory sets of duplicates are also available to compare with the Skyline original assays; a pulp sent to Amax Laboratory in Climax from diamond drillhole 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 in the 2005 report. In general, the results showed good correlation and high sampling variability for MoS_2.

During the Mosquito 2007 and 2008 drill campaigns blanks, standards were routinely inserted into the sample stream to monitor QA/QC at the primary laboratory ALS Chemex. In addition, the laboratory reported internal blanks, standards and duplicates which showed excellent agreement.

Results from the 2007 QA/QC program, as reported in Holmgren and Giroux (2008) showed good agreement.

The results from the 2008 QA/QC program were reported in the "Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report" dated May 13, 2009 and filed on SEDAR on May 14, 2009, the following sections have been summarised from that report:

14.2.1 2008 Blanks (Holmgren & Giroux, 2009)

During the 2008 diamond drill program blank samples were inserted in the samples stream at about a 1 in 20 frequency. A total of 235 were analyzed for MoS_2, Cu, Ag, Re, Ga, W, Fe and S. Holmgren & Giroux (2009) reported that the results were very good with no anomalies produced.

14.2.2 2008 Internal Lab Standards (Holmgren & Giroux, 2009)

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

14.2.3 2008 Internal Pulp Checks (Holmgren & Giroux, 2009)

ALS Chemex also routinely runs duplicate checks on sample pulps. Over the 2007-2008 drill program a total of 143 check samples were run for MoS_2. Holmgren & Giroux (2009) reported that the results were excellent with all but a few samples falling on an equal value line.

14.2.4 2008 Mosquito Standards (Holmgren & Giroux, 2009)

As explained in Section 12, CDN Labs prepared a set of Standards using drill core from the CUMO property.

Results for Standard CUMO1, the medium grade standard for Mo and highest grade for Cu, show one questionable result (see Figure 17). The assay for sample 396353 (2006 sample) was 0.049 % Mo with a corresponding low 0.01 % Cu indicating something was wrong with both analyses (Holmgren & Giroux (2009) suggested this was pointing to perhaps a numbering error on the Standard Sample). The remaining results are reasonable with most falling between the mean ± 2 standard deviations.

Results for Standard CUMO2 a higher grade Mo and low grade Cu standard show reasonable results for Cu and a couple of higher than normal Mo assays.

The results for Standard CUMO3 are also reasonable with more noise in the Cu analysis but no large variations. The Mo results are reasonable for low grade Mo values.

14.2.5 2008 Reject Duplicates (Holmgren & Giroux, 2009)

During the 2008 drill program second and third splits were taken from 154 rejects and re-assayed by the primary laboratory first by ICP_MS61 and then by XRF. Due to high volumes of samples submitted to the primary laboratory (ALS Chemex), 31 samples were run at a second laboratory SGS with a similar procedure. As all checks were completed by the same Laboratory in both cases the checks serve as a measure of sampling variability comparing 3 splits from the same crushed rejects.

14.3 2009 - 2010 drill program

QA/QC procedures on the 2009 and 2010 drill program included blanks, standards, internal laboratory standards, laboratory internal pulp checks, and re-splits sent to second laboratories.

14.4 Verification by Snowden

In March 2011, Snowden completed an audit of the quality control (“QC”) and quality assurance (“QA”) program related to the sampling for a Mineral Resource Estimate just on the samples taken in 2009 and 2010.

Overall, the data was found to be accurate relative to the laboratory certificates. There were some minor discrepancies that were noted and resolved and the resource database was updated. The QA/QC data collected is adequate to support the Mineral Resource definition, although in some cases it is not ideal and more work should be done to enable the data to be better used for resource categorization.

The audit looked at the program implemented by Mosquito to provide assurance that the data and analysis completed for the CUMO project was accurate and reproducible. This involved analysis of the drillhole and assay database, review of the certificate data received directly from the laboratory and analysis of the blank, standard and duplicate assays completed.

The CUMO database consists of 3215 analyses from drill core samples in 2009 and 2010. All of the assay results were compared to the independently sourced sample certificates. The analyses provided in the CUMO database were used to create the Mineral Resource estimate. There were minor discrepancies between assay certificates and the database for Cu and Mo. The database was revised and updated and used for the resource estimation. As well, Snowden verified all conversion formulas for Mo to MoS_2 and the units used in tables.

Mosquito inserted three types of QA/QC samples; blanks, standards and duplicates. These QA/QC samples are ideally interspersed regularly into the drill core sampling to provide a check on the sampling and laboratory biases that could exist and materially affect the results. QA/QC samples would ideally be “blind” to the laboratory, meaning the laboratory would not know the difference between a QA/QC sample and any regular sample.

Blank samples are used to highlight contamination between samples. Standard samples are of known value (reference) material, where the laboratory accuracy can be estimated by comparing the analysis to the expected standard deviation of the reference material. Duplicate samples help to quantify the repeatability of the analysis.

Two kinds of QA/QC sample were analyzed by Snowden; internal Mosquito data and laboratory data. The laboratory QA/QC data was used to supplement the CUMO data.

14.4.1 2009 - 2010 Blanks

During the 2009 and 2010 diamond drill program, blank samples were inserted in the samples stream at about a 1 in 20 frequency. A total of 112 samples were analyzed for MoS_2, Cu, Ag, Re, W, and Mo. The results were generally good with only two samples showing evidence of significant levels of contamination for Cu. A trend is also noted in the Mo results indicating low but increasing levels of contamination over time possibly due to wear of steel equipment containing Mo used in the preparation of the samples or analytical instrument drift. The graphs for Cu and MoS_2 are shown below in figures 14-1 and 14-2 respectively.

Figure 14.1 Copper in Blank Samples from 2009 and 2010 drill program

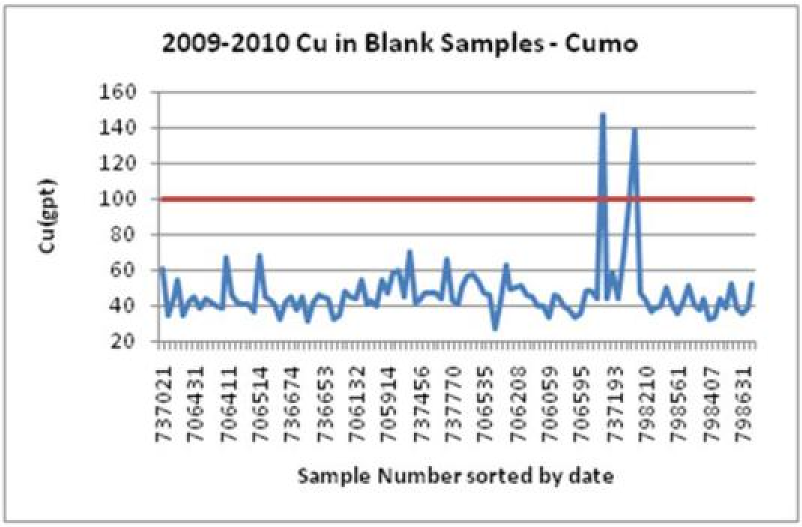
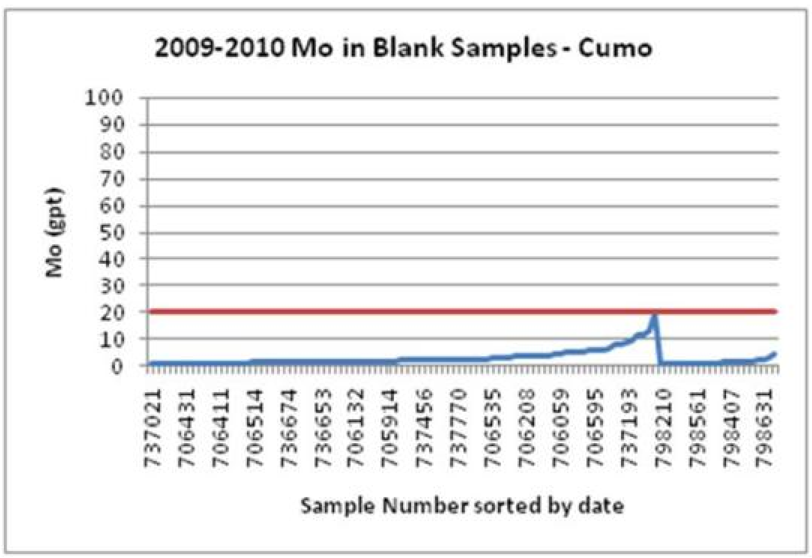


Figure 14.2 Mo in Blank Samples from 2009 and 2010 drill program



14.4.2 2009 – 2010 Mosquito standards

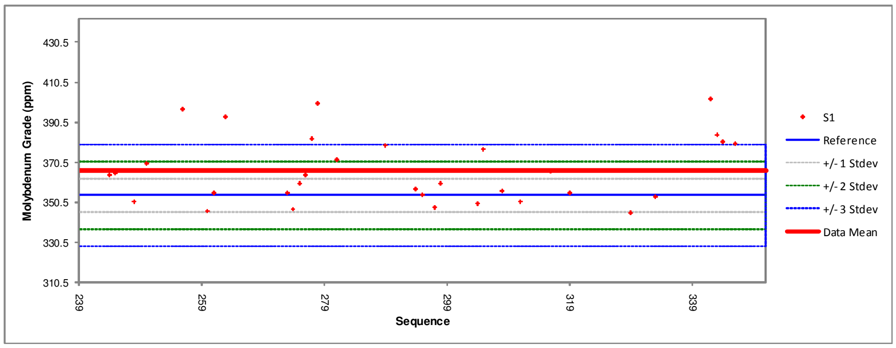
Certified standard samples are used to measure the accuracy of analytical processes and are composed of material that has been thoroughly analysed to accurately determine its grade within known error limits. A standard is considered to have failed if the assay result is above or below three standard deviations of the mean certified standard value defined by the standard manufacturer. If a standard has failed then it may be necessary to re-analyse the sample batch associated with the standard.

A total of 110 standard samples have been submitted at a frequency of 1 in every 20 samples. Mosquito uses the commercial standards for Mo, Cu, and Ag as described in section 12.

Results for Standard 1, (a medium grade Mo and high grade for Cu standard), are shown in Figure 14.4 and Figure 14.5. Results for Cu a generally acceptable with only one sample failing. However results for Mo are not optimal with several samples falling outside of the ±3SD tolerance limits.

Mosquito's QA / QC protocols require that all QA/QC samples (i.e. blank, duplicate and standards) must present results which exceed the accepted tolerance limits before an assay batch is rejected and re-assayed. As a result no batches were rejected and re-assayed during the 2009 and 2010 drilling programs.

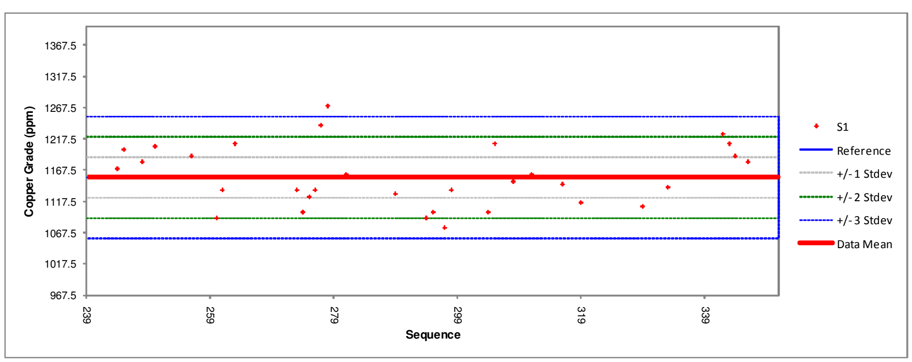
Figure 14.3 Standard 1 Mo (ppm) results 2009 - 2010



Summary statistics

Standard Reference Statistics	
Mean	354
Standard Deviation	8.5
Relative 95% confidence	4.71%
Standard Assayed Statistics	
Number of data	32
Mean	366.22
Standard deviation	16.71
Relative 95% confidence	8.94%
MAD from certified mean	14.78
<=3SD from certified mean	75.00%
<=2SD from certified mean	65.63%
<=1SD from certified mean	46.88%

Figure 14.4 Standard 1 Cu (ppm) results 2009 - 2010

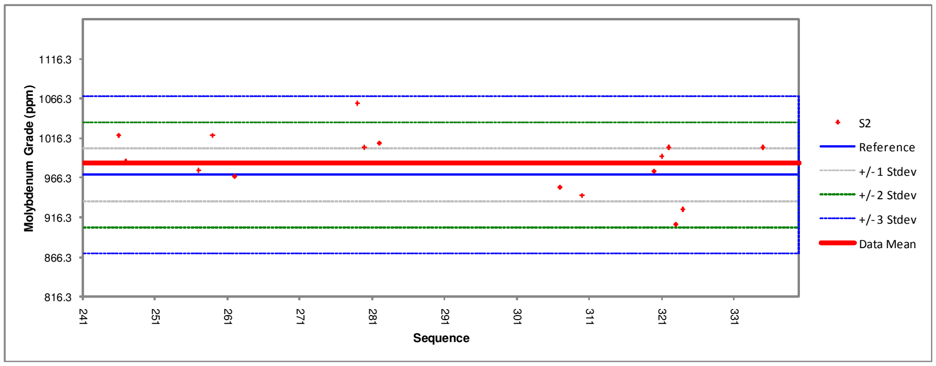


Summary statistics

Standard Reference Statistics	
Mean	1155
Standard Deviation	32.5
Relative 95% confidence	5.52%
Standard Assayed Statistics	
Number of data	32
Mean	1156.5625
Standard deviation	48.82
Relative 95% confidence	8.27%
MAD from certified mean	40.63
<=3SD from certified mean	96.88%
<=2SD from certified mean	81.25%
<=1SD from certified mean	43.75%

The results for Standard 2 (higher grade Mo and low grade Cu standard) are shown in Figure 14.5 and Figure 14.6. Results for Cu and Mo are acceptable but higher levels of variation are observed in the Cu analysis.

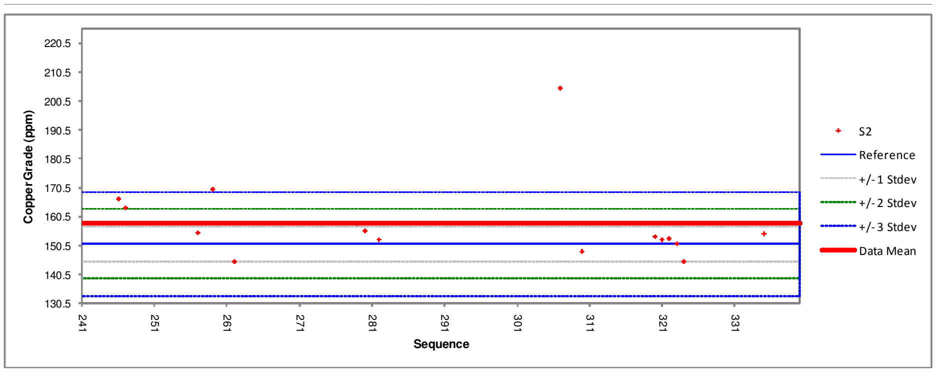
Figure 14.5 Standard 2 Mo (ppm) results 2009 - 2010



Summary statistics

Standard Reference Statistics	
Mean	970
Standard Deviation	33
Relative 95% confidence	6.67%
Standard Assayed Statistics	
Number of data	16
Mean	984.63
Standard deviation	38.79
Relative 95% confidence	7.72%
MAD from certified mean	33.38
<=3SD from certified mean	100.00%
<=2SD from certified mean	93.75%
<=1SD from certified mean	43.75%

Figure 14.6 Standard 2 Cu (ppm) results 2009 - 2010

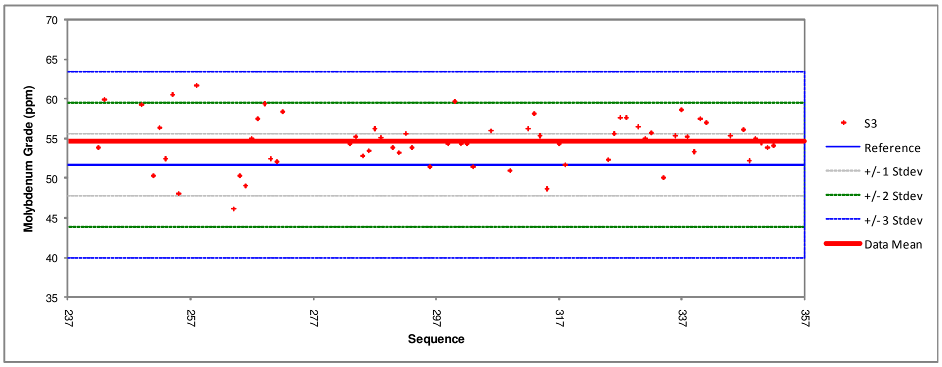


Summary statistics

Standard Reference Statistics	
Mean	151
Standard Deviation	6
Relative 95% confidence	7.79%
Standard Assayed Statistics	
Number of data	16
Mean	158.0625
Standard deviation	14.296707
Relative 95% confidence	17.73%
MAD from certified mean	8.875
<=3SD from certified mean	87.50%
<=2SD from certified mean	75.00%
<=1SD from certified mean	56.25%

Results for Standard 3, (a low grade Mo and low grade Cu standard), are shown in Figure 14.7 and Figure 14.8. Results for Mo show approximately 10 unacceptable results for Mo (i.e. samples which fall outside of the >±3SD thresholds) and two unacceptable results for Cu (i.e. >+3SD).

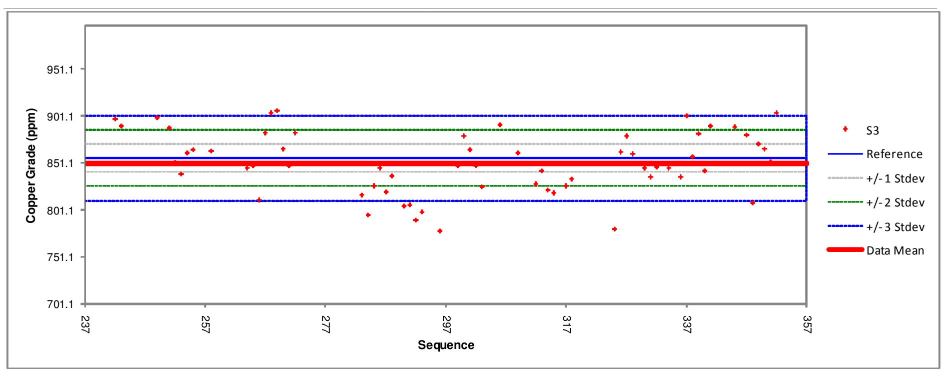
Figure 14.7 Standard 3 Mo (ppm) results 2009 - 2010



Summary statistics

Standard Reference Statistics	
Mean	51.7
Standard Deviation	3.9
Relative 95% confidence	14.79%
Standard Assayed Statistics	
Number of data	63
Mean	54.64
Standard deviation	3.13
Relative 95% confidence	11.24%
MAD from certified mean	3.60
<=3SD from certified mean	100.00%
<=2SD from certified mean	93.65%
<=1SD from certified mean	61.90%

Figure 14.8 Standard 3 Cu (ppm) results 2009 - 2010



Summary statistics

Standard Reference Statistics	
Mean	856
Standard Deviation	15
Relative 95% confidence	3.43%
Standard Assayed Statistics	
Number of data	63
Mean	850.90
Standard deviation	32.53
Relative 95% confidence	7.49%
MAD from certified mean	26.84
<=3SD from certified mean	80.95%
<=2SD from certified mean	60.32%
<=1SD from certified mean	36.51%

14.4.3 2009 – 2010 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 drillhole core samples are crushed in the first step in the preparation stage, two duplicate samples are taken for roughly every 20th sample being analysed by splitting the crushed half core. Mosquito has been taking coarse reject duplicates since 2006. Coarse reject duplicates are submitted to measure the precision of the sample preparation and analysis process. The first duplicate undergoes the same analytical procedure as the original sample (ICP-MS61) , while the second duplicate is analysed for molybdenum and copper using X-ray Florescent technique.

167 duplicate samples were submitted in 2009 and 2010, for a submission frequency rate of 1 in 20 samples.

A number of plots and graphs can be used to quantify precision and bias in the duplicate samples. These plots include:

- Scatter plot: assesses the degree of scatter of the duplicate result plotted against the original (first) assay value, which allows for bias characterisation and regression calculations.
- Precision plot: half of the absolute difference (HAD) of the sample pair values plotted against their average. The reference line indicates different levels of precision.
- Relative difference plot: relative difference of the paired values divided by their average.
- Ranked HARD plot: half absolute relative difference of samples plotted against their ranked value (samples are ordered from lowest to highest grade and ranked by percentile). For field duplicate samples in high nugget style deposits, the sample threshold is accepted to be 30% or below at the 90th percentile.
- The results of the Mosquito 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 within 10%. The results of the field duplicate samples are shown in Figure 14.9, Figure 14.10, and Figure 14.11.

Figure 14.9 Ag duplicate samples graphs

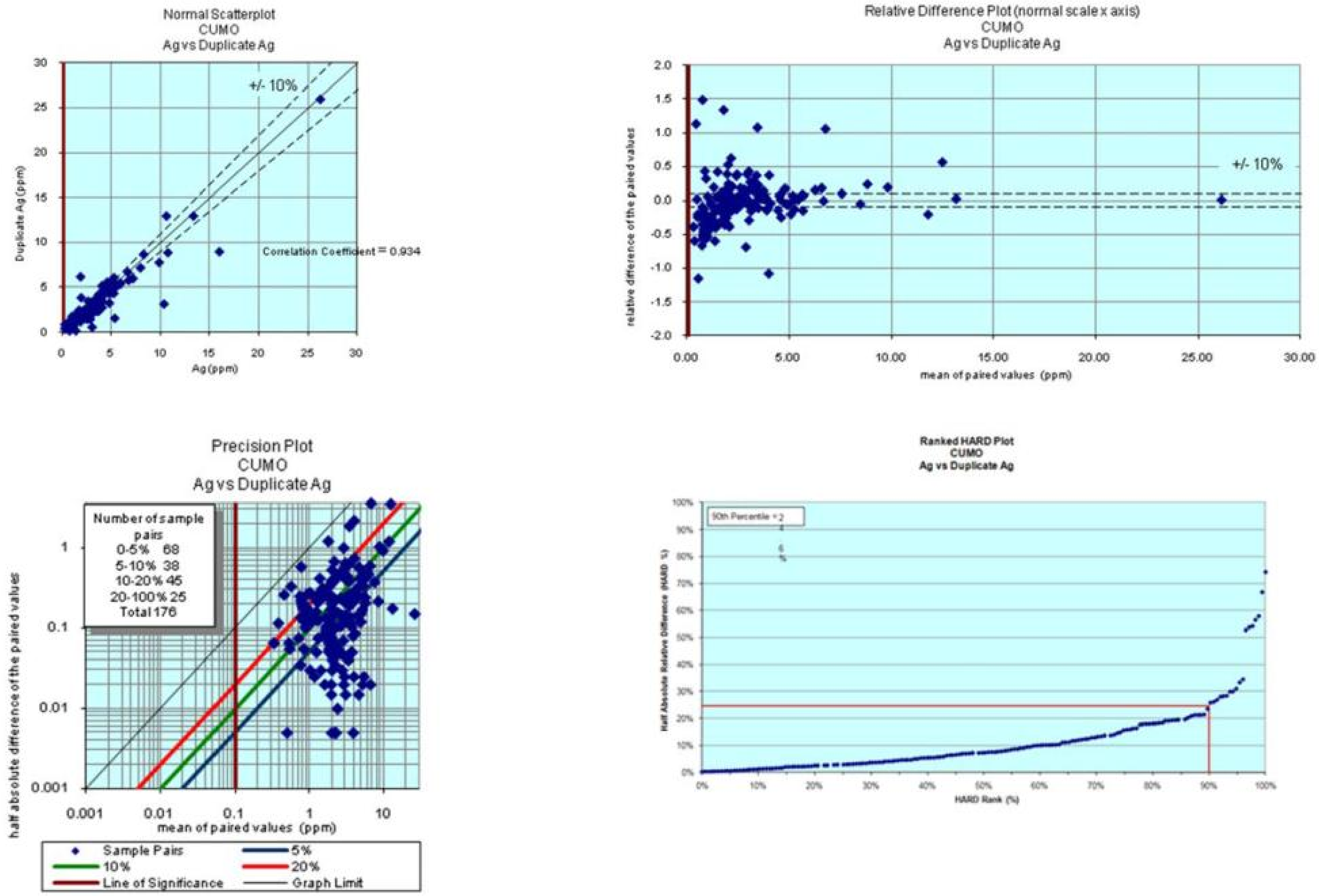


Figure 14.10 Cu duplicate samples graphs

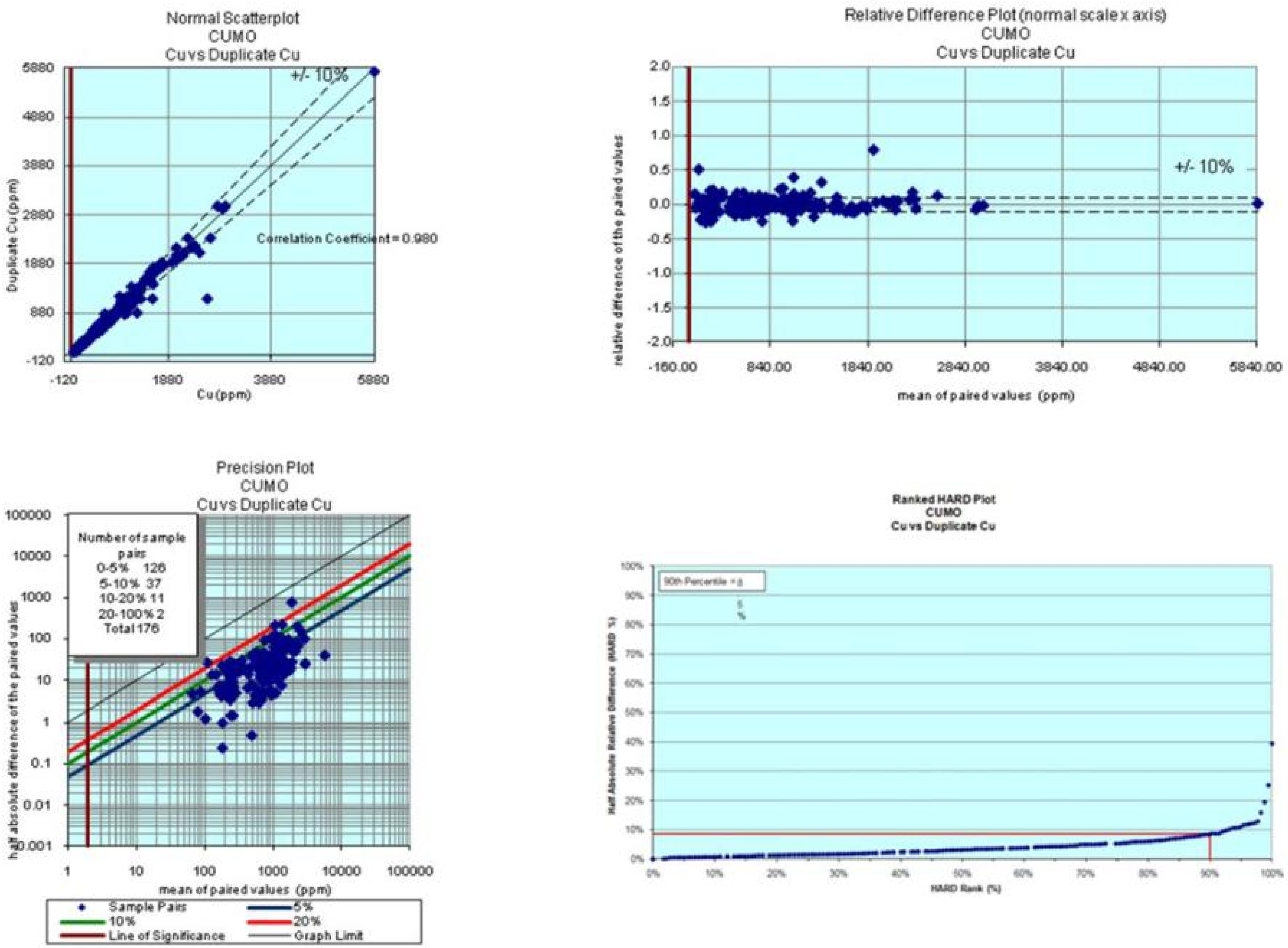
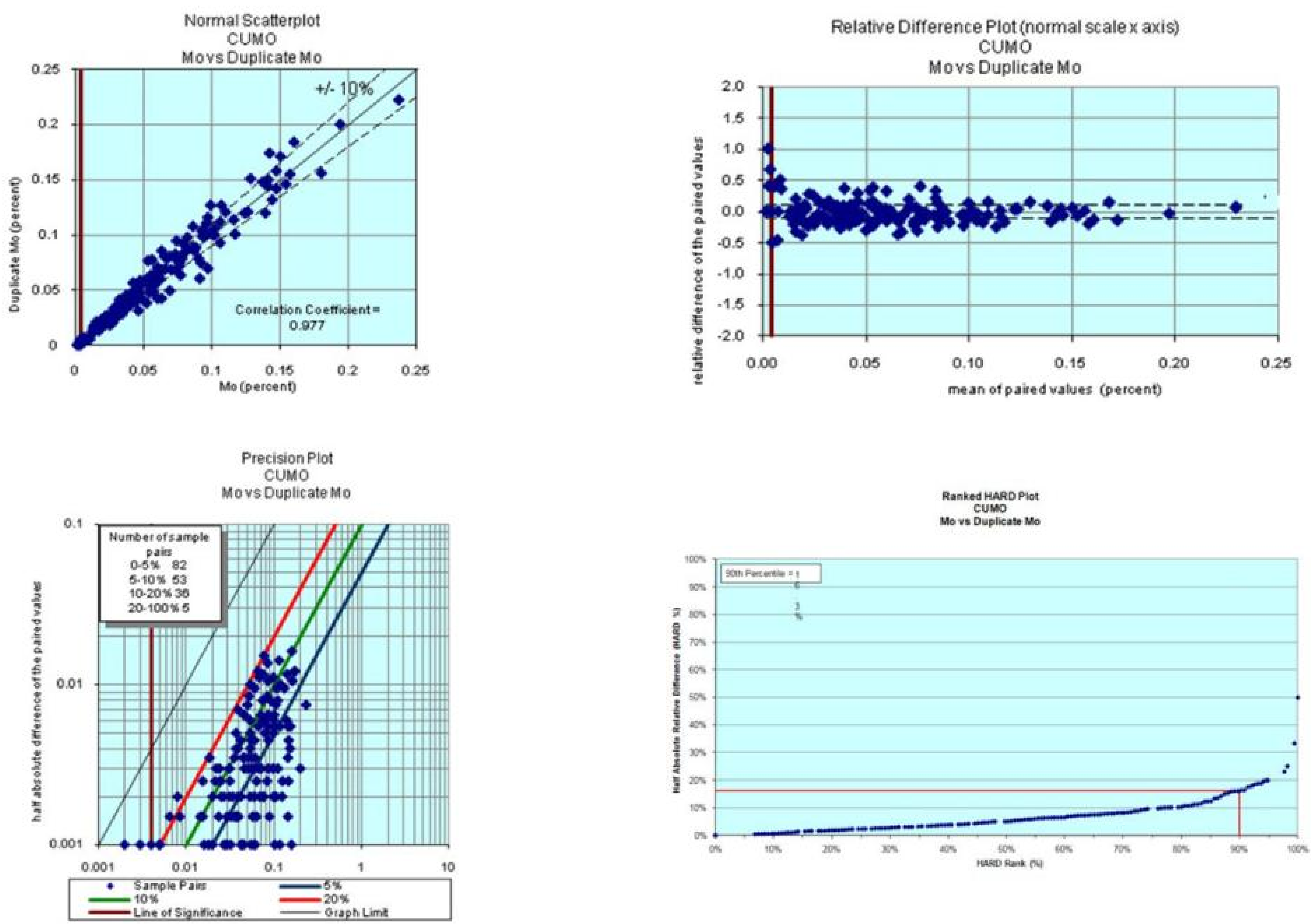


Figure 14.11 Mo duplicate samples graphs



14.5 Statements regarding verification

Snowden considers the type of QA/QC samples (i.e. standards, blank, and coarse crushed duplicates) submitted for the CUMO Project to be of a reasonable standard. The QA/QC results from the 2009 and 2010 blanks and coarse crushed duplicates do not indicate any significant source of bias or cross contamination.

However as noted there are some standard samples from 2009 and 2010 which presented values outside of the ±3SD tolerance limits in standard 1. This appears to be consistent with the earlier as well as the new data, but this variation in Snowden's opinion is consistent with an estimate with only moderate confidence as per the Indicated classification as applied.

Snowden recommends that a QA/QC protocol is adopted which rejects and re-analyses entire sample batches where any of the submitted QA/QC samples (i.e. duplicate, blank or standard) exceeds the accepted tolerance limits.

15 Adjacent properties

There are no adjacent properties applicable to the CUMO Project for disclosure in this report.

16 Mineral processing and metallurgical testing

This section is reproduced in total for completeness from "CUMO Property Preliminary Economic Assessment Throughput Scoping Study Report, November 18, 2009". For the purposes of this resource estimate update, it has been assumed that the metallurgical performance for the "msi" zone will be similar to the "Mo" zone. No metallurgical test work has been undertaken on the "msi" zone to date and therefore the assumed metallurgical performance has not been validated.

16.1 Metallurgical testing

16.1.1 Introduction

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. However, the existing test work data are considered suitable for a conceptual study and the comminution data are considered adequate for a conceptual milling 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 has been used to develop the process design criteria used in plant design.

The CUMO ores are of moderate competency and hardness, and amenable to grinding in a conventional SAG/ball milling circuit with pebble crushing (SABC). The mineralogy is fine grained and test work to date indicates the requirement for a fine target grind size to achieve adequate liberation for flotation.

Acid Based Accounting (ABA) 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 costs savings in the tailings handling circuit and a major reduction in the environmental impact of the project.

16.1.2 Sample selection

Mosquito began collecting metallurgical samples for 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 drillholes CO6-27, CO6-28 and CO6-29 and collected as individual 10-foot samples of quarter core selected as representative of the three mineralised 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 43-101 compliant 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".

16.1.3 Test work program

The 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 ore types in the CUMO deposit. 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 metallurgical testing were undertaken on the CUMO ore body 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 ore type composites.

Table 16.1 summarises 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 ore. To date no samples have had 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 ores.

Table 16.1 Summary of Comminution Test Work Data

Comminution Characteristics		Cu-Ag	Cu-Mo	Mo	Design
Specific gravity	t/m^3	2.64	2.60	2.60	2.64
SPI®	min	84.5	73.0	70.8	84.5
SMC DWI	kW/m^3	n/a	n/a	n/a	7.40
Crushing work index	kWh/t	n/a	n/a	n/a	15.8
Bond rod mill work index	kWh/t	n/a	n/a	n/a	15.8
Bond ball mill work index (closing screen 106 µm)	kWh/t	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 upper percentile competency and ensure a robust design. This premise will need to be tested in the next phase of study as more detailed mine schedule information and ore comminution characteristics become available.

Flotation test work results

Flotation test work was completed prior to the commencement of the Conceptual Study, commencing with rougher kinetic flotation testing and culminating with locked cycle testing of the major ore types. Only bulk sulfide flotation 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.

16.1.4 Conceptual Study Flotation Test Work

The Conceptual Study flotation test work program was divided into three phases:

- rougher flotation
- open circuit cleaner flotation
- locked cycle flotation.

Rougher Flotation

Initially, a series of rougher flotation tests were conducted to determine the sensitivity of the ore 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 copper/silver ore 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 exhibit little sensitivity to grind size.
- Target elements showed little sensitivity to grind size for the copper/molybdenum ore, 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 molybdenum ore type exhibited significant sensitivity to grind size. Although the sensitivity of the molybdenum was lower, the finer grind resulted in an increase in molybdenum recovery.
- Sulfur assays on the concentrates from the copper/silver and copper/molybdenum ores indicate the presence of a floatable sulfide gangue mineral; most likely pyrite (no sulfur assays were available for the molybdenum ore).

The results of these tests are summarised in Table 16.2.

Table 16.2 Baseline flotation results for CUMO composite samples

Ore Type	Test No.	Feed		Concentrate Grade			Concentrate Recovery		
		% Cu	g/t Mo	% Cu	% Mo	g/t Ag	% Cu	% Mo	% Ag
Cu-Ag	VF1-1	0.16	213	1.22	0.18	39	76.5	87.7	78.0
	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	74.0
	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	77.0	94.4	64.4
	VF3-2	0.03	1135	0.44	1.75	12	83.1	96.9	71.8

The tests indicate that these ores were amenable to flotation, resulting in good recovery of target mineral species into a low mass concentrate stream. The sensitivity of the ores to primary grind size indicates that a fine grind will be required to ensure good recovery. Additional grind sensitivity test work should be included in subsequent testing to optimise 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 in the ore, reporting to final concentrate, a non-specific sulfide collector (SIBX) was used for the cleaner flotation testing.

The fine grain structure of the ores 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 optimisation tests are summarized in Table 16.3.

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Table 16.3 Cleaner flotation results for CUMO composite samples

Ore Type	Test No.	Feed		Concentrate Grade			Concentrate Recovery		
		% Cu	g/t Mo	% Cu	% Mo	g/t Ag	% Cu	% Mo	% Ag
Cu-Ag	VF1-3	0.14	176	19.8	3.32	596	49.6	68.2	49.0
	VF1-4	0.16	185	15.3	2.30	462	64.0	81.3	64.9
	VF1-5	0.15	175	16.4	2.68	539	55.6	79.0	41.2
Cu-Mo	VF2-3	0.12	392	18.0	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

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 ore 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 ore 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 ore types for project evaluation.

Locked Cycle Test Work at Design Conditions

Flotation results from the optimisation 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 ore 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 ore 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 ore types, the results are summarised in Table 16.4.

Table 16.4 Locked cycle test results

Ore Type	Test No.	Feed		Concentrate Grade			Concentrate Recovery		
		% Cu	g/t Mo	% Cu	% Mo	g/t Ag	% Cu	% Mo	% Ag
Cu-Ag	VF1-LCT1	0.16	190	13.0	2.00	357	62.5	82.0%	71.7%
Cu-Mo	VF2-LCT1	0.12	401	16.4	5.66	324	90.7	93.8%	80.0%
MO	VF3-LCT1	0.04	1065	5.1	21.6	122	71.6	99.6%	59.3%

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 maximise their rejection whilst maintaining target recoveries.

16.1.5 Grade and Recovery Predictions

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 oxidised and non-oxidised 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 oxidised and non-oxidised material.

Ausenco has reviewed the specified recoveries and believes that they are reasonable for a bulk concentrate from the CUMO ore types. However, as discussed, the concentrate grades achieved from the tests do not reflect those required to achieve saleable concentrates and have been adjusted for the plant design and economic evaluation on the assumption that additional test work will further optimise flotation metallurgy, allowing higher concentrate grades to be achieved with minimal impact on recovery. This assumption will require confirmation and testing during subsequent project phases.

To produce saleable concentrates from the CUMO bulk concentrates, separation of the molybdenum and copper into separate concentrates is required. To date no test work has been undertaken to determine the actual concentrate grades and recoveries achievable after separation, nor to determine what process steps are required to achieve adequate separation.

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.

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 ores are shown in Table 16.5. These figures include bulk concentrate recovery, copper/molybdenum flotation separation and ferric chloride leach recovery.

Table 16.5 Grade/recovery predictions for CUMO ores

Ore 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	1.0	55

16.2 Mineral Processing

16.2.1 General

The CUMO process plant and associated service facilities will process ROM ore delivered to the primary crusher, to produce separate copper and molybdenum sulfide concentrates and tailings. The proposed process encompasses crushing and grinding of the ROM ore, bulk rougher and cleaner flotation, regrinding, molybdenum separation and dewatering of copper/molybdenum sulfides. Molybdenum sulfides will be further processed downstream in a roaster to produce a saleable molybdenum oxide concentrate. 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 Tailings Storage Facility (TSF).

The design incorporates a multiple grinding line approach with the ability to expand flotation and further downstream processes as needed. The process includes a gyratory crusher, stockpile conveyor, coarse ore stockpile, SAG and ball mill grinding circuit, bulk flotation circuit including regrind, molybdenum flotation circuit, concentrate dewatering, molybdenum concentrate leach circuit, molybdenum roasting, concentrate load-out and tailings thickening facilities.

The concentrator will use a conventional processing flow sheet and industry standard equipment. Concentrator 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.

16.2.2 Design Criteria Summary

The overall approach was to design a robust process plant that could be scaled up easily to the various tonnage scenarios proposed, and deliver good value for capital. The key project and ore specific criteria for the plant design and operating costs are provided in Table 16.6.

Table 16.6 Summary of the process plant design criteria

Criteria		Units	Design			
			50 kt/d (short tons)	100 kt/d (short tons)	150 kt/d (short tons)	200 kt/d (short tons)
Crusher Feed		kt/d (short tons)	50	100	150	200
		Mt/y (metric tons)	16.6	33.1	49.7	66.2
Crusher Availability		%	65	65	65	65
Crusher Throughput		t/h	2 907	5 814	8 721	11 629
Crusher Selection	Size		60 x 89	60 x 110	60 x 110	60 x 110
	No		1.0	1.0	2.0	2.0
Mill Throughput		Mt/y (metric tons)	16.6	33.1	49.7	66.2
Mill/Flotation Availability		%	92	92	92	92
Mill Throughput		metric t/h	2 054	4 108	6 162	8 216
Physical Characteristics	BWI	kWh/t (metric)	15.8	15.8	15.8	15.8
	SPI®	Mins	84.5	84.5	84.5	84.5
Grind Size	P_{80}	µm	63	63	63	63
Head Grade (Design)		% Cu	0.10	0.10	0.10	0.10
		% MoS_2	0.11	0.11	0.11	0.11
		g/t Ag	2.87	2.87	2.87	2.87
Flotation Recovery (Cu-Ag Ore)	Copper	%	64.3	64.3	64.3	64.3
	Silver	%	70.0	70.0	70.0	70.0
	Molybdenum	%	83.0	83.0	83.0	83.0
Flotation Recovery (Cu-Mo Ore)	Copper	%	85.0	85.0	85.0	85.0
	Silver	%	78.0	78.0	78.0	78.0
	Molybdenum	%	92.0	92.0	92.0	92.0
Flotation Recovery (Mo Ore)	Copper	%	72.0	72.0	72.0	72.0
	Silver	%	55.0	55.0	55.0	55.0
	Molybdenum	%	95.0	95.0	95.0	95.0
Cu Circuit Residence time	Roughers	Mins	27.5	27.5	27.5	27.5

Table 16.6 cont. Summary of the process plant design criteria

Criteria		Units	Design			
			50 kt/d (short tons)	100 kt/d (short tons)	150 kt/d (short tons)	200 kt/d (short tons)
	Cleaner 1	Mins	10	10	10	10
	Cleaner Scav.	Mins	2.5	2.5	2.5	2.5
	Cleaner 2	Mins	10	10	10	10
	Cleaner 3	Mins	5	5	5	5
Mo Circuit Residence time	Roughers	Mins	35	35	35	35
	Cleaner 1	Mins	25	25	25	25
	Cleaner Scav.	Mins	25	25	25	25
	Cleaner 2	Mins	25	25	25	25
	Cleaner 3	Mins	25	25	25	25
Cu Concentrate Filtration Rate		$kg/m^2/h$	262	262	262	262
Concentrates Thickening Flux		$t/m^2/h$	0.1	0.1	0.1	0.1
Mo Concentrate Filtration Rate		$kg/m^2/h$	356	356	356	356
Tailings Thickening Flux		$kg/m^2/h$	800	800	800	800
Tailings Thickener Underflow Density		% w/w	65	65	65	65
Collector Consumption (SIBX)		g/t (short ton)	66	66	66	66
Collector Consumption (Aero 3302)		g/t (short ton)	59	59	59	59
Activator Consumption (Moly Oil)		g/t (short ton)	51	51	51	51
Frother Consumption (X-133)		g/t (short ton)	67	67	67	67
Lime Consumption		kg/t (short ton)	0.18	0.18	0.18	0.18
Flocculant Consumption (Concentrate and tailings)		g/t (short ton)	15	15	15	15
SAG Mill Media Consumption		kg/t (short ton)	0.25	0.25	0.25	0.25
Ball Mill Media Consumption		kg/t (short ton)	0.55	0.55	0.55	0.55
Regrind Mill Media Consumption		kg/t (short ton)	0.04	0.04	0.04	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.

Plant design basis

The key criteria selected for the plant design are:

- Treatment of 50,000 short tons per day (50 kt/d), 100 kt/d (short tons), 150 kt/d (short tons) and 200 kt/d (short tons). These are approximately equivalent to 45,000 metric tonnes per day, 91 kt/d (metric tonnes), 136 kt/d (metric tonnes) and181 kt/d (metric tonnes)
- Design availability of 92% (after ramp-up), being 8,059 operating hours per year, with standby equipment in critical areas
- Sufficient plant design flexibility for treatment of all ore types at design throughput.

The selection of these parameters is discussed in detail below.

16.2.3 Throughput and Availability

Four different throughput scenarios were nominated by Mosquito to evaluate different corporate investment hurdles. Ausenco has nominated an overall plant availability of 92% or 8,059 h/y. This is an industry standard for a large, multi-train, flotation plant with moderately abrasive ore. Benchmarking indicates that similar plants have consistently achieved this level.

16.2.4 Processing Strategy

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

16.2.5 Head Grade

The plant is designed to treat various tonnages of primary ore with a maximum head grade of 0.08% Cu and 0.07% Mo (0.11% MoS_2).

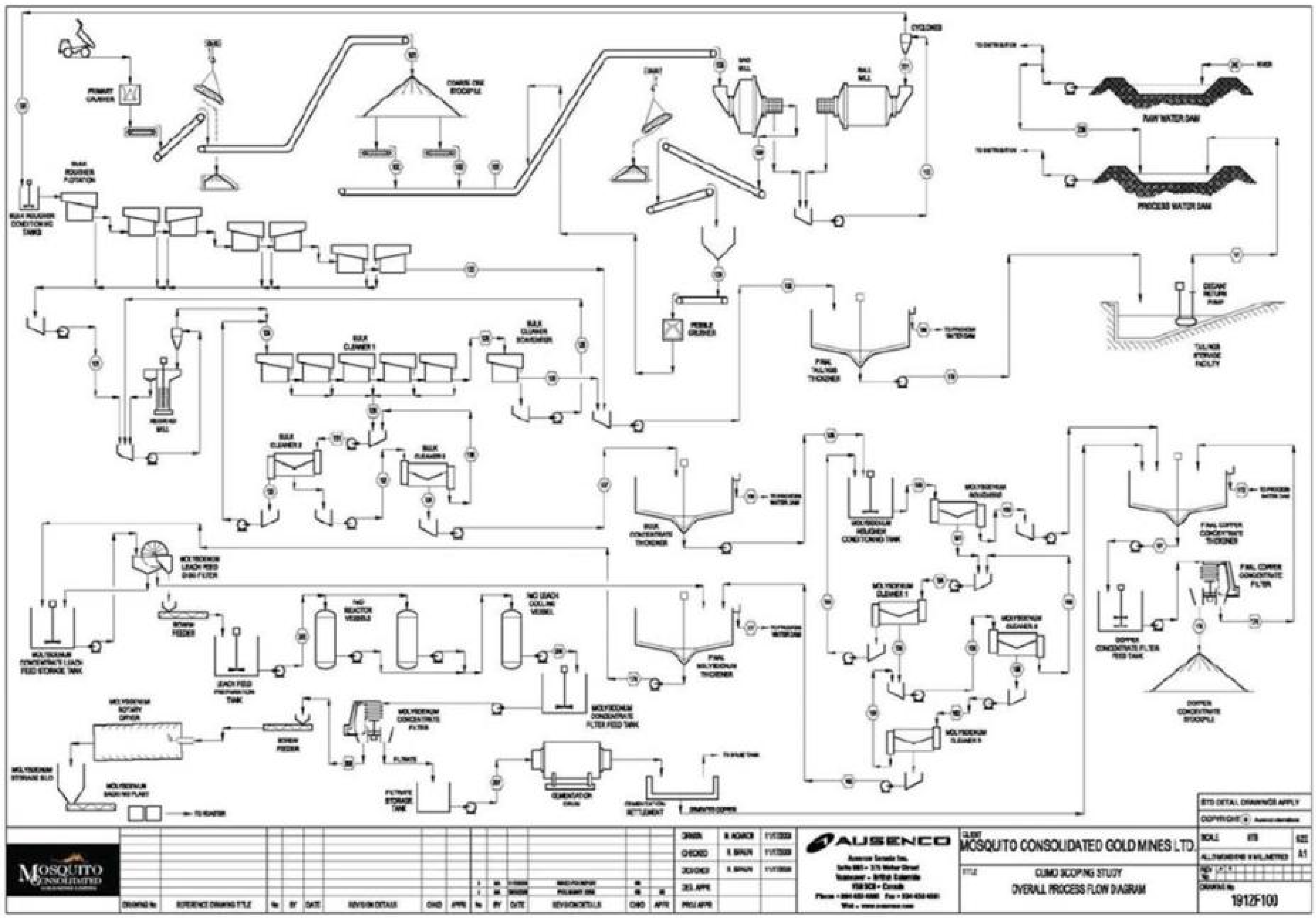
16.3 Flow Sheet Development and Equipment Sizing

The process plant flow sheet design for the CUMO circuit was conceptually based on those of comparable large flotation plants.

Figure 16.1 shows a process schematic for the CUMO plant.

Details of the flow sheet design and selection of major equipment for the various options are discussed in the sections below.

Figure 16.1 CUMO Process Plant Process Schematic



16.3.1 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 maximise the economies of scale possible with large equipment. However, the general design consists of a number of 50 kt/d (short tons) modules to achieve the differing throughput rates. Each module typically consists of the following unit processes:

- Ore 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 stockpile feed conveyor
- Conical stockpile of crushed ore with a live capacity of 18 h, with two apron feeders per grinding train, each capable of feeding 120% of the full mill throughput
- A 22 MW SAG mill, 11.58 m diameter with 7.60 m EGL, in closed circuit with pebble crushing
- Pebble crushing will be comprised of 2 MP800's 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 diameter with 12.19 m EGL, in closed circuit with hydrocyclones, grinding to a product size of nominally 80% passing (P80) 63 µm
- Bulk rougher flotation consisting of 200 m3 forced air tank flotation cells to provide a total of 28 minutes of retention time
- Rougher concentrate regrinding in 3 off 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 m3 forced air tank flotation cells to provide a total of 13 minutes of retention time
- Bulk cleaner 2 flotation cells consisting of 8 m3 trough shaped flotation cells to provide a total of 10 minutes of retention time
- Bulk cleaner 3 flotation cells consisting of 8 m3 trough shaped flotation cells to provide a total of 5 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

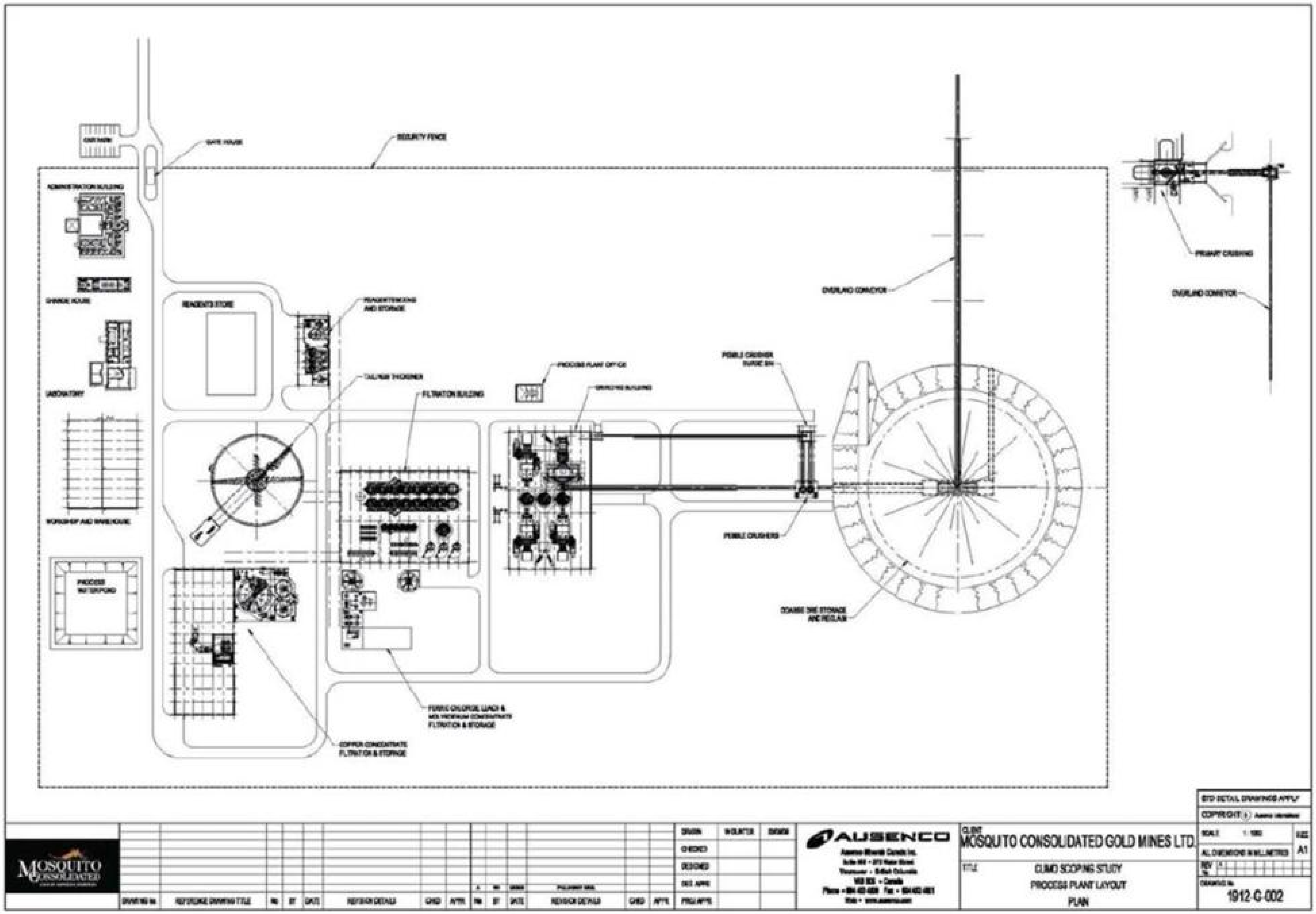
- Molybdenum ferric chloride leach in 4 000 U.S. gallon, glass lined steel leach reactors followed by drying and storage in bulk 1 ton bags
- Tailings thickening in a high rate thickener to an underflow density of 65% solids TSF for process tailings in a conventional dam
- Raw process plant water supply 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)
- 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.

16.3.2 Process Plant Layout

Basic layouts have been prepared based on a near pit crusher, overland conveyor and an SABC circuit. The layout for the 50 kt/d module is shown below in Figure 16.2.

The circuit layout has taken cognisance of the site topography and worked within the bounds imposed by preliminary locations of the pit, stockpiles and waste dumps.

Figure 16.2 CUMO Process Plant Layout



17 Mineral Resource and Mineral Reserve estimates

17.1 Disclosure

Mineral Resources reported in Section 17 were prepared and supervised by Mr Ivor Jones, a Senior Principal Consultant of Snowden and a Qualified Person as defined under NI 43-101. Documentation of the work was reviewed by Mr. Andy Ross, Senior Principal Consultant for Snowden's Perth office.

Snowden is independent of Mosquito.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. No Mineral Reserves are reported in this Technical Report.

This report uses definitions from and follows the guidelines of the CIM Definition Standards for Mineral Resources and Mineral Reserves and NI 43-101 Form F1. The Project has no mine design or defined economic parameters at this stage.

17.1.1 Known issues or factors that materially affect Mineral Resources

Snowden is unaware of any issues or factors that could materially affect the Mineral Resource estimates. These conclusions are based on the following:

Environmental and permitting

Mosquito has represented that there are no outstanding environmental issues pending against the Project other than those which are described in detail in section 4.3 of this report.

Legal

Mosquito has reported that there is an outstanding legal issue pertaining to the project that is currently being addressed in court (refer to section 4.3).

Title

Mosquito has represented that there are no outstanding title issues pending against the Project.

Taxation

Mosquito has represented that there are no outstanding taxation issues pending against the Project.

Socio-economic

Mosquito has represented that the Project has strong local community support.

Marketing

Mosquito has represented that there are no known marketing issues which could affect the Mineral Resources.

Political

Mosquito has represented that there are no known political issues with the Project.

Mining

Mosquito has represented that there are no known technical mining factors which could affect the Mineral Resources.

Metallurgical

Ausenco has demonstrated the potential for processing CUMO mineralization via a conventional process plant encompassing crushing, grinding and flotation (refer to Section 16).

Infrastructure

Mosquito has represented that there are no known infrastructure factors which could affect the project

17.2 Assumptions, methods and parameters

17.2.1 Software

The software used for the resource estimation was Datamine.

17.2.2 Supplied data

Mosquito provided raw drillhole data, including survey, assay, collar, geology and bulk density in Excel spreadsheet format, surface topography data in AutoCAD DXF format, and relevant technical documentation. Geology model outlines and a 50 ft x 50 ft x 50 ft geological block model, in csv format, coded by geology, was also provided to Snowden by Mosquito for the CUMO project.

17.2.3 Data preparation

Snowden prepared a de-surveyed drillhole file from the collar, survey, lithology, and assay data provided by Mosquito. The current resource estimation has been conducted using the drillhole data delivered to Snowden as follows:

The pre-2009 database was delivered directly from Gary Giroux on December 21, 2010 in order to maintain the integrity of the data; while the data for the 2009-2010 drilling was delivered during February 2011 following receipt of all assay data from the 2010 program. A location map showing drillholes available for the April 2011 Mineral Resource estimate is shown in Table 6.2. The drillholes used in the CUMO Mineral Resource estimate are shown in Table 6.1.

After conducting a block dimension study, Snowden generated a geological block model with a block size of 100 ft x 100 ft x 50 ft, using the block model extents, geological codes and volume percentage information, which was provided in csv format by Mosquito.

17.2.4 Data validation

Validation checks in Datamine mining software included searches for overlaps or gaps in sample and geology intervals, inconsistent drillhole identifiers, and missing data. No errors were noted.

17.2.5 Data transformations

The composite data was transformed to the pre-fault location based on displacement of marker beds across the fault boundaries prior to variogram analysis. No transformation or rotation has been done on the data for the block estimation.

17.2.6 Geological interpretation

Snowden performed a visual validation of the block model both in section and plan view comparing the block model with wireframe slices representing each of the domain boundaries. After resolving some minor issues with the data provided by Mosquito, this block model was used for the resource estimation process.

17.2.7 Definition of grade estimation domains

Grade estimation domains, which are subsets of the sample data, ensure that samples used for estimating a block grade are from the same population as the point of estimation. A grade population may be defined by attributes such as spatial location, lithology, mineralisation style, and structural boundaries.

Five mineralization domains have been defined for use in the April 2011 CUMO resource model. The five domains are as follows:

- an oxide domain (oxide)
- a copper-gold domain (cuag)
- a copper-molybdenum domain (cumo)
- a molybdenum domain (mo)
- and a low grade molybdenum-copper domain (msi)

These domains were defined through a process of geological interpretation and statistical analysis and the abbreviated codes have been assigned to samples, composites and into the geological block model for use in geostatistical analysis, grade estimation and resource reporting.

For both geological and statistical reasons, a soft boundary was used for copper and silver between the cuag and cumo domains, and for molybdenum between the cumo, mo, and msi domains.

17.2.8 Block model set-up

The block model parameters used to generate the grade model was based on the block model provided by Mosquito. A summary of the block model parameters is shown in Table 17.1.

Table 17.1 CUMO block model parameters

Direction	Minimum (ft.)	Maximum (ft.)	Increment (ft.)
Easting	214,600	225,000	100
Northing	114,250	123,250	100
Elevation	3,275	7,075	50

17.2.9 Exploratory data analysis

Compositing of assay intervals

Sample lengths were composited to ensure that the samples used in statistical analyses and estimations have similar support (i.e. length). CUMO drillholes were sampled at various interval lengths depending on the length of intersected geological features. Sample lengths were examined and a composite length of 20 ft was selected as this corresponds with the most frequently sampled length interval (Table 17.2). The composited and raw sample data were compared to ensure no sample length loss or metal loss had occurred.

The Datamine COMPDH downhole compositing process was used to composite the samples within the estimation, the MODE parameter was set to a value of 1 to allow adjustment of the composite length while keeping it as close as possible to the selected composite interval (i.e. 20 ft).). See Table 17.3.

Table 17.2 shows the summary statistics for molybdenum, copper, silver and tungsten by domain.

Table 17.2 Sample summary statistics

Domain	FIELD	NUMTRACE	MINIMUM	MAXIMUM	MEAN	STANDDEV	STANDERR
cuag	MOS2	1935	0.00	0.32	0.02	0.00	0.02
	CUPC	1912	0.00	0.77	0.09	0.01	0.08
	AGPPM	1937	0.01	345.00	3.01	85.08	9.22
	WPPM	1933	0.10	520.00	31.05	729.84	27.02
	LENGTH	1937	2.00	30.00	10.78	7.99	2.83
cumo	MOS2	3011	0.00	1.09	0.05	0.00	0.04
	CUPC	3009	0.00	0.92	0.11	0.01	0.07
	AGPPM	2996	0.01	744.00	3.16	203.98	14.28
	WPPM	3001	0.10	411.00	46.41	998.85	31.60
	LENGTH	3012	2.00	40.00	11.32	12.35	3.51
mo	MOS2	2365	0.00	0.99	0.11	0.00	0.07
	CUPC	2365	0.00	0.59	0.05	0.00	0.04
	AGPPM	2341	0.01	494.00	1.82	108.61	10.42
	WPPM	2342	0.10	890.00	45.66	1444.22	38.00
	LENGTH	2365	1.50	60.00	12.16	21.21	4.61
msi	MOS2	283	0.00	0.17	0.06	0.00	0.03
	CUPC	275	0.00	0.20	0.03	0.00	0.04
	AGPPM	280	0.01	182.00	1.88	125.37	11.20
	WPPM	283	3.30	1980.00	39.45	13838.28	117.64
	LENGTH	283	3.80	30.00	12.58	21.28	4.61
ox	MOS2	812	0.00	0.16	0.02	0.00	0.02
	CUPC	804	0.00	0.71	0.06	0.00	0.06
	AGPPM	807	0.01	19.05	2.10	3.84	1.96
	WPPM	796	0.10	375.00	26.90	862.41	29.37
	LENGTH	814	3.50	79.40	11.73	25.43	5.04

Table 17.3 Composite summary statistics

Domain	FIELD	NUMTRACE	MINIMUM	MAXIMUM	MEAN	STANDDEV	STANDERR
cuag	AGPPM	1052	0.19	176.37	3.00	6.90	0.21
	CUPC	1047	0.00	0.54	0.09	0.07	0.00
	LENGTH	1052	13.00	21.17	19.85	0.40	0.01
	MOS2	1052	0.00	0.19	0.02	0.02	0.00
	WPPM	1048	4.55	268.37	30.91	23.07	0.71
cumo	AGPPM	1699	0.02	316.84	3.15	8.83	0.21
	CUPC	1714	0.00	0.63	0.11	0.07	0.00
	LENGTH	1714	9.50	20.55	19.90	0.40	0.01
	MOS2	1714	0.00	0.80	0.05	0.04	0.00
	WPPM	1703	0.50	234.79	46.06	26.24	0.64
mo	AGPPM	1418	0.07	139.99	1.84	4.97	0.13
	CUPC	1441	0.00	0.31	0.05	0.04	0.00
	LENGTH	1441	18.33	20.46	19.96	0.20	0.01
	MOS2	1441	0.00	0.52	0.11	0.06	0.00
	WPPM	1419	2.64	400.28	47.00	30.62	0.81
msi	AGPPM	178	0.04	89.55	1.92	7.54	0.57
	CUPC	181	0.00	0.20	0.03	0.04	0.00
	LENGTH	181	4.00	20.19	19.66	1.26	0.09
	MOS2	181	0.00	0.13	0.06	0.03	0.00
	WPPM	181	3.66	994.43	38.60	74.00	5.50
OX	AGPPM	486	0.01	19.05	2.21	1.93	0.09
	CUPC	488	0.00	0.37	0.06	0.06	0.00
	LENGTH	488	15.00	21.60	19.57	0.80	0.04
	MOS2	488	0.00	0.13	0.02	0.02	0.00
	WPPM	477	0.10	210.00	27.32	28.09	1.29

17.2.10 Extreme value treatment

Top-cuts were applied to composites prior to the ordinary kriging grade estimation process. The values shown in Table 17.4 were determined from a combination of histograms at the point where the distribution begins to breakdown and become erratic and from a top cut analysis spreadsheet where it was possible to determine the effect on the mean grade and CV as well as the number of samples that are affected by the top-cut.

Table 17.4 Top-cuts applied by geological domain

Geological Domain	Metal	NO.Of. Samples	Capped Value	Capped mean	Uncapped mean(%)	Metal lost(%)	Capped CV	Uncapped CV
oxide	MoS_2%	491	0.1	0.02	0.02	0.01	1.07	1.09
	Cu%	491	0.3	0.06	0.06	1	0.9	1
	Ag (gpt)	489	8	2.17	2.21	2	0.8	0.9
	W (gpt)	480	115	26	27.23	4	0.8	1
cuag	MoS_2%	1052	0.11	0.018	0.018	0	0.894	0.936
	Cu%	1052	0.45	0.09	0.09	0	0.76	0.77
	Ag (gpt)	1052	13	2.7	3	10	0.8	2.3
	W (gpt)	1048	128	30.64	30.91	1	0.7	0.75
cumo	MoS_2%	1715	0.18	0.05	0.05	1	0.6	0.8
	Cu%	1715	0.5	0.11	0.11	0	0.6	0.6
	Ag (gpt)	1699	13.5	2.8	3.15	11	0.7	2.8
	W (gpt)	1703	185	45.98	46.06	0.18	0.6	0.6
mo	MoS_2%	1441	0.4	0.11	0.11	0	0.5	0.5
	Cu%	1441	0.25	0.05	0.05	0	0.8	0.8
	Ag (gpt)	1418	8	1.64	1.84	11	0.8	2.7
	W (gpt)	1419	190	46.65	46.99	1	0.6	0.7
msi	MoS_2%	181	-	0.06	0.06	0	0.5	0.5
	Cu%	181	0.15	0.03	0.03	0.02	1.3	1.34
	Ag (gpt)	178	4	1.11	1.92	42	1	3.9
	W (gpt)	181	80	32.72	38.6	15	0.5	1.9

17.2.11 Variogram analysis

Spatial grade continuity, as indicated by the variogram, is an important consideration when assigning Resource confidence classification. Variogram characteristics strongly influence estimation quality parameters such as kriging efficiency and regression slope.

The composite data was transformed to the pre-fault location based on marker beds displaced across fault boundaries. Normal Scores variograms were generated using the transformed data for all domains, then the normal scores variogram models were back-transformed into the original distribution. For molybdenum, data within the cumo, mo and msi domains were combined for the purpose of variogram generation and modeling, due to the transitional nature of the molybdenum grades across the domain boundaries. Similarly, data for copper, silver and tungsten within the cumo and cuag domains were also combined.

The back-transformed variogram parameters are tabulated in Table 17.5.

Table 17.5 Back transformed variogram model parameters

Grade	Domain	Orientation Studio 3 ZXZ angles	Nugget	Structure 1 Sill	Structure 1 Range	Structure 2 Sill	Structure 2 Range	Structure 3 Sill	Structure 3 Range
MoS2	OXIDE	0			810		1000		
		0	0.16	0.4	1300	0.5	1600		
		55			32		200		
	CUAG	-30			276		898		
		110	0.18	0.3	940	0.5	1200		
		-90			800		850		
	CUMO_MO_MSI	-30			50		661		950
		120	0.2	0.2	300	0.3	600	0.3	3000
		-90			400		450		500
Cu	OXIDE	-25			1348		2015		
		170	0.1	0.5	895.5	0.4	1532		
		-130			87.5		228		
	CUMO	-30			129		1806		3000
		130	0.14	0.1	740	0.4	1021	0.3	9999
		-100			350		600		1200
	Mo-MSI	-20			50		570		3000
		120	0.3	0.2	300	0.5	650	0.3	9999
		-90			300		400		1200
Ag	OXIDE	-25			200.5		1798		
		170	0.09	0.4	250.5	0.5	705		
		0			47		344.5		
	CUMO_CUAG	-30			1320		1600		1650
		130	0.17	0.2	550	0.3	983	0.3	9999
		-90			560		600		2140
	Mo-MSI	-30			40		860		
		120	0.06	0.3	400	0.7	2150		
		-90			200		480		
W	OXIDE	0			391		595		
		0	0.22	0.4	246	0.4	1685		
		55			48		189		
	CUMO_CUAG	-30			320		672.5		900
		130	0.18	0.2	200	0.4	442.5	0.2	9999
		-110			365.5		429.5		526

17.2.12 Kriging neighborhood analysis

Kriging neighborhood analysis (KNA) is the name given to the process of systematically optimizing the kriging parameters via the examination of conditional bias or smoothing statistics prior to undertaking the actual grade estimation of the domains.

The conditional bias statistics examined are as follows:

- Kriging Efficiency (KE)
- Slope of the regression (PSLOPE)

KE compares the kriging variance against the block variance. If these two variance values are similar, then the set of parameters and the data configuration used to estimate grades is likely to produce a locally accurate estimate of grade.

PSLOPE describes the expected regression slope between the actual and the estimated grades. If PSLOPE is close to one, then a strong relationship is expected between the estimated grades and the actual grades (theoretically calculated).

Both of these statistics provide an indication of how well the set of parameters and the data configuration can predict local block grades. These data statistics however can be distorted (lowered) in the case of narrow ore bodies and as such Snowden prefers to compile several kriging runs, varying settings such as block size for comparison. The analysis is then undertaken using the KE and slope values from the multiple kriging runs as a relative measure (i.e. selecting the highest values).

17.2.13 Block Size

KNA was undertaken using the MoS_2 and Cu grades from the cumo domain to determine the optimum block size. The study was undertaken using the following settings:

- For the MoS_2 KNA analysis, the variogram parameters were taken from the cumo-mo-msi variogram model.
- For the Cu KNA analysis, the variogram parameters were taken from the cumo-cuag variogram model.
- The search rotation parameters were taken from the variogram rotation parameters while the maximum search distances were set to 600 ft. by 1500 ft. by 250 ft. in the X, Y, and Z directions respectively. These search distances exceed the average drill spacing.
- The model was constrained to within the following area as only limited data is available outside this area.
  - 219,220 ft. E – 221,060 ft. E
  - 119,180 ft. N – 120,380 ft. N
  - 4330 ft. RL – 5400 ft. RL
- The minimum and maximum numbers of composites to be used were set to 25 and 30 respectively.
- Un-estimated model blocks that did not meet the minimum number of composite criteria were not allocated values and were thus excluded from the data analysis.
- Separate block models (parent cell only) were compiled and estimated for each KNA iteration.
- All of the blocks were discretized using points arranged in a 4 by 4 by 4 matrix regardless of block size or dimensions. Except in cases of extreme contrasts in block axis lengths (e.g. 200:1), KNA stats are relatively insensitive to the discretization settings that are used.
- Only composites located within the mineralized domains were used to compile the KNA statistics. The KNA stats will be distorted (lowered) by “edge effects”.

The results from the study for MoS_2 are summarized in Table 17.6, while a plot of the results is presented in Figure 17.1. The results for Cu are summarized in Table 17.7 and Figure 17.2.

Table 17.6 Kriging analysis study – block size analysis for MoS_2

Block size (X by Y by Z) ft.	Number of blocks	Mean Kriging Efficiency	Mean slope
50x50x50	3366	0.23	0.76
100x100x50	874	0.24	0.75
150x150x50	379	0.26	0.75
200x200x50	184	0.27	0.75
250x250x50	137	0.27	0.73
300x300x50	90	0.29	0.72
350x350x50	81	0.29	0.71
400x400x50	51	0.28	0.69

Figure 17.1 Kriging analysis study – block size analysis for MoS2

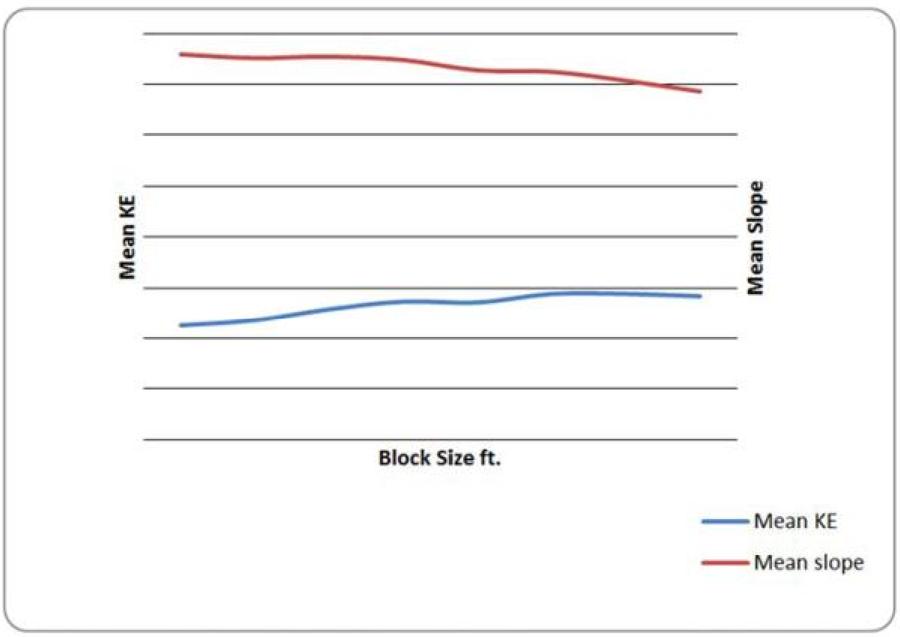
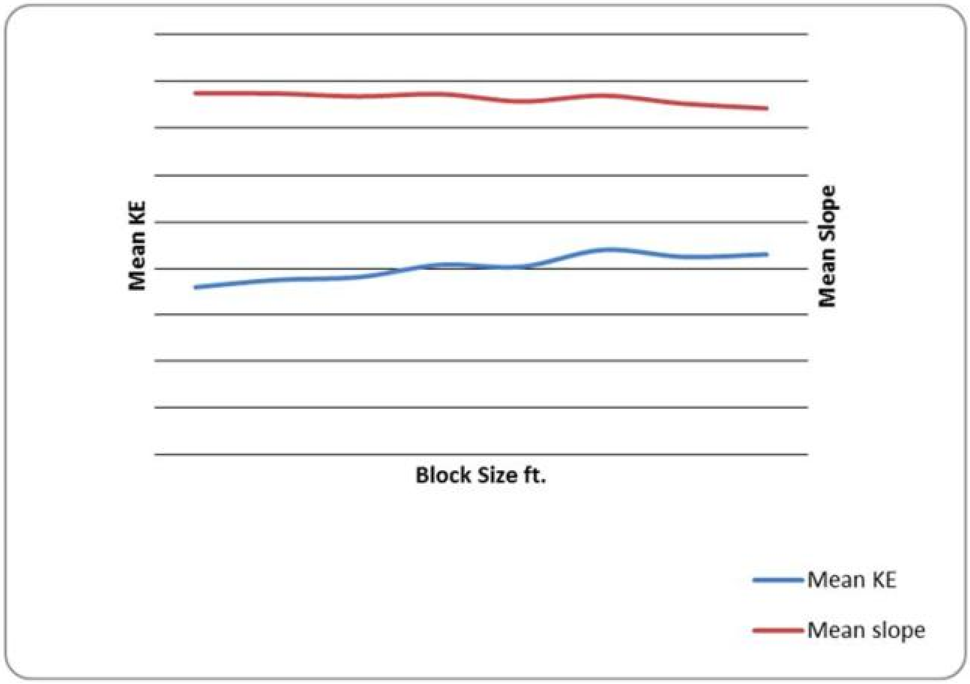


Table 17.7 Kriging analysis study – block size analysis for Cu

Block size (X by Y by Z) ft.	Number of blocks	Mean Kriging Efficiency	Mean slope
50x50x50	8625	0.36	0.77
100x100x50	2184	0.38	0.77
150x150x50	967	0.38	0.77
200x200x50	511	0.41	0.77
250x250x50	344	0.40	0.76
300x300x50	216	0.44	0.77
350x350x50	205	0.42	0.75
400x400x50	139	0.43	0.74

Figure 17.2 Kriging analysis study – block size analysis for Cu



As the average drill spacing was around 600 ft by 600 ft, a block size of 50 ft by 50 ft in the X and Y directions was considered by Snowden too small in order to estimate a robust value. However, at block sizes in excess of 100 ft by 100 ft, the slope of regression was starting to decrease (Cu) and Mosquito was concerned about the large size of the block. A block size of 100 ft by 100 ft was subsequently chosen for the estimation as it appeared to meet the requirements of the statistics as well as those of Mosquito.

17.2.14 Grade interpolation

Ordinary kriging was used to estimate grades for molybdenum, copper, silver and tungsten using both soft and hard boundaries as shown in Table 17.8.

Table 17.8 Summary of boundaries used for the estimation

Grade	Domain	Description
MoS_2	cuag	Using only composite samples occurring in CUAG and the CUAG variogram model for MoS_2.
	cumo	Using composite samples occurring in cumo, mo, and msi and the cumo_mo_msi variogram model for MoS_2.
	mo	Using composite samples occurring in mo, cumo and the cumo_mo_msi variogram model for MoS_2.
	msi	Using composite samples occurring in msi and the cumo_mo_msi variogram model for MoS_2.
	oxide	Using only composite samples occurring in the oxide zone and the oxide variogram model for MoS_2.
Cu	cuag	Using composite samples occurring in cuag and cumo and the cumo-cuag variogram model for Cu.
	cumo	Using composite samples occurring in cumo and cuag and the cumo-cuag variogram model for Cu.
	mo	Using only composite samples occurring in MO and the cumo-cuag variogram model for Cu.
	msi	Using composite samples occurring in msi and the mo_msi variogram model for Cu.
	oxide	Using only composite samples occurring in the oxide zone and the oxide variogram model for Cu.
Ag	cuag	Using composite samples occurring in cuag and cumo, and the cumo_cuag variogram model for Ag.
	cumo	Using only composite samples occurring in cumo and the cumo_cuag variogram model for Ag.
	mo	Using only composite samples occurring in mo and the cumo_cuag variogram model for Ag.
	msi	Using composite samples occurring in msi and the mo_msi variogram model for Ag.
	oxide	Using only composite samples occurring in the oxide zone and the oxide variogram model for Ag.
W	cuag	Using only composite samples occurring in cuag and the cumo_cuag variogram model for W.
	cumo	Using only composite samples occurring in cumo and the cumo_cuag variogram model for W.
	mo	Using only composite samples occurring in mo and the cumo_cuag variogram model for W.
	msi	Using composite samples occurring in msi and the cumo_mo_msi variogram model for W.
	oxide	Using only composite samples occurring in the oxide zone and the oxide variogram model for W.

The orientations of the search ellipses used for the grade estimation were derived from the variogram orientations. The ranges for the primary search volume were equal to ¼ of the variogram range as summarized in Table 17.9.

Table 17.9 Primary search parameters for grade estimation

Grade	Domain(s)	Search distance (ft.)	Orientation Studio 3 ZXZ angles
MoS_2	cumo mo msi	238 750 125	-30 120 -90
	cuag	225 300 213	-30 110 -90
	oxide	250 400 50.5	0 0 55
Cu	cuag cumo mo msi	750 2,500 300	-30 130 -100
	oxide	503 383 57	-25 170 -130
Ag	cuag cumo mo msi	413 2,500 535	-30 130 -90
	oxide	450 176 86	-25 170 0
W	cuag cumo mo msi	225 2,500 132	-30 130 -110
	oxide	149 421 47	0 0 55

The minimum and maximum numbers of composites were set to 10 and 30 respectively. The minimum number of composites had to be derived by selecting composites from at least two drillholes.

If the requirement for the minimum number of composites was not met, the search volume ranges were multiplied by a factor of 2 (equivalent to ½ the variogram range) and a second search volume defined. Similarly if the requirement for the minimum number of composites was not met for the second pass the search volume ranges were multiplied by a factor of 4 (equivalent to the full variogram range).

For the ordinary Kriging, a block discretisation of 8 by 8 by 8 was used in the X, Y and Z directions respectively.

If no grade was estimated for a particular block then a value of 0 was assigned.

17.2.15 Model validation

The block model was validated by completing a series of visual inspections and statistical checks, namely:

- Statistical validation (global)
  - The average of the grades in the block model and the composites were compared.
  - The metals distribution in the composites and the blocks were compared.
  - Comparison of average composite grades with average block estimates – swath plots.
  - Comparison of local "well-informed" block grades with composites contained within those blocks and comparison of panel grades with composites contained within those panels.
- Visual validation
  - The block model was visually checked via several iterations to optimise the search parameters and to make sure that grades were estimated properly with no unexpected extensions of high grades.

The average estimated blocks grades are identical to the average composite grades, with good correlation (70 percent and up) between the estimates and the composites on a block by block basis.

Comparisons of estimated and composite grades within large 200 ft by 200 ft by 50 ft panels were made. Again, the averages of those two datasets are identical.

17.2.16 Resource classification

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

> "*a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge*".

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. Snowden considers that the CUMO deposit is amenable for open pit extraction.

In order to determine the quantities of material offering reasonable prospects for economic extraction from an open pit, Snowden used a Whittle pit optimizer to evaluate the profitability of each resource block based on selected optimization parameters from the Thompson Creek mine (i.e. a comparable an open pit molybdenum project located in Idaho).

The optimization parameters included: ore mining and processing costs of $7.52 per processed ton, overall pit slope angles of 45 degrees, metallurgical recoveries (Table 17.10), and appropriate dilution and offsite costs and royalties. The metals prices for the pit shell for different metals are presented in Table 17.11. The reader is cautioned that the results from the conceptual pit optimization work are used solely for the purpose of reporting Mineral Resources that have "reasonable prospects" for economic extraction by an open pit method and do not represent an attempt to estimate Mineral Reserves.

To take into account the four main economic metals, Mo, Cu, Ag, W a Recoverable Value (RCV) was calculated for each block based on reasonable metal prices, estimated grades and estimated recoveries in each of five domains.

Table 17.10 Recovery table for different geological domains

Zone	Cu Recovery	MoS2 Recovery	Ag Recovery	W Recovery
oxide	60	80	65	0
cuag	68	86	75	35
cumo	85	92	78	35
mo	72	95	55	35
msi	72	95	55	35

Table 17.11 Metal prices used in model and for optimisation

Element	Metal price used in the model for RCV	Metal price used for pit shell optimization
Mo	$16/lb	$25/lb
Cu	$2/lb	$3/lb
Ag	$12/oz	$20/oz
W	$7/lb	$10/lb

Classification of Mineral Resources for the April 2011 CUMO resource model was undertaken by evaluating the confidence in the following sets of information:

- drillhole data
- the geological interpretation and geological continuity
- data density and orientation
- spatial grade continuity
- estimation quality

The resource model has been classified into Inferred Resource and Indicated Resource categories according to CIM Standards. The process for classification is as follows:

- A three dimensional perimeter was digitised around continuous areas of blocks which were estimated during the first and second estimation passes. All blocks within this perimeter were assigned an Indicated Resource classification.
- Blocks outside of this perimeter which were estimated during the third estimation pass were assigned an Inferred Resource classification. Remaining blocks which do not meet the criteria outlined above were not assigned a classification.

17.2.17 Resource reporting

Based on the recoveries and metal prices (Table 17.10, Table 17.11), and bulk density values, a dollar equivalent value, which is referred to as the Recovered Value (RCV) has been estimated for each block. The formulae used for RCV values are shown in Table 17.12 and also in section 11.2.

Table 17.12 CUMO RCV Calculation formula for different domains

Zone	Density	RCV
oxide	12.24	(230.202*MOCAP)+(25.2*CUCAP)+(0.2275*AGCAP)
cumo	12.3	(264.73*MOCAP)+(35.7*CUCAP)+(0.273*AGCAP)+(WCAP*0.0049)
cuag	12.18	(247.47*MOCAP)+(28.56*CUCAP)+(0.2625*AGCAP)+(WCAP*0.0049)
mo	12.32	(273.365*MOCAP)+(30.24*CUCAP)+(0.192*AGCAP)+(WCAP*0.0049)
msi	12.44	(269*MOCAP)+(30.24*CUCAP)+(0.192*AGCAP)+(WCAP*0.0049)

MOCAP = estimated Mo grade (top-cut)

CUCAP = estimated Cu grade (top-cut)

AGCAP = estimated Ag grade (top-cut)

WCAP = estimated W grade (top-cut)

Snowden considers it appropriate to report the Mineral Resources at a cut-off grade of $2.50 RCV.

No Mineral Reserves have been estimated at this time. Additional studies will be required to determine technical, economic, legal, environmental, socio-economic, and governmental factors. These modifying factors are normally included in a prefeasibility study and are a pre-requisite for conversion of Mineral Resources to Mineral Reserves. The CIM Standards (CIM, 2005) describe completion of a Preliminary Feasibility Study as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves.

A summary of the grade and tonnage figures at a range of cut-off grades by classification level are presented in Table 17.13 and Table 17.14. The selected cut-off grade for reporting of Mineral Resources of $2.50 RCV is highlighted in bold. Tonnage-grade curves by resource classification level are shown in Figure 17.3 and Figure 17.4.

Table 17.13 Global Indicated Resource within the optimum pit shell

RCV Cutoffs ($)	Million TONS	MoS_2 (%)	Cu (%)	Ag (gpt)	W (gpt)	RCV	Ag (M.Oz)	Cu (M.lb)	MoO_3 (M.lb)	W (M.IB)
0.50	2,052	0.058	0.080	2.37	38.63	18.94	142	3291	2144	159
1.00	2,052	0.058	0.080	2.37	38.63	18.94	142	3291	2144	159
1.50	2,050	0.058	0.080	2.37	38.67	18.96	142	3290	2144	159
2.00	2,037	0.059	0.081	2.38	38.89	19.07	141	3281	2144	158
2.50	2,026	0.059	0.081	2.38	39.06	19.16	141	3272	2143	158
3.00	2,006	0.059	0.081	2.39	39.29	19.32	140	3254	2142	158
3.50	1,978	0.060	0.082	2.40	39.67	19.55	138	3225	2139	157
4.00	1,952	0.061	0.082	2.41	39.97	19.76	137	3205	2136	156
4.50	1,923	0.062	0.083	2.42	40.28	19.99	136	3179	2131	155
5.00	1,894	0.062	0.083	2.42	40.59	20.23	134	3147	2126	154
5.50	1,868	0.063	0.084	2.43	40.83	20.43	132	3122	2121	153
6.00	1,841	0.064	0.084	2.44	41.07	20.65	131	3089	2114	151
6.50	1,809	0.065	0.084	2.44	41.34	20.91	129	3048	2107	150
7.00	1,774	0.066	0.085	2.45	41.61	21.19	127	3000	2097	148
7.50	1,737	0.067	0.085	2.45	41.85	21.48	124	2943	2087	145
8.00	1,701	0.068	0.085	2.45	42.05	21.77	121	2884	2076	143
8.50	1,670	0.069	0.085	2.44	42.19	22.02	119	2827	2066	141
9.00	1,640	0.070	0.084	2.43	42.29	22.27	116	2769	2056	139
9.50	1,614	0.070	0.084	2.43	42.40	22.48	114	2717	2046	137
10.00	1,589	0.071	0.084	2.42	42.51	22.68	112	2669	2036	135
10.50	1,560	0.072	0.084	2.41	42.66	22.91	110	2609	2024	133
11.00	1,531	0.073	0.083	2.40	42.84	23.14	107	2551	2011	131
11.50	1,500	0.074	0.083	2.38	43.05	23.39	104	2489	1995	129
12.00	1,469	0.075	0.083	2.38	43.27	23.63	102	2432	1979	127
12.50	1,439	0.076	0.082	2.37	43.51	23.87	99	2372	1961	125
13.00	1,410	0.077	0.082	2.35	43.77	24.10	97	2317	1944	123
13.50	1,378	0.078	0.082	2.35	44.07	24.34	94	2260	1924	121
14.00	1,347	0.079	0.082	2.34	44.31	24.59	92	2202	1903	119
14.50	1,316	0.079	0.081	2.33	44.55	24.84	89	2141	1880	117
15.00	1,282	0.080	0.081	2.31	44.77	25.10	86	2076	1856	115
15.50	1,248	0.082	0.080	2.30	44.96	25.37	84	2009	1830	112
16.00	1,212	0.083	0.080	2.29	45.14	25.66	81	1939	1801	109
16.50	1,177	0.084	0.080	2.27	45.32	25.94	78	1872	1772	107
17.00	1,139	0.085	0.079	2.25	45.52	26.24	75	1799	1739	104
17.50	1,101	0.086	0.079	2.24	45.78	26.55	72	1730	1705	101
18.00	1,061	0.087	0.078	2.23	46.08	26.88	69	1658	1667	98
18.50	1,018	0.089	0.078	2.21	46.41	27.25	66	1581	1625	94
19.00	973	0.090	0.077	2.20	46.76	27.64	62	1499	1579	91
19.50	926	0.092	0.076	2.17	47.08	28.07	59	1411	1530	87
0.50	2,052	0.058	0.080	2.37	38.63	18.94	142	3291	2144	159

Table 17.14 Global Inferred Resource within the optimum pit shell

RCV Cutoffs($)	Million tons	MoS_2 (%)	Cu (%)	Ag (gpt)	W (gpt)	RCV ($)	Ag (M.Oz)	Cu (M.lb)	MoO_3 (M.lb)	W (M.lb)
0.50	5,024	0.026	0.063	1.94	9.12	9.43	285	6331	2352	92
1.00	4,719	0.028	0.066	1.99	9.71	9.99	273	6235	2352	92
1.50	4,448	0.029	0.068	2.03	10.28	10.52	263	6059	2351	91
2.00	4,232	0.031	0.069	2.06	10.77	10.97	255	5868	2349	91
2.50	**3,947**	**0.033**	**0.070**	**2.11**	**11.51**	**11.60**	**243**	**5553**	**2343**	**91**
3.00	3,754	0.035	0.071	2.13	12.02	12.05	233	5305	2337	90
3.50	3,525	0.037	0.071	2.13	12.66	12.62	219	4971	2328	89
4.00	3,258	0.039	0.070	2.11	13.46	13.35	201	4547	2312	88
4.50	3,067	0.042	0.069	2.10	14.03	13.92	188	4231	2296	86
5.00	2,915	0.043	0.068	2.10	14.43	14.40	178	3974	2279	84
5.50	2,768	0.045	0.067	2.09	14.79	14.88	169	3717	2259	82
6.00	2,623	0.047	0.065	2.10	15.08	15.39	160	3424	2237	79
6.50	2,523	0.049	0.064	2.09	15.23	15.75	154	3249	2216	77
7.00	2,402	0.051	0.064	2.10	15.35	16.20	147	3081	2182	74
7.50	2,252	0.053	0.064	2.11	15.53	16.80	139	2875	2136	70
8.00	2,145	0.054	0.063	2.11	15.54	17.25	132	2700	2102	67
8.50	2,046	0.056	0.062	2.10	15.50	17.69	125	2553	2067	63
9.00	1,962	0.058	0.062	2.09	15.40	18.07	119	2421	2034	60
9.50	1,873	0.059	0.061	2.07	15.45	18.49	113	2291	1996	58
10.00	1,781	0.061	0.060	2.03	15.54	18.94	105	2142	1956	55
10.50	1,693	0.063	0.059	1.97	15.61	19.39	97	1995	1915	53
11.00	1,618	0.065	0.058	1.93	15.80	19.80	91	1873	1877	51
11.50	1,551	0.066	0.057	1.91	15.94	20.16	86	1778	1839	49
12.00	1,487	0.067	0.057	1.89	16.14	20.53	82	1693	1800	48
12.50	1,432	0.069	0.057	1.88	16.28	20.84	78	1621	1765	47
13.00	1,380	0.070	0.056	1.86	16.41	21.15	75	1559	1729	45
13.50	1,333	0.071	0.056	1.85	16.51	21.43	72	1501	1695	44
14.00	1,286	0.072	0.056	1.85	16.59	21.71	70	1448	1659	43
14.50	1,244	0.073	0.057	1.86	16.64	21.96	68	1409	1624	41
15.00	1,200	0.073	0.057	1.87	16.63	22.22	66	1374	1586	40
15.50	1,156	0.074	0.058	1.89	16.64	22.49	64	1342	1546	38
16.00	1,113	0.075	0.059	1.90	16.63	22.75	62	1310	1505	37
16.50	1,063	0.076	0.060	1.90	16.70	23.05	59	1270	1458	36
17.00	1,008	0.077	0.060	1.90	16.86	23.40	56	1219	1404	34
17.50	953	0.079	0.062	1.91	17.01	23.75	53	1174	1347	32
18.00	898	0.080	0.063	1.93	17.14	24.12	51	1135	1287	31
18.50	847	0.081	0.064	1.94	17.18	24.48	48	1089	1232	29
19.00	799	0.082	0.065	1.93	17.16	24.82	45	1037	1180	27
19.50	753	0.083	0.065	1.93	17.15	25.16	42	982	1128	26
20.00	703	0.085	0.066	1.94	17.27	25.55	40	927	1070	24

Figure 17.3 Grade-Tonnage curve - Indicated Resources

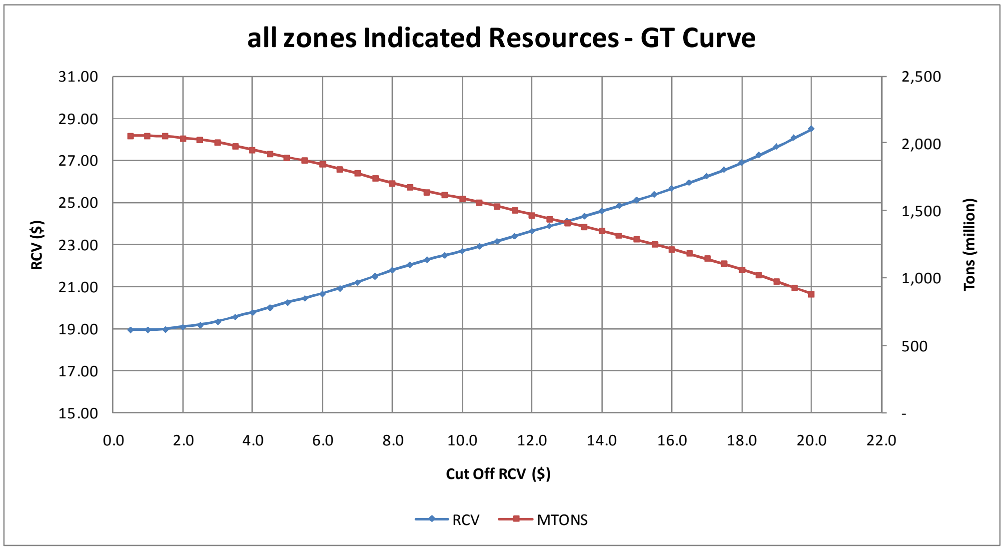
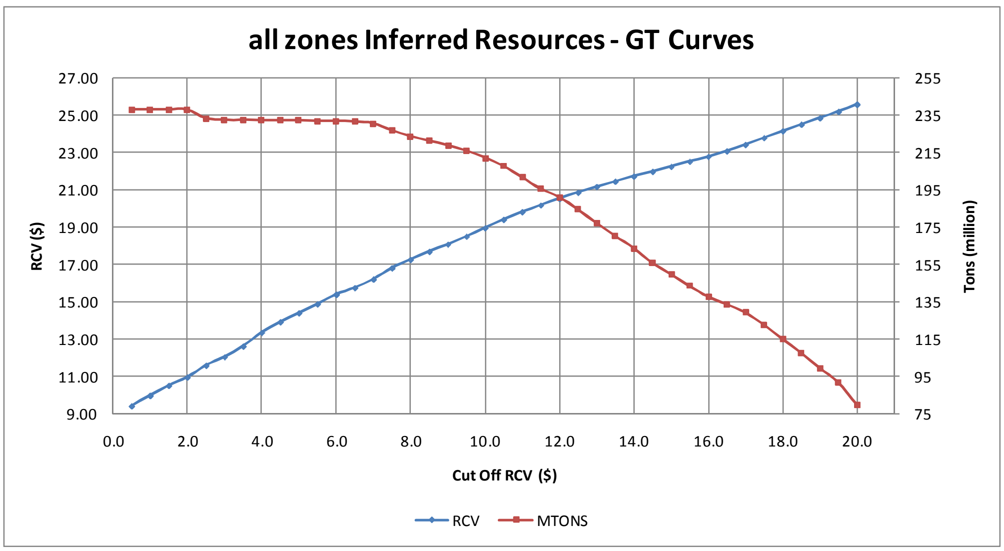


Figure 17.4 Grade-Tonnage Curve - Inferred Resources



17.2.18 Comparison with 2009 Resource

Table 17.15 provides a summary of the Mineral Resources reported for the CUMO Property in March 2009. The 2009 Mineral Resources were reported using an RCV (previously referred to as GRV) cut-off of $1.00. At this RCV cut-off some 1.44 Bt of Indicated Resources at a grade of 0.07 % MoS_2 and 2.51 Bt of Inferred Resources at a grade of 0.05 % MoS_2 were reported.

The 2011 Mineral Resources have been reported at a RCV cut-off of $2.50. In the calculation of the 2011 RCV values there have been changes to the assumed metal prices (See section 17.15 for details). Also in 2011 an ultimate pit shell has been calculated using mining and processing costs from similar deposits this has been used to constrain the reported Mineral Resources.

Changes to the geological model and mineralization envelope following the inclusion of the 11 drillholes completed in 2009-2010 have resulted in a significant increase in the size of the mineralized area inside which a reportable Mineral Resources for the CUMO Property can be calculated. Therefore the Mineral Resources calculated from the 2011 resource model update at a $2.50 RCV cut-off value there are 2.03 Bt of Indicated Resources at a grade of 0.06 % MoS_2 and 3.95 Bt of Inferred Resources at a grade of 0.03 % MoS_2. Table 17.16 and Table 17.17 provide a summary of the Indicated and Inferred Mineral Resources for the CUMO Property in May 2011.

Table 17.15 2009 Indicated and Inferred Mineral Resources at a $1 GRMV Cut-off (Mosquito, News Release, March 18th 2009, as filed on SEDAR)

Category	GRMV Cutoff ($)	TONS millions	Recovered MoS_2 Equiv. %*	Recovered Cu Equiv. %*	MoS_2 %	Cu %	Ag ppm	W ppm
Indicated	$1.00	1,444.8	0.068	0.68	0.071	0.08	2.26	44.06
Inferred	$1.00	2,512.2	0.051	0.51	0.051	0.07	2.16	34.93

Table 17.16 2011 Indicated Mineral Resources at a $2.50 RCV Cut-off value

Category	GRMV Cutoff ($)	Zone	TONS millions	Recovered MoS_2 Equiv. %*	Recovered Cu Equiv. %*	MoS_2 %	Cu %	Ag ppm	W ppm
Indicated	$2.50	Oxide	129	0.024	0.16	0.016	0.07	2.20	26.46
Indicated	$2.50	CuAg	466	0.031	0.21	0.019	0.10	3.03	27.59
Indicated	$2.50	CuMo	595	0.067	0.46	0.055	0.11	3.04	45.38
Indicated	$2.50	Mo	743	0.097	0.67	0.095	0.05	1.68	45.10
Indicated	$2.50	MSI	93	0.055	0.38	0.056	0.02	0.74	25.22
Indicated	$2.50	Total (all zones)	2,026	0.067	0.46	0.059	0.08	2.38	39.06

Table 17.17 2011 Inferred Mineral Resources at a $2.50 RCV Cut-off value

Category	GRMV Cutoff ($)	Zone	TONS millions	Recovered MoS_2 Equiv. %*	Recovered Cu Equiv. %*	MoS_2 %	Cu %	Ag ppm	W ppm
Inferred	$2.50	Oxide	232	0.021	0.14	0.014	0.06	1.91	13.64
Inferred	$2.50	CuAg	1,913	0.019	0.13	0.010	0.09	2.51	6.48
Inferred	$2.50	CuMo	918	0.074	0.51	0.054	0.08	2.15	17.86
Inferred	$2.50	Mo	650	0.070	0.48	0.068	0.02	1.36	15.47
Inferred	$2.50	MSI	233	0.058	0.40	0.061	0.01	0.94	14.68
Inferred	$2.50	Total (all zones)	3,946	0.038	0.26	0.033	0.07	2.11	11.51

18 Other relevant data and information (Ausenco, 2009)

This section is reproduced in total for completeness from "CUMO Property Preliminary Economic Assessment Throughput Scoping Study Report, November 18, 2009". The base estimate date of the "2009 PEA Technical Report" is July 2009. No re-evaluation or allowance, for foreign exchange rate fluctuations or cost escalation since that date, has been made to either the capital or operating cost estimates. Whilst this part of the report was prepared in 2009, Snowden considers it to remain relevant and valid as there have been no significant changes to the assumptions, and the grade of the mineralisation remains similar to the 2009 estimates. The one change that has been made is that the size of the deposit has increased significantly, and it is Snowden's opinion that this can only improve the confidence in the results of the PEA.

This assessment considers four options for plant throughput rates from 50 000 short tons per day (kt/d) to 200 kt/d and has developed conceptual pit shell designs, scoping-level TSF sizing, scheduling, order of magnitude plant, mining and TSF capital cost estimates to an accuracy of ±35%, as well as indicative operating costs for each treatment rate through the plant. At this stage the final size and shape of the deposit has not been fully determined and a fixed 40 year mine life has been considered with the varying plant throughput options.

18.1 Mining Operation Design

Mining at CUMO is conceptually designed as an open pit mine using the typical drill –blast – load – haul methods utilized at most large-tonnage, low-grade, open-pit porphyry deposits.

For this Preliminary Economic Assessment (PEA), four productions options were examined. These included:

- 50 000 short tons of ore per day (t/d)
- 100 000 t/d (short tons)
- 150 000 t/d (short tons)
- 200 000 t/d (short tons)

The mining operations conceptually will utilize rotary drills to drill blast holes and electrical shovels to load the blasted material into mechanical rigid-frame, rear-dump mine trucks.

18.1.1 Equipment Specifications

Equipment specifications were determined for the various production scenarios using information published by InfoMine in CostMine (2009) for a series of typical open pit mine models. The equipment was selected based on total tons moved per day for all categories of mined material, ore, stockpile material, and waste (Table 18.1). These were compared to the equipment selections in the CostMine mine models for similar sized operations. This information was then compared to the equipment selections detailed for similar sized operations to verify the equipment selections reflected industry standards.

Table 18.1 Mining Rates for Equipment Specifications

Production Rate	Ore	Stockpile	Waste	Prestrip	Total excluding Prestrip	Moved per Day
	(Mt)	(Mt)	(Mt)	(Mt)	(Mt)	(Mt)
50 kt/d	714	1,065	1,354	670	3,133	0.22
100 kt/d	1,425	1,667	2,316	823	5,408	0.38
150 kt/d	2,160	2,158	2,890	904	7,207	0.50
200 kt/d	2,880	2,045	2,954	968	7,879	0.55

The selection of the size of the equipment fleet is based on the following assumptions:

- The mining rate is considered constant over the 40 year mine life
- Waste and stockpile material will be hauled from the pit; ore will be hauled to pit-edge crusher, crushed, and conveyed to the mill for processing
- Truck capacities were chosen to minimize fleet size.

As the mining rate increases, the strip ratio decreases, resulting in an incrementally smaller increase in material moved per day than milled. This, together with equipment size increases for higher mining rates results in considerable economies of scale advantages for the higher mining rates.

18.1.2 Pit Design

The pit designs are conceptual and were provided to Vector by Mosquito. Although, the pit design parameters are not supported by any geotechnical rock mass data, Vector has reviewed the designs and consider then reasonable for this level of study. The conceptual pit design pit slopes shown below are the same for all four production scenarios.

Table 18.2 Pit slope design criteria

Sector	Criteria
South wall –	6300 to 5300 feet 45 degree wall
	5300 to 4300 feet 40 degree wall
	4300 to 3500 feet 35 degree wall
East wall –	5450 to 4450 feet 45 degree wall
	4450 to 3500 feet 40 degree wall
North wall –	5700 to 4700 feet 45 degree wall
	4700 to 3700 feet 40 degree wall
	3700 to 3500 feet 35 degree wall
West wall –	5700 to 4700 feet 45 degree wall
	4700 to 3700 feet 40 degree wall
	3700 to 3500 feet 35 degree wall

Bench heights in ore are conceptually 50 feet as defined by Mosquito’s pit models. Pit roads are not included in the client’s mine design. All in-pit ramps were assumed at 10% grade for the purposes of determining haul profile distances out of the pit.

Vector accepts the client's assertion that these pit designs reflect the three dimensional distribution of the in-place mineralization and as such Vector utilized these pit designs in estimating the CAPEX and OPEX mining costs for this PEA.

The following discussion summarizes Vector's understanding of the pit design procedure completed by the client. The pit design utilized the block model data from the resource report of Holmgren and Giroux (2009). The data from the resource model was plotted using AutoCAD to define the distribution of the blocks in three dimensions. Each block was assigned a gross revenue value based on assay data, assumed mill recovery and assumed metal prices. Based on the block model created by Holmgren and Giroux (2009) there were a percentage of the blocks within the conceptual pit boundaries that did not have a grade or a value assigned to them. Mosquito assigned a grade or value to these blocks based on the average grade of the blocks for that bench. The pits were then designed by assuming cutoff grades for mill ore, stockpile material, and waste and assigning a category to each block.

Table 18.3 is a summary of the cutoff grades for each category for each production scenario.

Table 18.3 Cutoff Grades for Pit Design Criteria

Scenario	Waste ($/ton recoverable metal)	Stockpile ($/ton recoverable metal)	Ore ($/ton recoverable metal)
50 kt/d	<$10.00	Yrs 1-17 ≥$10 <$22.50 Yrs 18-40 ≥$10 <$20.00	Yrs 1-17 ≥ $22.50 Yrs 18-40 ≥ $20.00
100 kt/d	<$7.50	Yrs 1-9 ≥$7.50 <$22.50 Yrs 10-40 ≥$7.50 <$20.00	Yrs 1-9 ≥ $22.50 Yrs 10-40 ≥ $20.00
150 kt/d	<$7.50	Yrs 1-6 ≥$7.50 <$22.50 Yrs 7-40 ≥$7.50 <$20.00	Yrs 1-6 ≥ $22.50 Yrs 7-40 ≥ $20.00
200 kt/d	<$7.50	Yrs 1-6 ≥$7.50 <$22.50 Yrs 7-40 ≥$7.50 <$20.00	Yrs 1-6 ≥ $22.50 Yrs 7-40 ≥ $20.00

Based on the pit slopes of the conceptual model, the outer pit boundaries for each level were established to capture the majority of the ore blocks. The blocks in each category were then summed for each bench elevation.

This data was provided to Vector as a series of EXCEL spreadsheets for each production scenario detailing the tons of ore, waste, and stockpile material by bench elevation. The sheets also detailed a mine schedule by year. It was this information that was used to determine a yearly mining rate for the combined categories which was used in a factored analysis to determine mining costs for each production scenario.

The pit models do not include bench widths or haul road locations, the models have not been optimized. Vector's understanding is that the assignment of each block to one of three categories, ore, stockpile, or waste, is based solely on the value of the recoverable metal in the block and does not consider the cost of mining the material above a block including pre-strip.

18.2 TSF Design

The TSFs are the North, Middle and South facilities to be located south of the mill site. The TSFs were developed to store tailings for the four plant throughput options of 50, 100, 150 and 200 kt/d (short tons) assuming production proceeds at a 365 days/year basis for 40 years. The tailings impoundments were sized for generally 10 percent more than the tailings solids production to account for the volume taken by the tailings water, water pool, design storm water and dry freeboard.

The TSFs will be developed in stages by constructing starter dams and raising the dams using the downstream construction method. The 50 kt/d plant throughput option will require the construction of the North TSF in three stages. The 100 kt/d option will require the construction of the North TSF in two stages and the Intermediate Middle TSF in two stages. The 150 kt/d option will require the construction of the North TSF in two stages and the Ultimate Middle TSF in two stages. The 200 kt/d option will require the construction of the North TSF in a single stage and the South TSF in three stages.

The fill required to construct the tailings dams is assumed to consist mostly of waste rock generated from mine pre-stripping operations. Rock fill dams constructed using the downstream method are utilized for this conceptual design due to the considerable height of the planned dams, the relatively high seismicity of the project area, the lack of geotechnical data for the dam sites, and the abundance of waste rock. Other dam construction methods and materials may be studied in the Feasibility Design once the project parameters and characteristics are better defined and rigorous engineering analyses are conducted.

Unlined tailings impoundments were considered in this conceptual design since it is Vectors understanding, based on the preliminary Acid Base Analysis (ABA) test work conducted on flotation tailings by SGS, that the tailings will most likely be inert and seepage water quality is acceptable for release to the environment. Consideration may be given in the Feasibility Study to lining critical portions of the impoundments to minimize water seepage loss.

18.3 Waste Dump Design

The waste dump will be developed in the area south of the ultimate mine pit and will accommodate approximately 2.6 billion tons of waste rock, which exceeds the maximum that may be generated minus the material used for tailings dam and water storage dam construction. For the purposes of this study the waste material has been assumed to be benign based on the preliminary ABA test work conducted on flotation tailings by SGS.

The waste dump plan will be updated in the Feasibility Study for the plant throughput selected for the project and the dump design details will be provided at that time.

18.4 Low-Grade Ore Stockpile Design

The amounts of low-grade ore estimated for the four plant throughput options vary from 1.2 to 2.4 billion tons. The start-up site for the low-grade ore stockpile will be east of the mine pit, and an expansion site located south of the pit and east of the waste dump will be utilized if additional storage of low-grade stockpile is required in later years of mine life. The exact amounts of material that can be stored in these sites were not calculated and will be determined in the Feasibility Study when the project parameters are better defined. For the purposes of this study the stockpiled material has been assumed to be benign based on the preliminary ABA test work conducted on flotation tailings by SGS.

18.5 Environmental Considerations

No environmental considerations were investigated as part of this report. Assumptions on the possible environmental impacts of the project have been made where required for this study, as detailed in the relevant sections of this report. Detailed remediation and reclamation plans will need to be addressed in subsequent studies.

18.6 Taxes and Royalties

All values are calculated based on Earnings Before Interest Tax Depreciation and Amortisation (EBITD&A). No royalties were taken into consideration.

18.7 Capital Cost Estimate

The concept study estimate is based on a circuit consisting of open pit mining, primary gyratory crushing, coarse ore stockpiling, SAG and ball milling with pebble crushing (SABC), bulk flotation followed by copper-molybdenum separation and conventional tailings disposal. Molybdenum concentrates are further processed at an off-site roaster to produce molybdenum oxide, rhenium metal and sulfuric acid.

The capital costs for development of the project increase as the design throughput increases. The capital cost for development of the mine (pre-strip cost), is relatively insensitive to the size of the operation and the other capital items; mining fleet, concentrator; tailings storage facilities, roaster and site ancillary buildings do allow some reduction in capital intensity (cost per unit throughput) to be achieved i.e. economies of scale. The ±35% accuracy total project capital cost with a base date of July 2009 for each throughput option are summarized below in Table 18.4 and discussed in detail in sections below.

Table 18.4 Summary of Initial Capital Costs

Capital Cost		Design			
		50 kt/d (short tons)	**50 kt/d (short tons)**	**50 kt/d (short tons)**	**50 kt/d (short tons)**
Plant capital	$USM	590	1000	1500	2900
Roaster capital	$USM	120	200	270	350
Mining fleet capital	$USM	100	200	270	270
Preproduction costs (incl. Prestrip)	$USM	750	700	640	660
Tailings	$USM	40	80	80	160
Total Initial Capital	**$USM**	**1600**	**2200**	**2800**	**3400**

18.8 Mining Capital Costs

18.8.1 Introduction

Capital mining costs for CUMO were developed by Vector Engineering, Inc. (Vector) for equipment, haul roads and site work, pre-production stripping, buildings required to support the mining operations, working capital, and engineering and management. The guide for estimating these capital costs was the CostMine (2009) books published by InfoMine.

18.8.2 Equipment Requirements and Capital Costs

The equipment requirements for the conceptual operations at CUMO were determined by a combination of:

- Factored analysis of the equipment lists from CostMine for the mine cost models using total tons moved per day as the common factor
- Analysis of conceptual haul profiles based on the conceptual pit designs including:
  - Hauling waste and stockpile material to the waste and stockpile storage areas selected for this study
  - Hauling ore to an edge-of-pit crusher
- Productivity of the haul fleet
- Review of the equipment requirements for similar operations and projects including operating mines at Thompson Creek (MineCost, 2009), and Morenci (MineCost, 2009), and feasibility or pre-feasibility studies at Mt, Hope (3M Engineering and Technology, 2007), Augusta Rosemont (3M Engineering and Technology, 2007A), Creston (3M Engineering and Technology, 2009), and Angostura (GRD Minproc, 2009).

The numbers of the various pieces of equipment required are a function of the size of the haul fleet which in turn is a function of total tons moved on a daily basis. At CUMO this includes ore, stockpile material, and waste. The following assumptions were made in estimating the size of the haul fleet for each production scenario. They are:

The conceptual "typical" haul profile includes loading time, hauling time, turning time, dumping time, and return time

- Loading, turning and dumping times were assumed to aggregate 7 minutes total for all four production scenarios;
- Haul speeds from the pit to the destination
  - 15 mph from the mining face to the pit ramp
  - 8 mph up the ramp
  - 15 mph from the pit edge to the final destination
- Ore was hauled from the pit to a pit-edge crusher and stockpile and waste material was hauled to the stockpile and waste storage areas respectively
- Return times were calculated at an assumed speed of 15 mph
- Availability of trucks was estimated to be 80%

Based on the calculated haul fleet requirements, estimates were made for the additional equipment necessary to produce sufficient material to meet the production requirements and support the haul fleet. Assumptions made in making this estimate were:

- The maximum number of haul units was determined based on the conceptual haul profiles
- It was assumed the maximum number of haul units would not be required immediately.

- The useful life for the equipment was assumed as follows:
  - Cable Shovels – 20 years
  - Haul Trucks – 11 years
  - Rotary Drills – 10 years
  - Bulldozers, Graders, Water Tankers – 12 years
  - All other equipment except pumps – 7 years
  - Pumps – 2.5 years

Table 18.6 to Table 18.8 show the initial (pre-production) equipment requirements for each production scenario, additional replacement equipment will be required throughout the duration of the life of the project; these costs have been included as sustaining capital.

Table 18.5 50 kt/d Mine Equipment Capital Costs

50 kt/d (short tons)					
Equipment	**Parameter**	**Size**	**Number**	**Cost / Unit US$M**	**Total Cost US$M**
Cable Shovels	cu meter	35.2	2	10.9	22
Rear End Dump Trucks	metric ton	218	21	3.05	64
Rotary Drills	centimetre	38.1	5	1.25	6.3
Bulldozers	kW	305	6	0.73	4.4
Graders	kW	160	3	0.32	1.0
Water Tankers	liter	53,000	1	0.74	0.7
Service trucks	kg gvw	20,500	5	0.06	0.3
Mechanics Truck	kg gvw	20,500	5	0.07	0.3
Tire trucks	kg gvw	20,500	3	0.16	0.5
Bulk Trucks	kg/minute	600	3	0.04	0.1
Light Plants	kW	10.1	4	0.02	0.1
Pumps	kW	93.2	0	0.03	0.0
Pickup trucks			16	0.02	0.3
TOTALS US$M					**100**

Table 18.6 100 kt/d Mine Equipment Capital Costs

100 kt/d (short tons)					
Equipment	**Parameter**	**Size**	**Number**	**Cost / Unit US$M**	**Total Cost US$M**
Cable Shovels	cu meter	61.2	2	17.9	36
Rear End Dump Trucks	metric ton	327	25	5.63	140
Rotary Drills	centimetre	38.1	6	1.25	7.5
Bulldozers	kW	305	5	0.73	3.7
Graders	kW	160	4	0.32	1.3
Water Tankers	liter	53,000	2	0.74	1.5
Service trucks	kg gvw	20,500	6	0.06	0.3
Mechanics Truck	kg gvw	20,500	5	0.07	0.3
Tire trucks	kg gvw	20,500	6	0.16	1.0
Bulk Trucks	kg/minute	600	3	0.04	0.1
Light Plants	kW	10.1	4	0.02	0.1
Pumps	kW	93.2	3	0.03	0.1
Pickup trucks			23	0.02	0.5
TOTALS US$M					**200**

Table 18.7 150 kt/d Mine Equipment Capital Costs

150 kt/d (short tons)					
Equipment	**Parameter**	**Size**	**Number**	**Cost / Unit US$M**	**Total Cost US$M**
Cable Shovels	cu meter	61.2	3	17.9	54
Rear End Dump Trucks	metric ton	327	36	5.63	200
Rotary Drills	centimetre	38.1	5	1.25	6.3
Bulldozers	kW	305	5	0.73	3.7
Graders	kW	160	4	0.32	1.3
Water Tankers	liter	53,000	1	0.74	0.7
Service trucks	kg gvw	20,500	9	0.06	0.5
Mechanics Truck	kg gvw	20,500	6	0.07	0.4
Tire trucks	kg gvw	20,500	9	0.16	1.4
Bulk Trucks	kg/minute	600	3	0.04	0.1
Light Plants	kW	10.1	4	0.02	0.1
Pumps	kW	93.2	3	0.03	0.1
Pickup trucks			30	0.02	0.6
TOTALS US$M					**270**

Table 18.8 200 kt/d Mine Equipment Capital Costs

200 kt/d (short tons)					
Equipment	**Parameter**	**Size**	**Number**	**Cost / Unit US$M**	**Total Cost US$M**
Cable Shovels	cu meter	61.2	3	17.9	54
Rear End Dump Trucks	metric ton	327	35	5.63	200
Rotary Drills	centimetre	38.1	7	1.25	8.8
Bulldozers	kW	305	6	0.73	4.4
Graders	kW	160	5	0.32	1.6
Water Tankers	liter	53,000	2	0.74	1.5
Service trucks	kg gvw	20,500	10	0.06	0.6
Mechanics Truck	kg gvw	20,500	6	0.07	0.4
Tire trucks	kg gvw	20,500	10	0.16	1.6
Bulk Trucks	kg/minute	600	4	0.04	0.2
Light Plants	kW	10.1	6	0.02	0.1
Pumps	kW	93.2	3	0.03	0.1
Pickup trucks			34	0.02	0.7
TOTALS US$M					**270**

18.8.3 Non-Equipment Capital Costs

Table 18.9 is a summary of the estimated capital costs excluding equipment. With the exception of pre-stripping costs, these were estimated by factored analysis from the CostMine (2009) mine models. There has been no estimation of additional sustaining capital for the mine, other than that estimated for equipment replacement.

Table 18.9 Mining Capital Costs Excluding Equipment

Category	**50 kt/d (US$M)**	**100 kt/d (US$M)**	**150 kt/d (US$M)**	**200 kt/d (US$M)**
Haul Roads/Site Work	27	35	42	43
Preproduction Stripping	610	540	490	500
Buildings				
Repair and Maintenance Shop	15	20	25	28
Tire Shop	0.2	0.3	0.4	0.5
Anfo Storage	0.3	0.6	0.7	0.8
Working Capital (1year)	85	81	77	75
Engineering and management	15	14	13	13
TOTAL US$M	**750**	**700**	**640**	**660**

Pre-stripping is substantial in the conceptual pit design and mine schedule provided by Mosquito for all four production scenarios. The pre-stripping schedule provided is designed to move the prestrip in three years in all four production scenarios. Further Mosquito has developed a pre-stripping schedule that includes pre-stripping by both the owner and a contractor. Table 50 shows the amount of prestrip material for each production scenario along with the total tons to be moved by the owner and also by the contractor. In addition Table 50 shows the cost of the prestrip operations.

Table 18.10 Mining Pre-Strip Costs

	50 kt/d		100 kt/d		150 kt/d		200 kt/d	
	Owner	**Contract**	**Owner**	**Contract**	**Owner**	**Contract**	**Owner**	**Contract**
Prestrip tons (million)	232	438	401	422	533	371	583	385
Cost/ton (US$)	0.92	0.82	0.40	0.82	0.27	0.82	0.25	0.82
Total Cost (US$M)	213	359	162	346	146	304	148	316
Mob/DeMob (US$M)	35	-	35	-	35	-	35	-
Subtotals (US$M)	248	359	197	346	181	304	183	316
TOTAL US$M	**610**		**540**		**490**		**500**	

As shown in Table 18.10, the tonnages moved by the owner increases while the contractor's tonnage stays relatively constant. It also shows overall prestripping costs actually decrease as the tonnage increases for two interrelated reasons.

- The owner is moving a higher percentage of the tons of pre-strip as the tonnages increases while the contractor's percentage at the higher mining cost decreases
- The cost of mining for the owner decreases at a rate faster than the rate of tonnage increase.

Engineering and management costs are estimated at 2% of the total mining capital costs before engineering and management added into the total. Once more detail is known about the engineering required, this number can be refined but at a scoping level study, these estimates should be within ±35% accuracy.

18.9 Process Plant Capital Costs

A summary of the estimated capital cost for the processing plant and on-site ancillary facilities is provided in Table 18.11 and Table 18.12 for the roaster and ancillary facilities, which exclude any escalation or foreign currency fluctuations and are current day costs only (3Q 2009).

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 18.11 Summary of Plant Capital Cost Estimate

AREA	Throughput Option			
	50 kt/d	**100 kt/d**	**150 kt/d**	**200 kt/d**
Direct Costs				
Site Development	10	17	24	30
Concentrator	310	620	900	1 200
Concentrator Services	21	32	41	50
Concentrator Infrastructure	41	58	71	83
Molybdenum Plant	23	34	44	52
Dams and Tailings Line	9	14	17	21
Spares and First Fill	16	9	31	38
TOTAL Direct Costs	***430***	***780***	***1 100***	***1 500***
Indirect Costs				
Temporary Construction Facilities	16	22	26	29
EPCM	72	130	180	230
Pre-production Owner's Costs	21	38	53	67
Project Fee	13	23	34	44
Contingency	42	78	110	150
TOTAL Indirect Costs	***170***	***290***	***410***	***510***
TOTAL US$M	**590**	**1 000**	**1 500**	**2 000**

Table 18.12 Summary of Roaster Capital Cost Estimate

AREA	Throughput Option			
	50 kt/d	**100 kt/d**	**150 kt/d**	**200 kt/d**
Direct Costs				
Site Works	5	8	11	13
Feed Handling	8	13	17	21
Molybdenum Roaster	21	42	62	83
Rhenium Recovery	20	32	43	52
Acid Plant	22	30	38	47
Gas Scrubbing	-	-	-	-
TOTAL Direct Costs	***75***	***130***	***170***	***220***
Indirect Costs				
Temporary construction facilities	8	13	17	22
EPCM	15	25	34	43
Pre-production Owner's costs	5	8	10	13
Project Fee	2	4	5	6
Contingency	16	26	36	45
TOTAL Indirect Costs	***45***	***75***	***100***	***130***
TOTAL US$M	**120**	**200**	**270**	**350**

The following is a brief methodology for the determination of capital cost estimates for the CUMO process plant, roaster and related ancillary infrastructure.

The CUMO circuit capital cost estimate was derived by factoring the mechanical equipment costs, which are defined in the concept study mechanical equipment list. 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 using factors developed from the Ausenco data base of projects.

18.9.1 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 base date of the cost estimate is 15th of July 2009. The estimate is expressed in United States Dollars.

For reference, the currency conversions rates used during the estimate preparation are:

- 1.00 US$ = CAD 1.09
- 1.00 US$ = AUD 1.225
- 1.00 US$ = EUR 0.713

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 have been excluded.

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.

18.9.2 Contingency

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

18.9.3 Owner’s Costs

Owner’s costs have been excluded from this estimate.

18.9.4 Project Fee

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

18.9.5 Escalation

Escalation provision past Q3 2009 has not been included in the estimate.

18.10 Tailings Capital Costs

The TSF capital cost estimate was based on conceptual-level material quantity and cost estimates. The estimates for construction of the TSFs for the four plant throughput options for the LOM are presented in Table 18.13.

Table 18.13 TSF Capital Cost Summary LOM

		50 kt/d		100 kt/d		150 kt/d		200 kt/d	
Description	**Unit**	**Quantity**	**US$M**	**Quantity**	**US$M**	**Quantity**	**US$M**	**Quantity**	**US$M**
Rough Grade Surface	Myd^2	1.9	2.5	5.3	6.9	6.7	8.7	5.5	7.1
Prepare Ground Surface to Receive Fill	Myd^2	1.9	1.0	5.3	2.6	6.7	3.4	5.5	2.7
Underdrains	ft	7,000	0.2	14,000	0.4	15,000	0.4	13,000	0.3
Low-permeability Core Fill	Myd^3	8.4	42	16.8	84	20.5	100	25.1	130
Drain Filter Fill	Myd^3	8.1	49	16.5	99	22.8	140	25.0	150
Rock Fill	Myd^3	112.9	68	307.9	180	494.3	300	412.6	250
Riprap	Myd^3	0.0	0.1	0.0	0.2	0.0	0.2	0.0	0.2
Seepage Collection Ponds	Ea.	3	0.2	4	0.2	4	0.2	5	0.3
TOTAL US$M			**160**		**380**		**550**		**540**

The majority of the unit rates was based on experience with similar projects and is to ±35% accuracy (Q3 2009). Costs for some items were assumed for this level of design and should suffice for the required level of accuracy. Other assumptions are noted below including that material shrinkage or bulking was not considered in calculating the site grading earthwork quantities.

The cost estimates assume that liquefiable foundation soils will be removed from the valley bottoms within the tailings dam footprints and replaced with rock fill. The presence of unsuitable foundation soils and the soils areal extent and depth will be evaluated in the Feasibility Study by geotechnical site investigations. The cost estimates will be adjusted based on the results of the investigations.

The cost estimates in the Table 18.13 are for unlined TSFs. It is estimated that lining the TSFs would cost an additional 20 to 30 percent of the unlined construction cost with the largest TSF having the highest lining cost as a percentage of the total cost.

18.11 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 feasibility study. The following items are excluded from this study:

- Power generation.
- Project acquisition costs.
- Feasibility study costs.
- Legal fees.
- Corporate costs.
- Exploration, geotechnical and sterilisation costs.
- Water compensation.
- Borefield or raw water dam.
- General and administration (G&A) cost estimates (included in financial model).
- Construction Camp.
- Plant or infrastructure outside of the battery limits.
- All Owner payable taxes, government and other charges.
- License and Royalty fees.
- No allowances are made for special incentives (schedule, safety or others).
- Sustaining or deferred capital costs (included in financial model).
- 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 and machinery breakdown etc.
- Costs for community relations and services.
- Any bridges or tunnels, permanent or temporary.
- Maintenance of all roads & 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.

18.12 Operating Cost Estimate

The total project operating costs for the different throughput options are summarised in Table 18.14. The costs are presented as Life of Mine (LOM) averages per short ton of ore processed.

Table 18.14 Summary of LOM Operating Costs

Operating Cost (million US $ per year)				
Description	**50 kt/d**	**100 kt/d**	**150 kt/d**	**200 kt/d**
Mining cost of mill feed	$13	$18	$21	$27
Mining cost of stockpile material	$29	$27	$26	$22
Mining cost of waste	$39	$40	$35	$32
Total Mining Cost	**$81**	**$85**	**$81**	**$81**
Plant	$91	$169	$251	$331
General & Administration	$5	$7	$8	$9
Closure and Reclamation Cost Allowance	$1	$2	$3	$4
Subtotal -Mine site Costs	**$178**	**$263**	**$344**	**$425**
Roaster	$17	$32	$48	$60
Realization costs	$8	$13	$19	$26
TOTAL OPERATING COST	**$200**	**$310**	**$410**	**$510**
TOTAL UNIT OPERATING COST ($/short ton milled)[1]	**$11.2**	**$8.6**	**$7.6**	**$7.1**
TOTAL UNIT OPERATING COST ($/short ton milled excluding stockpile mining cost)[4]	**$9.6**	**$7.8**	**$7.2**	**$6.8**

The estimate was prepared with a base date of July 2009 to an accuracy level of ±35%. Various parties contributed to the estimates as detailed below. These estimates exclude sustaining capital expenditure requirements, but include realisation costs associated with sale of final products.

18.12.1 Mining Operating Costs

CUMO mining costs have been estimated by Vector based on a factored analysis of the costs estimated for similar large open pit operations. Estimated or actual mining costs for five large open pit mining projects were used. The numbers were taken from both published and proprietary information.

The production numbers for CUMO used in the tables reflect the bench plans and mining schedule as discussed in Section 17.1.2. Pre-strip has been removed from the estimate of daily tonnage moved to arrive at the average daily tonnage moved that was used to calculate the mining costs.

Table 18.15 is a summary of the base case mining costs for CUMO for each of the scenarios before modification for site specific conditions for CUMO. Table 18.16 shows the amount of material moved for each scenario for the LOM. Based on a 40 year mine life with 360 work days per year the total tons moved per day were calculated. Using the average amount of material moved per day for the LOM and using the analysis of the costs for similar large open pit operations, a base case cost per ton moved was calculated for each production scenario without regard to site specific layout or equipment selection.

[2] *see Section 17.12.1 for detailed explanation*

Table 18.15 Base Case Mining Cost Summary

Production Rate	Ore	Stockpile	Waste	Prestrip	Total excluding Prestrip	Moved per Day	Mining Cost	Mining Cost
	Mt	(Mt)	(Mt)	(Mt)	Mt	Mt	$/ton moved	$/ton milled
50 kt/d	714	1,065	1,354	670	3,133	0.22	0.87	3.8
100 kt/d	1,425	1,667	2,316	823	5,408	0.38	0.58	2.2
150 kt/d	2,160	2,158	2,890	904	7,207	0.50	0.43	1.4
200 kt/d	2,880	2,045	2,954	968	7,879	0.55	0.38	1.0

The base case numbers include costs for drilling and blasting, loading, hauling, roads and dumps, and miscellaneous. Drilling and blasting, loading, roads and dumps and miscellaneous are assumed to be NOT site specific.

Haul costs ARE site specific. Haul costs must be modified to reflect the site layout for CUMO. These base case haul numbers were used as a starting point to estimate haul numbers specific to CUMO. The incremental increases in haul times and costs were calculated for each typical haul profile.

Table 18.16 Haul Stockpile and Waste and Convey Ore

Option		Drill & Blast	Loading	Hauling	Roads & Dumps	Other	Total	Total Costs Mining US$M	Ave Mining $/ton Moved
50 kt/d	Ore	0.18	0.11	0.30	0.07	0.08	0.74	530	**1.0**
	Stockpile	0.18	0.11	0.65	0.07	0.08	1.09	1 200	
	Waste	0.18	0.11	0.69	0.07	0.08	1.13	1 500	
100 kt/d	Ore	0.12	0.07	0.21	0.04	0.06	0.51	730	**0.6**
	Stockpile	0.12	0.07	0.36	0.04	0.06	0.65	1 100	
	Waste	0.12	0.07	0.37	0.04	0.06	0.67	1 500	
150 kt/d	Ore	0.09	0.05	0.16	0.03	0.04	0.38	830	**0.5**
	Stockpile	0.09	0.05	0.26	0.03	0.04	0.48	1 000	
	Waste	0.09	0.05	0.27	0.03	0.04	0.49	1 400	
200 kt/d	Ore	0.08	0.05	0.18	0.03	0.04	0.37	1 100	**0.4**
	Stockpile	0.08	0.05	0.23	0.03	0.04	0.42	870	
	Waste	0.08	0.05	0.23	0.03	0.04	0.43	1 300	

A review of the yearly mining costs shows that across all four production options, the yearly mining operating costs are nearly constant. Table 18.17 below shows the total cost per annum is nearly constant while the cost on a per ton basis declines with an increase in production.

The nearly constant per annum mining cost is a function of the decrease in costs as the mining rate increases and a proportional increase in the total tons moved. For the 50 kt/d option the daily tons moved average 217 500 t/d (short tons), while at 200 kt/d option the daily tons moved average 547 000 t/d (short tons); an increase of approximately 2.5 times while the mining costs per ton mined decrease approximately 2.7 times. This is a function of pit design and ore body configuration that results in lower strip ratios as the tonnage mined increases.

The decrease in mining costs is a result of economies of scale as the mining rate increases. As the mining rate increases, the size of the equipment increases and unit operating costs decrease resulting in overall lower operating costs on a per unit basis, especially in those areas of fixed costs.

Table 18.17 Summary of Mining Operating Costs

	Design			
	50 kt/d	**100 kt/d**	**150 kt/d**	**200 kt/d**
Total Cost Per annum (US$M)	81	84	81	81
Cost per ton of mill feed (US$)	4.5	2.3	1.5	1.1
Cost per ton of mill feed and stockpile (US$)	2.9	1.6	1.0	0.8

For the purposes of this assessment a fixed mine and plant life of 40 years has been selected to conduct the economic comparison despite the fact that the mine is not exhausted under any of the current proposed mining rates.

18.12.2 Mining Operating Cost Comparison

Overall average annual mining operating costs are approximately US$80M (Table 18.17), which equates to $4.5/short ton of ore processed for the 50 kt/d option, falling to $1.1/short ton for the 200 kt/d option. This is higher than other comparable operations, since lower grade material that is normally processed immediately or stockpiled and processed after the pit is exhausted is not included in the processing schedules developed for CUMO to date. For comparative purposes, the cost per ton of material mined (ore, low grade stockpile and waste but excluding pre-strip) is about $1.0/short ton for the 50 kt/d option falling to $0.4/short ton for the 200 kt/d option; these costs are comparable to similar sized operations.

If the processing plant life were extended beyond the current 40 years and the stockpiled material treated, the mining cost per ton of ore would be reduced to approximately $2.9 for the 50 kt/d option falling to $0.8 for the 200 kt/d option (excluding stockpile reclamation and rehandling costs), which are similar to comparable operations.

However, due to the long life of the CUMO operation, this operating scenario has been excluded from this analysis. The ability to extend the life of the processing plant beyond the current 40 year life is considered project upside that requires additional investigation during future study phases.

18.12.3 Process Plant Operating Costs

The total process operating costs have been developed on an annual basis throughout the life of the mine. Cost estimates were generated for each of the different throughput scenarios 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 ore treated for the Project is outlined in Table 18.18. The costs have been divided into the key cost centres.

All figures have been based on the study estimates applying as of the third quarter 2009 (calendar year).

Table 18.18 Estimated Plant Average Operating Costs

SUMMARY	50 kt/d	100 kt/d	150 kt/d	200 kt/d
Labour	0.5	0.2	0.2	0.2
Power	1.9	1.9	1.9	1.9
Maintenance Materials and Services	0.8	0.8	0.7	0.7
Reagents & Consumables	1.8	1.8	1.8	1.8
Miscellaneous	0.0	0.0	0.0	0.0
TOTAL US$/t (short tons)	**5.0**	**4.7**	**4.7**	**4.6**

Labour

Site labour costs are provided by Ausenco 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 as determined by Ausenco. 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.

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 Mosquito at US$0.063/kWh, based on information from the Thompson Creek Mine (Thompson Creek Mine Model, MineCost (2009)).

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.

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 ore 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.

18.13 Economic Analysis

Variability analyses were conducted using different metal prices, and varying capital and operating costs to determine the effect of these variables on the project economics. These analyses were conducted on the basis of the assumptions as listed below in Table 18.19.

Note that the preliminary 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 assessment will be realized.

Table 18.19 Base Assumptions for Economic Analysis

Parameter	Unit	Value
Plant throughput, t/d (short tons)	Option 1	50 000
	Option 2	100 000
	Option 3	150 000
	Option 4	200 000
Life of project, years		40
MoS_2 Grades – % Averages[5]		0.048 – 0.039
Copper Grades – % Averages[1]		0.100 – 0.087
Silver Grades – g/t Averages[1]		2.38 – 2.25
Concentrate molybdenum grade, %		51.7
Concentrate rhenium grade, g/t		35
Concentrate copper grade, %		22.4
Concentrate silver grade, g/t		
Process plant molybdenum recovery, %		Table 20
Process plant copper recovery, %		Table 20
Process plant silver recovery, %		Table 20
Moisture of molybdenum concentrate for transport, %		0
Moisture of copper concentrate for transport, %		10
Molybdenum transport cost, US$/t con	- to roaster	5.44
Molybdenum transport cost, US$/t Molybdenum Oxide	- to market	5.44
Sulfuric Acid transport cost, US$/t	- to market	27.22
Molybdenum roaster recovery, %		99
Roaster acid recovery,%		99
Sulfuric acid grade, % H_2SO_4		94
Roaster Rhenium recovery, %		90
Copper transport cost, US$/t con	- Road	30
	- Sea	0
Smelter costs, US$/t con		70
Base copper refining costs, US$/lb Cu		0.07
Silver refining costs, US$/oz Ag		0.40
Payable, %	Copper	96.5

[5] Average grades vary with throughput option

Table 18.19 cont. Base Assumption for Economic Analysis

Parameter	Unit	Value
	Silver	93.0
Royalties (% NSR)		0
Interest, %		Not Applicable
Taxation, %		Not Applicable
Depreciation, %		Not Applicable
Amortisation, %		Not Applicable
NPV discount rate, %		5
Base Molybdenum price, $US/lb		16.0
Base copper price, $US/lb		2.10
Base silver price, $US/oz		12
Base acid price, $US/t		135
Base rhenium price, $US/kg		6500
Base capital cost, US $M	Option 1	1 600
	Option 2	2 200
	Option 3	2 800
	Option 4	3 400
Total operating cost, $USM	Option 1	8 000
	Option 2	12 400
	Option 3	16 400
	Option 4	20 400
Sustaining capital cost, $USM	Option 1	800
	Option 2	1 700
	Option 3	2 500
	Option 4	2 600

18.13.1 Economic Analysis (Base Case)

The base case economic analysis, based on the estimates of capital and operating costs and assumptions as listed in Table 18.20 indicates that, given the current estimated mining and plant operating costs, as well as capital cost estimates, the internal rate of return (%IRR), Net Present Value at 5% discount rate (NPV5), payback period (years), discounted payback period at 5% and operating costs per pound of molybdenum oxide are as shown below in Table 60. All values are calculated based on Earnings Before Interest Tax Depreciation and Amortisation (EBITD&A).

Table 18.20 Base Case Economic Analysis

Economic parameters (EBITD&A)	Throughput Option			
	50 kt/d	**100 kt/d**	**150 kt/d**	**200 kt/d**
NPV (US$M @ 5%)	4	10	16	21
IRR%	19	29	36	39
Simple Payback Period (years)	4.9	3.0	2.3	2.0
Discounted Payback Period (years @ 5%)	6.1	3.6	2.7	2.3
Total Operating Costs per lb of Molybdenum Oxide Equivalent	**5.5**	**4.3**	**3.9**	**3.8**

18.13.2 Sensitivity analysis (Metal Prices)

A basic sensitivity analysis was conducted on the economic effects of various metal price scenarios. The following Table 18.21 shows a matrix of the various metal prices used in the scenarios analysed.

Table 18.21 Metal Price Sensitivity

Metal	Units	Metal Prices		
		High	Medium[6]	Low
Molybdenum Oxide	US$/lb	28	16	7.5
Copper	US$/lb	3.5	2.1	1.5
Silver	US$/Oz (troy)	15	12	9.0
Rhenium	US$/kg	10 000	6 500	2 500
Sulfuric Acid	US$/t (short ton)	235	135	85

A further sensitivity analysis was conducted on the basis of cyclical metal prices, with average prices similar to the medium prices shown in Table 18.21, but assuming that the operation commences production on the commencement of the upturn in metal prices.

The annual metal prices used in this scenario are summarised in Table 18.22.

Table 18.22 Cyclical Metal Price Scenario

Year	Molybdenum Oxide	Copper	Silver	Rhenium	Sulfuric Acid
	US$/lb	US$/lb	US$/oz	US$/kg	US$/t
1	$22.0	$3.0	$12.0	$6,500.0	$125.0
2	$25.0	$3.3	$13.5	$8,250.0	$180.0
3	$28.0	$3.5	$15.0	$10,000.0	$235.0
4	$21.2	$2.8	$13.0	$7,500.0	$185.0
5	$14.3	$2.2	$11.0	$5,000.0	$135.0
6	$7.5	$1.5	$9.0	$2,500.0	$85.0
7	$11.8	$2.0	$10.5	$4,500.0	$110.0
8	$16.0	$2.5	$12.0	$6,500.0	$135.0
9	$18.0	$2.7	$12.0	$6,500.0	$131.7
10	$20.0	$2.8	$12.0	$6,500.0	$128.3
11	$22.0	$3.0	$12.0	$6,500.0	$125.0
12	$25.0	$3.3	$13.5	$8,250.0	$180.0
13	$28.0	$3.5	$15.0	$10,000.0	$235.0
14	$21.2	$2.8	$13.0	$7,500.0	$185.0
15	$14.3	$2.2	$11.0	$5,000.0	$135.0
16	$7.5	$1.5	$9.0	$2,500.0	$85.0
17	$11.8	$2.0	$10.5	$4,500.0	$110.0
18	$16.0	$2.5	$12.0	$6,500.0	$135.0
19	$18.0	$2.7	$12.0	$6,500.0	$131.7
20	$20.0	$2.8	$12.0	$6,500.0	$128.3
21	$22.0	$3.0	$12.0	$6,500.0	$125.0
22	$25.0	$3.3	$13.5	$8,250.0	$180.0
23	$28.0	$3.5	$15.0	$10,000.0	$235.0
24	$21.2	$2.8	$13.0	$7,500.0	$185.0
25	$14.3	$2.2	$11.0	$5,000.0	$135.0
26	$7.5	$1.5	$9.0	$2,500.0	$85.0
27	$11.8	$2.0	$10.5	$4,500.0	$110.0
28	$16.0	$2.5	$12.0	$6,500.0	$135.0
29	$18.0	$2.7	$12.0	$6,500.0	$131.7
30	$20.0	$2.8	$12.0	$6,500.0	$128.3
31	$22.0	$3.0	$12.0	$6,500.0	$125.0
32	$25.0	$3.3	$13.5	$8,250.0	$180.0
33	$28.0	$3.5	$15.0	$10,000.0	$235.0
34	$21.2	$2.8	$13.0	$7,500.0	$185.0
35	$14.3	$2.2	$11.0	$5,000.0	$135.0
36	$7.5	$1.5	$9.0	$2,500.0	$85.0
37	$11.8	$2.0	$10.5	$4,500.0	$110.0
38	$16.0	$2.5	$12.0	$6,500.0	$135.0
39	$18.0	$2.7	$12.0	$6,500.0	$131.7
40	$20.0	$2.8	$12.0	$6,500.0	$128.3
Average	**$16.2**	**$2.7**	**$12**	**$6,326**	**$145**

A matrix of the IRR for the four throughput options for each of the metal pricing scenarios is shown below in Table 18.23.

Table 18.23 IRR Sensitivity to Metal Pricing

Metal Price Scenario	%IRR Sensitivity (EBITD&A basis)			
	50 kt/d	100 kt/d	150 kt/d	200 kt/d
High	36	51	60	66
Cyclical	26	39	49	54
Medium	19	29	36	39
Low	3	9	12	15

A matrix of the Project NPV5 for the four throughput options for each of the metal pricing scenarios is shown below in Table 18.24.

Table 18.24 NPV5 Sensitivity to Metal Pricing

Metal Price Scenario	NPV5 Sensitivity US$M (EBITD&A basis)			
	50 kt/d	100 kt/d	150 kt/d	200 kt/d
High	10 000	22 000	35 000	45 000
Cyclical	5 200	12 000	21 000	27 000
Medium	3 800	9 700	16 000	21 000
Low	-500	1 100	2 900	4 400

18.13.3 Sensitivity analysis (Variability)

A further sensitivity analysis was conducted to ascertain the effect of variability of the following parameters:

- molybdenum oxide price
- copper price
- rhenium price
- sulfuric acid
- capital cost
- operating cost.

The variation of molybdenum oxide, copper, rhenium and sulfuric acid prices specified are listed below in Table 18.25.

Table 18.25 Metal Pricing for Sensitivity Analysis

Molybdenum Oxide	Copper	Rhenium	Sulfuric Acid
US$/lb	US$/lb	US$/kg	US$/t
12.00	1.75	1 500	35
14.00	2.00	3 000	85
16.00 – Base case	2.10 – Base case	6 500 – Base case	135 – Base case
18.00	2.25	8 000	175
20.00	2.50	10 000	200

The operating and capital costs were varied from the base costs by -20, -10, 10 and 20%. This variation was at Ausenco's discretion.

This analysis was conducted by varying one parameter at a time to determine an IRR and NPV. The results of this analysis are depicted in Figure 18.1 and Figure 18.2 for the 50 kt/d (short ton) throughput option.

Figure 18.1 50 kt/d Throughput IRR Sensitivity

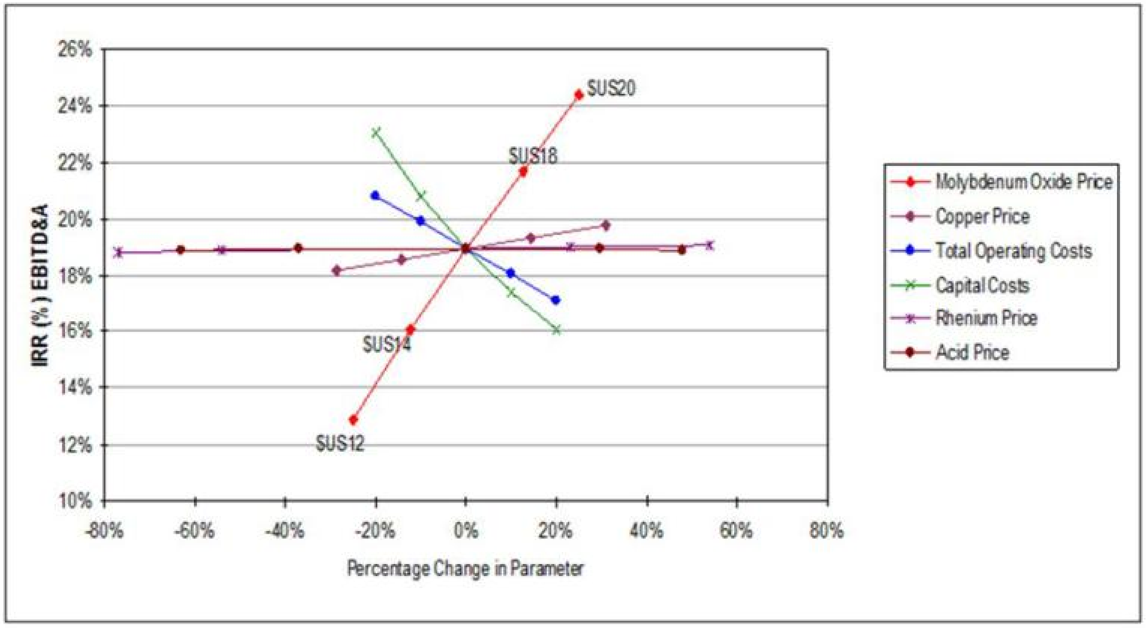
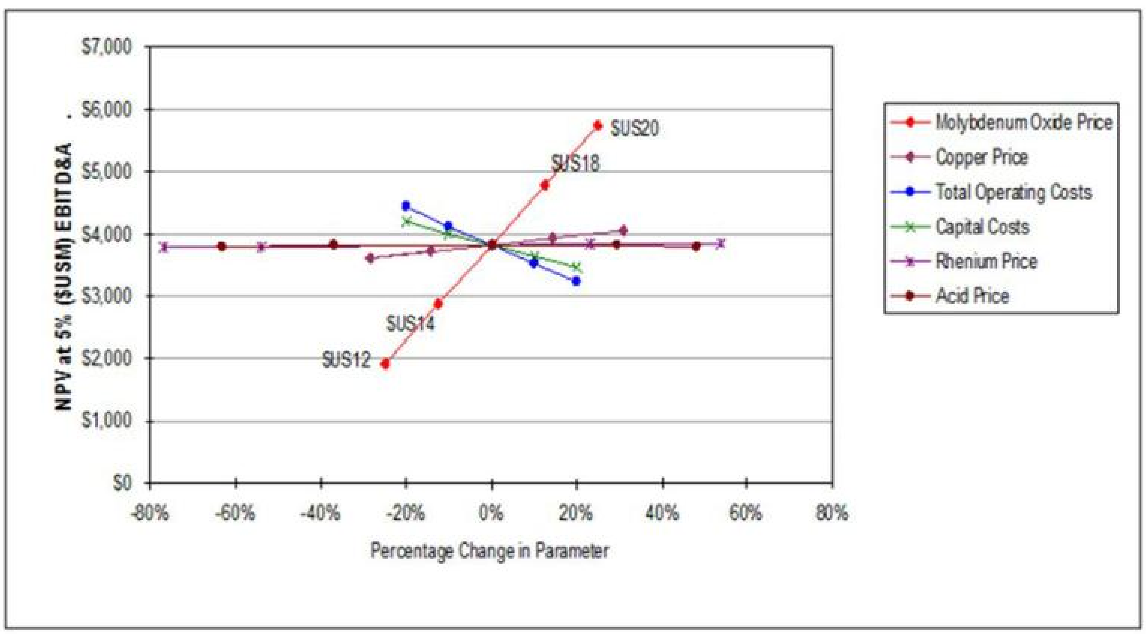


Figure 18.2 50 kt/d Throughput NPV Sensitivity



In the two figures above, the point at which all lines meet is the base case (see assumptions above). The lines emanating out from this point show the influence of varying the different parameters from that base. It can be seen that varying the copper price causes minor variations in the NPV, as this line is relatively flat. The rhenium and sulfuric acid lines are almost horizontal, indicating that the prices of these products have almost no impact on the project economics. The capital and operating cost lines are moderately steeper, indicating reasonable sensitivity to both project capital and operating costs. However, the molybdenum oxide price slope is relatively steep; indicating this to be the most sensitive parameter for the project.

The same sensitivity analysis was conducted for the other three throughput scenarios (See Figure 18.3 through to Figure 18.8). The relative sensitivities for variations in the parameters tested are very similar for all throughput options.

Figure 18.3 100 kt/d Throughput IRR Sensitivity

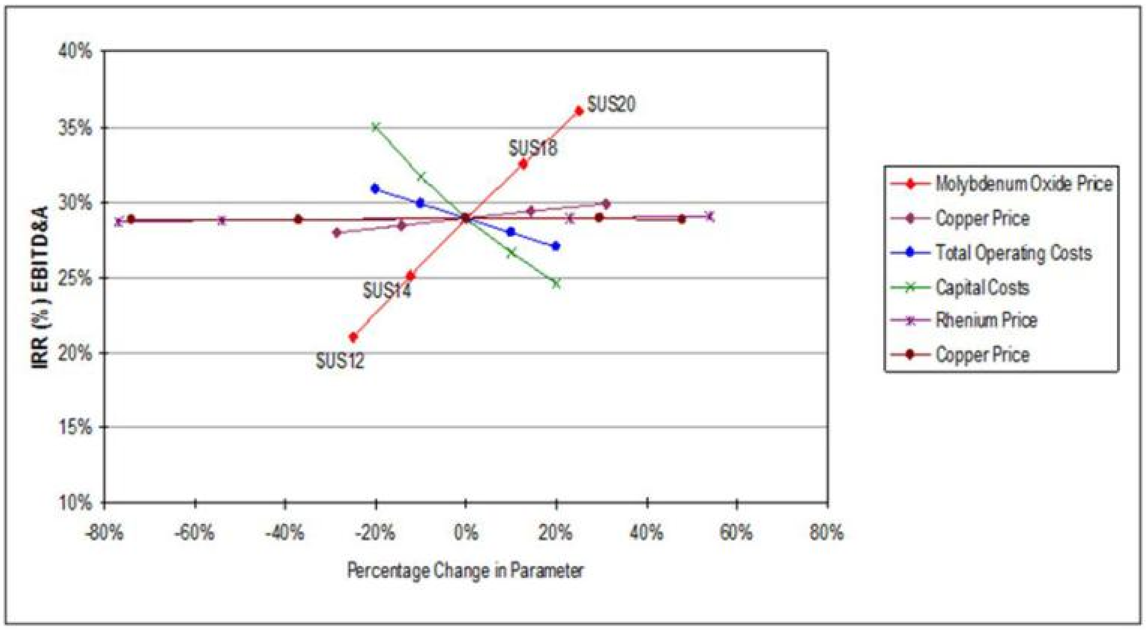


Figure 18.4 100 kt/d Throughput NPV Sensitivity

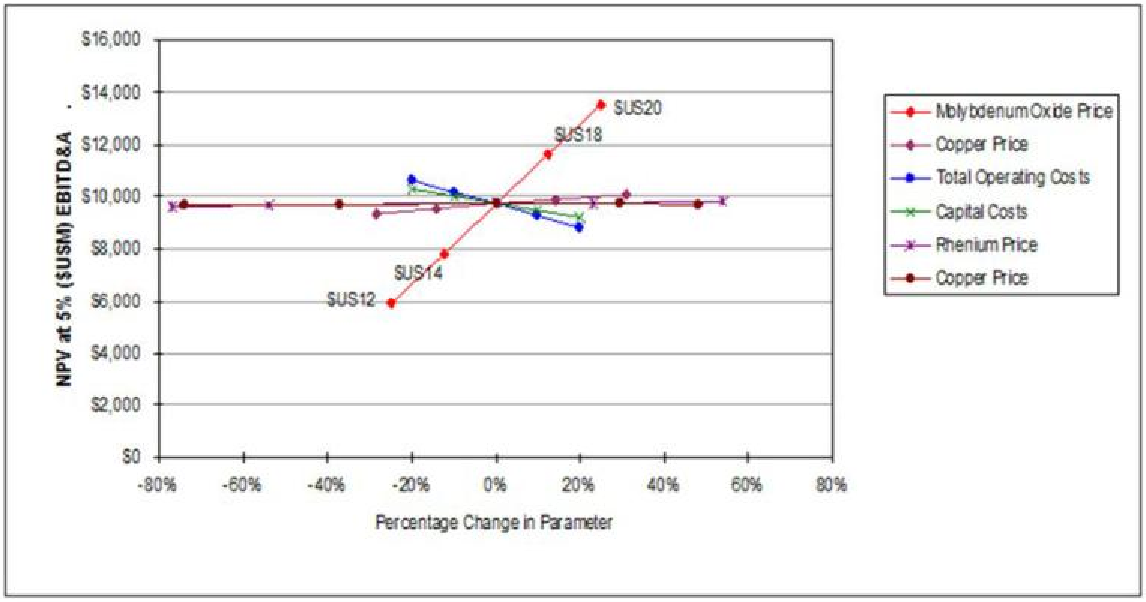


Figure 18.5 150 kt/d Throughput IRR Sensitivity

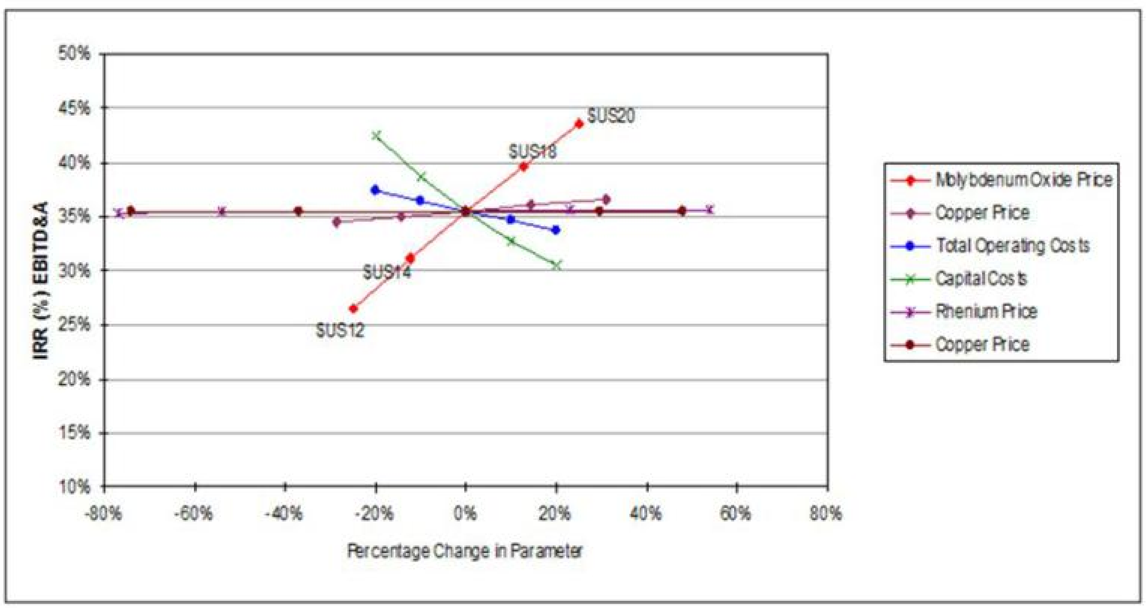


Figure 18.6 150 kt/d Throughput NPV Sensitivity

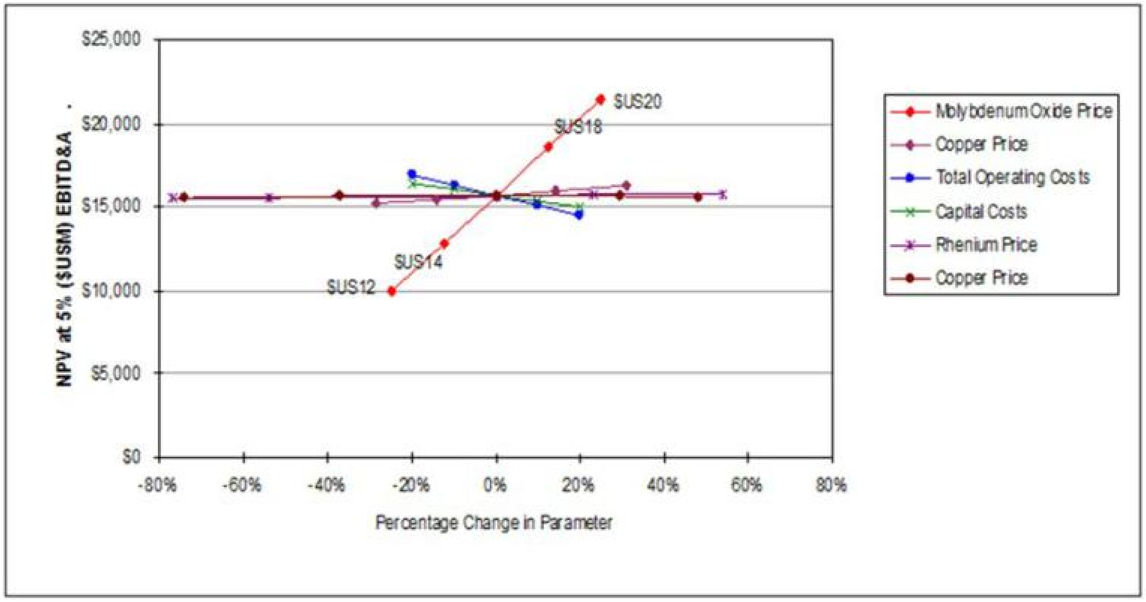


Figure 18.7 200 kt/d Throughput IRR Sensitivity

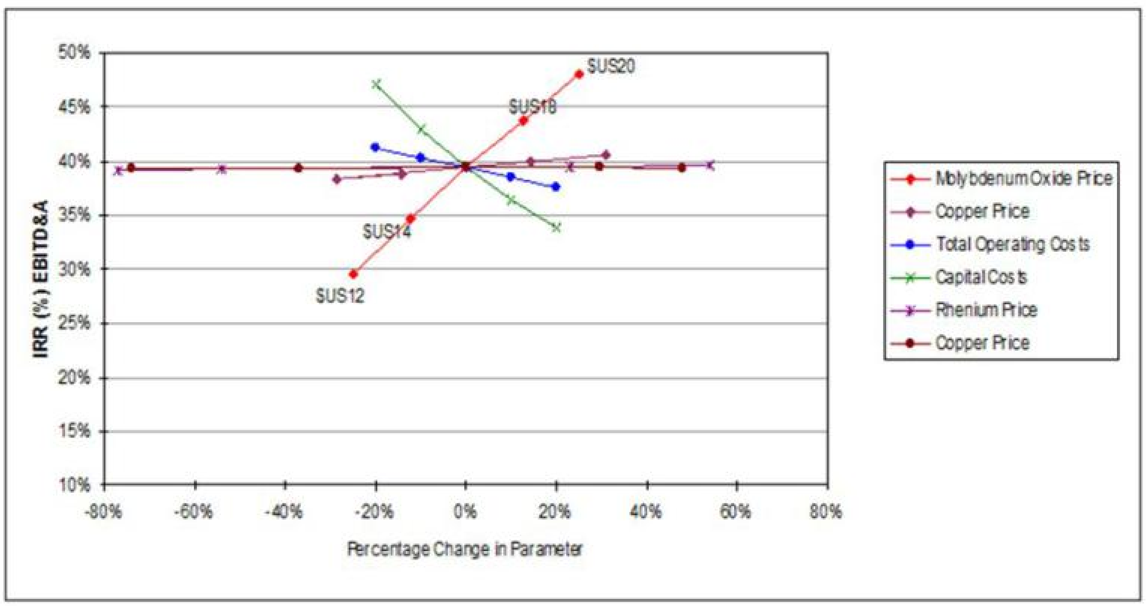
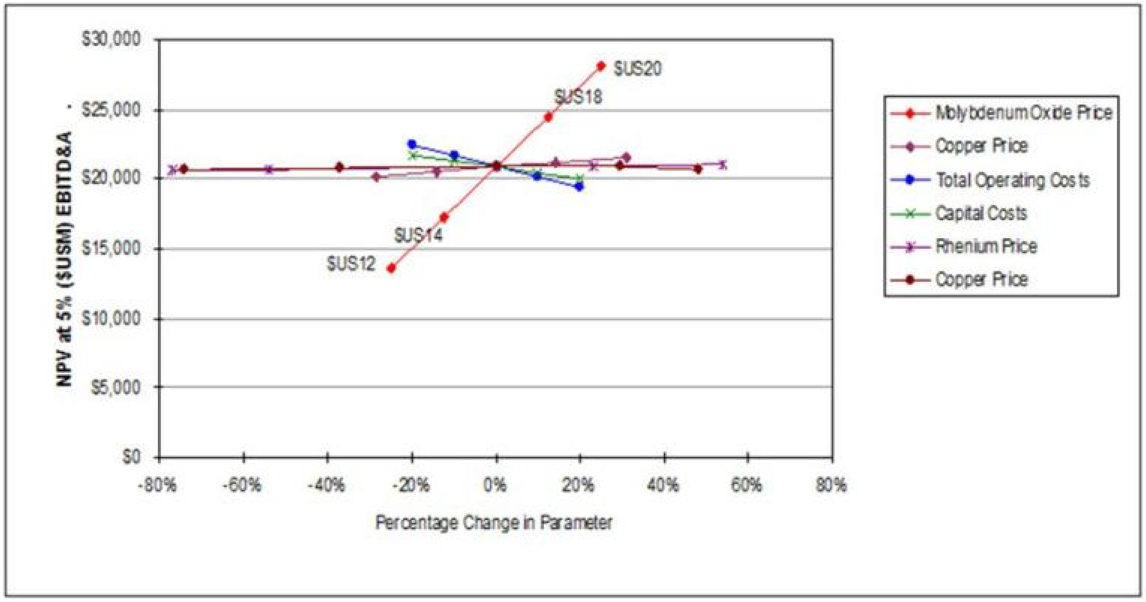


Figure 18.8 200 kt/d Throughput NPV Sensitivity



19 Interpretation and conclusions

The 2011 Mineral Resource represents an update of the CUMO resource estimate completed after the 2010 drill program utilizing the additional twelve diamond drillholes totalling 7,000 metres (22,968 ft).

For this estimate, variography was conducted after the major post mineral fault blocks were rotated back into their pre fault positions based on marker horizons. This allowed for evaluation of data on either side of major faults and resulted in the determination of more realistic anisotropic grade continuity ranges

The drilling undertaken in 2009 and 2010 has allowed for a significant increase in the overall quantity of mineralisation that has sufficient confidence to be classified as a Mineral Resource. The additional data has also resulted in longer continuity ranges being demonstrated in the variograms. This demonstrated continuity means that a significant portion of the mineralisation can be classified as an Indicated Mineral Resource.

Based on the Mineral Resources defined to date the CUMO project continues to be a notable project and it is therefore recommended that the project be advanced to feasibility stage. The drilling program recommended by Ausenco in November 2009 is in progress and is proposed to be carried out over a minimum time frame of two years with an estimated cost of $72.5 million.

20 Recommendations

The following recommendations are made for the further advancement of the Project:

- The lateral and down dip extensions of mineralization in CUMO are not well established. More drilling is required to define the full extent of the mineralization. The drilling program defined as by Ausenco in November 2009 (Ausenco, 2009) is in progress and may help define the lateral and down dip extensions to the mineralization.
- Infill drilling is required to improve the confidence in the geological interpretation and to define the shape of mineralization domains. This additional drilling will also help in demonstrating grade continuity with variography to support resource estimation and higher confidence resource classification;
- Snowden recommends a more rigorous approach to quality control and quality assurance including:
    - Reject and re-analyse sample batches where any of the QAQC samples types (i.e. blanks, duplicates, or standards) returns analytical results outside of the certified or specified control limits.
    - Compile and monitor the analytical quality control data as the data are received from the assay laboratory to ensure the detection of errors and, if necessary, trigger appropriate remediation measures.
    - Prepare a periodical report and use industry standard criteria for rejecting the batches and document all of the rejected samples in the QAQC report.
    - Use field duplicate samples instead of coarse rejects in order to assess the potential errors associated with sample preparation.
- More bulk density samples should be taken for this deposit. Snowden recommends the preparation of adequate bulk density samples (at least 10 percent of the samples) throughout the extent of the deposit. Bulk density measurements should be routinely taken on core samples from all rock types to augment the existing data and confirm variability of specific gravity between each geological domain.
- Snowden recommends the preparation of a validated solids model, in software such as Datamine or Gemcom, for different zones and using these solids for updating the resource estimation purpose.
- Snowden agrees with the proposal by Ausenco in 2009 that further bench scale metallurgical tests, including petrography, grinding, and milling, to assess the variability of the various mineralized zones to metallurgical processes.
- Continue the modelling of fault interpretations for use in future resource estimations.
- Snowden further recommends that Mosquito undertake a drillhole spacing study at CUMO using conditional simulation to quantify the optimal drillhole spacing required to achieve a range of estimation qualities. Some close-spaced drilling should be performed in a representative mineralized domain to characterise the short-range behaviour of the mineralisation.

20.1 Recommended Work Programme

Mosquito's work plan (costed in Table 20.1), which Snowden endorses and recommends includes the following work programme:

i. **2012 Diamond Drill Program** – drill off an area approximately 1,000 m square and 300 m deep to improve the confidence in approximately *500,000,000 tonnes* of mineralization. 15,000 m of core drilling will be within this area, 5,000 m of core drilling will be "step outs" to test potential high grade targets within the deposit and to test for the limits of the deposit. At the time of this report, the deposit is open in all directions. Three core holes will be twinned with Reverse Circulation (RC) drilling in order to correlate and calibrate the metal recoveries between the 2 types of drilling. This should provide a resource of a sufficient size to prepare a reliable *Pre-feasibility Study*.

   Drill Program Costs – Core drilling is estimated at $400/m while RC drilling is estimated at $150/m.

   Metallurgical Sampling – The 3 twinned core drill holes will be drilled with PQ (85mm) size core; the balance of the core holes will be drilled with HQ (63 mm) size core. It is expected that approximately 1,000 m of PQ core will be available for metallurgical testing.

ii. **2013 Diamond Drill Program -** drill off a second area approximately 1,000 m square and to a depth of 300 m to improve the confidence (and therefore classification) of an additional 500,000,000 tonnes. 15,000 m of core drilling will be within this area, 5,000 m of core drilling will be "step outs" to test potential high grade targets within the deposit and to test for the limits of the deposit. Because of the areal extent of the deposit, the deposit is expected to still remain open in all directions after the 2012 program and will probably remain open in all directions after the 2013 program. This should provide a resource of a sufficient size to prepare a reliable Feasibility Study.

iii. **2012 Reverse Circulation Drill Program** - drill off within the same area as the core drilling, an area approximately 1,000 m square and to a depth of 300 m to improve the confidence (and therefore classification) of an additional 500,000,000 tonnes. 16,000 m of RC drilling will be within this area, but located between the core holes. Three RC holes will be twinned with core drilling in order to correlate and calibrate the metal recoveries between the 2 types of drilling. Three RC holes will also be used as groundwater monitoring wells in support of environmental baseline studies for production permitting. A total of 18 groundwater monitoring wells have been budgetted for permitting.

   Metallurgical Sampling – The 3 RC holes drilled for groundwater monitoring will 150 mm diameter holes. All other RC holes will be 115 mm diameter. All RC holes will be fully sampled for metallurgical testing.

iv. **2013 Reverse Circulation Drill Program** - drill off a second area approximately 1,000 m square and to a depth of 300 m to improve the confidence (and therefore classification) of an additional 500,000,000 tonnes., 15,000 m of RC drilling will be within this area, but located in between the core holes. Additional RC holes will also be used as groundwater monitoring wells in support of environmental baseline studies for production permitting.

v. **2012 Geology** – Limited reconnaissance work over those areas that may be suitable for mine infrastructure. This will include geological mapping, limited ground geophysics and soil geochemistry sampling.

vi. **2013 Geology** – Extensive reconnaissance work over the entire property to evaluate areas proposed for tailings and waste rock. This will include LIDAR (for those areas where detailing topography is not available), airborne geophysics (Magnetometer, EM, and radiometric surveys), detailed geological mapping and soil geochemistry sampling over selected areas.

vii. **2012 Metallurgical Test Program** – Sample collection, from RC drilling, is expected to be completed in October, 2012. Approximately 300 tonnes of core and RC chip samples from the three mineralized zones (Ag, Cu-Ag, and Mo) within the deposit will be shipped to Hazen Research (Denver, CO) for metallurgical testing. Mineralogy will be identified, bench scale testing, including flotation and autoclave processing for flow sheet design will be undertaken and a pilot plant test scaling up the bench work will completed. It is expected that approximately 300 kg of concentrate product will be produced and available for market studies.

viii. **2013 Metallurgical Test Program** – The metallurgical test program is expected to take approximately six months in total. Work in 2013 will be a continuation of the program initiated in the fall of 2012 and is expected to be completed in March 2013. The aim of the metallurgical program is to produce the mill flow sheet and mill process equipment selection.

ix. **2012 Baseline Environment** – Surveys and sampling to establish the environmental baseline, that is, the existing conditions on the property, are being initiated in 2012. Studies include local weather, wetlands, soils, surface and groundwater, stream sediments, fisheries, invertebrates, terrestrial species (birds, mammals), large mammals and vegetation. Particular attention will be paid to contamination from previous human activity such as mining, forestry and ranching operations. Mining has taken place throughout the region since the mid to late 19th century culmination in extensive dredging of Grimes Creek downstream of CuMo. It is necessary to quantify the impact of these activities before the possible cumulative effects that could result from CuMo can be assessed.

x. **2013 Baseline Environment** – A minimum of 2 years of data is required to properly establish the environmental baseline. All investigative programs will continue through the year.

xi. **2012 Tailings and Waste Rock** – Tailings and waste rock chemistry and disposal will be evaluated in conjunction with the metallurgical test program. This will include geochemical characterization, acid base accounting, kinetic testing (humidity cells and column tests) and toxicity testing. Kinetic testing requires up to 2 years for meaningful results. This program will be initiated in 2012 and completed in conjunction with the Feasibility Study.

xii. **2013 Tailings and Waste Rock** - Preliminary design of tailings impounds and waste dumps will be included in the Pre-feasibility Study. One year of kinetic results will be available for the Pre-feasibility Study.

xiii. **Land Acquisition** – Expansion of the property to provide a larger buffer around the deposit, plant facilities and alternate locations for tailings and waste rock dumps.

xiv. **Land Acquisition** - Expansion of the property to provide a larger buffer around the deposit, plant facilities and alternate locations for tailings and waste rock dumps.

xv. **Pre-feasibility Study** – A Scoping Study was prepared in November, 2009. Results of this investigation, with an initial financial analysis of the Indicated Mineral Resource were promising. Building on the Scoping Study, a Pre-feasibility Study will be prepared that will include a preliminary mine plan, more detailed engineering and an expanded exploration program that will target upgrading a portion of the resource to the measured category. This study will expand plans and factor known costs from existing projects to an accuracy of 20 - 30%.

Table 20.1 Costing of recommended work plan for 2012 and 2013.

	2012	2013
Drilling - Core	8,264,000i	3,680,000ii
Drilling - RC	2,250,000iii	3,150,000iv
Geology	600,000v	900,000vi
Metallurgy	500,000vii	500,000viii
Baseline Environmental Monitoring and Permitting	1,000,000ix	1,000,000x
Tailings and Waste Rock	250,000xi	500,000xii
Land Acquisition	3,000,000xiii	2,000,000xiv
Pre-feasibility Study Documents		1,750,000xv
Feasibility Study Documents		
Corporate	2,000,000	2,000,000
Totals (Annual)	17,864,000	15,480,000
Pre-feasibility		17,614,000
Feasibility		
Contingency (10%)		1,761,000
Totals		19,375,000

Note, notations i to xv relate to items in the recommended work programme (Section 20.1)

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22 Date and Signatures

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23 Certificates

CERTIFICATE of QUALIFIED PERSON

(a) I, Ivor W.O. Jones, Senior Principal Consultant and Group General Manager – Geosciences of Snowden Mining Industry Consultants Inc., 87 Colin St., West Perth, Western Australia; do hereby certify that:

(b) I am a co-author of the technical report titled Mosquito Consolidated Gold Mines Limited: CUMO Project Resource Estimation Update dated June 13, 2011 and amended June 20, 2012 (the 'Technical Report') relating to the Cumo property.

(c) I graduated with an Honours Degree in Bachelor of Science in Geology from Macquarie University in Sydney Adelaide in 1986. In 2001 I graduated with a Master of Science degree in Resource Evaluation from the University of Queensland. I am a Fellow and Chartered Professional (Geology) of the Australasian Institute of Mining and Metallurgy. I have worked as a geologist continuously for a total of 25 years since my graduation from university.

I have read the definition of 'qualified person' set out in National Instrument 43-101 ('the Instrument') and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements of a 'qualified person' for the purposes of the Instrument. I have been involved in mining and resource evaluation consulting practice for 20 years, including resource evaluation of porphyry deposits for at least 5 years.

(d) I visited the Cumo property on one occasion on 23 November, 2010.

(e) I am responsible for the preparation of sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 13, 14, 15, 16.1.2, 17, 18.5, 19 and 20 of the Technical Report.

(f) I am independent of the issuer as defined in section 1.4 of the Instrument.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

(i) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Perth, Western Australia this 20 Day of June, 2012.

Signed

Ivor W.O. Jones

Ivor W.O.Jones

CERTIFICATES OF QUALIFIED PERSONS

Charles J. Khoury, P.E.

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I, Charles Khoury, P.E., certify that I am a Senior Geotechnical Engineer at Golder Associates Inc., 44 Union Boulevard, Suite 300, Lakewood, Colorado 80228, USA.

This certificate applies to the Technical Report titled "**CUMO Project Resource Estimation Update**" dated June 13, 2011 and amended June 20, 2012.

My qualifications and relevant experiences are that:

a) I graduated with a Master of Science in Civil Engineering from the University of Kentucky, May 1987.

b) I am a member of the SME and ASCE and a Professional Engineer in the States of Colorado and Nevada.

c) I have worked as a Geotechnical Engineer for a total of 24 years.

d) I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.

e) I have not visited the Property.

f) I am responsible for the preparation of Sections 18.2, 18.3, 18.4, and 18.10 of the Technical Report which have been reproduced in their entirety from the "CUMO Property Preliminary Economic Assessment Throughput Scoping Study" report dated November 18, 2009.

g) At the time of the writing of the "CUMO Property Preliminary Economic Assessment Throughput Scoping Study" I was employed as a Senior Geotechnical Engineer by Vector Engineering of Golden, Colorado, USA, and it was in this capacity that I authored the above mentioned sections.

h) I am independent of the issuer per Section 1.4 of NI 43-101.

i) I have not had prior involvement with the property that is the subject of the Technical Report.

j) I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with that instrument.

k) As of the date of the certificate, to the best of my knowledge, information and belief, the aforementioned Sections 18.2, 18.3, 18.4, and 18.10 of the Technical Report contain all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated:	June 20, 2012
Signature of Qualified Person:	**[Signed]**
Name of Qualified Person:	Charles J. Khoury

CERTIFICATE OF QUALIFIED PERSONS

Telephone:	303 986-6950
Facsimile:	303 987-8907
E-mail:	rkehmeier@pincock.com

I, Richard J. Kehmeier, B.Sc. Geo. Eng, M.Sc. Geology, CPG, certify that I am a Principal Geologist at Pincock Allen & Holt, 195 South Union Blvd., Suite 950, Lakewood, Colorado 80228, U.S.A.

This certificate applies to the Technical Report titled "**CUMO Project Resource Estimation Update**" dated June 13, 2011 and amended June 20, 2012.

My qualifications and relevant experiences are that:

a) I graduated with a Batchelor of Science in Geological Engineering in 1970 and a Masters of Science in Geology in 1973 from the Colorado School of Mines, Golden, Colorado.
b) I am a member of AIPG
c) I have worked as a Geologist for a total of 41 years.
d) I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.
e) I have not visited the Property.
f) I am responsible for the preparation of Sections 18.1, 18.8, and 18.12.1 of the Technical Report which have be reproduced in their entirety from the "CUMO Property Preliminary Economic Assessment Throughput Scoping Study" report dated November 18, 2009.
g) At the time of the writing of the "CUMO Property Preliminary Economic Assessment Throughput Scoping Study" I was employed asa Senior Geologist by Ausenco / Vector Engineering of Golden, Colorado and it was in this capacity that I authored the above mentioned sections.
h) I am independent of the issuer per Section 1.4 of NI 43-101.
i) I have not had prior involvement with the property that is the subject of the Technical Report.
j) I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with that instrument.
k) As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated:	June 20 2012
Signature of Qualified Person:	**[Signed and Sealed]**
Name of Qualified Person:	Richard J. Kehmeier

CERTIFICATE OF QUALIFIED PERSONS

Telephone:	778 374 6999
Facsimile:	778 374 6998
E-mail:	kevin.scott@ausenco.com

I, Kevin Scott, P.Eng. do certify that I am Manager, Process and Studies for Ausenco Solutions Canada Inc., 855 Homer St., Vancouver, British Columbia, V6B 2W2, Canada.

This certificate applies to the Technical Report titled "**CUMO Project Resource Estimation Update**" dated June 13, 2011 and amended June 20, 2012.

My qualifications and relevant experiences are that:

a) I am a graduate of University of British Columbia, Vancouver, Canada in 1989 with a Bachelor of Applied Science degree in Metals and Materials Engineering.

b) I am registered as a Professional Engineer in the Province of British Colombia (Licence # 25314) and the Province of Ontario (License # 90443342).

c) I have worked as a Metallurgist continuously for a total of 22 years since my graduation from University. My relevant experience for the purpose of the Technical Report is:

   1. Reviews and reports as a metallurgical consultant on a number of mining operations and projects for due diligence and financial monitoring requirements
   2. Process engineer at three Canadian base metals mineral processing operations
   3. Senior metallurgical engineer working for three multi-national engineering and construction companies on feasibility studies and in engineering design of mineral processing plants in Canada and South America
   4. Senior process manager in charge of process design and engineering for a metallurgical processing plant in South America.

d) I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101) and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of this NI 43-101.

e) I did not visit the property.

f) I am responsible for the preparation of Sections 16 (except sub-section 16.1.2), 18.6, 18.7, 18.9, 18.11, 18.12.2, 18.12.3: and 18.13 of the Technical Report which have be reproduced in their entirety from the "CUMO Property Preliminary Economic Assessment Throughput Scoping Study" report dated November 18, 2009.

g) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Mosquito Consolidated Gold Mines, or any associated or affiliated entities.

h) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Mosquito Consolidated Gold Mines, or any associated or affiliated companies.

i) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Mosquito Consolidated Gold Mines, or any associated or affiliated companies.

j) I am independent of the Issuer applying the test set out in Section 1.5 of the NI 43-101.

k) I have read National Instrument 43-101 and Form 43-101F1, and confirm that the Technical Report has been prepared in compliance with that instrument and form.

l) To the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make this technical report not misleading.

Dated:	June 20 2012
Signature of Qualified Person:	**[Signed]**
Name of Qualified Person:	Kevin Scott