American CuMo Mining Corporation

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# CuMo project throughput Scoping Study Report

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# CUMO PROPERTY PRELIMINARY ECONOMIC ASSESSMENT

# THROUGHPUT SCOPING STUDY REPORT

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**REVISION NUMBER E**


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# 1 SUMMARY

## 1.1 Introduction and History

The CUMO deposit is a molybdenum-copper deposit situated 60 kilometres (37 miles) 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 meters (36,025.8 feet) in 22 diamond drill holes. A geologically inferred historic resource of 1.36 billion tonnes at 0.092% $MoS_2$ ***(Non Compliant with 43-101 – see History)*** was calculated by block modeling 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. Mosquito has completed 14,729 meters (44,188 feet) of drilling in 19 diamond drill holes.

## 1.2 Geology

The CUMO deposit is located at the southwestern end of the Idaho-Montana Porphyry Belt. Igneous complexes in this belt are interpreted to be related to an Eocene, intra-arc rift, and are characterized by alkalic rocks in the northeast, mixed alkalic and calc-alkalic rocks in the middle, and calc-alkaline rocks in the southwest. The CUMO deposit is typical of large, dispersed, low-grade molybdenum ± copper porphyry deposits that are associated with hybrid magmas typified by fluorine-poor, differentiated monzogranite igneous complexes. Due to their large size, the total contained economic molybdenum in these types of deposits can be equivalent to or exceed that of high grade molybdenum deposits. 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 three distinct metal zones within the deposit. These zones were previously interpreted by Amax as distinct ore shells that were produced by separate intrusions. Re-interpretation of down-hole histograms for Cu, Ag and Mo suggests the metal zones are part of a single, large, concentrically zoned system with 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 the above zonation indicates the current area being drilled is located on the north side of a large system extending 4.5 km (15,000 feet) in diameter, of which only a small part (1 km or 3000 feet) has been drilled.

## 1.3 Resource Estimate

A resource estimate update was completed at the request of Mosquito based on a total of 42 diamond drill holes totalling 76,436 ft. Of these 11 diamond drill holes were completed in 2008. A geologic model separating the CUMO Deposit into three 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 three geologic domains: a near surface Cu-Ag zone, a deeper Cu-Mo zone and a still deeper Mo zone. Statistics on each variable in each Domain led to the capping of assays based on the grade distribution within each Domain. Uniform down hole 50 ft. composites were produced for each domain. For variography the major post mineral fault blocks were rotated back to their original position using marker beds. Semivariograms were produced for each variable within each domain based on the samples original pre fault locations. A block model with block dimensions of 50 ft. was superimposed on the mineralized domains. Grade was interpolated into blocks by ordinary kriging. A tonnage factor was determined for each domain based on multiple specific gravity determinations. Individual blocks were classified as Indicated or Inferred based on their location relative to drill hole composites. To take into account the four main economic minerals estimated a Gross Recoverable Value (GRV) was calculated for each block based on reasonable metal prices and estimated recoveries in each of the oxide zone, Cu-Ag zone, Cu-Mo zone and Mo zone.

The resource is summarized below in Table 1 for GRV cutoffs.

**Table 1: Summary of CUMO Resource**

| Cutoff | Indicated | | | | Inferred | | | |
|---|---|---|---|---|---|---|---|---|
| | Millions (tons) | Grade > Cutoff | | | Millions (tons) | Grade > Cutoff | | |
| GRV $US | | $MoS_2$ (%) | Cu (%) | Ag g/t | | $MoS_2$ (%) | Cu (%) | Ag g/t |
| <7.50 | 210 | 0.017 | 0.08 | 2.3 | 840 | 0.013 | 0.07 | 2.2 |
| 7.5-20 | 580 | 0.045 | 0.09 | 2.6 | 840 | 0.042 | 0.08 | 2.3 |
| >20 | 660 | 0.110 | 0.06 | 1.9 | 830 | 0.097 | 0.06 | 2.0 |

The GRV is based on:

$MoS_2$ – Molybdenum is sold as molybdenum trioxide ($MoO_3$) which has higher Mo content. Forecasts are for $MoO_3$ to rise to $16 in 2010 and to $20 in 2011 (CPM group, Feb.2009). The Chinese have stated that they will not be selling their $MoO_3$ for less than $15/lb due to their production costs. The price used for the GRV calculations (these prices were subsequently revised for the economic modelling; refer to Section 17.13 for details) in this study for $MoO_3$ is $15/lb. $MoO_3$ is calculated from $MoS_2$ by the following: Pounds Mo = $MoS_2$ * 20 / 1.6681 and then Pounds $MoO_3$ = Pounds Mo * 1.5

Cu – A copper price of $1.50 / lb was used

Ag – A silver price of $12.00 / oz was used

W – A tungsten price of $8.50 / lb was used

The metal recoveries used in the GRV calculations were a function of metal domains as summarised in Table 2. (Metal recoveries have been revised for plant metallurgical design and economic modelling as detailed in Section 15.1.5)

**Table 2: Metal Recoveries for GRV Calculation**

| | %Recoveries in Oxides | %Recoveries in Cu-Ag Domain | %Recoveries in Cu-Mo Domain | %Recoveries in Mo Domain |
|---|---|---|---|---|
| Cu | 60.0 | 68.0 | 87.0 | 80.0 |
| Ag | 70.0 | 73.0 | 78.0 | 55.0 |
| W | 35.0 | 35.0 | 35.0 | 35.0 |
| Mo | 80.0 | 85.0 | 92.0 | 95.0 |

## 1.4 Preliminary Capital and Operating Cost Estimates

Mosquito is currently undertaking a diamond drill program designed to expand the identified resource and convert its inferred mineral resources to indicated and measured. It has also used the recent (May 2009) resource estimate to conduct a Preliminary Economic Assessment (scoping/sizing study) at various production rates ranging from 50 000 to 200 000 short tons per day to determine the most economic production rate prior to commencing detailed feasibility study work. Note that mineral resources that are not mineral reserves do not have demonstrated economic viability.

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 costs, with a base date of July 2009 for each throughput option, are summarised below in Table 3 and discussed in detail in Section 17. These estimates should be assessed against the study battery limits, exclusions and scope as detailed in the relevant sections of this report.

**Table 3: Summary of Initial Capital Costs**

| Capital Cost | | Design | | | |
|---|---|---|---|---|---|
| | | 50 kt/d (short tons) | 100 kt/d (short tons) | 150 kt/d (short tons) | 200 kt/d (short tons) |
| Plant capital | $USM | 590 | 1 000 | 1 500 | 2 900 |
| Roaster capital | $USM | 120 | 200 | 270 | 350 |
| Mining fleet capital | $USM | 100 | 200 | 270 | 270 |
| Preproduction costs (inc Prestrip) | $USM | 750 | 700 | 640 | 660 |
| Tailings | $USM | 40 | 80 | 80 | 160 |
| **Total Initial Capital** | **$USM** | **1 600** | **2 200** | **2 800** | **3 400** |

The total project operating costs for the different throughput options are summarised in Table 4. The costs are presented as Life of Mine (LOM) averages per short ton of ore processed.

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 in Section 17. These estimates exclude sustaining capital expenditure requirements, but include realisation costs associated with sale of final products. Sustaining capital expenditure has been included as part of the economic assessments.

**Table 4: Summary of LOM Operating Costs**

| Operating Cost | | Design 50 kt/d (short tons) | 100 kt/d (short tons) | 150 kt/d (short tons) | 200 kt/d (short tons) |
|---|---|---|---|---|---|
| Mining | $US/t ore | 4.5 | 2.3 | 1.5 | 1.1 |
| Plant | $US/t ore | 5.0 | 4.7 | 4.6 | 4.6 |
| Roaster | $US/t ore | 0.9 | 0.9 | 0.9 | 0.8 |
| Closure and Reclamation Allowance | $US/t ore | 0.1 | 0.1 | 0.1 | 0.1 |
| General and Administration | $US/t ore | 0.3 | 0.2 | 0.2 | 0.1 |
| **Total Site Operating Costs** | **$US/t ore** | **10.8** | **8.2** | **7.2** | **6.7** |
| Realisation Costs | $US/t ore | 0.4 | 0.4 | 0.4 | 0.4 |
| **Total Unit Operating Costs** | **$US/t ore** | **11.2** | **8.6** | **7.6** | **7.1** |
| **Total Unit Operating Costs (excluding stockpile mining cost)** | **$US/t ore** | **9.6** | **7.8** | **7.2** | **6.8** |

For the purposes of these assessments 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. The average mining costs per ton of mill feed are higher than typically seen for comparable operations as the stockpiled material is not milled during the 40 year life of the study.

If the mining cost per ton of mill feed excluding stockpile costs is considered (Table 5), it can be seen that the unit mining costs are similar to comparable operations (see Table 31). These mining costs are summarised in Table 5 and discussed in detailed in Section 17.12.1.

**Table 5: Summary of Mining Operating Costs**

| | Design 50 kt/d (short tons) | 100 kt/d (short tons) | 150 kt/d (short tons) | 200 kt/d (short tons) |
|---|---|---|---|---|
| Total cost per annum (US$M) | 81 | 84 | 81 | 80 |
| Cost per ton of mill feed (US$) | 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 |

## 1.5 Preliminary Economic Analysis

Overall, the economic performance of the project (as measured by the IRR, NPV and payback period etc.) improves as the design throughput increases. These data are summarised below in Table 6 and discussed in detail, together with the metal prices and assumptions used in the calculations in Section 17.13. All values are calculated based on Earnings Before Interest Tax Depreciation and Amortisation (EBITD&A). 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 6: Base Case Economic Analysis**

| **Economic parameters (EBITD&A)** | **Throughput Option** | | | |
|---|---|---|---|---|
| | **50 kt/d (short tons)** | **100 kt/d (short tons)** | **150 kt/d (short tons)** | **200 kt/d (short tons)** |
| NPV (US$Billion @ 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** |

The economic metrics continue to improve as the design throughput increases, showing that even higher throughputs would give higher NPV and higher IRR. However at 100 to 150 kt/d, CUMO would be very large for a green-fields base metals project, with a matching high capital cost. A project of larger scale would likely encounter difficulties in obtaining financing and a more reasonable design throughput for future studies is in the 100 to 150 kt/d range.

## 1.6 Recommendations

Based on the resources defined to date and this preliminary economic analysis, it is recommended that the CUMO project be advanced to feasibility stage. The recommended program is proposed to be carried out over a minimum time frame of two years at an estimated cost of **$72.5M (US$).**

## 2 INTRODUCTION AND TERMS OF REFERENCE

Ausenco Minerals Canada Inc. (Ausenco) and Vector Engineering Inc. (Vector) were contracted by Mosquito to assist in production of a Preliminary Economic Assessment (PEA) on the CUMO Property in Boise County, Idaho, based on the previously filed NI 43-101 Report (Holmgren and Giroux, 2009).

The scope of work includes an open cut copper-molybdenum (Cu-Mo) mine, conventional processing plant, molybdenum roaster, associated services and utilities, supporting infrastructure, tailings storage facilities (TSF) and waste stockpiles.

This report considers four options for plant throughput rates from 50 000 short tons per day (kt/d) to 200 kt/d and has developed preliminary 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.

The material found in this technical report is an amalgamation of previous reports, program updates, consultant reports, and corporate releases that were available for review. There were no limitations put on the authors in preparation of this report with respect to the property vendor or Mosquito's information. Reports and data were obtained from all parties. The authors have relied heavily on information presented by Mosquito, and in particular the report titled "Summary Report on the CUMO Property, Boise County, Idaho, USA, Technical Report" dated May 13, 2009.

The geology of this immediate area of Idaho is poorly documented in the professional literature and there are very few pertinent papers available for review.

Co-author Jackie Holmgren visited the site between November 29 and December 2, 2008 and on August 22, 2008. 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. Co-authors Gary Giroux, Robert Braun, Charlie Khoury and Richard Kehmeier have not visited the site.

The following table, Table 7, identifies several important terms and abbreviations used in this report.

**Table 7: Abbreviations and symbols**

| Unit | Abbreviation or Symbol |
|---|---|
| average | ave |
| day | d |
| degree Celsius | °C |
| degree Fahrenheit | °F |
| diameter | dia |
| foot | ft |
| gram | g |
| grams per litre | g/L |
| hectare | ha |
| hour | h |
| inside diameter | ID |
| kilogram | kg |
| kiloPascal | kPa |
| kilowatthour | kWh |
| life of mine | LOM |
| litre | L |
| maximum | max |
| metre | m |
| metre per second | m/s |
| metre per second squared | $m/s^2$ |
| metric tons | t (metric tons) |
| metric tons per hour | t/h (metric tons) |
| micron | $\mu$m |
| miles per hour | mph |
| minimum | min |
| minute | min |
| mole percent | mol % |
| molecular mass (weight) | mol wt |
| parts per billion | ppb |
| parts per million | ppm |
| pounds | lb |
| run of mine | ROM |
| second | s |
| specific gravity | SG |
| square metre | $m^2$ |
| short tons per hour[1] | t/h |
| short tons | t |
| troy ounces | oz |
| volume by volume | v/v |
| weight (mass) | wt |
| weight (mass) percent | wt % |
| weight by mass | w/w |
| weight by volume | w/v |
| year | y |

[1] unless noted otherwise all tons are short tons.

## 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 by the following:

- The Qualified Persons responsible for the property description, accessibility and site history, together with geological estimates (exploration, drilling, sampling and data verification) and resource estimation were Jackie Holmgren of Roche Zaune Exploration and Gary Giroux of Giroux Consultants Ltd, specifically sections 1.1, 1.2, 1.3, 1.6, 2, 3. 4, 5, 6, 7, 8, 9, 11, 12, 13, 14, 15.1.2, 16, 17.5, 19.1, 19.6 and 19.7 of this report.
- The Qualified Person responsible for open cut mine capital and operating cost estimating was Richard Kehmeier of Vector, specifically sections 10, 17.1, 17.8 17.12.1 and 19.3 of this report.
- Metallurgical testing conducted by SGS Canada Inc.
- The Qualified Person responsible for the metallurgical plant design, capital and operating cost estimates and economic analysis was Robert Braun of Ausenco, specifically sections 1.4, 1.5, 15 (except section 15.1.2), 17.6, 17.7, 17.9, 17.11, 17.12.2, 17.12.3, 18, 19.2, 19.5 and 19.8 of this report.
- The Qualified Person responsible for the surface waste disposal and Tailings Storage Facilities was Charlie Khoury of Vector, specifically sections 17.2, 17.3, 17.4, 17.10 and 19.4 of this report.

The authors believe that the information provided and relied upon for preparation of this report is accurate at the time of the report and that the interpretations and opinions expressed in them 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 on 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

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 CUMO property is located approximately 59 kilometres (37 miles) northeast of the city of Boise, Idaho, USA (Figure 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 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 597500E, 4,876,000N (NAD 27 CONUS).

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, 600ft by 1500ft claims (20.66 acres each). However, the total includes twenty-seven fractional claims where the new claims were staked over existing claims. The claims are shown in Figure 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 block, 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 $125/claim must be paid to the BLM prior to Aug 31, 2009 or all claims will expire on Sept 1, 2009. At $125/claim, the company must make annual payments to the BLM of US$43,000 to keep all claims in good standing.

### 4.1 Ownership Agreements

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 become 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.

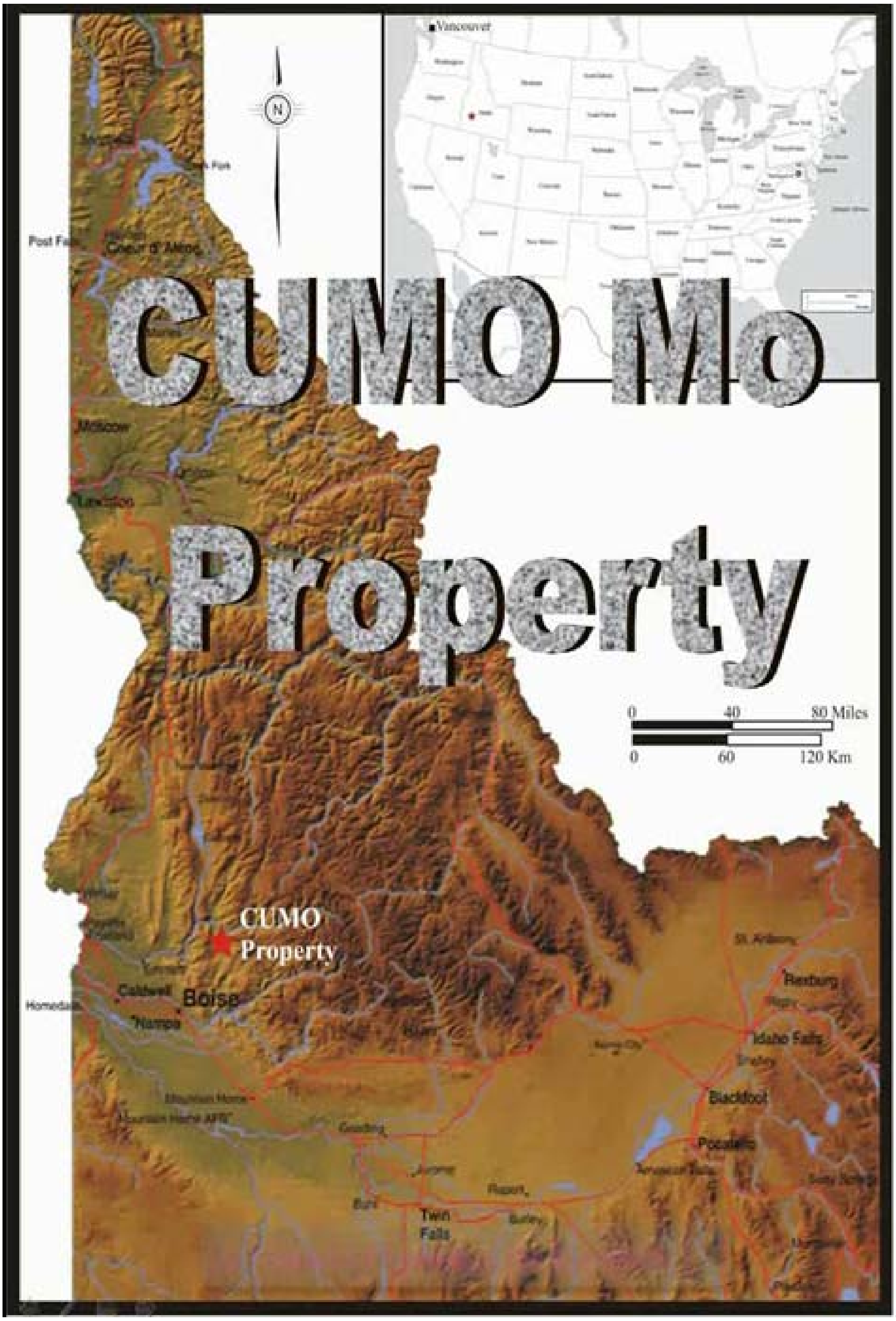



Figure 1: CUMO Property Location Map.

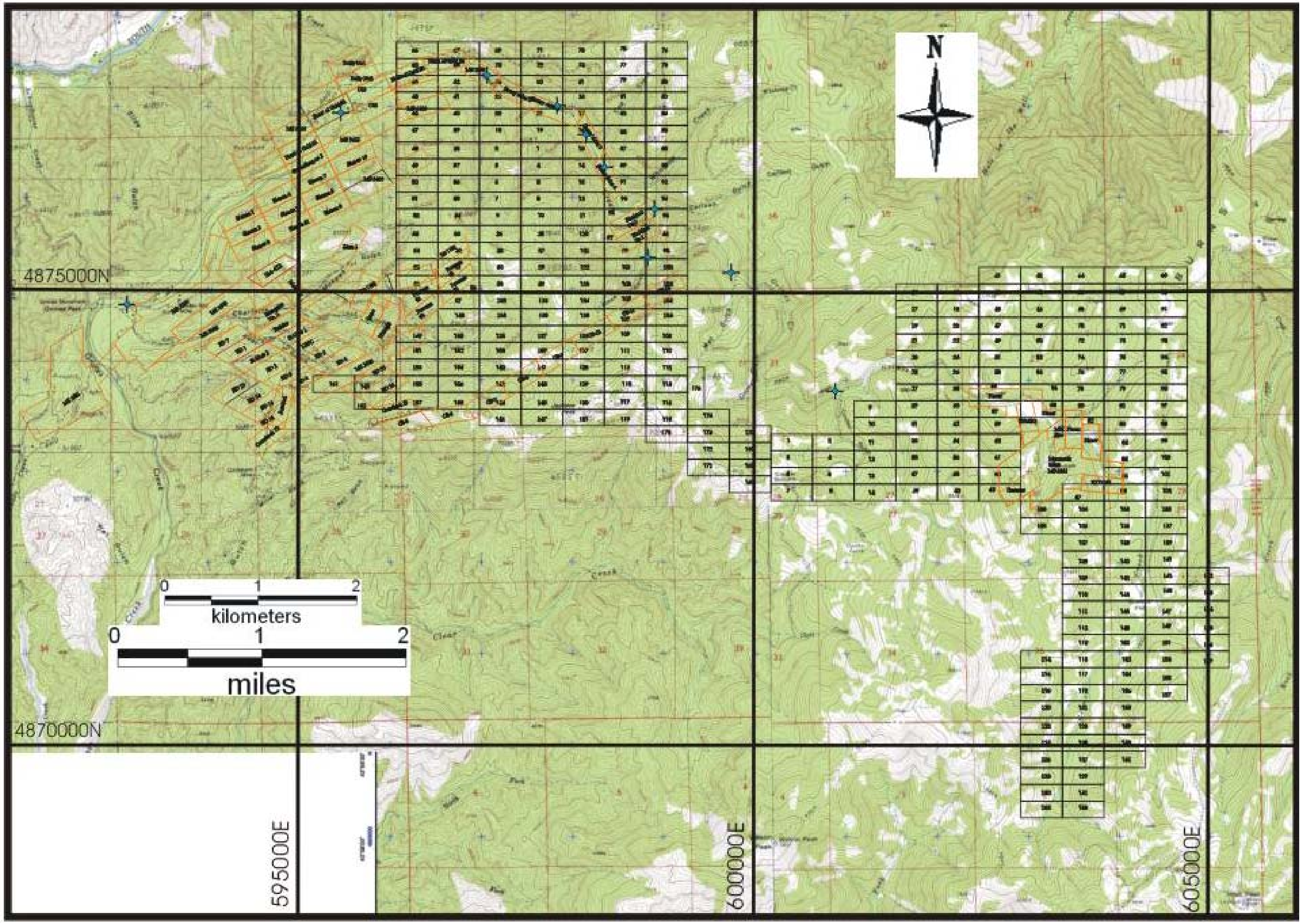



**Figure 2: Claim location map for the CUMO property.**
**Note: Claims indicated by colored outline are not currently part of the property.**

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.

## 4.2 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. Track mounted auger drilling and no new road clearing would fit in this category according to USFS personnel.
- Environmental assessment (EA) requires an in depth study with 30 days for public comment, plus additional time for appeal. Drilling with an RC rig using water, new road construction, etc., would require this level of permit. USFS personnel suggest that one year may be required to receive a permit. 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 US Forest Service, 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.

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.

## 5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

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.

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 5,100 feet (1700 meters) to 7,200 feet (2,400 meters).

The climate is defined by summer temperatures to a maximum of 100° F and cold, windy winters with lows to -10° F. Precipitation is moderately light with an average rainfall of 30 inches (<1 metre) and an average snowfall of approximately 140 inches (3.6 meters). 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 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 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 30,822 feet of drilling (Table 8), 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 3 different grids.

**Table 8: Summary of Historic Drilling**

| Year | Company | Holes | Footage | Meters | Comments |
|---|---|---|---|---|---|
| 1969 | Midwest | 4 | 378 | 115.2 | rotary holes shallow due to water |
| 1970 | Midwest | 0 | 653 | 199.0 | 2 rotary holes deepened with core to 400' depth |
| 1971 | Midwest | 1 | 2251 | 686.1 | one core hole deepened further to 1884 ft |
| 1972 | Midwest | 3 | 1892 | 576.7 | one core hole deepened from 810-1416 ft |
| 1974 | Amax | 1 | 805 | 245.4 | hole 9-9A |
| 1975 | Amax | 1 | 2382 | 726.0 | hole 10 |
| 1976 | Amax | 2 | 4343 | 1323.7 | one vertical, other 1340ft @-45 |
| 1977 | Amax | 3 | 5861 | 1786.4 | 3 vertical DDH 1804-2124 feet deep |
| 1978 | Amax | 3 | 6774 | 2064.7 | 3 vertical DDH 2132-2361 feet deep |
| 1979 | Amax | 2 | 4823 | 1470.0 | vertical DDH to 2543 foot depth |
| 1980 | Amax | 2 | 2630 | 801.6 | RC holes |
| 1981 | Amax | 3 | 3204 | 976.6 | vertical DDH 1,000 to 1,193 foot depths |
| Total | | 26 | 35,996 | 10,971 | |

Based on the 26 drill holes a resource block model was constructed in 1983, extending between local grid coordinates 17,000 to 25,000 east and 16000 to 23000 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 drill hole assay composites and grade zone boundaries. Kriging was performed within a 1500 foot horizontal search limited to 300 feet vertically (Table 9).

**Table 9: CUMO Historical Resource, 1982 AMAX Block Model**

| Cutoff Grade (% MoS2) | Million Tons | Average Grade (%MoS2) |
|---|---|---|
| 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). Nevertheless, Climax is considered to be a reliable source and therefore the estimate is considered relevant as to the tonnage and grade potential.

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.

# 7 GEOLOGICAL SETTING

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.

A description of the "Geological Setting" 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 following is additional information that may duplicate, in part, previous Technical Reports.

## 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 and others, 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 3) 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 4), 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 and others, 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.

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.

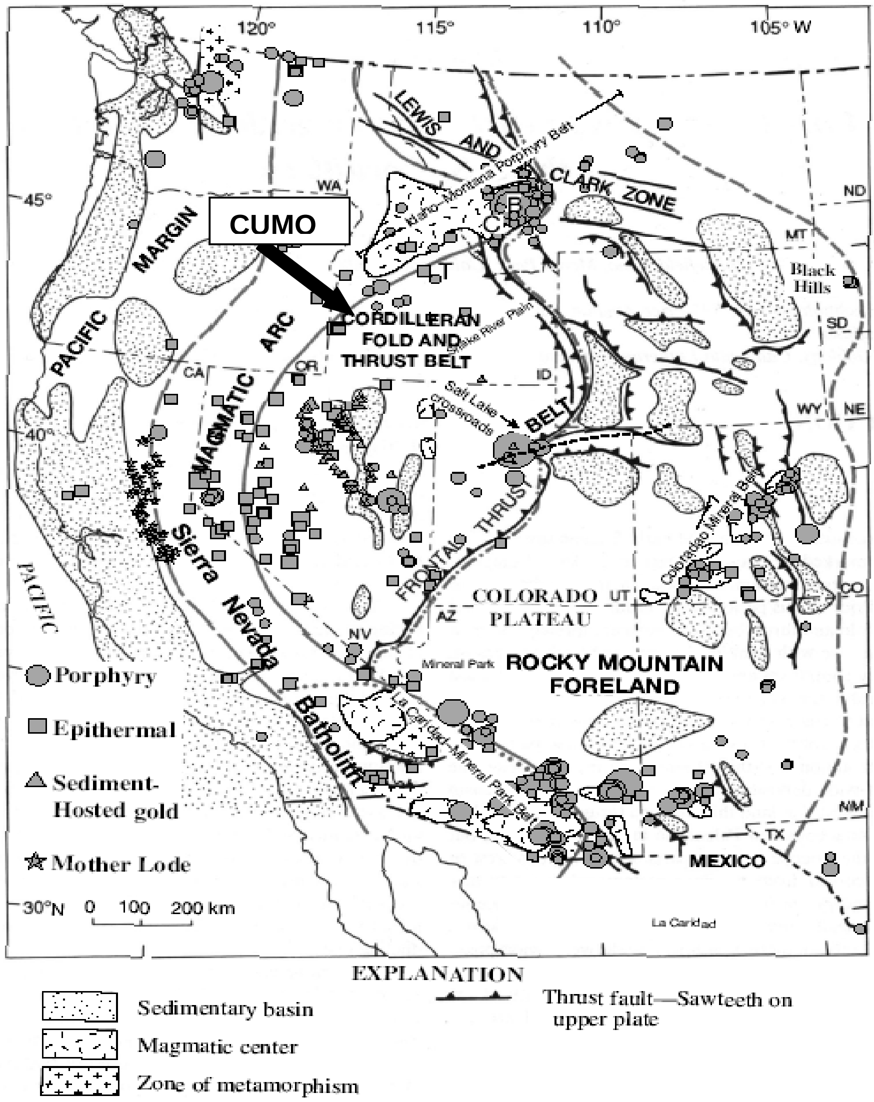



FIG. 1. Map of the western United States cordillera showing ore deposits superimposed on major tectonic elements and Laramide igneous zones, sedimentary basins, and metamorphic belts. The western United States is divided into four generalized geologic provinces (boundaries shown as heavy solid and dashed 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 3: Tectonic map of the western United States (Hildenbrand and others, 2000)**

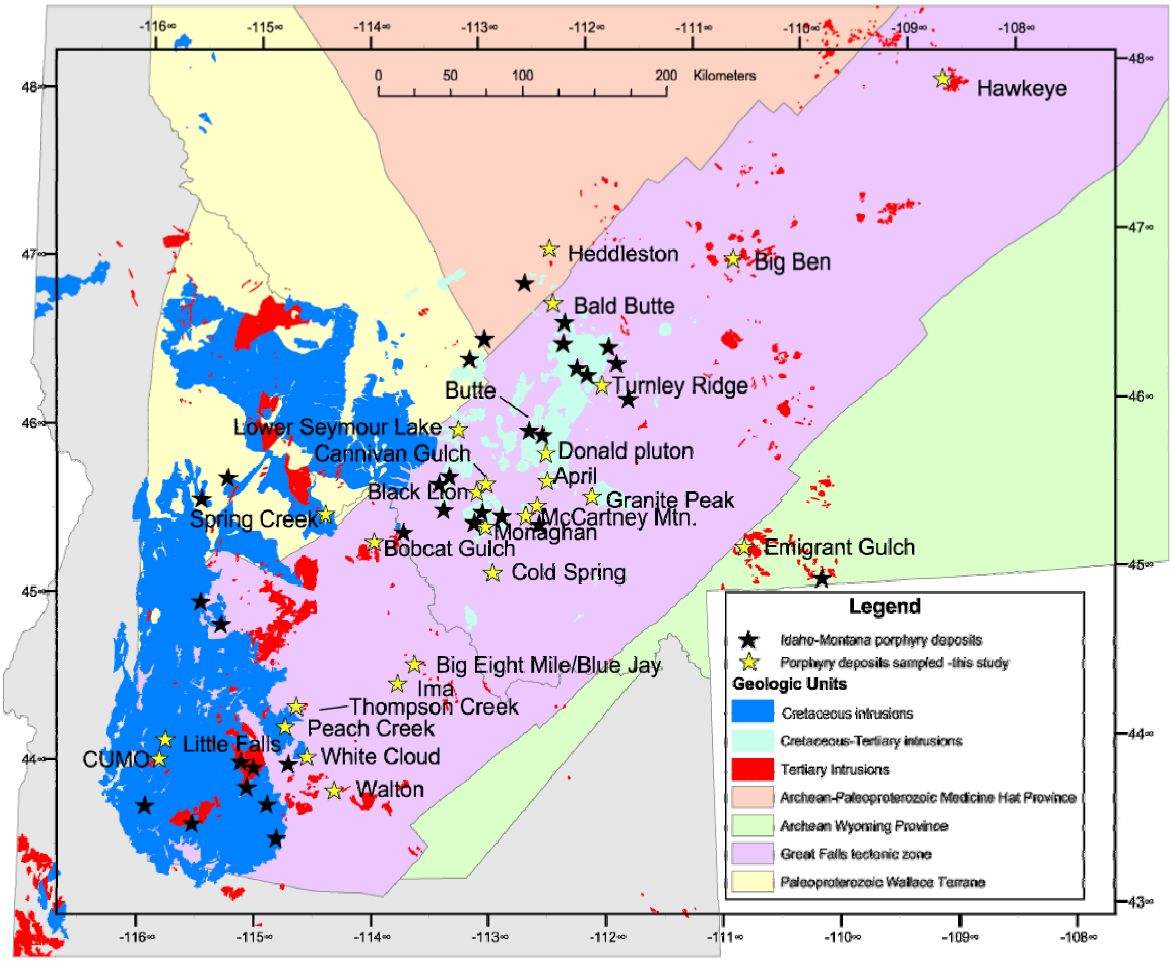



**Figure 4: Distribution of Idaho-Montana Porphyry deposits in relation to the Great Falls Tectonic Zone.**
**(From Lund and others, 2005).**

Mineral deposits in the Idaho-Montana Porphyry Belt (also called the Transverse Porphyry Belt of Idaho-Montana by Carten and others, 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 and others, 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 ond others, 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).

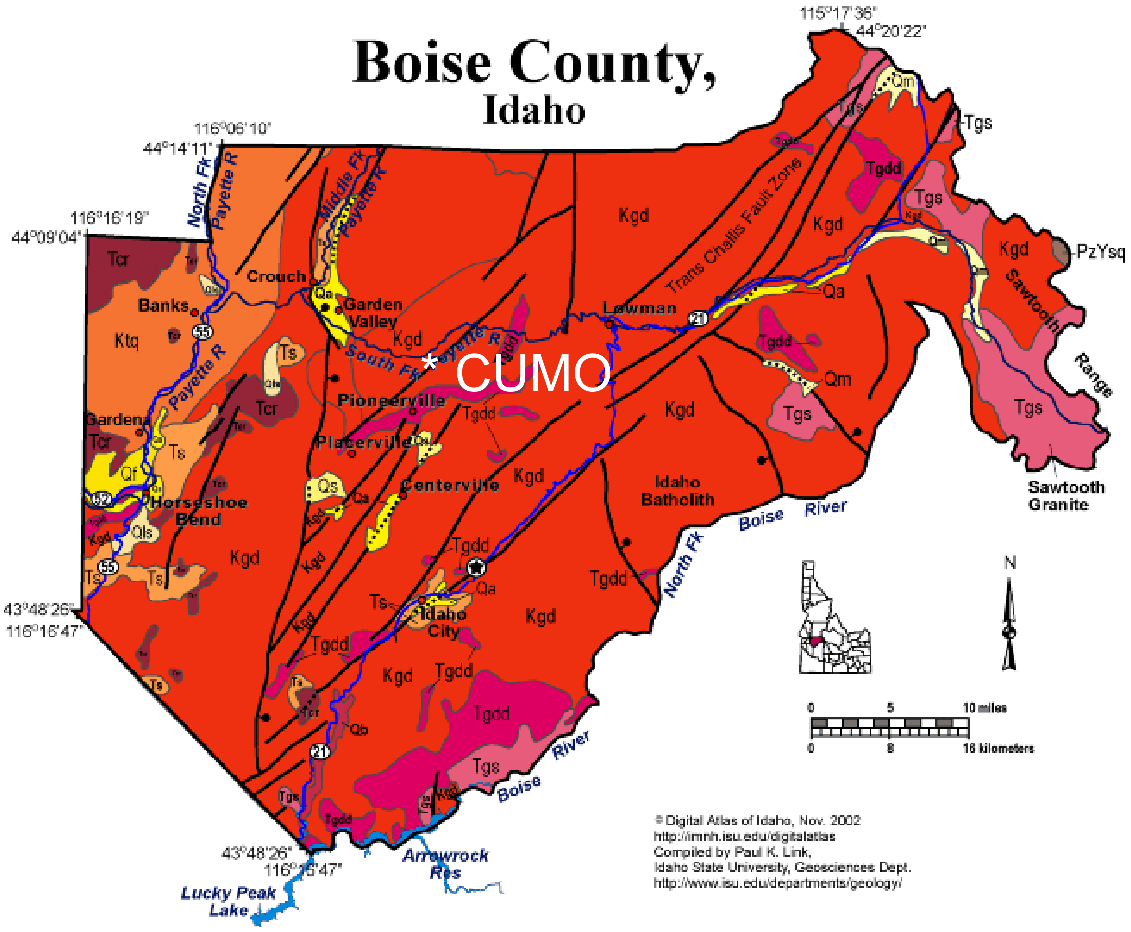
The CUMO deposit is situated within the Atlanta Lobe of the Idaho batholith. The western margin of the Atlanta lobe is strongly folded and metamorphosed into gneissic rocks, which are well exposed near McCall. The western side is composed of tonalite, 95 to 85 million years old. The batholith core is 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 and others, 2006), and adjoining Idaho City 30 x 60 quadrangle map (Kilsgaard and others, 2001), and compiled into the Boise County map of the digital Atlas of Idaho (Figure 5).

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 and others, 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.

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 and others, 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 and others, 1985).




**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. |

**Figure 5: Geology of Boise County, Idaho, showing geologic setting of CUMO deposit.**
(Modified from: http://imnh.isu.edu/digitalatlas/counties/boise/geomap.htm)

Hypabyssal equivalents of the granites include numerous rhyolite dikes that are concentrated along the trans-Challis fault system (Killsgaard and others, 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 and others, 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:

**Table 10: Summary of Rock units at CUMO**

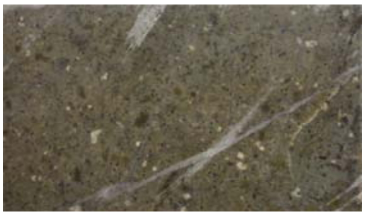
| Unit | Age | Rock Type | Texture | Grain Size (groundmass) |
|---|---|---|---|---|
| Tl | Tertiary | lamprophyre | porphyritic | fine |
| Td | Tertiary | diabse | massive, amydaloidal | aphanitic |
| Tr | Tertiary | rhyolite | massive to flow-banded | aphanitic to fine |
| TpE | Tertiary | biotite 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 dike types (TpA-TpF) overlap, but show a general trend from early, phenocryst-rich porphyries with large phenocrysts, to young, phenocryst-poor porphyries with small phenocrysts".

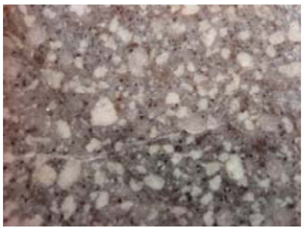
In 2006, three main intrusive types were observed in the holes drilled, including equigranular quartz monzonite, quartz monzonite porphyry, and intrusive breccia. Mafic dikes were also intersected locally. The equigranular quartz monzonite is considered to be the Idaho batholith (unit Kqm) and locally 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 Figure 6.

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.



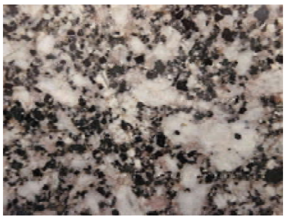
a) Porphyry unit Tbqmp1 (Amax TpF) C40-08: 158ft



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)

**Figure 6: Core photographs of felsic porphyry types recognized in the 2008 program.**

# 8 DEPOSIT TYPES

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 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 and others, 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 million tons at 0.085% MoS2 and very few deposits exceed 500 million tons) 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 tons.

The CUMO deposit is primarily of economic interest for its Mo content but contains significant values of Cu and Ag. According to Carten and others (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 and others, 1993). For the Granite-related Mo-Cu (>0.05%Mo) class of deposits the CUMO deposit ranks highest in terms of total potential contained molybdenum (tonnes x grade), based on the historical resource. Compared to all porphyry copper-molybdenum deposits (model type 21a) listed in the USGS world database (Singer and others (2005)), the CUMO deposit ranks fourth in terms of total potential contained molybdenum, based on the historical Amax resource (Table 11).

### Table 11: Ranking of Open Pit Resources Under Exploration or Development (2008).

| deposit name | Meas+Ind tons(millions | inferred tons(millions) | total tons(millions) | Cu % | Mo % | Au gms/T | Ag gms/T | Re gms/T | Cu Eq. % | Gross Value $/ton | Lbs mos2 (millions) | lbs mo (millions) | total $ (millions) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Cumo_Total | 1,374.4 | 2,246.0 | 3,957.0 | 0.07 | 0.038 | | 2.22 | | 0.67 | $19.98 | 4,561.7 | 2,734.7 | $79,048.99 |
| Cumo_$7.50 | 1,234.8 | 1,667.9 | 2,902.7 | 0.07 | 0.044 | | 2.19 | | 0.76 | $22.83 | 4,296.0 | 2,575.4 | $66,266.70 |
| Cumo $10 | 1,150.0 | 1,401.8 | 2,551.8 | 0.07 | 0.049 | | 2.12 | | 0.82 | $24.69 | 4,133.9 | 2,478.2 | $63,012.13 |
| Jinduicheng | 910 | | 910 | 0.03 | 0.102 | 0.00 | 0.00 | | 1.56 | $46.80 | 3,096.66 | 1,856.40 | $42,588.00 |
| mt toleman | 1,565 | 340 | 1905.0 | 0.09 | 0.047 | 0 | 0 | | 0.80 | $23.85 | 2,987.1 | 1,790.7 | $45,434.25 |
| Cumo_Amax Historic | | 1,500 | 1,500 | 0.07 | 0.056 | | 0.06 | | 0.91 | $27.44 | 2,802.4 | 1,680.0 | $41,161.50 |
| Mt Hope | 966 | 191 | 1,157 | | 0.068 | | | | 1.02 | $30.60 | 2,624.8 | 1,573.5 | $35,404.20 |
| Pebble West | 3,026 | 1,130 | 4,156 | 0.26 | 0.015 | 0.31 | 0.00 | 0.000 | 0.67 | $20.13 | 2,079.8 | 1,246.8 | $83,666.33 |
| Sierrita | 1,830 | | 1,830 | 0.26 | 0.030 | 0.03 | 1.20 | 0.057 | 0.74 | $22.26 | 1,831.6 | 1,098.0 | $40,736.58 |
| Toqupala | 1,161 | | 1161.0 | 0.668 | 0.040 | | | | 1.27 | $38.04 | 1,549.3 | 928.8 | $44,164.63 |
| Chuquicamata (remainin | 700 | | 700 | 1.53 | 0.065 | 0.01 | 5.00 | | 2.57 | $77.13 | 1,518.0 | 910.0 | $53,993.84 |
| Spinifex ridge | 1048.8 | 0 | 1048.8 | 0.08 | 0.043 | | 2.16 | | 0.75 | $22.51 | 1,504.6 | 902.0 | $23,604.29 |
| shaft creek | 1,542 | | 1,542 | 0.28 | 0.021 | 0.18 | 1.54 | | 0.71 | $21.41 | 1,072.8 | 643.1 | $33,015.06 |
| Climax (remaining) | 150 | 25 | 175 | | 0.167 | | | | 2.51 | $75.15 | 975.0 | 584.5 | $13,151.25 |
| Cajone | 1,261 | | 1261.3 | 0.61 | 0.020 | | | | 0.91 | $27.30 | 841.6 | 504.5 | $34,434.52 |
| Thompson creek | 372 | | 372 | | 0.063 | | | | 0.95 | $28.35 | 781.0 | 468.2 | $10,534.86 |
| Mineral Park | 520 | | 520 | 0.13 | 0.039 | | 2.74 | | 0.75 | $22.41 | 677.0 | 405.9 | $11,660.47 |
| Bingham -left | 557 | | 557 | 0.54 | 0.033 | 0.27 | 2.52 | | 1.23 | $36.79 | 613.2 | 367.6 | $20,493.85 |
| endako | 368 | | 368 | | 0.050 | | | | 0.75 | $22.50 | 613.0 | 367.5 | $8,268.75 |
| Bagdad | 1,600 | | 1,600 | 0.40 | 0.010 | 0.00 | 0.97 | 0.000 | 0.56 | $16.86 | 533.8 | 320.0 | $26,974.89 |
| sonora | 94 | 93 | 187 | 0.05 | 0.081 | | | | 1.27 | $37.95 | 504.3 | 302.3 | $7,082.78 |
| atlin | 213 | | 213 | | 0.063 | | | | 0.95 | $28.35 | 447.7 | 268.4 | $6,038.55 |
| Quellaveco | 947 | | 947.0 | 0.94 | 0.014 | | | | 1.15 | $34.50 | 442.3 | 265.2 | $32,671.50 |
| magistral | 196 | 55 | 251 | 0.52 | 0.041 | | | | 1.14 | $34.05 | 343.2 | 205.7 | $8,543.15 |
| Gibralter | 965 | | 965 | 0.32 | 0.010 | 0.07 | 0.90 | 0.000 | 0.52 | $15.68 | 321.9 | 193.0 | $15,126.69 |
| Island copper | 377 | | 377 | 0.41 | 0.017 | 0.19 | 1.40 | 0.032 | 0.80 | $23.86 | 213.8 | 128.2 | $8,995.62 |
| Max | 43 | | 43 | | 0.120 | | | | 1.80 | $54.00 | 171.7 | 103.0 | $2,316.60 |
| lucky ship | 45 | 17 | 62 | | 0.068 | | | | 1.02 | $30.60 | 139.5 | 83.6 | $1,881.90 |
| Poplar | 116 | | 116 | 0.32 | 0.009 | 0.10 | | | 0.52 | $15.45 | 34.8 | 20.9 | $1,792.25 |

The following 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). 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 sulfide content.

**PORPHYRY Cu+/-Mo+/-Au**
L04
by Andre Panteleyev
British Columbia Geological Survey

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.

## 8.1 Identification

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

**COMMODITIES (BY-PRODUCTS):** 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).

## 8.2 Geological Characteristics

**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 sulfide minerals are present, generally in subordinate amounts. The mineralization is spatially, temporally and genetically associated with hydrothermal alteration of the hostrock intrusions and wallrocks.

**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.

**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.

**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.

**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).

**DEPOSIT FORM:** Large zones of hydrothermally altered rock contain quartz veins and stockworks, sulfide-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):

Volcanic type deposits (e.g. Island Copper) 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.

Classic deposits (e.g., Berg) 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 sulfide 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, sulfide 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.

**TEXTURE / STRUCTURE:** Quartz, quartz-sulfide and sulfide veinlets and stockworks; sulfide grains in fractures and fracture selvages. Minor disseminated sulfides commonly replacing primary mafic minerals. Quartz phenocrysts can be partially resorbed and overgrown by silica.

**ORE MINERALOGY** (Principal and subordinate): Pyrite is the predominant sulfide 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.

**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.

**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) .

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

**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.

**ASSOCIATED DEPOSIT TYPES:** Skarn Cu (K01), porphyry Au (K02), epithermal Au-Ag in low sulfidation type (H05) or epithermal Cu-Au-Ag as high-sulfidation 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 sulfide mantos and replacements in carbonate and non- carbonate rocks (M01, M04), placer Au (C01, C02).

**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.

## 8.3 Exploration Guides

**GEOCHEMICAL SIGNATURE:** 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 sulfur, mainly in the form of metal sulfides, chiefly pyrite.

**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 sulfide-poor systems the ore itself provides the only significant IP response.

**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).

### 8.4 Economic Factors

**TYPICAL GRADE AND TONNAGE:** 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 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.

A description of the "Geological Setting" 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 following is additional information that may duplicate, in part, previous Technical Reports.

### 9.1 District Mineralization

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 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 sulfides that occur within the Idaho batholith and are associated with weak wall rock alteration. Associated sulfides 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 sulfides, 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 kilometres/6 to 10 miles), 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 meter) wide sulfide 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 and other, 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 and others, 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 and others, 1989).

## 9.2 Property Mineralization

Mineralization on 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 centimeters in width, with an overall thickness averaging 0.3-0.4 cm. Pyrite and/or chalcopyrite are commonly associated with molybdenite although molybdenite can occur alone without other metallic mineralization. Chalcopyrite occurs in quartz-pyrite + molybdenite veinlets, in magnetite + pyrite as well as in pyrite-biotite +quartz +magnetite veins with secondary biotite halos. Scheelite is common on the property and closely parallels the distribution of molybdenite (Baker, 1983). Figure 7 and Figure 8 show examples of mineralization at CUMO from the recent drill holes.

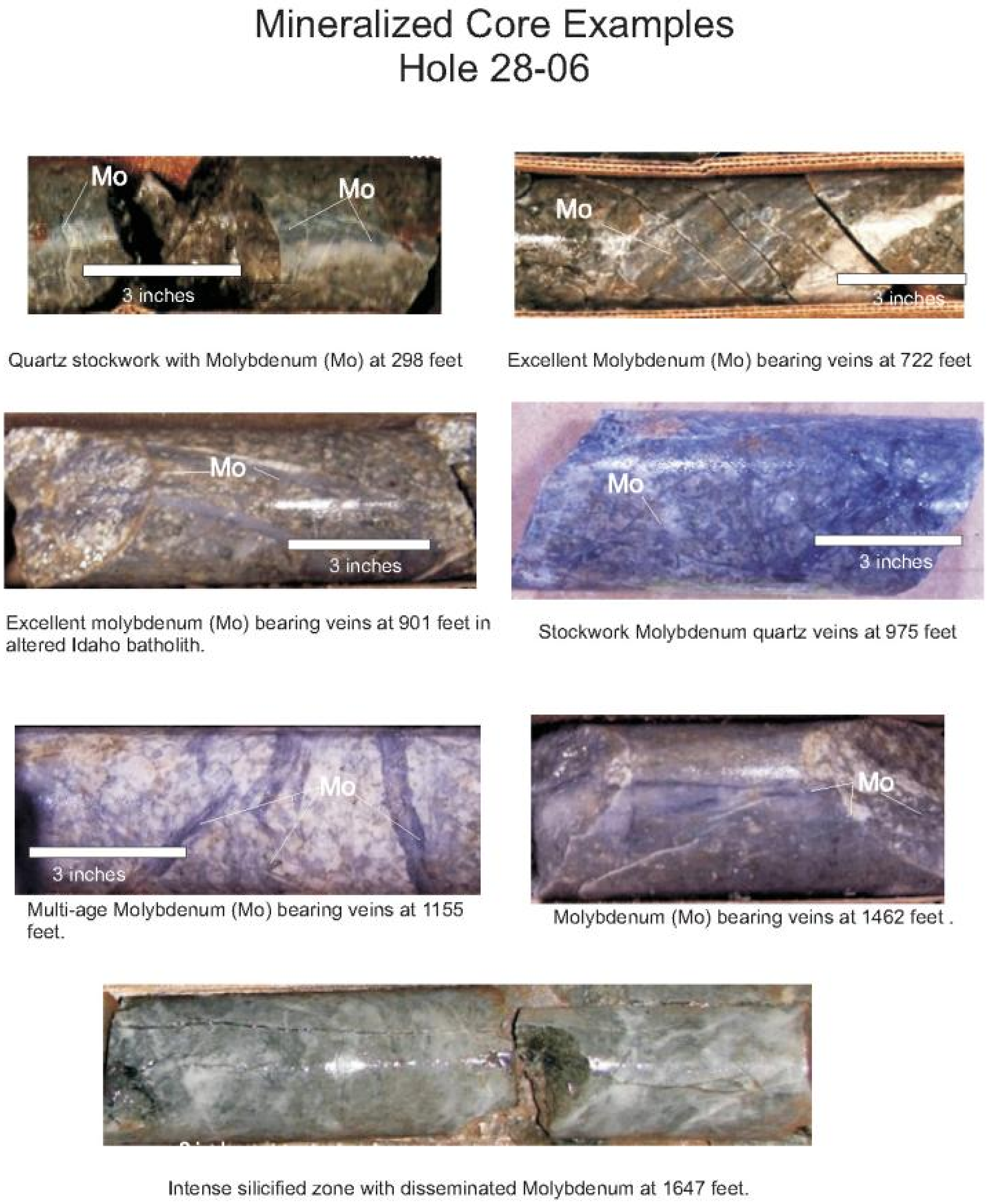
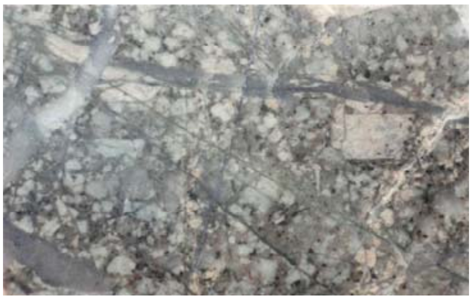



**Figure 7: Photographs of mineralized core from the CUMO 2006 program, hole C06-28.**



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)

**Figure 8: Photographs of molybdenite mineralization in 2008 drill core.**

Compilation of Amax data on the frequency of veins mapped on surface as well as their mineral constituents was presented by Giroux and others (2005) and is shown in Figure 9. A concentric pattern is clearly evident, which is also shown by the distribution of anomalous Mo and Cu rock geochemical results (Figure's 10a and 10b). 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.

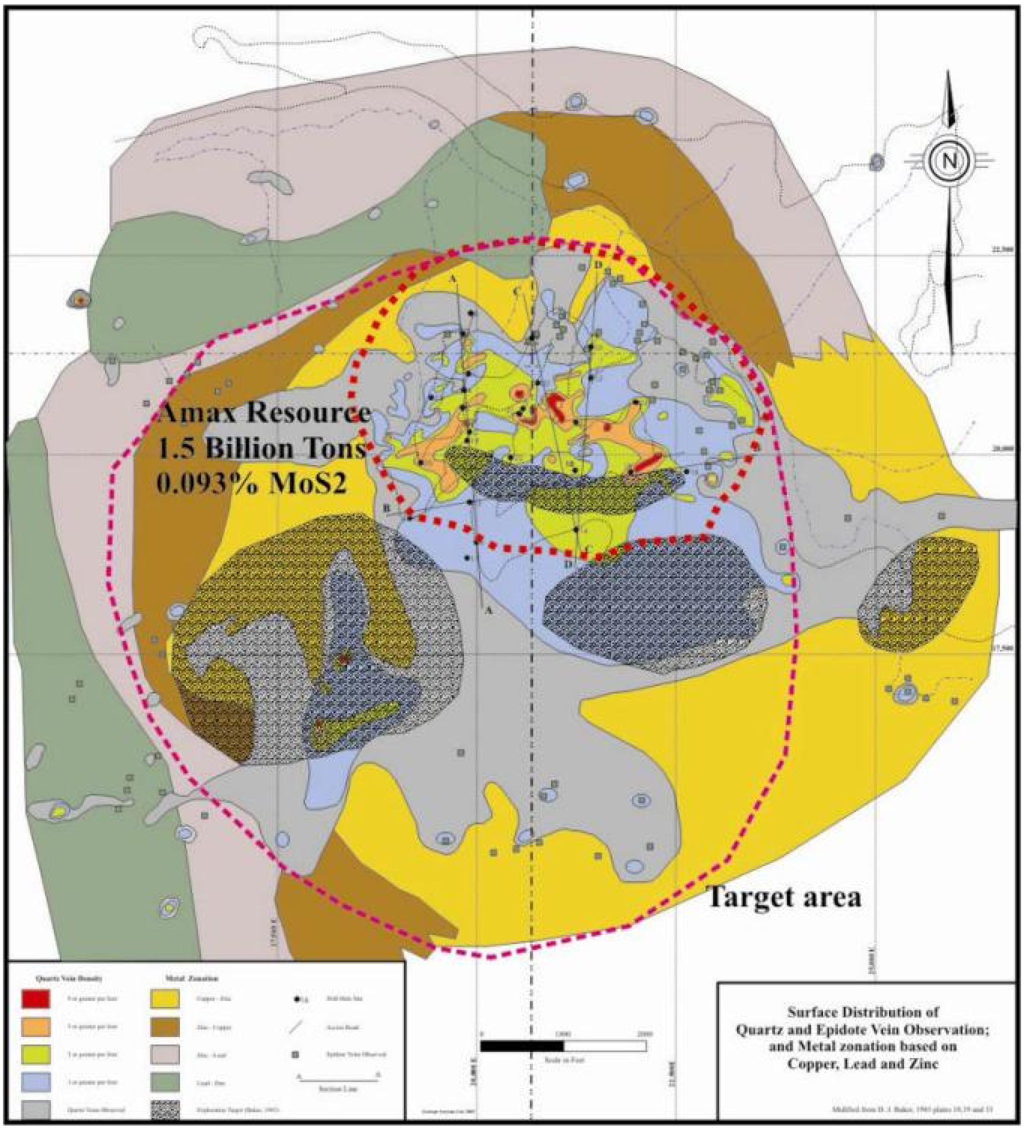
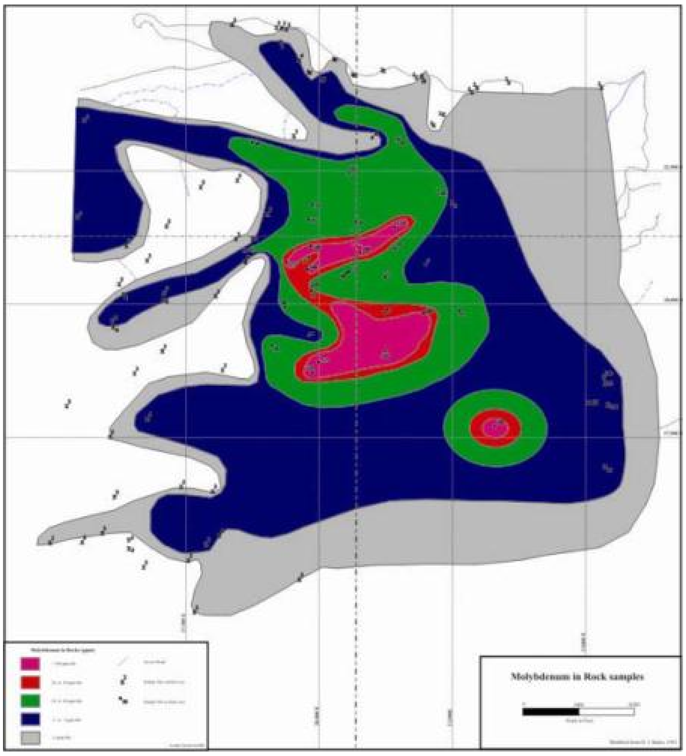
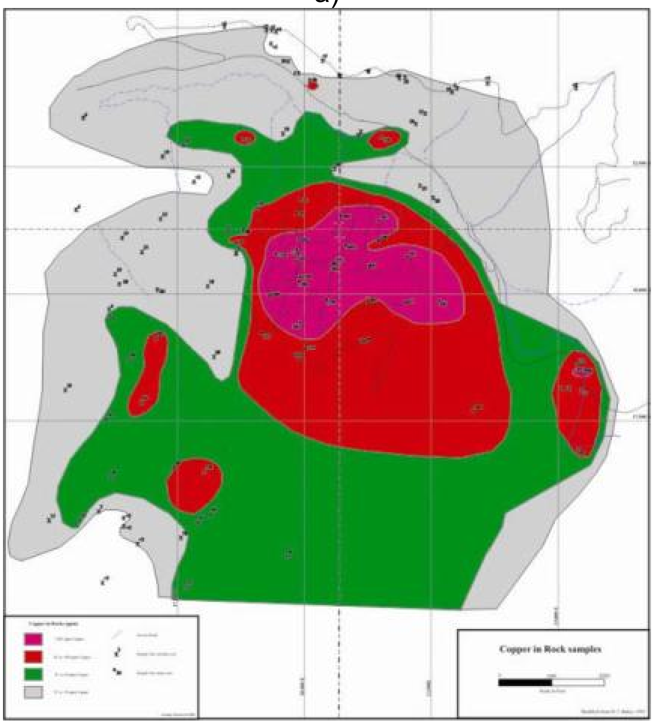



**Figure 9: Surface Distribution of Quartz and Epidote Veinlets and Metal Zonation**



a)



b)

**Figure 10: Geochemical distribution of Mo (a) and Cu (b) in Surface Rock Chip Samples**

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.) and indicates the current area being drilled 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 has 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

Diamond drilling is currently ongoing on the CUMO property and the results will be reported in a resource update when the current program concludes and all data is compiled. The current PEA study is based on those resources reported by Holmgren and Giroux, 2009.

# 11 DRILLING

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.

## 11.1 Analysis of Historic Drill Data

Variography was completed on historic Amax data prior to drilling by Giroux and others (2005). The study was based on 23 diamond drill holes and 3 reverse circulation drill holes with 139 down hole surveys and 2,356 assays for MoS2 and Cu.

It was found that the vertical direction produced good semivariograms for both $MoS_2$ and Cu. Nested spherical models were fit to the downhole (Az 0 Dip -90) direction and showed good structures for both $MoS_2$ and Cu with longest range of 400 and 350 ft respectively. There was insufficient data to determine representative semivariograms in the horizontal plane (Giroux and others, 2005). The results suggest that closer drill hole spacing is required to achieve representative semivariograms in the horizontal plane, in order to determine the drill hole spacing required for resource estimation.

The current average drill spacing is approximately 700 feet (213m). Although the horizontal range may be anticipated to be greater than that in the vertical direction, the longest vertical range can be used as an initial target for maximum hole spacing. The range of 350 feet reported for Cu is therefore suggested as a target for maximum hole spacing at the initial stage.

## 11.2 Year 2006, 2007 and 2008 Drilling Programs

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 Company delivered a notice of termination in respect of the CUMO Property. The option on the project was terminated when the second hole was at a depth of 600 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 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), +2000 foot, diamond drill holes.

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

**Table 12: 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 holes were surveyed down the hole at regular intervals using a Reflex survey instrument

Figure 11 shows the locations of all holes drilled to date in the deposit

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

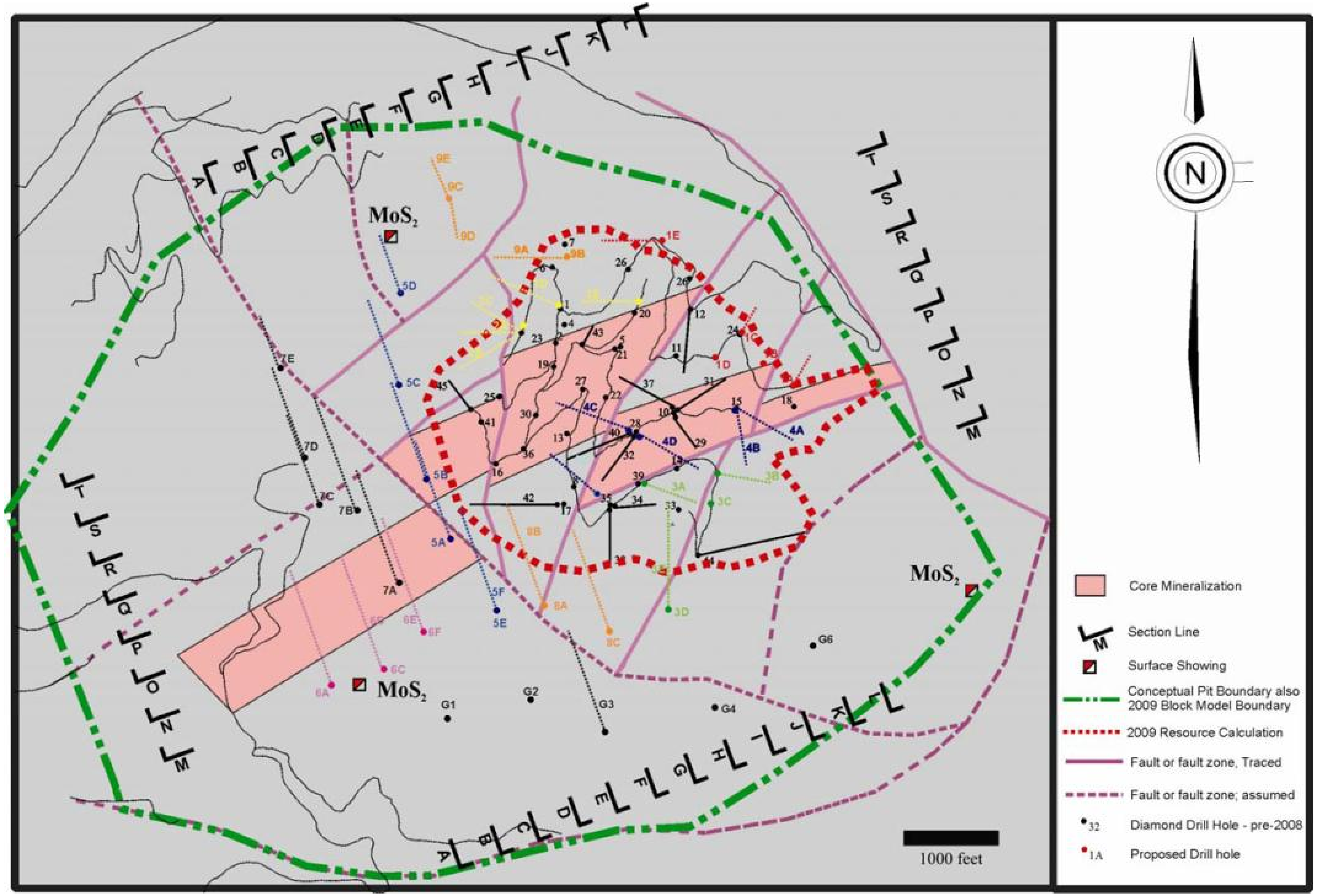



**Figure 11: Map Showing the Location of Completed and Proposed Drill Holes**

A summary of significant intersections for all the CUMO drilling are given in Table 14. 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, it was decided to calculate both a copper and molybdenum equivalent for the intercepts. Both equivalents are required as the deposit is zoned as described above. The following outlines the calculations were involved:

Copper equivalent (Cu. Equiv.) and Molybdenite Equiv. (MoS2 Equiv.) are based on the following metal prices (all in US$): Copper $1.50/lb, Molybdenum Oxide ($15/lb), Silver $0.35/gram and Tungsten $0.22/gram.($7.00 per lb)

Other factors include 1% = 20 pounds/short ton; 1 ppm = 1 gram/metric ton;
1000 ppb =1 ppm = 1 gram/metric ton.

Molybdenum is sold as either ferro-molybdenite or molybdenum oxide.

The price used is $15 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.

Metallurgical recoveries used in calculation are as follows for each metal zone. Recoveries are slightly lower that those currently reported by SGS in their recent metallurgical study. These recoveries and metal values have only been used for the calculation of recovered metal values for the drilling intercepts and in the block model. Additional analyses of recoveries and modified metal prices have been used in the economic analyses as discussed in Section 17.13 of this report.

**Table 13: Metallurgical Recoveries For Recovered Metal Value Calculation**

| Zone | Cu% | MoS2% | Ag % | W % |
|---|---|---|---|---|
| Oxide | 60% | 80% | 70% | 35% |
| CuAg | 68% | 85% | 73% | 35% |
| CuMo | 87% | 92% | 78% | 35% |
| Mo | 80% | 95% | 55% | 35% |

## 11.3 Formulas

Recoveries for the metals are taken from Table 13 above for each assay/block in a particular zone and are value percentages/100.

GRV= ((Cu*20*$* recv)+((MoS2*20*(1.5/1.6681)*$(MoO3)* recv)+(Ag*$* recv)+(W*$* recv))

Recovered Cu. Equiv. = GRMV / ($(Copper) *20)

Recovered MoS2. Equiv. = GRMV / ((1.6681/1.5)* $($MoO_3$)*20)

## Table 14: Significant Intersections from CUMO Drilling

| Hole | From | To | Length | From | To | Length | Zone | recv | recv | MoO3 Equiv | MoS2 | Cu | Ag | Re | W | Recovered |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Name | feet | feet | feet | meters | meters | meters | | Cu Equiv. | MoS2 equiv. | lbs | % | % | g/t | ppm | ppm | Metal value US$ |
| C71-01 | 231 | 1884 | 1653 | 70 | 574 | 504 | main | 0.61 | 0.067 | 1.2 | 0.059 | 0.12 | 2.59 | 0.00 | 46 | $18 |
| C71-01 | 390 | 470 | 80 | 119 | 143 | 24 | sub | 0.90 | 0.100 | 1.8 | 0.099 | 0.14 | 2.56 | 0.00 | 44 | $27 |
| C71-01 | 1700 | 1884 | 184 | 518 | 574 | 56 | sub | 0.92 | 0.101 | 1.8 | 0.100 | 0.08 | 1.21 | 0.00 | 54 | $27 |
| C72-05 | 450 | 1416 | 966 | 137 | 432 | 294 | main | 0.65 | 0.072 | 1.3 | 0.060 | 0.13 | 4.46 | 0.00 | 75 | $20 |
| C74-09 | 460 | 805 | 345 | 140 | 245 | 105 | main | 0.81 | 0.089 | 1.6 | 0.077 | 0.12 | 7.16 | 0.00 | 71 | $24 |
| C75-10 | 220 | 2160 | 1940 | 67 | 658 | 591 | main | 0.88 | 0.097 | 1.8 | 0.099 | 0.05 | 1.43 | 0.00 | 48 | $26 |
| C76-11 | 140 | 2428 | 2288 | 43 | 740 | 698 | main | 0.67 | 0.074 | 1.3 | 0.074 | 0.05 | 1.55 | 0.00 | 36 | $20 |
| C76-11 | 1300 | 1960 | 660 | 396 | 597 | 201 | sub | 1.10 | 0.122 | 2.2 | 0.127 | 0.03 | 0.77 | 0.00 | 58 | $33 |
| C76-12 | 98 | 1430 | 1332 | 30 | 436 | 406 | main | 0.41 | 0.045 | 0.8 | 0.041 | 0.06 | 1.66 | 0.00 | 45 | $12 |
| C77-13 | 680 | 1804 | 1124 | 207 | 550 | 343 | main | 0.98 | 0.109 | 2.0 | 0.111 | 0.05 | 1.98 | 0.00 | 49 | $30 |
| C77-14 | 780 | 2124 | 1344 | 238 | 647 | 410 | main | 1.02 | 0.112 | 2.0 | 0.114 | 0.06 | 1.84 | 0.00 | 65 | $30 |
| C77-14 | 1200 | 1960 | 760 | 366 | 597 | 232 | sub | 1.33 | 0.148 | 2.7 | 0.151 | 0.06 | 1.91 | 0.00 | 74 | $40 |
| C77-15 | 600 | 1933 | 1333 | 183 | 589 | 406 | main | 1.01 | 0.112 | 2.0 | 0.113 | 0.06 | 1.73 | 0.00 | 57 | $30 |
| C77-15 | 1260 | 1880 | 620 | 384 | 573 | 189 | sub | 1.30 | 0.144 | 2.6 | 0.153 | 0.02 | 0.75 | 0.00 | 69 | $39 |
| C78-16 | 1000 | 2132 | 1132 | 305 | 650 | 345 | main | 0.82 | 0.091 | 1.6 | 0.093 | 0.04 | 1.86 | 0.00 | 32 | $25 |
| C78-17 | 1160 | 2282 | 1122 | 354 | 695 | 342 | main | 0.63 | 0.069 | 1.3 | 0.064 | 0.08 | 2.55 | 0.00 | 40 | $19 |
| C78-18 | 1400 | 2361 | 961 | 427 | 720 | 293 | main | 1.16 | 0.129 | 2.3 | 0.129 | 0.08 | 2.71 | 0.00 | 41 | $35 |
| C79-19 | 120 | 2280 | 2160 | 37 | 695 | 658 | main | 0.92 | 0.102 | 1.8 | 0.101 | 0.08 | 2.27 | 0.00 | 49 | $28 |
| C79-20 | 165 | 1800 | 1635 | 50 | 549 | 498 | main | 0.70 | 0.077 | 1.4 | 0.069 | 0.11 | 3.83 | 0.00 | 52 | $21 |
| C81-25 | 190 | 1011 | 821 | 58 | 308 | 250 | main | 0.71 | 0.079 | 1.4 | 0.070 | 0.13 | 2.42 | 0.00 | 58 | $21 |
| C81-25 | 740 | 1011 | 271 | 226 | 308 | 83 | sub | 0.89 | 0.099 | 1.8 | 0.090 | 0.14 | 2.98 | 0.00 | 84 | $27 |
| C81-26 | 30 | 750 | 720 | 9 | 229 | 220 | main | 0.48 | 0.053 | 1.0 | 0.034 | 0.18 | 7.58 | 0.00 | 28 | $14 |
| C06-27 | 120 | 1849 | 1729 | 37 | 564 | 527 | main | 0.77 | 0.085 | 1.5 | 0.084 | 0.06 | 1.6 | 0.02 | 49 | $23 |
| C06-27 | 1080 | 1849 | 769 | 329 | 564 | 234 | sub | 1.16 | 0.128 | 2.3 | 0.133 | 0.04 | 0.99 | 0.04 | 59 | $35 |
| C06-28 | 50 | 1690 | 1640 | 15 | 515 | 500 | main | 0.89 | 0.098 | 1.8 | 0.097 | 0.07 | 1.92 | 0.05 | 54 | $27 |
| C06-28 | 840 | 1240 | 400 | 256 | 378 | 122 | sub | 1.40 | 0.155 | 2.8 | 0.162 | 0.03 | 0.98 | 0.09 | 68 | $42 |
| C07-29 | 190 | 2230 | 2040 | 58 | 680 | 622 | main | 0.95 | 0.105 | 1.9 | 0.103 | 0.08 | 2.13 | 0.05 | 53 | $29 |
| C07-29 | 1180 | 1790 | 610 | 360 | 546 | 186 | sub | 1.46 | 0.162 | 2.9 | 0.169 | 0.04 | 1.2 | 0.08 | 37 | $44 |

**Table 13: Significant Intersection from CUMO Drilling (Continued)**

| Hole Name | From feet | To feet | Length feet | From meters | To meters | Length meters | Zone | recv Cu Equiv. | recv MoS2 equiv. | MoO3 Equiv lbs | MoS2 % | Cu % | Ag g/t | Re ppm | W ppm | Recovered Metal value US$ |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| C07-30 | 40 | 2386 | 2346 | 12 | 727 | 715 | main | 0.98 | 0.108 | 1.9 | 0.108 | 0.06 | 2.05 | 0.04 | 41 | $29 |
| C07-30 | 1180 | 1988 | 808 | 360 | 606 | 246 | sub | 1.59 | 0.177 | 3.2 | 0.185 | 0.04 | 1.46 | 0.07 | 37 | $48 |
| C07-31 | 22 | 2104 | 2082 | 7 | 641 | 635 | main | 0.61 | 0.067 | 1.2 | 0.064 | 0.07 | 1.76 | 0.02 | 43 | $18 |
| C07-31 | 780 | 1540 | 760 | 238 | 469 | 232 | sub | 0.74 | 0.082 | 1.5 | 0.081 | 0.05 | 1.45 | 0.03 | 45 | $22 |
| C07-32 | 22 | 2104 | 2082 | 7 | 641 | 635 | main | 1.00 | 0.111 | 2.0 | 0.109 | 0.09 | 2.26 | 0.04 | 61 | $30 |
| C07-32 | 780 | 1540 | 760 | 238 | 469 | 232 | sub | 1.19 | 0.132 | 2.4 | 0.129 | 0.10 | 2.62 | 0.05 | 77 | $36 |
| C07-33 | 722 | 2094 | 1372 | 220 | 638 | 418 | main | 0.30 | 0.033 | 0.6 | 0.026 | 0.07 | 2.01 | 0.01 | 48 | $9 |
| C07-33 | 1980 | 2094 | 114 | 604 | 638 | 35 | sub | 0.82 | 0.091 | 1.6 | 0.084 | 0.10 | 2.68 | 0.03 | 67 | $25 |
| C07-34 | 140 | 1769 | 1629 | 43 | 539 | 497 | main | 0.37 | 0.042 | 0.8 | 0.034 | 0.08 | 2.3 | 0.01 | 53 | $11 |
| C07-34 | 1550 | 1769 | 219 | 472 | 539 | 67 | sub | 0.71 | 0.078 | 1.4 | 0.074 | 0.09 | 2.36 | 0.02 | 67 | $21 |
| C08-35 | 120 | 2640 | 2520 | 37 | 805 | 768 | main | 0.54 | 0.060 | 1.1 | 0.057 | 0.06 | 1.73 | 0.02 | 37 | $16 |
| C08-35 | 420 | 2640 | 2220 | 128 | 805 | 677 | sub | 0.58 | 0.065 | 1.2 | 0.062 | 0.07 | 1.69 | 0.02 | 39 | $17 |
| C08-35 | 1730 | 2640 | 910 | 527 | 805 | 277 | sub | 0.81 | 0.089 | 1.6 | 0.089 | 0.05 | 1.37 | 0.03 | 35 | $24 |
| C08-36 | 560 | 2488 | 1928 | 171 | 758 | 588 | main | 0.79 | 0.087 | 1.6 | 0.088 | 0.05 | 1.42 | 0.03 | 34 | $24 |
| C08-36 | 920 | 2488 | 1568 | 280 | 758 | 478 | sub | 0.91 | 0.101 | 1.8 | 0.103 | 0.04 | 1.04 | 0.03 | 33 | $27 |
| C08-37 | 60 | 2195 | 2135 | 18 | 669 | 651 | main | 0.76 | 0.085 | 1.5 | 0.084 | 0.05 | 1.67 | 0.03 | 42 | $23 |
| C08-37 | 780 | 2130 | 1350 | 238 | 649 | 412 | sub | 0.90 | 0.100 | 1.8 | 0.104 | 0.02 | 1.17 | 0.04 | 41 | $27 |
| C08-38 | 170 | 2441 | 2271 | 52 | 744 | 692 | main | 0.32 | 0.035 | 0.6 | 0.029 | 0.06 | 4.4 | 0.00 | 32 | $10 |
| C08-39 | 310 | 2688 | 2378 | 95 | 819 | 725 | main | 0.89 | 0.098 | 1.8 | 0.099 | 0.06 | 1.38 | 0.03 | 52 | $27 |
| C08-39 | 900 | 2390 | 1490 | 274 | 729 | 454 | sub | 1.07 | 0.119 | 2.1 | 0.122 | 0.04 | 1.09 | 0.04 | 57 | $32 |
| C08-40 | 60 | 2252 | 2192 | 18 | 686 | 668 | main | 1.04 | 0.115 | 2.1 | 0.115 | 0.06 | 3.79 | 0.04 | 46 | $31 |
| C08-40 | 390 | 2080 | 1690 | 119 | 634 | 515 | sub | 1.17 | 0.129 | 2.3 | 0.129 | 0.06 | 4.27 | 0.05 | 45 | $35 |
| C08-40 | 1110 | 1820 | 710 | 338 | 555 | 216 | sub | 1.29 | 0.143 | 2.6 | 0.142 | 0.04 | 7.78 | 0.06 | 45 | $39 |
| C08-41 | 850 | 2830 | 1980 | 259 | 863 | 604 | main | 0.65 | 0.072 | 1.3 | 0.067 | 0.08 | 2.23 | 0.02 | 43 | $20 |
| C08-41 | 1490 | 2030 | 540 | 454 | 619 | 165 | sub | 0.99 | 0.110 | 2.0 | 0.107 | 0.08 | 2.99 | 0.03 | 38 | $30 |
| C08-41 | 2490 | 2830 | 340 | 759 | 863 | 104 | sub | 0.70 | 0.078 | 1.4 | 0.077 | 0.06 | 1.53 | 0.03 | 34 | $21 |
| C08-42 | 550 | 2707 | 2157 | 168 | 825 | 658 | main | 0.47 | 0.052 | 0.9 | 0.044 | 0.06 | 5.81 | 0.01 | 25 | $14 |
| C08-42 | 950 | 2707 | 1757 | 290 | 825 | 536 | sub | 0.50 | 0.056 | 1.0 | 0.047 | 0.07 | 6.78 | 0.01 | 27 | $15 |

**Table 13: Significant Intersection from CUMO Drilling (Continued)**

| Hole Name | From feet | To feet | Length feet | From meters | To meters | Length meters | Zone | recv Cu Equiv. | recv MoS2 equiv. | MoO3 Equiv lbs | MoS2 % | Cu % | Ag g/t | Re ppm | W ppm | Recovered Metal value US$ |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| C08-43 | 660 | 820 | 160 | 201 | 250 | 49 | sub | 0.71 | 0.078 | 1.4 | 0.071 | 0.11 | 3.14 | 0.03 | 45 | $21 |
| C08-44 | 1125 | 2840 | 1715 | 343 | 866 | 523 | main | 0.27 | 0.029 | 0.5 | 0.028 | 0.02 | 0.89 | 0.01 | 29 | $8 |
| C08-44 | 2560 | 2690 | 130 | 780 | 820 | 40 | sub | 0.49 | 0.054 | 1.0 | 0.055 | 0.02 | 1.47 | 0.01 | 20 | $15 |
| C08-45 | 170 | 1796 | 1626 | 52 | 547 | 496 | main | 0.33 | 0.037 | 0.7 | 0.021 | 0.15 | 3.08 | 0.00 | 42 | $10 |
| C08-45 | 1010 | 1796 | 786 | 308 | 547 | 240 | sub | 0.44 | 0.048 | 0.9 | 0.032 | 0.18 | 3.05 | 0.00 | 40 | $13 |

Note: Holes 33 was just entering the higher grade MO zone when stopped.
Hole 34 had not yet reached the Mo zone and will be continued in 2008.
Rhenium was not assayed for prior to 2006
Recv are recovered values with recoveries built in based on the zone as detailed in Table 13

The 2006 - 2008 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 - 2008 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 down-hole histograms for Cu, Ag and Mo suggests the metal zones may be a part of a single, large, concentrically zoned system with an upper copper-silver zone, underlain by a transitional copper-molybdenum zone, in turn underlain by a lower molybdenum-rich zone (Figure 12).

Three-dimensional modeling of the above zonation was conducted by Mr. Shaun Dykes (P.Geo.), which indicates the current area being drilled is located on the north side of a large system extending 4.5 km (15,000 feet) in diameter, of which only a small part (1 km or 3000 feet) has been drilled (Figure 13).

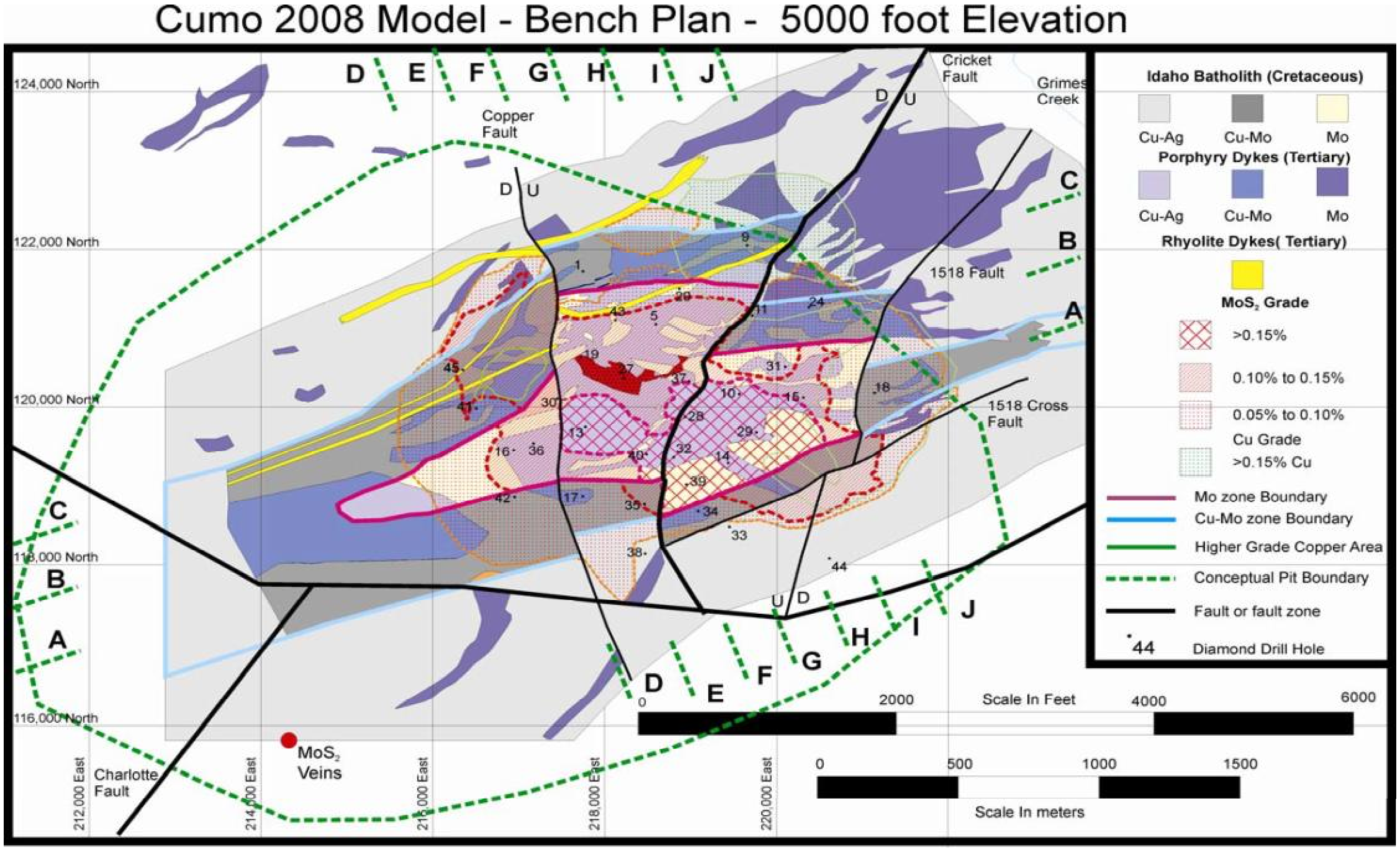



**Figure 12: Geology Bench Plan At 5000 Ft Elevation**

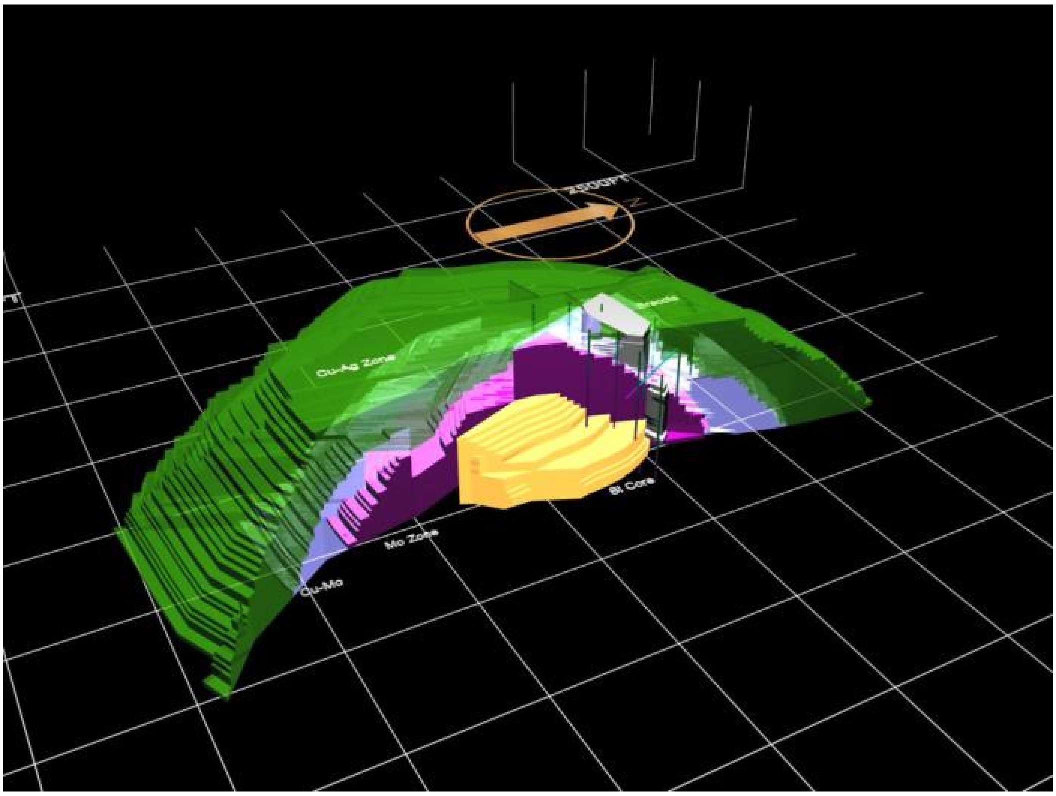



**Figure 13: Snapshot of 3D model of CUMO Deposit Showing Concentric Pattern of Metal zones.**

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

## 12 SAMPLING METHOD AND APPROACH

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.

Sampling was restricted during 2006, 2007 and 2008 to Diamond Drill Hole (DDH) core and metallurgical sampling of previously drilled DDH core. Standard core sampling methods were employed for both drill core and metallurgical samples. The companies approach was based upon the tried and true methods of drilling, sampling and assaying to physically define an ore body.

DDH drill core was placed in wooden core boxes during the 2006, 2007 and 2008 drilling seasons. In 2008, Mosquito's staff, over seen 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 drill hole.

A core technician using a standard rock saw samples the core using typical procedures. The technician uses safety equipment such as goggles and earplugs. 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.

MoS2 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 MoS2 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.

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 &

[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 lighted and is seldom without occupancy by Mosquito staff. The doors are locked when the building is unoccupied.

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 15).

**Table 15: 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.

# 13 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

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.

## 13.1 Analysis

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 lighted 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 lighted, 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.

## 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.

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

### 14.1 Historical Checks

As reported in the June 2005 report (Cavey et. al. 2005) 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 lab sets of duplicates are also available to compare with the Skyline original assays; a pulp sent to Amax Lab in Climax from diamond drill hole assays, a second split at the drill of reverse circulation drill cuttings and a selected set of samples sent to Hazen Laboratory. The results from all comparisons are presented in the 2005 report. In general, the results showed good correlation and high sampling variability for $MoS_2$.

During the Mosquito 2007 drill campaign blanks, standards were routinely inserted into the sample stream to monitor QA/QC at the primary laboratory ALS Chemex. In addition the Lab reported internal blanks, standards and duplicates which showed excellent agreement. Results from the 2007 QA/QC program reported in (Holmgren and Giroux, 2008) showed good agreement.

### 14.2 2008 Drill Program

QA/QC procedures on the 2008 drill program included blanks, standards, internal lab standards, lab internal pulp checks, and re-splits sent to second labs.

#### 14.2.1 Blanks

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. The results were very good with no anomalies produced. The graphs for $MoS_2$ and Cu are shown below in Figure 14 and Figure 15.

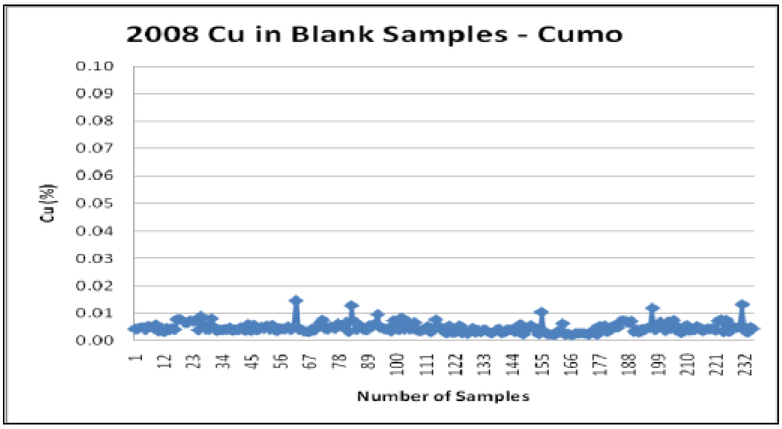



**Figure 14: $MoS_2$ in Blank Samples from 2008 Drill Program CUMO**

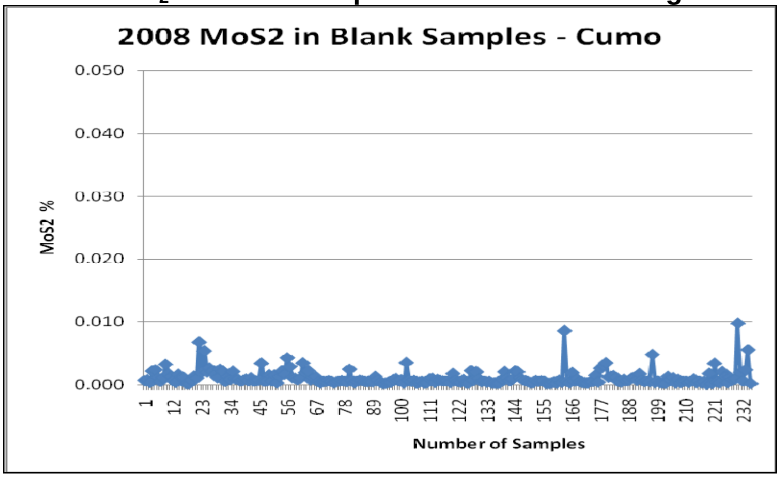



**Figure 15: Cu in Blank Samples from 2008 Drill Program CUMO**

### 14.2.2 Internal Lab Standards

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 Internal Pulp Checks

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 $MoS_2$. Figure 16 below shows the results are excellent with all but a few samples falling on an equal value line. The best fit regression line mirrors the equal value line.

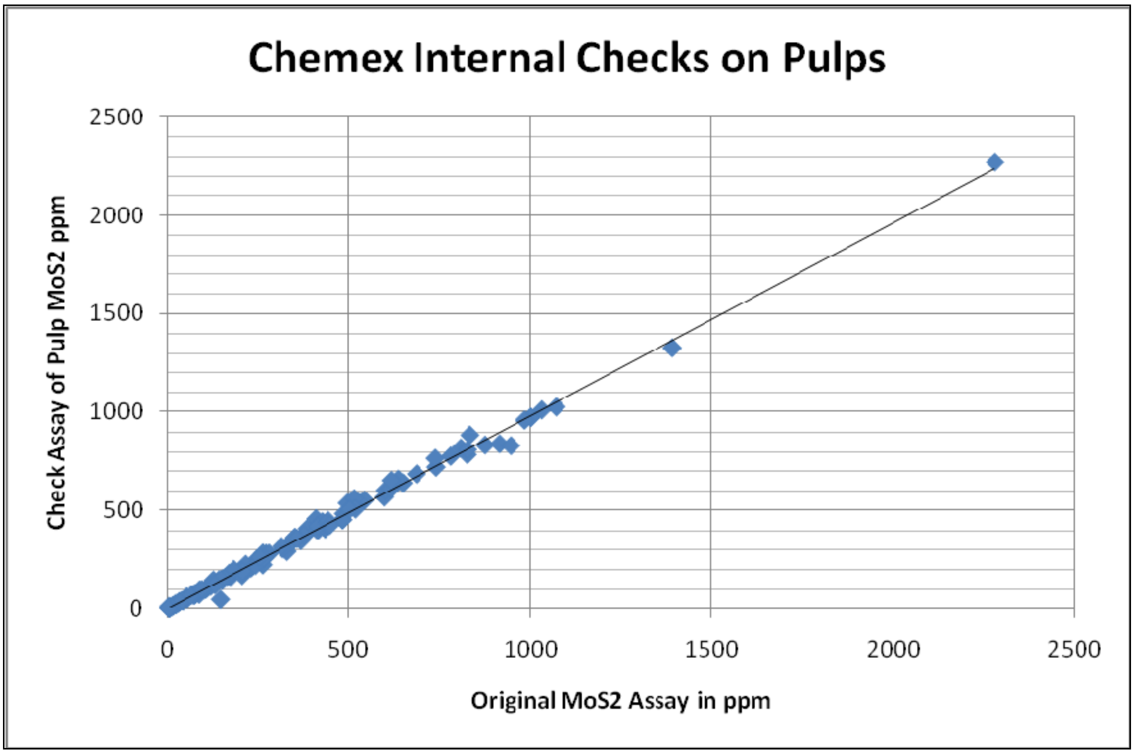



**Figure 16: Scatter plot of Chemex Internal Duplicates for $MoS_2$**

### 14.2.4 Mosquito Standards

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). Sample 396353 (2006 sample) reports 0.049 % Mo with a corresponding low 0.01 % Cu indicating something was wrong with both analysis 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 (see Figure 18).

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 (see Figure 19).

### 14.2.5 2008 Reject Duplicates

During the 2008 drill program second and third splits were taken from 154 rejects and re-assayed by the primary Lab first by ICP_MS61 and then by XRF. Due to high volumes of samples submitted to the primary Lab ALS Chemex, 31samples 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.

The results are presented as a series of scatter plots with all variables reported in ppm and are shown in Appendix 1.

The results for the re-splits on Mo run by ALS Chemex, both by ICP, are excellent with a correlation coefficient of 0.9933 and no bias indicated (the reduced major axis (RMA) regression line mirrors the equal value line). The average ICP to ICP precision is ± 24.4 %.

Comparing re-splits for Mo run by ALS Chemex analyzed by ICP and a check analysis by XRF also shows excellent agreement. The correlation coefficient is 0.9944 and again the RMA regression line mirrors the equal value line indicating no bias present. The precision on ICP to XRF is ± 22.3%.

Copper run at Chemex by ICP and compared to a re-split run by ICP showed excellent agreement with a correlation coefficient of 0.9981. The RMA regression line is slightly above the equal value line but samples are scattered about the line equally and no bias is indicated. The precision on ICP original to ICP check is ± 15 %.

Copper ICP compared to copper from XRF, both run at Chemex, show a high correlation coefficient of 0.9978 and a slight indication of bias with XRF slightly higher in values above 200 ppm (the RMA regression line is pulled slightly above the equal value line). The precision on ICP original to XRF check is ±15.9 %.

Silver original ICP compared with a second split also run by ICP showed excellent agreement with a correlation coefficient of 0.9978. The RMA regression line is slightly below the equal value line but no bias is indicated. The precision on Ag is ±15 %.

A similar set of comparisons was made for the 31 samples sent to SGS Laboratory.

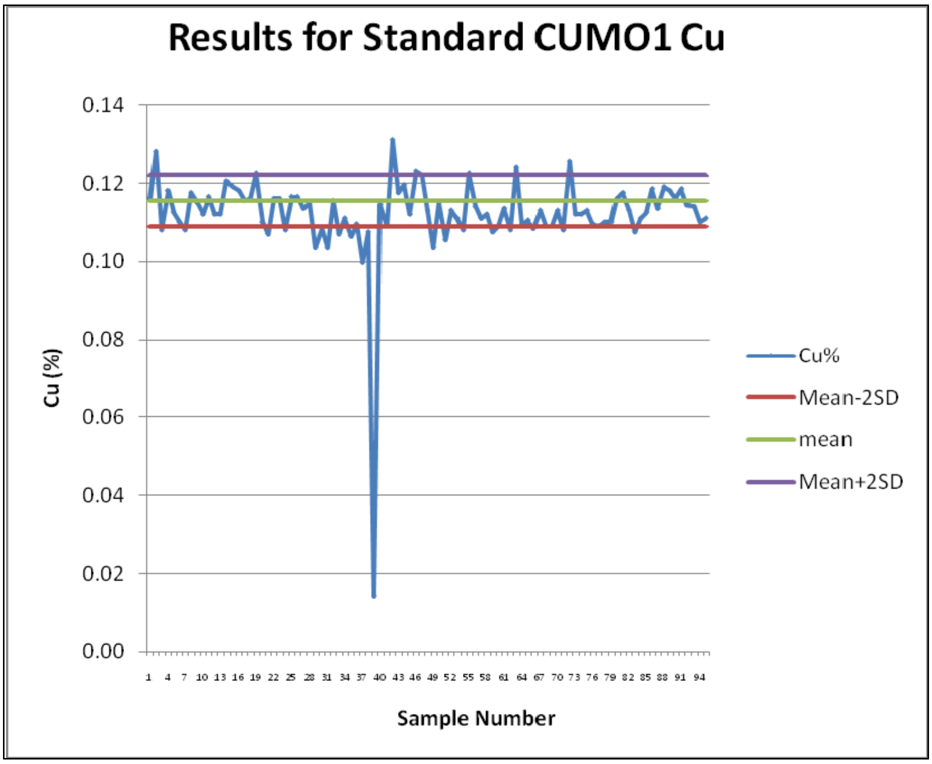
A comparison of Mo from the original ICP analysis with an ICP on a split from rejects shows the RMA regression line pulled above an equal value line by two high values. In general however there is no bias indicated. The correlation coefficient is 0.9960 and the precision is ± 24.6%.

A comparison of Mo from the original SGS ICP analysis with an SGS XRF analysis from a second split of rejects showed good agreement with a correlation coefficient of 0.9829. The RMA regression line is pulled above an equal value line by one high sample but no bias is indicated. The precision between the two analysis is ± 52.8 % indicating more scatter about the RMA regression line and a number of low XRF readings.

The comparison between SGS original sample and ICP check sample for copper is excellent with a coefficient of correlation of 0.9944. There is no indication of bias with the RMA regression line nearly identical to the equal value line. The precision between the two estimates is ± 12.6 %.

The comparison between SGS original samples and SGS XRF check samples is not as good. A bias is clearly indicated with XRF showing higher values than ICP above 300 ppm. The RMA regression line shows a proportional bias relative to the equal value line. The coefficient of correlation is reasonable at 0.9884 with the precision between the two estimates of ± 18 %.

The SGS checks on Ag comparing the original sample with a second split from the rejects show a fair degree of scatter but no bias. The correlation coefficient is 0.9491 and the precision on the two samples is ± 45.4 %.



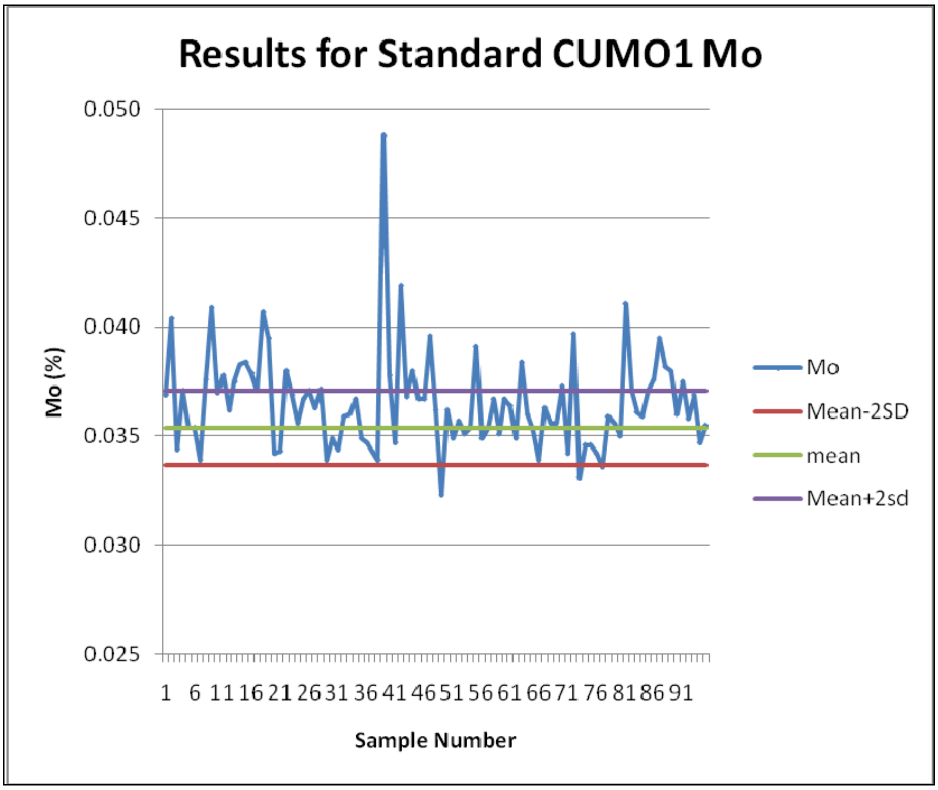
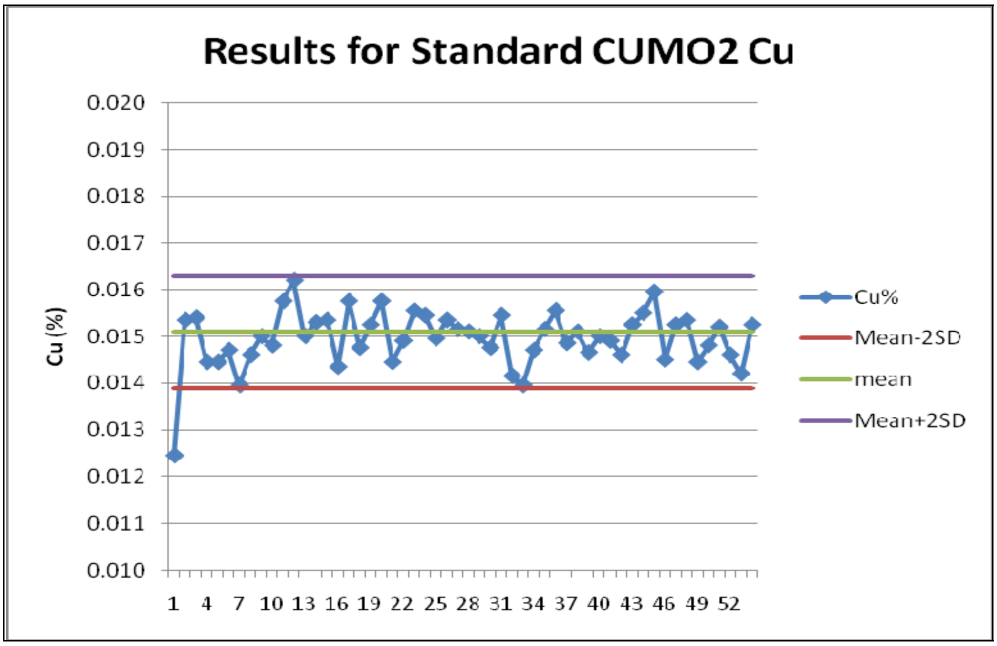




**Figure 17: Results for Standard CUMO1**



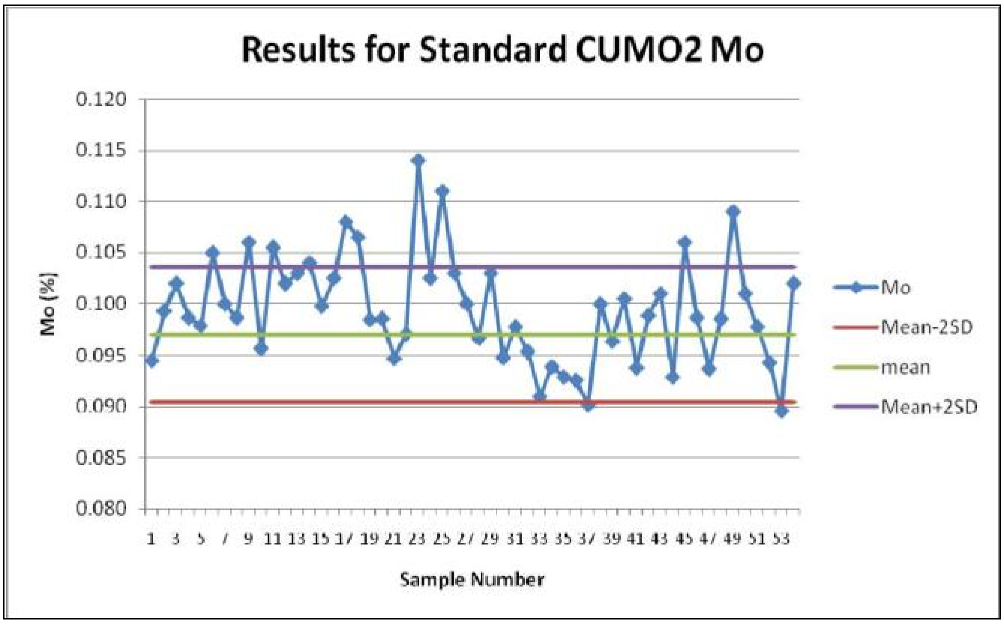
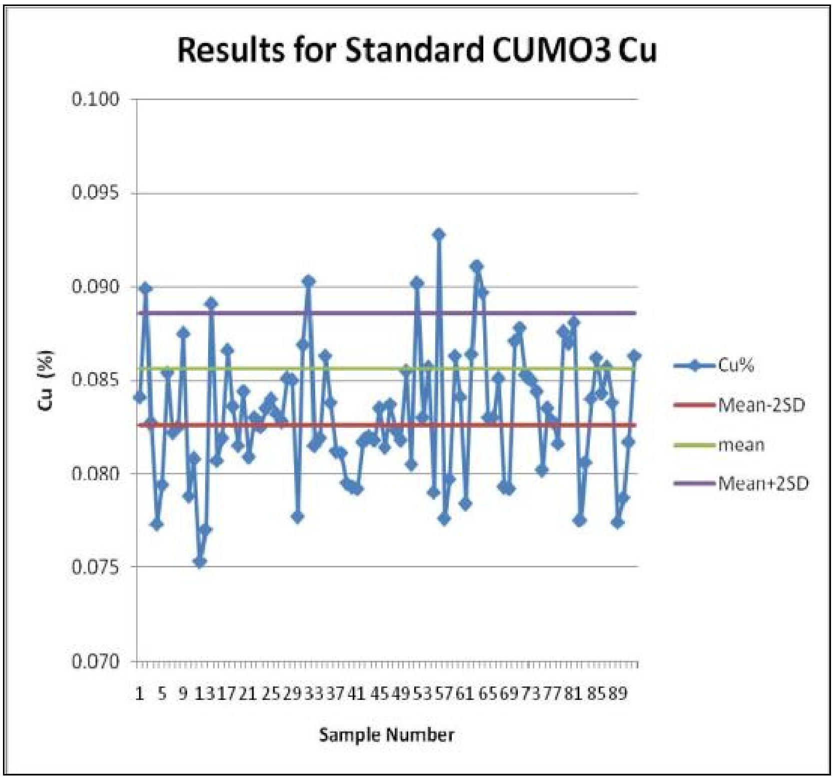




**Figure 18: Results for Standard CUMO2**



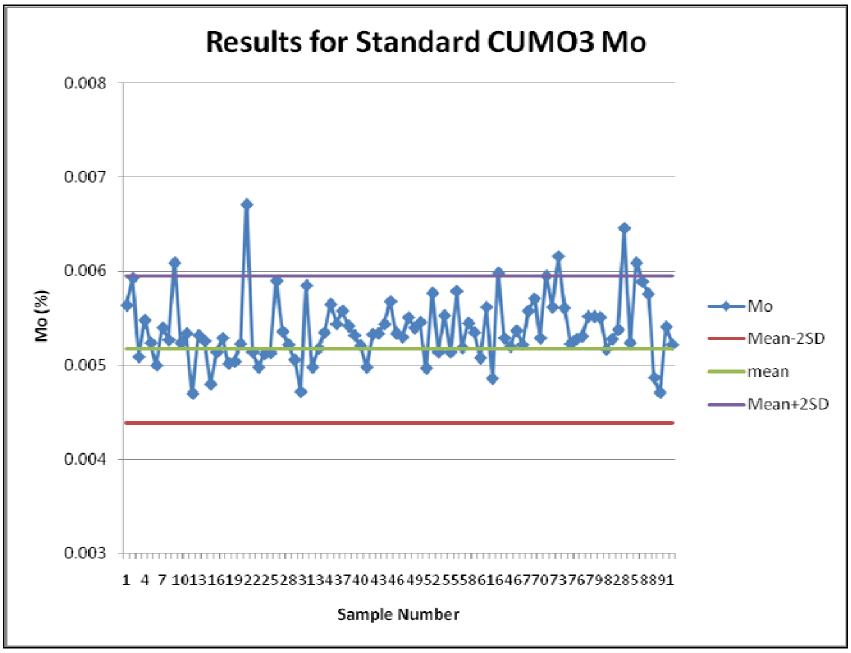




**Figure 19: Results for Standard CUMO3**

# 15 MINERAL PROCESSING AND METALLURGICAL TESTING

## 15.1 Metallurgical Testing

### 15.1.1 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.

### 15.1.2 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 drill holes 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".

### 15.1.3 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" February 18, 2009 by SGS Canada Inc.

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, and
- Bench scale flotation testing consisting of rougher kinetic flotation, cleaner flotation and locked-cycle tests, supplemented with mineralogical examination.

### a) 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 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: Summary of Comminution Test Work Data**

| Comminution Characteristics | | Cu-Ag | Cu-Mo | Mo | Design |
|---|---|---|---|---|---|
| Specific gravity | t/m³ | 2.64 | 2.60 | 2.60 | 2.64 |
| SPI® | min | 84.5 | 73.0 | 70.8 | 84.5 |
| SMC DWI | kW/m³ | n/a | n/a | n/a | 7.40 |
| 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 |

Normally, when a significant amount of variability testing is undertaken, the design comminution characteristics are selected to ensure that the majority of the ore body can be treated at the nominated design rate. This is typically achieved by selecting upper percentile comminution characteristics as the basis for comminution circuit design. This is based on the premise that the hardest ores can be blended with softer ore during normal mining and stockpile operations, allowing the plant to achieve the nameplate capacity at all times.

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.

### b) 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.

### 15.1.4 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.

#### a) 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 17.

**Table 17: Baseline Flotation results for CUMO Composite Samples**

| Ore Type | | Test No. | | Feed | | Concentrate Grade | | Concentrate Recovery | |
|---|---|---|---|---|---|---|---|---|---|
| | | % Cu | g/t Mo | % Cu | %t 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.

### b) 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 summarised in Table 18.

**Table 18: 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 indicates 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.

#### c) 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 19.

**Table 19: 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.

### 15.1.5 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.

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 20. These figures include bulk concentrate recovery, copper/molybdenum flotation separation and ferric chloride leach recovery.

**Table 20: 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 |

## 15.2 Mineral Processing

### 15.2.1 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.

### 15.2.2 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 21.

### Table 21: 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 |
| | Cleaner 1 | Mins | 10 | 10 | 10 | 10 |
| | Cleaner Scav. | Mins | 2.5 | 2.5 | 2.5 | 2.5 |

| Criteria | | Units | Design | | | |
|---|---|---|---|---|---|---|
| | | | 50 kt/d (short tons) | 100 kt/d (short tons) | 150 kt/d (short tons) | 200 kt/d (short tons) |
| | 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 summarised below.

### 15.2.3 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) and 181 kt/d (metric tonnes);
- Design availability of 92% (after ramp-up), being 8,059 operating hours per year, with standby equipment in critical areas, and
- Sufficient plant design flexibility for treatment of all ore types at design throughput.

The selection of these parameters is discussed in detail below.

### 15.2.4 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.

### 15.2.5 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.

### 15.2.6 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% $MoS_2$).

## 15.3 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 20 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.

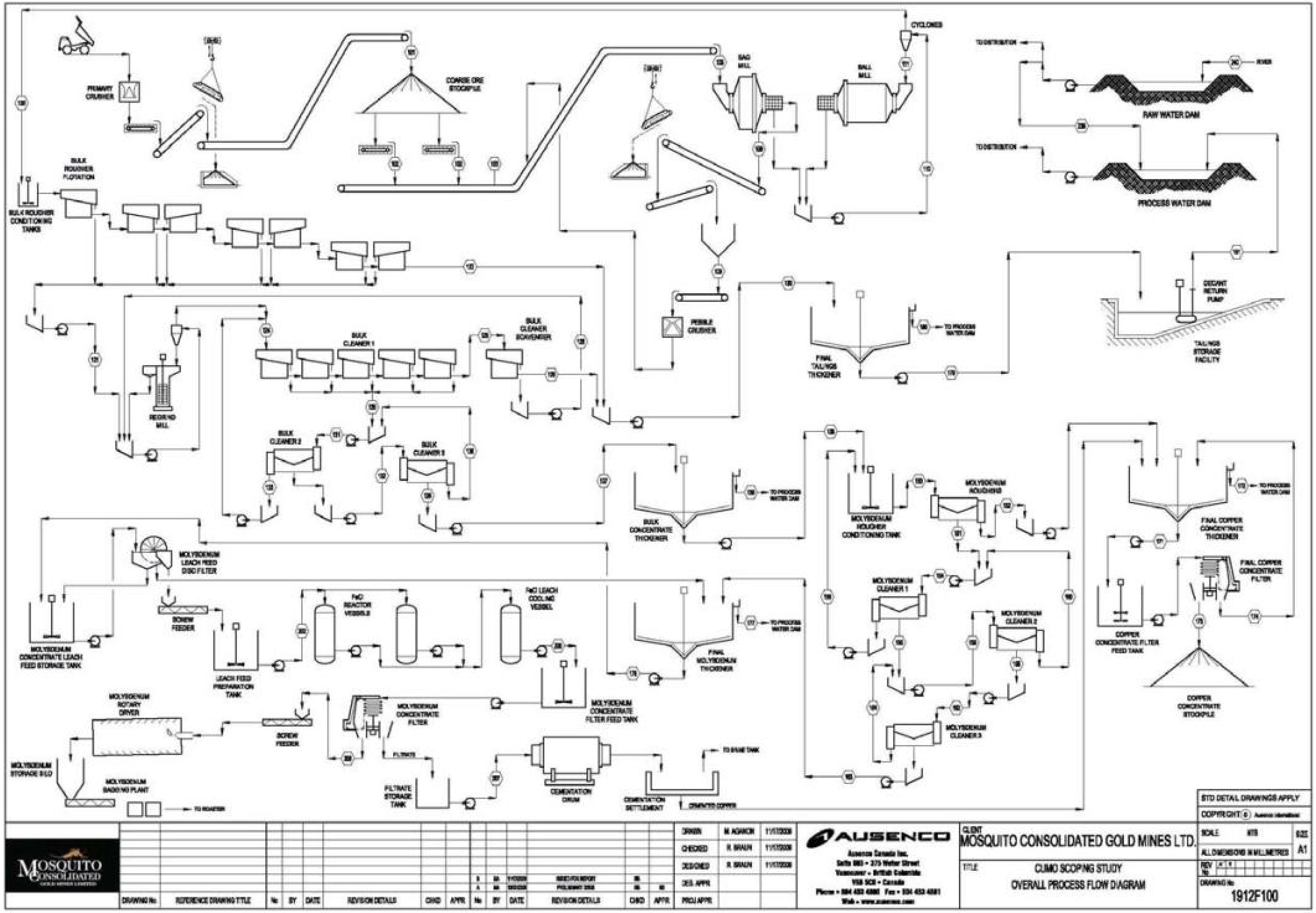



**Figure 20: CUMO Process Plant Process Schematic**

### 15.3.1 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 $m^3$ 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 $m^3$ forced air tank flotation cells to provide a total of 13 minutes of retention time;
- Bulk cleaner 2 flotation cells consisting of 8 $m^3$ trough shaped flotation cells to provide a total of 10 minutes of retention time;
- Bulk cleaner 3 flotation cells consisting of 8 $m^3$ trough shaped flotation cells to provide a total of 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, and
- Plant, instrument and flotation air services and associated infrastructure.

### 15.3.2 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 21.

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.

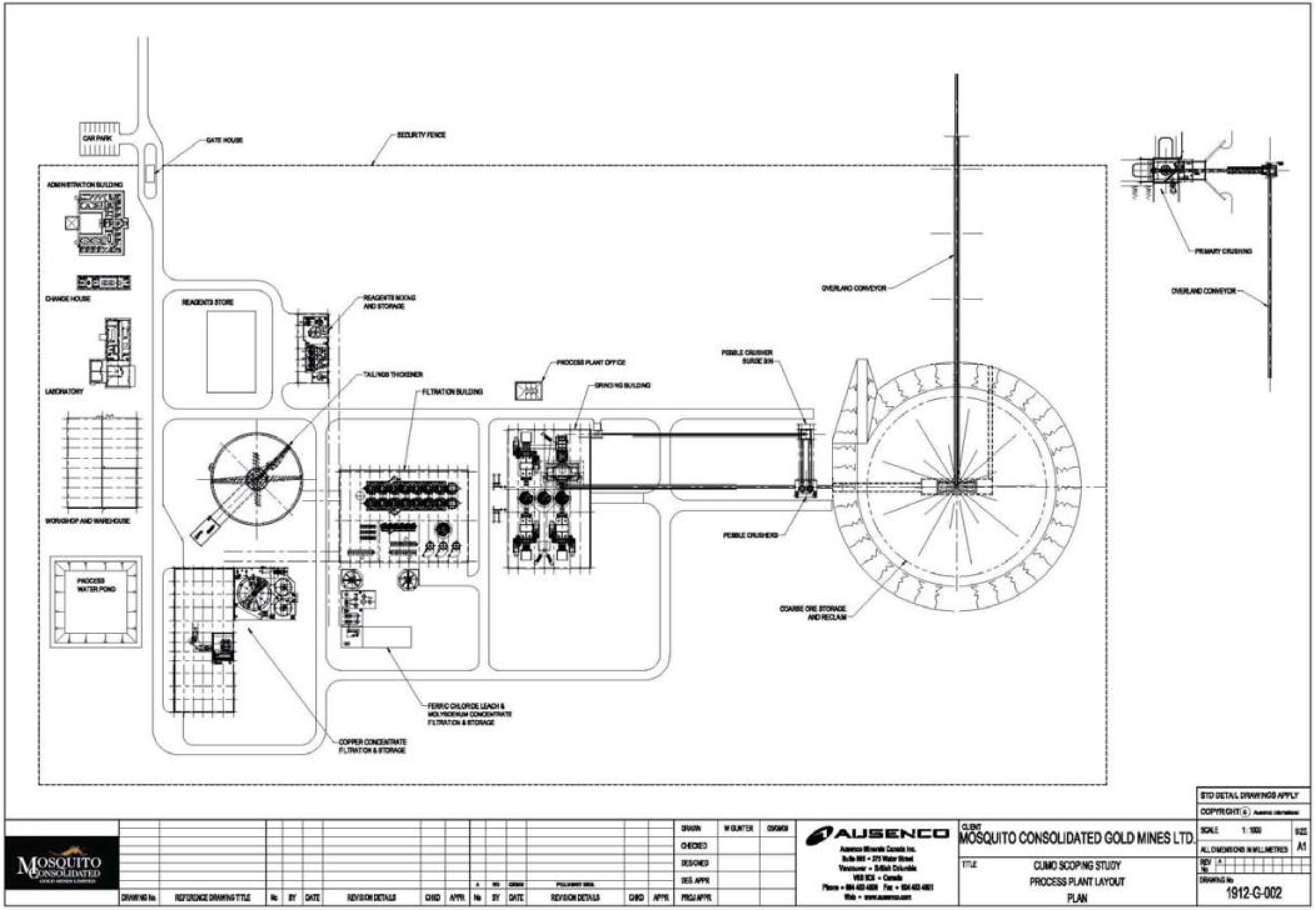



**Figure 21: CUMO Process Plant Layout**

# 16 MINERAL RESOURCE ESTIMATION

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 2009 CUMO Resource estimate represents an update of the 2008 estimate (Holmgren and Giroux, 2008) based on an additional 11 new diamond drill holes completed in 2008.

## 16.1 Data Analysis

A total of 42 diamond drill holes over a combined total of 76,436 ft and 3 reverse circulation drill holes were provided with 632 down hole surveys and 6,619 assays for $MoS_2$ and Cu. For this resource estimation the 3 reverse circulation holes were not used (see Appendix 2 for a list of drill holes used in the Estimate). The basic assay statistics for diamond drill holes are presented below in Table 22.

**Table 22: Summary of Assay Statistics**

| | $MoS_2$ (%) | Cu (%) |
|---|---|---|
| Number | 6,619 | 6,619 |
| Mean | 0.061 | 0.078 |
| Standard Deviation | 0.061 | 0.069 |
| Minimum | 0.0005 | 0.001 |
| Maximum | 1.09 | 0.920 |
| Coefficient of Variation | 1.00 | 0.85 |

The molybdenum and copper mineralization at CUMO lies in three distinct mineral zones with an oxidized layer on top. More or less from top to bottom there occurs in most drill holes an Oxide Zone, Cu-Ag zone, a Cu-Mo zone and a Mo zone. While the oxide zone has been modeled for metallurgical reasons it has been combined with the Cu-Ag zone for estimation purposes. There are also several post mineral dykes that are large enough and continuous enough to be modeled. The Cu and $MoS_2$ grades can be sorted by Zone. Silver and tungsten assays are shown for the same mineral zones. Values for $MoS_2$ and Cu reported as 0.000 were assigned values of 0.0005% and 0.001 % respectively. Silver values reported as 0.000 were set to 0.01 g/t while tungsten values reported as 0.000 were set to 0.1 ppm.

### Table 23: Summary of Assay Statistics for Cu and $MoS_2$ Sorted by Zone

| | Cu–Ag Zone | | Cu-Mo Zone | | Mo Zone | | Dyke | |
|---|---|---|---|---|---|---|---|---|
| | $MoS_2$ (%) | Cu (%) | $MoS_2$ (%) | Cu (%) | $MoS_2$ (%) | Cu (%) | $MoS_2$ (%) | Cu (%) |
| Number | 1,492 | 1,492 | 2,504 | 2,504 | 2,223 | 2,223 | 53 | 53 |
| Mean | 0.018 | 0.084 | 0.049 | 0.100 | 0.108 | 0.047 | 0.007 | 0.018 |
| Standard Deviation | 0.022 | 0.071 | 0.047 | 0.071 | 0.067 | 0.042 | 0.019 | 0.032 |
| Minimum | 0.0005 | 0.001 | 0.0005 | 0.001 | 0.0005 | 0.001 | 0.0005 | 0.001 |
| Maximum | 0.315 | 0.71 | 1.09 | 0.92 | 0.99 | 0.59 | 0.13 | 0.15 |
| Coefficient of Variation | 1.21 | 0.85 | 0.95 | 0.71 | 0.63 | 0.89 | 2.52 | 1.78 |

### Table 24: Summary of Assay Statistics for Ag and W Sorted by Zone

| | Cu–Ag Zone | | Cu-Mo Zone | | Mo Zone | | Dyke | |
|---|---|---|---|---|---|---|---|---|
| | Ag (g/t) | W (ppm) | Ag (g/t) | W (ppm) | Ag (g/t) | W (ppm) | Ag (g/t) | W (ppm) |
| Number | 1,485 | 1,470 | 2,488 | 2,493 | 2,196 | 2,200 | 53 | 46 |
| Mean | 2.79 | 30.1 | 3.13 | 46.8 | 1.76 | 45.1 | 0.78 | 15.5 |
| Standard Deviation | 10.23 | 28.8 | 16.02 | 49.7 | 10.80 | 37.6 | 1.03 | 16.3 |
| Minimum | 0.01 | 0.1 | 0.01 | 0.1 | 0.01 | 0.1 | 0.01 | 2.1 |
| Maximum | 345.00 | 520.0 | 744.0 | 1980.0 | 494.0 | 890.0 | 4.40 | 65.0 |
| Coefficient of Variation | 3.66 | 0.96 | 5.12 | 1.06 | 6.15 | 0.83 | 1.32 | 1.06 |

To determine if capping was required and if so at what level the distribution of grades for each variable within each domain was examined using lognormal cumulative frequency plots.

In all cases multiple overlapping lognormal populations were present.

A similar strategy was applied to Cu, Ag and W. The capping levels for each variable are shown below.

### Table 25: Summary of Capping Levels by Domain

| Domain | Variable | Cap Level | Number Capped |
|---|---|---|---|
| Cu-Ag Zone | $MoS_2$ | 0.16 % | 3 |
| Cu-Mo Zone | $MoS_2$ | 0.40 % | 2 |
| Mo Zones | $MoS_2$ | 0.48 % | 6 |
| Cu-Ag Zone | Cu | 0.83 % | 0 |
| Cu-Mo Zone | Cu | 0.59 % | 4 |
| Mo Zones | Cu | 0.27 % | 6 |
| Cu-Ag Zone | Ag | 115 g/t | 2 |
| Cu-Mo Zone | Ag | 102 g/t | 5 |
| Mo Zones | Ag | 24 g/t | 5 |
| Cu-Ag Zone | W | 452 ppm | 1 |
| Cu-Mo Zone | W | 277 ppm | 4 |
| Mo Zones | W | 275 ppm | 4 |

## 16.2 50 Foot Composites

The bulk of the drill holes were assayed on 10 or 20 ft spacings. A 50 ft composite length was chosen to match a reasonable mining bench for this scale of deposit. This differs from the 2008 resource estimate where 20 ft composites were used. The statistics for 50 ft composites are shown in Table 26.

**Table 26: Summary of 50 ft Composite Statistics**

| | $MoS_2$ (%) | Cu (%) | Ag (g/t) | W (ppm) |
|---|---|---|---|---|
| **Cu-Ag Zone** | | | | |
| Number | 350 | 350 | 350 | 346 |
| Mean | 0.017 | 0.086 | 2.63 | 29.6 |
| Standard Deviation | 0.015 | 0.057 | 3.94 | 22.2 |
| Minimum | 0.001 | 0.001 | 0.01 | 0.1 |
| Maximum | 0.111 | 0.379 | 69.06 | 210.0 |
| Coefficient of Variation | 0.86 | 0.67 | 1.50 | 0.75 |
| **Cu-Mo Zone** | | | | |
| Number | 563 | 563 | 558 | 559 |
| Mean | 0.046 | 0.100 | 2.86 | 45.1 |
| Standard Deviation | 0.025 | 0.055 | 3.21 | 22.0 |
| Minimum | 0.003 | 0.005 | 0.23 | 10.7 |
| Maximum | 0.277 | 0.361 | 42.39 | 161.3 |
| Coefficient of Variation | 0.54 | 0.55 | 1.12 | 0.49 |
| **Mo Zone** | | | | |
| Number | 571 | 571 | 563 | 564 |
| Mean | 0.108 | 0.052 | 1.63 | 47.2 |
| Standard Deviation | 0.046 | 0.039 | 1.32 | 23.9 |
| Minimum | 0.025 | 0.003 | 0.09 | 5.0 |
| Maximum | 0.298 | 0.218 | 10.68 | 158.8 |
| Coefficient of Variation | 0.43 | 0.74 | 0.81 | 0.51 |
| **BBZ Zone** | | | | |
| Number | 13 | 13 | 13 | 13 |
| Mean | 0.026 | 0.007 | 1.40 | 23.2 |
| Standard Deviation | 0.016 | 0.005 | 2.06 | 5.9 |
| Minimum | 0.004 | 0.003 | 0.32 | 15.0 |
| Maximum | 0.054 | 0.020 | 8.06 | 34.0 |
| Coefficient of Variation | 0.61 | 0.76 | 1.47 | 0.26 |
| **Dykes** | | | | |
| Number | 4 | 4 | 4 | 4 |
| Mean | 0.002 | 0.003 | 0.28 | 6.1 |

## 16.3 Variography

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

Pairwise relative semivariograms were used to determine grade continuity for $MoS_2$, Cu, Ag and W in 50 ft composites. The semivariogram parameters are summarized in Table 27. The models for $MoS_2$ and Cu are shown in Appendix 3.

**Table 27: Parameters for Semivariogram Models at CUMO**

| Variable | Domains | Direction | C0 | C1 | C2 | Range a1 (ft) | Range a2 (ft) |
|---|---|---|---|---|---|---|---|
| $MoS_2$ | Cu-Mo and Mo Zone | Az 60 Dip 0 | 0.06 | 0.08 | 0.16 | 200 | 1800 |
| | | Az 150 Dip -55 | 0.06 | 0.08 | 0.16 | 150 | 1300 |
| | | Az 330 Dip -35 | 0.06 | 0.08 | 0.16 | 200 | 480 |
| | Cu-Ag Zone | Az 160 Dip 0 | 0.15 | 0.10 | 0.45 | 100 | 1000 |
| | | Az 70 Dip 0 | 0.15 | 0.10 | 0.45 | 400 | 500 |
| | | Az 0 Dip -90 | 0.15 | 0.10 | 0.45 | 200 | 600 |
| Cu | Cu-Ag and Cu-Mo Zone | Az 60 Dip 0 | 0.10 | 0.10 | 0.15 | 200 | 2000 |
| | | Az 150 Dip -55 | 0.10 | 0.10 | 0.15 | 300 | 1800 |
| | | Az 330 Dip -35 | 0.10 | 0.10 | 0.15 | 100 | 1000 |
| | Mo Zone | Az 60 Dip 0 | 0.05 | 0.20 | 0.17 | 60 | 400 |
| | | Az 150 Dip 0 | 0.05 | 0.20 | 0.17 | 200 | 800 |
| | | Az 0 Dip -90 | 0.05 | 0.20 | 0.17 | 600 | 800 |
| Ag | Cu-Ag and Cu-Mo Zone | Az 70 Dip 0 | 0.10 | 0.05 | 0.13 | 50 | 600 |
| | | Az 160 Dip 0 | 0.10 | 0.05 | 0.13 | 100 | 200 |
| | | Az 0 Dip -90 | 0.10 | 0.05 | 0.13 | 100 | 800 |
| | Mo Zone | Az 60 Dip 0 | 0.10 | 0.10 | 0.25 | 300 | 1100 |
| | | Az 150 Dip 0 | 0.10 | 0.10 | 0.25 | 200 | 600 |
| | | Az 0 Dip -90 | 0.10 | 0.10 | 0.25 | 400 | 600 |
| W | Cu-Mo and Mo Zone | Az 135 Dip 0 | 0.05 | 0.04 | 0.15 | 160 | 1200 |
| | | Az 45 Dip 0 | 0.05 | 0.04 | 0.15 | 100 | 400 |
| | | Az 0 Dip -90 | 0.05 | 0.04 | 0.15 | 300 | 1000 |
| | Cu-Ag Zone | Az 160 Dip 0 | 0.05 | 0.10 | 0.30 | 100 | 1200 |
| | | Az 70 Dip 0 | 0.05 | 0.10 | 0.30 | 80 | 600 |
| | | Az 0 Dip -90 | 0.05 | 0.10 | 0.30 | 200 | 500 |

## 16.4 Block Model

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

a) Lower Left Corner

| | | |
|---|---|---|
| 216,975 E | Column Size – 50 ft | 139 Columns |
| 116,725 N | Row Size – 50 ft | 108 Rows |

b) Top of Model

| | | |
|---|---|---|
| 6550 Elevation | Level Size – 50 ft | 60 Levels |

## 16.5 Grade Interpolation

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

$MoS_2$

- Estimated for Cu-Ag Domain using only composites from Cu-Ag Domain
- Estimated for Cu-Mo and Mo Domains using only composites from Cu-Mo and Mo Domains

Cu

- Estimated for Mo Domain using only composites from Mo Domain
- Estimated for Cu-Ag and Cu-Mo Domains using only composites from Cu-Ag and Cu-Mo Domains

Ag

- Estimated for Mo Domain using only composites from Mo Domain
- Estimated for Cu-Ag and Cu-Mo Domains using only composites from Cu-Ag and Cu-Mo Domains

W

- Estimated for Cu-Ag Domain using only composites from Cu-Ag Domain
- Estimated for Cu-Mo and Mo Domains using only composites from Cu-Mo and Mo Domains

Each kriging run was composed of 4 passes. The dimensions for the search ellipse, within each pass, were a function of the semivariogram range. Pass 1 required a minimum of 4 composites within a search ellipse of dimensions equal to ¼ of the semivariogram range. For blocks not estimated, the search ellipse was expanded to ½ the semivariogram range in pass 2 and again a minimum of 4 composites were required to estimate the block. In cases with a vertical search, for both pass 1 and 2 the vertical search distance was set at 75 ft to insure at least 2 holes were used. Pass 3 expanded the search ellipse to the entire range and a final 4th pass used double the range. In all cases if more than 16 composites were found the closest 16 were used. The search parameters for each run are listed below in Table 28. For Ag and W a fifth pass was used with search ellipses equal to the maximum search in Cu and MoS2, to produce a value for all blocks estimated for $MoS_2$ and Cu. This was due to the under-sampling of Ag and W relative to $MoS_2$ and Cu.

A grade for each of the four variables was estimated in a total of 401,908 blocks.

**Table 28: Summary of Kriging Search Parameters for each Domain**

| Domain | Variable | Pass | Number Of Blocks Estimated | Az/Dip | Dist. (ft) | Az/Dip | Dist. (ft) | Az/Dip | Dist. (ft) |
|---|---|---|---|---|---|---|---|---|---|
| Cu-Ag | $MoS_2$ | 1 | 5,223 | 160/0 | 250 | 70/0 | 125 | 0/-90 | 75 |
| | | 2 | 13,196 | 160/0 | 500 | 70/0 | 250 | 0/-90 | 75 |
| | | 3 | 41,082 | 160/0 | 1,000 | 70/0 | 500 | 0/-90 | 150 |
| | | 4 | 76,551 | 160/0 | 2,000 | 70/0 | 1,000 | 0/-90 | 150 |
| Cu-Mo & Mo | $MoS_2$ | 1 | 68,722 | 60/0 | 450 | 330/-35 | 120 | 150/-55 | 325 |
| | | 2 | 99,930 | 60/0 | 900 | 330/-35 | 240 | 150/-55 | 650 |
| | | 3 | 135,778 | 60/0 | 1,800 | 330/-35 | 480 | 150/-55 | 1,300 |
| Cu-Ag & Cu-Mo | Cu | 1 | 81,725 | 60/0 | 500 | 330/-35 | 250 | 150/-55 | 450 |
| | | 2 | 91,389 | 60/0 | 1,000 | 330/-35 | 500 | 150/-55 | 900 |
| | | 3 | 225,702 | 60/0 | 2,000 | 330/-35 | 1,000 | 150/-55 | 1,800 |
| Mo | Cu | 1 | 26,939 | 60/0 | 450 | 330/0 | 200 | 0/-90 | 75 |
| | | 2 | 41,213 | 60/0 | 900 | 330/0 | 400 | 0/-90 | 75 |
| | | 3 | 61,579 | 60/0 | 1,800 | 330/0 | 800 | 0/-90 | 150 |
| | | 4 | 71,897 | 60/0 | 3,600 | 330/0 | 1,600 | 0/-90 | 150 |
| Cu-Ag & Cu-Mo | Ag | 1 | 6,072 | 70/0 | 250 | 340/0 | 50 | 0/-90 | 75 |
| | | 2 | 17,150 | 70/0 | 500 | 340/0 | 100 | 0/-90 | 75 |
| | | 3 | 60,675 | 70/0 | 1,000 | 340/0 | 200 | 0/-90 | 150 |
| | | 4 | 71,511 | 70/0 | 2,000 | 340/0 | 400 | 0/-90 | 300 |
| | | 5 | 84,502 | 70/0 | 2,000 | 340/0 | 1,000 | 0/-90 | 1,800 |
| Mo | Ag | 1 | 12,855 | 60/0 | 275 | 330/0 | 150 | 0/-90 | 75 |
| | | 2 | 28,108 | 60/0 | 550 | 330/0 | 300 | 0/-90 | 75 |
| | | 3 | 57,161 | 60/0 | 1,100 | 330/0 | 600 | 0/-90 | 150 |
| | | 4 | 46,656 | 60/0 | 2,200 | 330/0 | 1,200 | 0/-90 | 150 |
| | | 5 | 18,341 | 60/0 | 3,600 | 330/0 | 1,600 | 0/-90 | 150 |
| Cu-Ag | W | 1 | 18,754 | 135/0 | 300 | 45/0 | 100 | 0/-90 | 75 |
| | | 2 | 44,677 | 135/0 | 600 | 45/0 | 200 | 0/-90 | 75 |
| | | 3 | 102,762 | 135/0 | 1,200 | 45/0 | 400 | 0/-90 | 150 |
| | | 4 | 64,581 | 135/0 | 2,400 | 45/0 | 800 | 0/-90 | 150 |
| | | 5 | 51,867 | 135/0 | 2,400 | 45/0 | 1,000 | 0/-90 | 150 |
| Cu-Mo & Mo | W | 1 | 7,058 | 160/0 | 300 | 70/0 | 150 | 0/-90 | 75 |
| | | 2 | 23,870 | 160/0 | 600 | 70/0 | 300 | 0/-90 | 75 |
| | | 3 | 42,136 | 160/0 | 1,200 | 70/0 | 600 | 0/-90 | 150 |
| | | 4 | 52,009 | 160/0 | 2,400 | 70/0 | 1,200 | 0/-90 | 150 |
| | | 5 | 7,126 | 160/0 | 2,400 | 70/0 | 1,200 | 0/-90 | 1,300 |

## 16.6 Bulk Density

Specific gravity determinations were made for CUMO for each grade Domain. The measurements were made using the weight in air/weight in water procedure. The results are summarized below in Table 29.

**Table 29 Specific Gravity Determination**

| Domain | Number of SG Determinations | SG Minimum | SG Maximum | Average SG (gm/cc) | Average TF (cu.ft/ton) | Average MoS2 (%) |
|---|---|---|---|---|---|---|
| Cu-Ag | 9 | 2.58 | 2.72 | 2.64 | 12.13 | 0.045 |
| Cu-Mo | 66 | 2.37 | 2.70 | 2.60 | 12.30 | 0.093 |
| Mo | 125 | 2.46 | 2.70 | 2.60 | 12.33 | 0.106 |

The tonnage factor for each block was a weighted average based on the domains tonnage factor and the amount of that domain within the block.

## 16.7 Classification

### 16.7.1 Introduction

Based on the study herein reported, delineated mineralization of the CUMO Property is classified as a resource according to the following definition from National Instrument 43-101

*"In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by CIM Council on August 20, 2000, as those definitions may be amended from time to time by the Canadian Institute of Mining, Metallurgy, and Petroleum."*

*"A **Mineral Resource** is a concentration or occurrence of natural, solid, inorganic or fossilized organic material 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 term Inferred is defined in NI 43-101 as follows:

"An '**Inferred Mineral Resource'** is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes."

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

### 16.7.2 Results

At CUMO geologic continuity has been established through diamond drilling. The concentric zonation and faults have been used to constrain the mineralization in a series of metal zones. Grade continuity within the metal domains has been established by semivariograms. The semivariogram analysis was completed after moving major fault blocks back to pre fault positions. The kriging procedure was completed on fault blocks in their current positions. Blocks estimated in Pass 1 and 2 using search ellipses up to a maximum of ½ the semivariogram range were classified as Indicated. All other blocks were classified as inferred.

To properly evaluate the CUMO Deposit with 4 metals occurring in different zones, a form of metal equivalent or Gross Recoverable Value (GRV) was used. This calculation used metal prices in US dollars and metal recoveries as follows:

$MoS_2$ – Molybdenum is sold as molybdenum trioxide ($MoO_3$) which has higher Mo content. Forecasts are for $MoO_3$ to rise to \$16 in 2010 and to \$20 in 2011 (CPM group, Feb.2009). The Chinese have stated that they will not be selling their $MoO_3$ for less than \$15/lb due to their production costs. The price used in this study for $MoO_3$ is \$15/lb. $MoO_3$ is calculated from $MoS_2$ by the following: Pounds Mo = $MoS_2$ * 20 / 1.6681 and then Pounds $MoO_3$ = Pounds Mo * 1.5

- Cu – A copper price of \$1.50 / lb was used
- Ag – A silver price of \$12.00 / oz was used
- W – A tungsten price of \$8.50 / lb was used

The metal recoveries used were a function of metal domains as follows:

**Table 30: Metal Recoveries for GRV Calculation**

| | %Recoveries in Oxides | %Recoveries in Cu-Ag Domain | %Recoveries in Cu-Mo Domain | %Recoveries in Mo Domain |
|---|---|---|---|---|
| Cu | 60.0 | 68.0 | 87.0 | 80.0 |
| Ag | 70.0 | 73.0 | 78.0 | 55.0 |
| W | 35.0 | 35.0 | 35.0 | 35.0 |
| Mo | 80.0 | 85.0 | 92.0 | 95.0 |

The equations to calculated GRV for each Domain were as follows:

GRV (oxides) = (Cu% * 18.0) + (Ag(g/t) * 0.25) + (W% * 0.01) + ($MoS_2$ * 215.81)
GRV (Cu-Ag) = (Cu% * 20.4) + (Ag(g/t) * 0.26) + (W% * 0.01) + ($MoS_2$ * 229.30)
GRV (Cu-Mo) = (Cu% * 26.1) + (Ag(g/t) * 0.27) + (W% * 0.01) + ($MoS_2$ * 248.19)
GRV (Mo) = (Cu% * 24.0) + (Ag(g/t) * 0.19) + (W% * 0.01) + ($MoS_2$ * 256.28)

For Blocks overlapping the domain boundaries a weighted average GRV was produced.

At the time the GRV calculations were completed, no economic evaluation had been undertaken, so an economic cutoff was unknown. A value in the non oxide material of US\$7.50 was highlighted as a possible open pit cutoff, based on similar size mines currently at the feasibility (Mt. Hope and Creston feasibility studies) or production (cost data from Producing Mines obtained from Mine Cost Data Models: www.minecost.com) (refer to Table 31 for details).

**Table 31: Operating Costs from Comparable Operations**

| Mine | Tons per Day | Operating Costs US$/t |
|---|---|---|
| Bingham | 163 000 | 6.25 |
| Sierrita | 143 000 | 4.53 |
| Mercator – Mineral Park | 50 000 | 4.57 |
| Baghdad | 75 000 | 5.10 |
| Mt. Hope | 60 625 | 6.81 |
| Thompson Creek | 25 000 | 11.63 |
| Highland Valley | 118 314 | 6.10 |
| Morenci – SXEW | 90 000 | 2.97 |
| Morenci – all | 203 314 | 4.75 |
| Morenci - sulfide | 68 000 | 16.08 |

The CUMO Resources is reported first for the oxide portion of the deposit in Table 32 and Table 33.

## Table 32: CUMO Oxide Domain - Indicated Resource

| Cutoff | Tons > Cutoff | Grade > Cutoff | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| GRV | (million tons) | MoS2 | Cu | Ag | W | GRV | Million | Million | Million | Million | Million |
| ($US) | | (%) | (%) | (g/t) | (ppm) | US$ | lbs. Mo | lbs MoO3 | lbs Cu | oz Ag | lbs W |
| 5.00 | 130 | 0.03 | 0.10 | 2.8 | 38 | 10 | 53 | 79 | 270 | 11 | 9.7 |
| 6.00 | 100 | 0.04 | 0.11 | 2.9 | 40 | 11 | 49 | 73 | 230 | 8.8 | 8.3 |
| 7.00 | 88 | 0.04 | 0.11 | 2.9 | 42 | 12 | 45 | 68 | 200 | 7.5 | 7.4 |
| 7.50 | 81 | 0.05 | 0.11 | 2.9 | 43 | 13 | 44 | 66 | 182 | 7.0 | 7.0 |
| 8.00 | 78 | 0.05 | 0.11 | 2.9 | 43 | 13 | 43 | 64 | 170 | 6.6 | 6.7 |
| 9.00 | 66 | 0.05 | 0.11 | 2.9 | 45 | 14 | 39 | 59 | 150 | 5.6 | 5.9 |
| 10.00 | 58 | 0.05 | 0.11 | 2.9 | 46 | 15 | 37 | 55 | 130 | 4.9 | 5.3 |
| 12.00 | 39 | 0.06 | 0.12 | 3.0 | 49 | 16 | 28 | 42 | 93 | 3.4 | 3.8 |
| 12.50 | 34 | 0.06 | 0.12 | 3.0 | 50 | 17 | 26 | 38 | 84 | 3.0 | 3.4 |
| 13.00 | 30 | 0.06 | 0.12 | 3.1 | 51 | 17 | 24 | 35 | 76 | 2.7 | 3.1 |
| 14.00 | 26 | 0.07 | 0.13 | 3.0 | 52 | 18 | 21 | 31 | 64 | 2.3 | 2.7 |
| 15.00 | 20 | 0.07 | 0.12 | 3.0 | 54 | 19 | 18 | 26 | 51 | 1.8 | 2.2 |
| 17.00 | 14 | 0.08 | 0.12 | 2.9 | 56 | 20 | 13 | 20 | 35 | 1.2 | 1.6 |
| 19.00 | 7.8 | 0.09 | 0.13 | 3.1 | 59 | 22 | 7.9 | 12 | 20 | 0.7 | 0.9 |
| 20.00 | 5.5 | 0.09 | 0.13 | 3.1 | 60 | 23 | 5.9 | 8.9 | 14 | 0.5 | 0.7 |
| 25.00 | 1.0 | 0.11 | 0.13 | 3.2 | 60 | 27 | 1.4 | 2.0 | 2.7 | 0.1 | 0.1 |

## Table 33: CUMO Oxide Domain - Inferred Resource

| Cutoff | Tons > Cutoff | Grade > Cutoff | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| GRV | (million tons) | MoS2 | Cu | Ag | W | GRV | Million | Million | Million | Million | Million |
| ($US) | | (%) | (%) | (g/t) | (ppm) | US$ | lbs. Mo | lbs MoO3 | lbs Cu | oz Ag | lbs W |
| 5.00 | 220 | 0.02 | 0.11 | 2.8 | 31 | 7 | 49 | 74 | 460 | 18 | 14 |
| 6.00 | 140 | 0.02 | 0.12 | 3.0 | 32 | 8 | 36 | 53 | 320 | 12 | 8.5 |
| 7.00 | 77 | 0.03 | 0.13 | 3.2 | 33 | 9 | 24 | 36 | 190 | 7.1 | 5.0 |
| 7.50 | 59 | 0.03 | 0.13 | 3.2 | 33 | 9 | 20 | 30 | 150 | 5.5 | 3.9 |
| 8.00 | 47 | 0.03 | 0.13 | 3.3 | 34 | 10 | 17 | 25 | 120 | 4.4 | 3.1 |
| 9.00 | 30 | 0.03 | 0.13 | 3.3 | 34 | 11 | 12 | 18 | 77 | 2.9 | 2.0 |
| 10.00 | 17 | 0.04 | 0.13 | 3.3 | 35 | 12 | 7.5 | 11 | 44 | 1.6 | 1.2 |
| 12.00 | 5 | 0.05 | 0.13 | 2.9 | 38 | 14 | 2.9 | 4.4 | 13 | 0.42 | 0.38 |
| 12.50 | 3.7 | 0.05 | 0.12 | 2.7 | 40 | 15 | 2.4 | 3.5 | 9.0 | 0.29 | 0.29 |
| 13.00 | 3 | 0.06 | 0.12 | 2.7 | 40 | 15 | 2.0 | 3.0 | 7.3 | 0.23 | 0.24 |
| 14.00 | 2 | 0.06 | 0.12 | 2.6 | 42 | 16 | 1.4 | 2.1 | 4.8 | 0.15 | 0.17 |
| 15.00 | 1.2 | 0.06 | 0.12 | 2.5 | 45 | 17 | 0.9 | 1.4 | 2.8 | 0.09 | 0.11 |
| 17.00 | 0.3 | 0.07 | 0.11 | 2.4 | 48 | 18 | 0.30 | 0.4 | 0.60 | 0.02 | 0.03 |
| 19.00 | 0.1 | 0.08 | 0.11 | 2.5 | 52 | 20 | 0.10 | 0.1 | 0.20 | 0.01 | 0.01 |
| 20.00 | 0.1 | 0.08 | 0.11 | 2.3 | 53 | 21 | 0.10 | 0.1 | 0.20 | 0.01 | 0.01 |

The non oxide portion of the deposit is reported in Table 34 and Table 35.
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**Table 34: CUMO Non Oxide Domains - Indicated Resource**

| Cutoff | Tons > Cutoff | Grade > Cutoff | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| GRV | (million tons) | MoS2 | Cu | Ag | W | GRV | Million | Million | Million | Million | Million |
| ($US) | | (%) | (%) | (g/t) | (ppm) | US$ | lbs. Mo | lbs MoO3 | lbs Cu | oz Ag | lbs W |
| 5.00 | 1300 | 0.08 | 0.07 | 2.2 | 46 | 22 | 1200 | 1700 | 1800 | 80 | 56 |
| 6.00 | 1300 | 0.08 | 0.07 | 2.2 | 46 | 23 | 1200 | 1700 | 1800 | 79 | 56 |
| 7.00 | 1200 | 0.08 | 0.07 | 2.2 | 46 | 23 | 1100 | 1700 | 1700 | 76 | 55 |
| **7.50** | **1200** | **0.08** | **0.07** | **2.2** | **46** | **24** | **1100** | **1700** | **1700** | **74** | **55** |
| 8.00 | 1200 | 0.08 | 0.07 | 2.2 | 46 | 24 | 1100 | 1700 | 1700 | 73 | 54 |
| 9.00 | 1100 | 0.08 | 0.07 | 2.2 | 46 | 24 | 1100 | 1700 | 1600 | 72 | 54 |
| 10.00 | 1100 | 0.09 | 0.07 | 2.2 | 47 | 24 | 1100 | 1700 | 1600 | 70 | 54 |
| 12.00 | 1100 | 0.09 | 0.07 | 2.2 | 47 | 25 | 1100 | 1600 | 1500 | 64 | 52 |
| 12.50 | 1100 | 0.09 | 0.07 | 2.1 | 47 | 26 | 1100 | 1600 | 1400 | 63 | 52 |
| 13.00 | 990 | 0.09 | 0.07 | 2.1 | 47 | 26 | 1100 | 1600 | 1400 | 61 | 51 |
| 14.00 | 940 | 0.09 | 0.07 | 2.1 | 48 | 26 | 1100 | 1600 | 1300 | 58 | 50 |
| 15.00 | 910 | 0.10 | 0.07 | 2.1 | 48 | 27 | 1000 | 1600 | 1200 | 55 | 49 |
| 17.00 | 810 | 0.10 | 0.07 | 2.0 | 48 | 28 | 980 | 1500 | 1100 | 48 | 46 |
| 19.00 | 710 | 0.11 | 0.07 | 2.0 | 48 | 30 | 910 | 1400 | 920 | 41 | 42 |
| 20.00 | 660 | 0.11 | 0.06 | 2.0 | 48 | 30 | 870 | 1300 | 840 | 37 | 40 |
| 25.00 | 460 | 0.12 | 0.06 | 1.9 | 49 | 34 | 680 | 1000 | 550 | 25 | 31 |

**Table 35: CUMO Non Oxide Domains - Inferred Resource**

| Cutoff | Tons > Cutoff | Grade > Cutoff | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| GRV | (million tons) | MoS2 | Cu | Ag | W | GRV | Million | Million | Million | Million | Million |
| ($US) | | (%) | (%) | (g/t) | (ppm) | US$ | lbs. Mo | lbs MoO3 | lbs Cu | oz Ag | lbs W |
| 5.00 | 2100 | 0.06 | 0.07 | 2.1 | 36 | 18 | 1500 | 2200 | 2900 | 130 | 71 |
| 6.00 | 2000 | 0.06 | 0.07 | 2.1 | 36 | 18 | 1400 | 2200 | 2800 | 120 | 70 |
| 7.00 | 1700 | 0.07 | 0.07 | 2.1 | 35 | 20 | 1400 | 2100 | 2400 | 110 | 67 |
| **7.50** | 1600 | **0.07** | **0.07** | **2.1** | **35** | **20** | **1400** | **2100** | **2300** | **99** | **66** |
| 8.00 | 1600 | 0.07 | 0.07 | 2.1 | 36 | 21 | 1400 | 2000 | 2100 | 94 | 65 |
| 9.00 | 1400 | 0.08 | 0.07 | 2.1 | 36 | 22 | 1300 | 2000 | 1900 | 87 | 63 |
| 10.00 | 1400 | 0.08 | 0.06 | 2.0 | 36 | 22 | 1300 | 2000 | 1800 | 82 | 62 |
| 12.00 | 1300 | 0.08 | 0.06 | 2.0 | 37 | 23 | 1300 | 1900 | 1600 | 77 | 60 |
| 12.50 | 1200 | 0.08 | 0.06 | 2.0 | 37 | 23 | 1300 | 1900 | 1600 | 75 | 60 |
| 13.00 | 1200 | 0.08 | 0.06 | 2.0 | 37 | 23 | 1300 | 1900 | 1600 | 74 | 59 |
| 14.00 | 1300 | 0.08 | 0.06 | 2.0 | 37 | 24 | 1200 | 1900 | 1500 | 71 | 58 |
| 15.00 | 1200 | 0.09 | 0.06 | 2.0 | 37 | 24 | 1200 | 1800 | 1400 | 67 | 57 |
| 17.00 | 1100 | 0.09 | 0.06 | 1.9 | 36 | 25 | 1100 | 1700 | 1200 | 60 | 53 |
| 19.00 | 890 | 0.10 | 0.06 | 2.0 | 36 | 26 | 1000 | 1500 | 1100 | 51 | 47 |
| 20.00 | 820 | 0.10 | 0.06 | 2.0 | 36 | 27 | 950 | 1400 | 1000 | 48 | 44 |
| 25.00 | 490 | 0.11 | 0.06 | 2.1 | 36 | 30 | 640 | 950 | 600 | 30 | 29 |

The Non Oxide Resource can also be broken down into individual Domains.

**Table 36: CUMO CU-AG Domain - Non Oxide Indicated Resource**

| Cutoff | Tons > Cutoff | Grade > Cutoff | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| GRV | (million tons) | MoS2 | Cu | Ag | W | GRV | Million | Million | Million | Million | Million |
| ($US) | | (%) | (%) | (g/t) | (ppm) | US$ | lbs. Mo | lbs MoO3 | lbs Cu | oz Ag | lbs W |
| 5.00 | 67 | 0.02 | 0.11 | 3.0 | 29 | 9 | 18 | 27 | 150 | 6 | 1.1 |
| 6.00 | 56 | 0.03 | 0.11 | 3.1 | 30 | 9 | 17 | 25 | 130 | 5 | 1.0 |
| 7.00 | 43 | 0.03 | 0.12 | 3.3 | 30 | 10 | 14 | 21 | 100 | 4 | 0.9 |
| **7.50** | **37** | **0.03** | **0.12** | **3.3** | **30** | **10** | **13** | **19** | **86** | **4** | **0.8** |
| 8.00 | 31 | 0.03 | 0.12 | 3.3 | 30 | 11 | 11 | 17 | 72 | 3 | 0.7 |
| 9.00 | 21 | 0.04 | 0.12 | 3.5 | 30 | 12 | 8.8 | 13 | 48 | 2 | 0.5 |
| 10.00 | 14 | 0.04 | 0.12 | 3.7 | 31 | 14 | 6.9 | 10 | 34 | 2 | 0.4 |
| 12.00 | 6.7 | 0.05 | 0.13 | 4.1 | 35 | 17 | 4.2 | 6.3 | 17 | 0.8 | 0.2 |
| 12.50 | 5.8 | 0.06 | 0.13 | 4.1 | 35 | 17 | 3.8 | 5.7 | 15 | 0.7 | 0.2 |
| 13.00 | 5.0 | 0.06 | 0.13 | 4.1 | 36 | 18 | 3.5 | 5.2 | 13 | 0.6 | 0.2 |
| 14.00 | 4.0 | 0.06 | 0.13 | 3.9 | 37 | 19 | 3.0 | 4.5 | 10 | 0.5 | 0.2 |
| 15.00 | 3.2 | 0.07 | 0.13 | 3.9 | 37 | 21 | 2.6 | 3.9 | 8.2 | 0.4 | 0.1 |
| 17.00 | 2.1 | 0.08 | 0.12 | 3.8 | 37 | 23 | 2.0 | 2.9 | 5.2 | 0.2 | 0.1 |
| 19.00 | 1.4 | 0.09 | 0.12 | 3.6 | 36 | 25 | 1.5 | 2.2 | 3.2 | 0.2 | 0.1 |
| 20.00 | 1.2 | 0.10 | 0.11 | 3.6 | 36 | 27 | 1.4 | 2.1 | 2.7 | 0.1 | 0.1 |
| 25.00 | 0.7 | 0.11 | 0.10 | 3.3 | 33 | 30 | 0.9 | 1.4 | 1.4 | 0.1 | 0.0 |

**Table 37: CUMO CU-AG Domain - Non Oxide Inferred Resource**

| Cutoff | Tons > Cutoff | Grade > Cutoff | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| GRV | (million tons) | MoS2 | Cu | Ag | W | GRV | Million | Million | Million | Million | Million |
| ($US) | | (%) | (%) | (g/t) | (ppm) | US$ | lbs. Mo | lbs MoO3 | lbs Cu | oz Ag | lbs W |
| 5.00 | 570 | 0.02 | 0.10 | 2.6 | 31 | 8 | 140 | 220 | 1100 | 43 | 8.8 |
| 6.00 | 460 | 0.02 | 0.10 | 2.7 | 30 | 8 | 120 | 180 | 960 | 36 | 7.6 |
| 7.00 | 310 | 0.03 | 0.12 | 2.8 | 28 | 9 | 92 | 140 | 700 | 25 | 5.6 |
| **7.50** | **230** | **0.03** | **0.12** | **2.9** | **27** | **10** | **76** | **110** | **570** | **20** | **4.5** |
| 8.00 | 180 | 0.03 | 0.13 | 3.1 | 26 | 10 | 61 | 91 | 440 | 16 | 3.6 |
| 9.00 | 100 | 0.04 | 0.12 | 3.2 | 26 | 12 | 42 | 63 | 250 | 9.3 | 2.4 |
| 10.00 | 51 | 0.04 | 0.13 | 3.5 | 27 | 14 | 26 | 39 | 130 | 5.2 | 1.4 |
| 12.00 | 20 | 0.06 | 0.12 | 3.7 | 29 | 19 | 15 | 22 | 47 | 2.1 | 0.7 |
| 12.50 | 18 | 0.07 | 0.12 | 3.6 | 29 | 20 | 14 | 21 | 40 | 1.8 | 0.7 |
| 13.00 | 16 | 0.07 | 0.11 | 3.6 | 30 | 21 | 13 | 19 | 34 | 1.6 | 0.6 |
| 14.00 | 11 | 0.08 | 0.10 | 3.2 | 30 | 23 | 11 | 17 | 23 | 1.1 | 0.5 |
| 15.00 | 11 | 0.09 | 0.10 | 3.2 | 30 | 24 | 11 | 16 | 21 | 1.0 | 0.5 |
| 17.00 | 8.5 | 0.09 | 0.10 | 3.3 | 30 | 26 | 10 | 14 | 18 | 0.8 | 0.4 |
| 19.00 | 7.1 | 0.10 | 0.11 | 3.4 | 30 | 27 | 9 | 13 | 15 | 0.7 | 0.4 |
| 20.00 | 6.3 | 0.10 | 0.11 | 3.5 | 30 | 28 | 8 | 12 | 14 | 0.6 | 0.4 |
| 25.00 | 4.3 | 0.12 | 0.11 | 3.6 | 29 | 31 | 6 | 8.9 | 10 | 0.5 | 0.3 |

### Table 38: CUMO CU-MO Domain - Non Oxide Indicated Resource

| Cutoff | Tons > Cutoff | Grade > Cutoff | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| GRV | (million tons) | MoS2 | Cu | Ag | W | GRV | Million | Million | Million | Million | Million |
| ($US) | | (%) | (%) | (g/t) | (ppm) | US$ | lbs. Mo | lbs MoO3 | lbs Cu | oz Ag | lbs W |
| 5.00 | 570 | 0.06 | 0.11 | 3.0 | 47 | 18 | 350 | 530 | 1100 | 46 | 19 |
| 6.00 | 460 | 0.06 | 0.11 | 3.0 | 47 | 18 | 350 | 530 | 1100 | 45 | 19 |
| 7.00 | 310 | 0.06 | 0.11 | 3.0 | 47 | 19 | 340 | 520 | 1100 | 43 | 18 |
| **7.50** | **230** | **0.06** | **0.11** | **3.1** | **47** | **19** | **340** | **510** | **1000** | **43** | **18** |
| 8.00 | 180 | 0.06 | 0.11 | 3.1 | 47 | 19 | 340 | 510 | 1000 | 42 | 18 |
| 9.00 | 100 | 0.06 | 0.11 | 3.1 | 47 | 19 | 340 | 510 | 1000 | 41 | 18 |
| 10.00 | 51 | 0.06 | 0.11 | 3.1 | 47 | 20 | 330 | 500 | 1000 | 40 | 18 |
| 12.00 | 20 | 0.07 | 0.11 | 3.1 | 48 | 21 | 310 | 470 | 900 | 36 | 16 |
| 12.50 | 18 | 0.07 | 0.11 | 3.1 | 49 | 21 | 310 | 460 | 870 | 35 | 16 |
| 13.00 | 16 | 0.07 | 0.11 | 3.1 | 49 | 21 | 310 | 460 | 840 | 34 | 16 |
| 14.00 | 11 | 0.07 | 0.12 | 3.2 | 49 | 22 | 290 | 430 | 780 | 31 | 15 |
| 15.00 | 11 | 0.07 | 0.12 | 3.2 | 50 | 23 | 280 | 410 | 720 | 29 | 14 |
| 17.00 | 8.5 | 0.08 | 0.12 | 3.2 | 50 | 24 | 240 | 370 | 600 | 24 | 12 |
| 19.00 | 7.1 | 0.09 | 0.12 | 3.2 | 51 | 26 | 210 | 310 | 460 | 18 | 10 |
| 20.00 | 6.3 | 0.09 | 0.12 | 3.2 | 51 | 27 | 190 | 280 | 400 | 16 | 9.2 |
| 25.00 | 4.3 | 0.11 | 0.12 | 3.2 | 52 | 31 | 120 | 180 | 220 | 9.0 | 5.9 |

### Table 39: CUMO CU-MO Domain - Non Oxide Inferred Resource

| Cutoff | Tons > Cutoff | Grade > Cutoff | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| GRV | (million tons) | MoS2 | Cu | Ag | W | GRV | Million | Million | Million | Million | Million |
| ($US) | | (%) | (%) | (g/t) | (ppm) | US$ | lbs. Mo | lbs MoO3 | lbs Cu | oz Ag | lbs W |
| 5.00 | 590 | 0.07 | 0.10 | 2.9 | 40 | 20 | 460 | 680 | 1200 | 49 | 23 |
| 6.00 | 580 | 0.07 | 0.10 | 2.9 | 40 | 20 | 460 | 680 | 1200 | 49 | 23 |
| 7.00 | 530 | 0.07 | 0.11 | 3.0 | 38 | 21 | 450 | 670 | 1100 | 46 | 23 |
| **7.50** | **520** | **0.07** | **0.11** | **3.0** | **38** | **22** | **450** | **670** | **1100** | **45** | **22** |
| 8.00 | 510 | 0.07 | 0.11 | 3.0 | 37 | 22 | 440 | 670 | 1100 | 45 | 22 |
| 9.00 | 510 | 0.07 | 0.11 | 3.0 | 37 | 22 | 450 | 670 | 1100 | 44 | 22 |
| 10.00 | 500 | 0.07 | 0.11 | 3.0 | 37 | 22 | 440 | 660 | 1100 | 44 | 22 |
| 12.00 | 490 | 0.08 | 0.11 | 3.0 | 37 | 23 | 430 | 650 | 1000 | 43 | 22 |
| 12.50 | 470 | 0.08 | 0.11 | 3.0 | 37 | 23 | 430 | 650 | 1000 | 42 | 22 |
| 13.00 | 470 | 0.08 | 0.11 | 3.0 | 37 | 23 | 430 | 650 | 1000 | 41 | 21 |
| 14.00 | 450 | 0.08 | 0.11 | 3.0 | 37 | 23 | 420 | 630 | 950 | 39 | 21 |
| 15.00 | 420 | 0.08 | 0.11 | 3.0 | 38 | 24 | 400 | 610 | 900 | 37 | 20 |
| 17.00 | 370 | 0.09 | 0.11 | 3.0 | 37 | 25 | 370 | 550 | 780 | 32 | 18 |
| 19.00 | 300 | 0.09 | 0.11 | 3.1 | 37 | 27 | 330 | 400 | 670 | 27 | 16 |
| 20.00 | 290 | 0.09 | 0.11 | 3.1 | 37 | 27 | 310 | 470 | 630 | 26 | 15 |
| 25.00 | 180 | 0.10 | 0.11 | 3.1 | 36 | 30 | 220 | 340 | 400 | 17 | 11 |

## Table 40: CUMO MO Domain - Non Oxide Indicated Resource

| Cutoff | Tons > Cutoff | Grade > Cutoff | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| GRV | (million tons) | MoS2 | Cu | Ag | W | GRV | Million | Million | Million | Million | Million |
| ($US) | | (%) | (%) | (g/t) | (ppm) | US$ | lbs. Mo | lbs MoO3 | lbs Cu | oz Ag | lbs W |
| 5.00 | 640 | 0.10 | 0.04 | 1.5 | 47 | 28 | 790 | 1200 | 570 | 28 | 36 |
| 6.00 | 640 | 0.10 | 0.04 | 1.5 | 47 | 28 | 790 | 1200 | 570 | 28 | 36 |
| 7.00 | 640 | 0.10 | 0.04 | 1.5 | 47 | 28 | 780 | 1200 | 560 | 28 | 36 |
| **7.50** | **640** | **0.10** | **0.04** | **1.5** | **47** | **28** | **780** | **1200** | **560** | **28** | **36** |
| 8.00 | 640 | 0.10 | 0.04 | 1.5 | 47 | 28 | 780 | 1200 | 560 | 28 | 36 |
| 9.00 | 640 | 0.10 | 0.04 | 1.5 | 47 | 28 | 790 | 1200 | 560 | 28 | 36 |
| 10.00 | 630 | 0.10 | 0.04 | 1.5 | 47 | 28 | 780 | 1200 | 560 | 28 | 36 |
| 12.00 | 620 | 0.10 | 0.04 | 1.5 | 47 | 28 | 770 | 1200 | 550 | 27 | 35 |
| 12.50 | 610 | 0.11 | 0.04 | 1.5 | 47 | 29 | 770 | 1200 | 540 | 27 | 35 |
| 13.00 | 610 | 0.11 | 0.04 | 1.5 | 47 | 29 | 770 | 1200 | 540 | 27 | 35 |
| 14.00 | 600 | 0.11 | 0.04 | 1.5 | 47 | 29 | 760 | 1100 | 530 | 26 | 35 |
| 15.00 | 590 | 0.11 | 0.04 | 1.5 | 47 | 29 | 760 | 1100 | 510 | 26 | 34 |
| 17.00 | 550 | 0.11 | 0.04 | 1.5 | 47 | 30 | 740 | 1100 | 490 | 24 | 33 |
| 19.00 | 510 | 0.12 | 0.05 | 1.5 | 47 | 31 | 700 | 1100 | 460 | 22 | 32 |
| 20.00 | 490 | 0.12 | 0.05 | 1.5 | 47 | 32 | 680 | 1000 | 440 | 21 | 31 |
| 25.00 | 360 | 0.13 | 0.05 | 1.5 | 48 | 35 | 560 | 840 | 330 | 16 | 25 |

## Table 41: CUMO MO Domain - Non Oxide Inferred Resource

| Cutoff | Tons > Cutoff | Grade > Cutoff | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| GRV | (million tons) | MoS2 | Cu | Ag | W | GRV | Million | Million | Million | Million | Million |
| ($US) | | (%) | (%) | (g/t) | (ppm) | US$ | lbs. Mo | lbs MoO3 | lbs Cu | oz Ag | lbs W |
| 5.00 | 870 | 0.08 | 0.03 | 1.4 | 37 | 22 | 850 | 1300 | 590 | 34 | 39 |
| 6.00 | 870 | 0.08 | 0.03 | 1.4 | 37 | 22 | 850 | 1300 | 590 | 34 | 39 |
| 7.00 | 860 | 0.08 | 0.03 | 1.4 | 37 | 22 | 850 | 1300 | 590 | 34 | 39 |
| **7.50** | **860** | **0.08** | **0.03** | **1.4** | **36** | **23** | **850** | **1300** | **580** | **34** | **39** |
| 8.00 | 860 | 0.08 | 0.03 | 1.4 | 36 | 23 | 850 | 1300 | 580 | 34 | 39 |
| 9.00 | 850 | 0.08 | 0.03 | 1.4 | 36 | 23 | 850 | 1300 | 580 | 34 | 39 |
| 10.00 | 840 | 0.08 | 0.03 | 1.4 | 36 | 23 | 840 | 1300 | 570 | 33 | 38 |
| 12.00 | 810 | 0.09 | 0.03 | 1.4 | 36 | 23 | 830 | 1300 | 540 | 32 | 38 |
| 12.50 | 800 | 0.09 | 0.03 | 1.4 | 36 | 23 | 820 | 1200 | 530 | 31 | 37 |
| 13.00 | 790 | 0.09 | 0.03 | 1.4 | 36 | 24 | 820 | 1200 | 520 | 31 | 37 |
| 14.00 | 770 | 0.09 | 0.03 | 1.4 | 36 | 24 | 810 | 1200 | 510 | 30 | 37 |
| 15.00 | 750 | 0.09 | 0.03 | 1.4 | 36 | 24 | 800 | 1200 | 490 | 29 | 36 |
| 17.00 | 690 | 0.09 | 0.03 | 1.4 | 36 | 25 | 760 | 1200 | 450 | 27 | 34 |
| 19.00 | 580 | 0.10 | 0.03 | 1.4 | 36 | 26 | 670 | 1000 | 380 | 23 | 30 |
| 20.00 | 540 | 0.10 | 0.03 | 1.4 | 36 | 27 | 630 | 940 | 350 | 21 | 28 |
| 25.00 | 310 | 0.11 | 0.03 | 1.4 | 36 | 30 | 410 | 610 | 200 | 13 | 18 |

# 17 OTHER RELEVANT DATA AND INFORMATION

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.

## 17.1 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.

### 17.1.1 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 42). 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 42: 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.

### 17.1.2 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.

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

**Table 43: 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 1-17 ≥ $22.50 |
| | | Yrs 18-40 ≥$10 <$20.00 | Yrs 18-40 ≥ $20.00 |
| 100 kt/d | <$7.50 | Yrs 1-9 ≥$7.50 <$22.50 | Yrs 1-9 ≥ $22.50 |
| | | Yrs 10-40 ≥$7.50 <$20.00 | Yrs 10-40 ≥ $20.00 |
| 150 kt/d | <$7.50 | Yrs 1-6 ≥$7.50 <$22.50 | Yrs 1-6 ≥ $22.50 |
| | | Yrs 7-40 ≥$7.50 <$20.00 | Yrs 7-40 ≥ $20.00 |
| 200 kt/d | <$7.50 | Yrs 1-6 ≥$7.50 <$22.50 | Yrs 1-6 ≥ $22.50 |
| | | Yrs 7-40 ≥$7.50 <$20.00 | 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.

### 17.2 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.

### 17.3 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.

## 17.4 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.

## 17.5 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.

## 17.6 Taxes and Royalties

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

## 17.7 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 44 and discussed in detail in sections below.

**Table 44: Summary of Initial Capital Costs**

| Capital Cost | | Design | | | |
|---|---|---|---|---|---|
| | | 50 kt/d (short tons) | 100 kt/d (short tons) | 150 kt/d (short tons) | 200 kt/d (short tons) |
| Plant capital | $USM | 590 | 1 000 | 1 500 | 2 900 |
| Roaster capital | $USM | 120 | 200 | 270 | 350 |
| Mining fleet capital | $USM | 100 | 200 | 270 | 270 |
| Preproduction costs (inc Prestrip) | $USM | 750 | 700 | 640 | 660 |
| Tailings | $USM | 40 | 80 | 80 | 160 |
| **Total Initial Capital** | **$USM** | **1 600** | **2 200** | **2 800** | **3 400** |

## 17.8 Mining Capital Costs

### 17.8.1 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.

### 17.8.2 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;

1) Loading, turning and dumping times were assumed to aggregate 7 minutes total for all four production scenarios;
2) Haul speeds from the pit to the destination:
    a) 15 mph from the mining face to the pit ramp;
    b) 8 mph up the ramp;
    c) 15 mph from the pit edge to the final destination;
3) 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;
4) Return times were calculated at an assumed speed of 15 mph;
5) 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:

1) The maximum number of haul units was determined based on the conceptual haul profiles;
2) It was assumed the maximum number of haul units would not be required until year 5 of the production schedule;
3) The useful life for the equipment was assumed as follows:
    a) Cable Shovels – 20 years;
    b) Haul Trucks – 11 years;
    c) Rotary Drills – 10 years;
    d) Bulldozers, Graders, Water Tankers – 12 years;
    e) All other equipment except pumps – 7 years;
    f) Pumps – 2.5 years;

Table 45 to Table 48 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 45: 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 46: 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 47: 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 48: 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** |

### 17.8.3 Non-Equipment Capital Costs

Table 49 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 49: 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 50: 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 50, 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.

1) 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;
2) 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.

## 17.9 Process Plant Capital Costs

A summary of the estimated capital cost for the processing plant and on-site ancillary facilities is provided in Table 51 and Table 52 for the roaster and ancillary facilities, which exclude any escalation or foreign currency fluctuations and are current day costs only (3Q2009).

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 51: Summary of Plant Capital Cost Estimate [3]**

| **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** |

[3] See body of document for scope and battery limits

**Table 52: 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.

### 17.9.1 Assumptions

#### a) 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.

#### b) 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

**c) 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.

**d) 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.

### 17.9.2 Contingency

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

### 17.9.3 Owner's Costs

Owner's costs have been excluded from this estimate.

### 17.9.4 Project Fee

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

### 17.9.5 Escalation

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

## 17.10 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 53.

**Table 53: 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 53 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.

## 17.11 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.

## 17.12 Operating Cost Estimate

The total project operating costs for the different throughput options are summarised in Table 54. The costs are presented as Life of Mine (LOM) averages per short ton of ore processed.

**Table 54: 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.

### 17.12.1 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 55 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 55 shows the amount of material moved for each scenario for the LOM. Based on a 40 year mine life

[4] see Section 17.12.1 for detailed explanation

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.

**Table 55: 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 56: 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 57 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 57: 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.

### 17.12.2 Mining Operating Cost Comparison

Overall average annual mining operating costs are approximately US$80M (Table 57), 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.

### 17.12.3 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 58. 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 58: 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** |

#### a) 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.

#### b) Power

Power is to be supplied to the mine site from the local power grid, provided by Idaho Power. Unit power cost rates have been supplied by Mosquito at US$0.063/kWh, based on information from the Thompson Creek Mine (Thompson Creek Mine Model, *MineCost* (2009)).

#### c) Maintenance Consumables and Services

Maintenance consumable costs were estimated as a percentage of the direct installed capital cost (percent factor). The factor is based on actual data from similar projects and takes into consideration an assumed bond abrasion index of 0.25.

#### d) Reagents and Consumables

Reagent consumptions have been estimated from metallurgical test work or comparable operations. Although reagent consumptions will vary according to metallurgical and production parameters, the average predicted consumptions, by 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.

### 17.13 Economic Analysis

Economic analysis spreadsheets are included as Appendix 4 for each of the individual throughput options considered in this report. 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 59.

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 59: 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$/t con | - to roaster | 5.44 |
| Molybdenum transport cost,<br>US$/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

| 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 |

### 17.13.1 Economic Analysis (Base Case)

The base case economic analysis, based on the estimates of capital and operating costs and assumptions as listed in Table 59 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 60: 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** |

### 17.13.2 Sensitivity analysis (Metal Prices)

A basic sensitivity analysis was conducted on the economic effects of various metal price scenarios. The following Table 61 shows a matrix of the various metal prices used in the scenarios analysed.

**Table 61: 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 61, 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 62.

[6] Base Case

**Table 62: 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 63.

**Table 63: 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 64.

**Table 64: 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 |

### 17.13.3 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 65:

**Table 65: 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 22 and Figure 23 for the 50 kt/d (short ton) throughput option.

**Figure 22: 50 kt/d Throughput IRR Sensitivity**

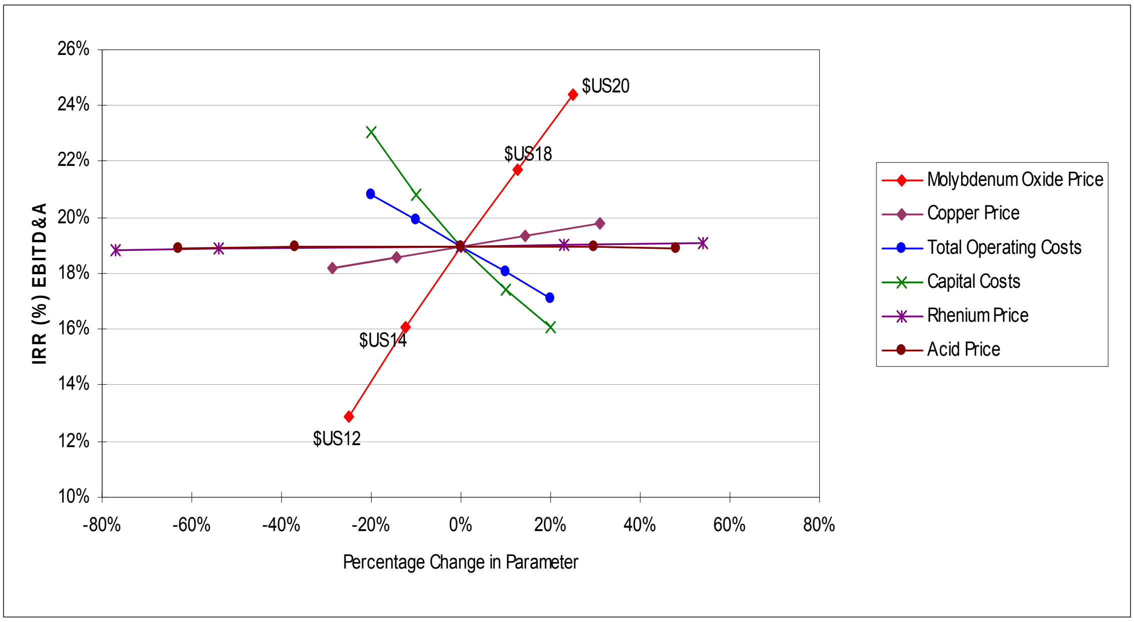
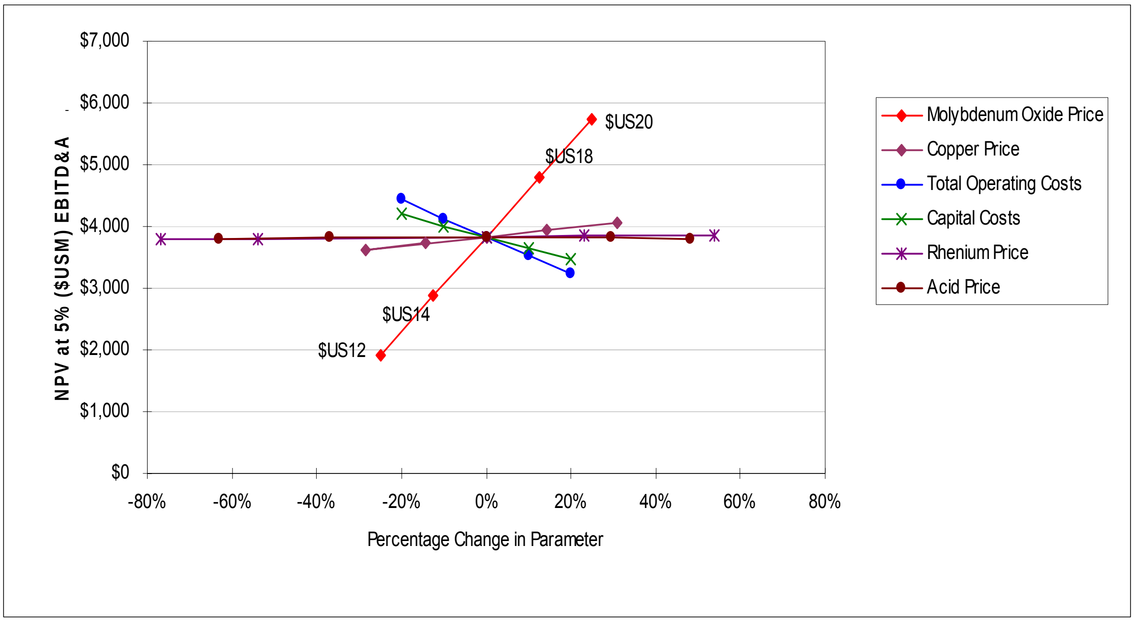


**Figure 23: 50 kt/d Throughput NPV Sensitivity**




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 24 through to Figure 29). The relative sensitivities for variations in the parameters tested are very similar for all throughput options.

**Figure 24: 100 kt/d Throughput IRR Sensitivity**

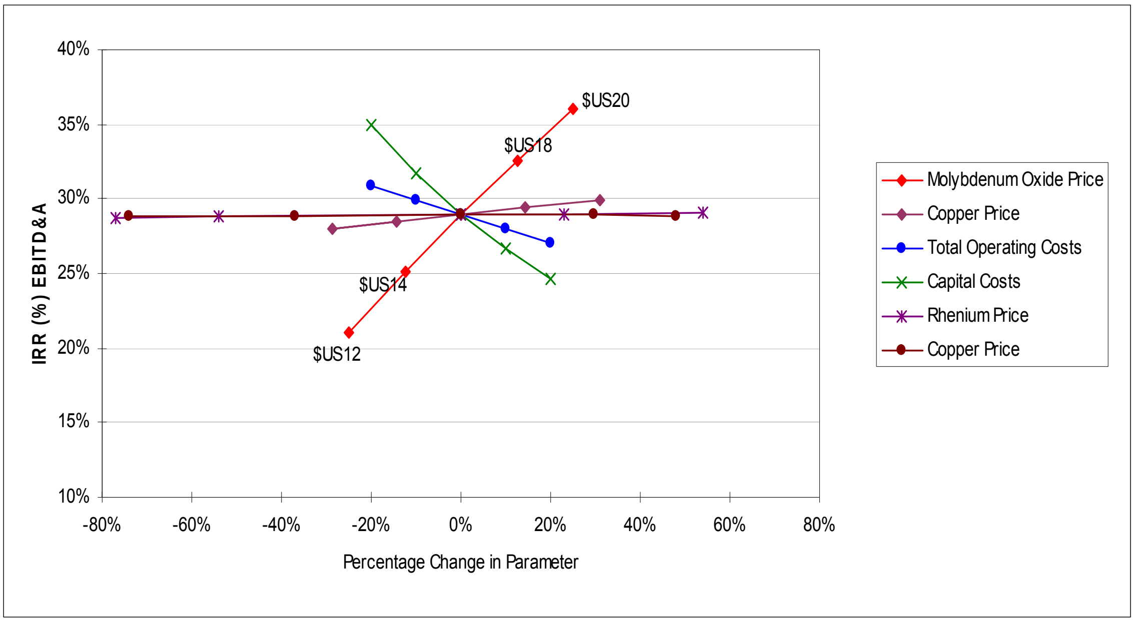



**Figure 25: 100 kt/d Throughput NPV Sensitivity**

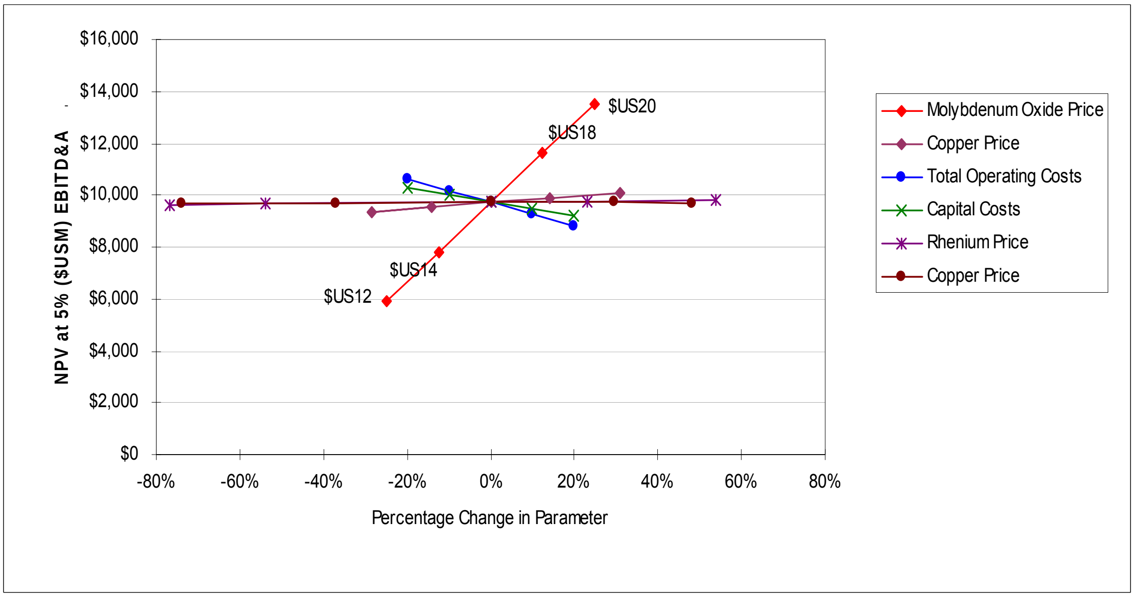


**Figure 26: 150 kt/d Throughput IRR Sensitivity**

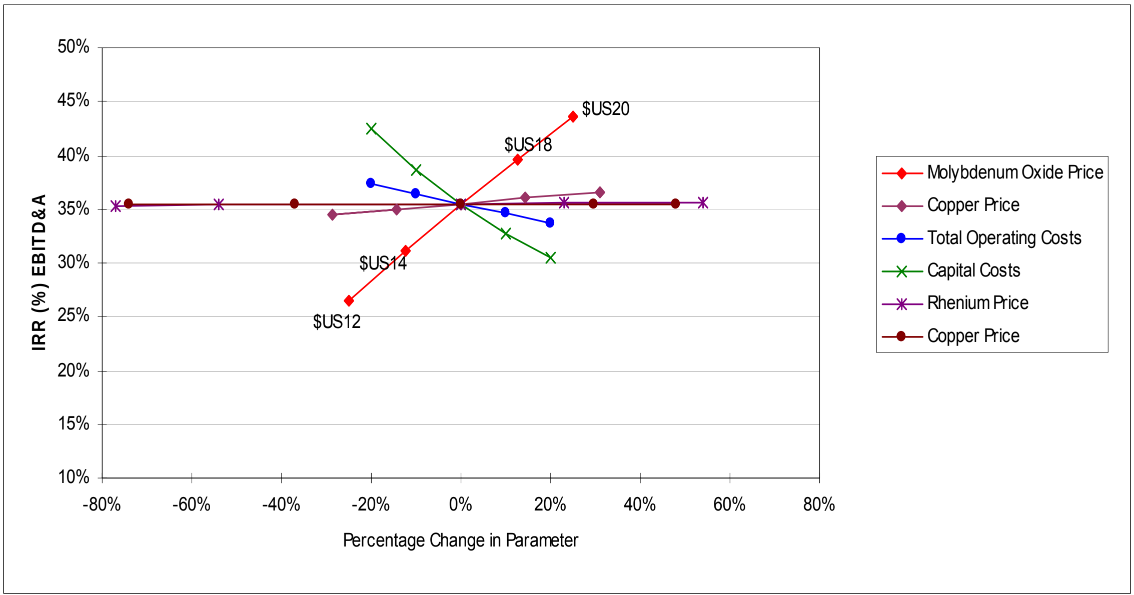



**Figure 27: 150 kt/d Throughput NPV Sensitivity**

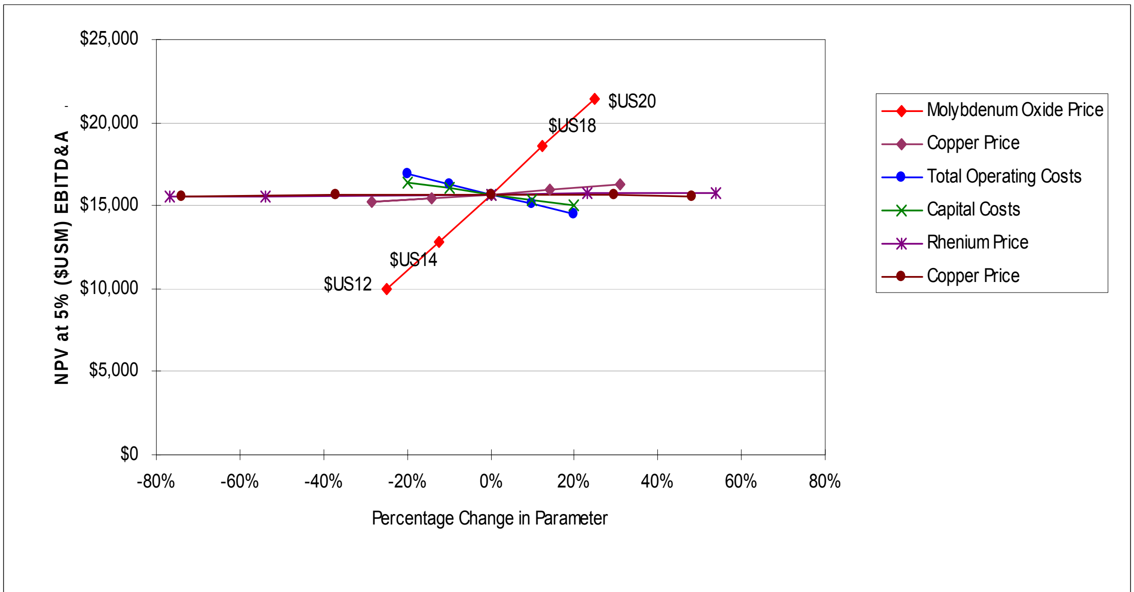


**Figure 28: 200 kt/d Throughput IRR Sensitivity**

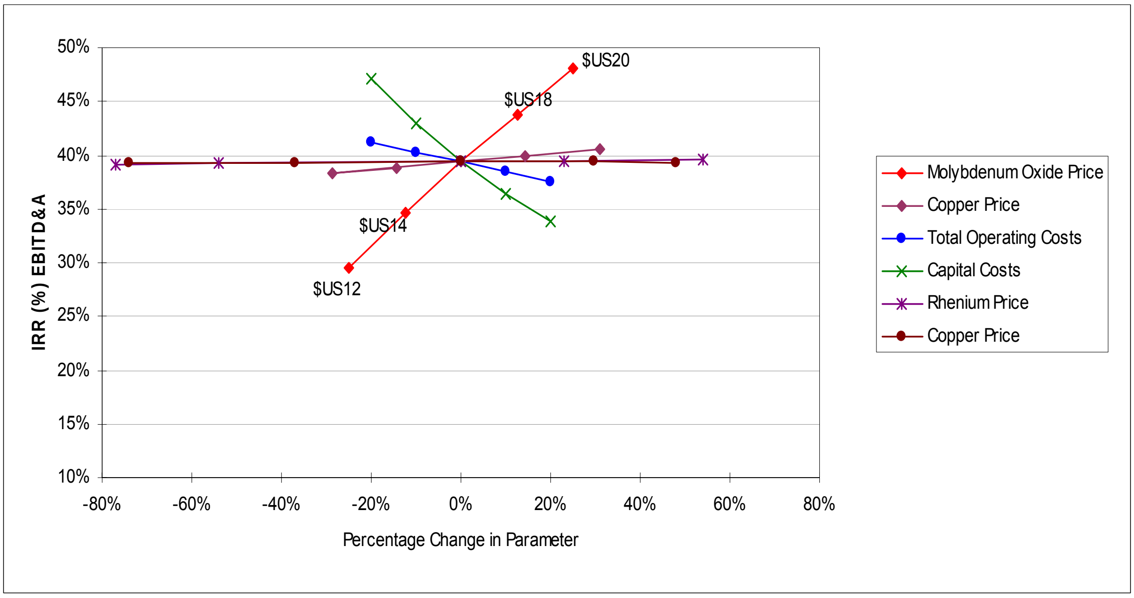
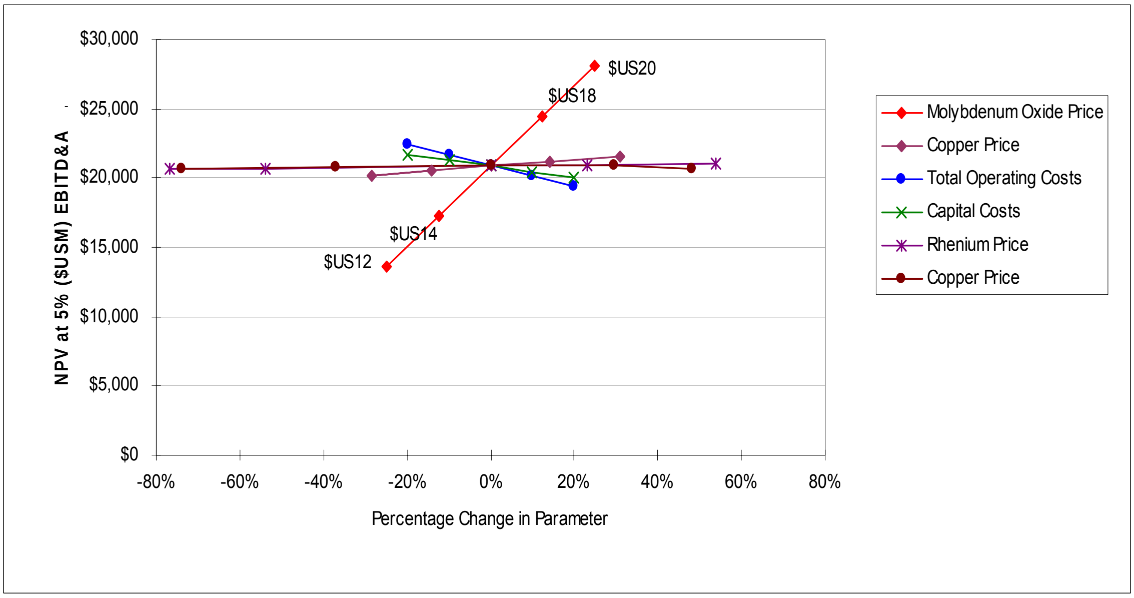



**Figure 29: 200 kt/d Throughput NPV Sensitivity**



## 18 INTERPRETATION AND CONCLUSIONS

Based on the existing 2009 resource estimate (Holmgren and Giroux 2009), this PEA has estimated the capital and operating costs for the mine, processing plant and related infrastructure to process between 50 000 and 200 000 short tons per day to determine the most economic production rate prior to commencing detailed feasibility study work.

Overall, the economic performance of the property (as measured by the IRR, NPV and payback period etc.) improves as the design throughput increases. These data are summarised below in Table 66 and discussed in detail, together with the metal prices and assumptions used in the calculations in Section 17. All values are calculated based on Earnings Before Interest Tax Depreciation and Amortisation (EBITD&A).

**Table 66: Base Case Economic Analysis**

| Economic parameters (EBITD&A) | Throughput Option | | | |
|---|---|---|---|---|
| | 50 kt/d (short tons) | 100 kt/d (short tons) | 150 kt/d (short tons) | 200 kt/d (short tons) |
| NPV (US$B @ 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** |

The economic metrics continue to improve as the design throughput increases, showing that even higher throughputs would give higher NPV and higher IRR. However, at 100 to 150 kt/d, CUMO would be very large for a green-fields base metals project, with a matching high capital cost. A project of larger scale would likely encounter difficulties in obtaining financing and a more reasonable design throughput for future studies is in the 100 to 150 kt/d range.

Based on the outcomes from this PEA, it is recommended that additional test work and analysis should be undertaken on the CUMO property, to determine whether the project can achieve the economic performance required by Mosquito at a Feasibility Study level. This should include, but not be limited to, the work outlined below in Section 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.

# 19 RECOMMENDATIONS

The following recommendations are based on the review of the work done to date.

## 19.1 Drilling

Exploration work consisting mainly of drilling is required to reach feasibility. It is estimated that a total of 45 additional holes for 125,600 feet plus an additional 5 geotechnical holes for 12,000 feet making a total of 137,600 feet of drilling will be required. This drilling is broken into the following categories.

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

The shortest time to complete this work will be two seasons using 7 drill rigs each season.

## 19.2 Site Selection

Several preliminary sites have been selected and need to be examined in detail in order to prepare an environmental study plan. These include mill, tailings and waste impoundment sites, potential low-impact Hydro-electric sites, housing and social structure sites, and finally mine and road access sites. Each site selection should be narrowed to 1 or 2 choices.

Once complete, a preliminary Plan of Operations can be created in order to start the environmental studies required for the feasibility study.

## 19.3 Preliminary Mine Design

The critical area for mine planning is mine design. At the present time the mine design is conceptual. The conceptual mine design by Mosquito assumes pit parameters that reflect pit parameters from other large open pits (i.e. Bingham Canyon, personal communication, Shaun Dykes, Mosquito). Using the conceptual parameters, Mosquito has produced pit designs for all four production scenarios that can be considered world class pits in terms of depths of excavation.

The data required to assess the overall stability of the pit wall has not been fully defined at this time necessitating a pit design for this study that is conceptual in nature. Stability is a function of rock quality, fracturing, and faulting as well as the width of the benches. The equivalent strength of the rock mass is a function of the fracturing and faulting as well as the mineral integrity of the rock itself.

To move toward a feasibility level study it will be necessary to develop the data required to quantify those aspects of the project that are presently conceptual in nature. The areas of focus for the recommendations are mine modelling and pit design. The recommendations are:

- Define by drilling the limits of the ore body in three dimensions and develop a resource model that contains a high proportion of the resources in the measured and indicated categories;
- Conduct geotechnical investigations on both the core and the surface rock exposures to establish rock mass characteristics to assist in future mine design;
- Develop an open pit mine model that incorporates information from the resource study and the geotechnical study to establish the parameters for the mine design;
- Develop an optimised pit design using detailed CAPEX and OPEX mining cost information based on a first principle approach and CAPEX and OPEX processing costs to estimate the financial performance of the project as an open pit operation.
- Investigate mine planning and site design options that will minimize the visual and environmental impact of the overall project.
- Review mining methods such as block cave mining that have the potential to improve the overall economic performance of the project.

## 19.4 Tailings and Stockpile design

Geotechnical field and laboratory testing programs should be conducted during the Feasibility Study to generate sufficient site-specific data to complete the Feasibility Design of the TSFs, waste dump and low-grade ore stockpile. The field investigation will be performed at the final locations of these facilities to characterize the geology, evaluate the subsurface soil, bedrock and groundwater conditions, estimate material limits and properties for use in the engineering analyses and designs of the facilities, and develop descriptions and parameters of construction materials. The investigation will consist of boreholes to be drilled with a drill rig and test pits to be excavated with a backhoe, and may also include potential onsite and offsite sources of borrow materials.

A seismic hazard assessment should also be performed for the project site and the results used in the Feasibility Design of the facilities. The assessment will provide design earthquake magnitude and peak ground acceleration for use in seismic stability analyses of the TSF dams and waste dump slopes. Other seismic parameters will also be estimated for use in the design of structures.

## 19.5 Metallurgical test work

Work carried out to date is sufficient to support the conceptual level design and costing. Further work will be required for a full Feasibility Study to generate design criteria.

For a detailed Feasibility Study, flotation and comminution variability test work across the ore body will be required to allow development of detailed models of plant throughput and grade/recovery that take into account variations in competency, mineralogy and head grade.

Recommended additional work identified as part of the Conceptual Study includes the following:

- Additional comminution test work on the major CUMO ore types.

- Rougher flotation test work for target grind size optimisation.
- A program of variability testing to determine the validity of the assumptions used for the comminution design.
- Review of comminution circuit selection and design by an external consultant such as Steve Morrell Comminution Consulting Pty. Ltd. (SMCC).
- Additional reagent and flotation flow sheet testing to reduce costs and improve metallurgy.
- Copper – Molybdenum separation flotation test work.
- Flotation variability and lock cycle testing to confirm metallurgical modelling.
- Test work to confirm concentrate thickening rates for concentrate thickener selection.
- Test work to confirm tailings thickening rates for tailings thickener selection.
- Test work to confirm concentrate filtration rates and optimise filter selection.
- Rheology test work to confirm tailings pumping, pipeline and distribution design.
- Bulk materials handling test work to optimise design of the ROM bin, crushed ore stockpile and reclaim facility.
- Confirmation of geotechnical conditions for engineering design purposes in the plant and TSF locations, particularly in the locations of heavy structures.

### 19.6 Environmental work

Once the mill and other sites have been identified, a Plan of Operations will need to be filed and base line environmental studies for the project started. This will lead to an Environmental Impact Statement being required to permit a mining operation.

In addition, an inter-agency governmental task force will need to be established to ensure all the various groups co-operate and communicate in a timely manner with each other.

### 19.7 Public Relations

Initiate a community relations program to establish the company as a good corporate citizen and disseminate positive information about the potential of this project. This would include preliminary discussions with local communities to minimize future issues related to on-going exploration and development.

### 19.8 Cost Estimate

Optimal timing for commencement of mine operations for the CUMO deposit is at the start of the next metal cycle for molybdenum. Given that the design, construction and permitting stages for placing CUMO into production are anticipated to take 3 to 4 years, and since a feasibility study needs to be completed prior to detailed design and construction, it is therefore critical to do the work required for feasibility as soon as possible.

A budget has therefore been estimated to accomplish the goals laid out in the shortest reasonable time frame (Section 19.8.1). The objective is to produce a feasibility study in three years. This would enable a mine to be developed in time to catch the next metal price cycle peak for molybdenum, anticipating a peak in 5 years.

The budget to achieve feasibility in 3 years is summarized as follows:

| | |
|---|---|
| 2010 budget | $25,000,000 |
| 2011 budget | $20,000,000 |
| 2012 budget | $27,500,000 |
| **Total ($US)** | **$72,500,000** |

*Note: This budget does not include funds for any activity beyond feasibility other than permitting. Capital and construction costs to production would be outlined in the feasibility study.*

### 19.8.1 COST ESTIMATES

| YEAR #1 – 2010 | | | |
|---|---|---|---|
| Diamond Drilling | | | |
| Delineation, infill, metallurgy | 25,298 meters (83,000 feet) | $100/ft | $8,300,000 |
| Mob-Demobilization | | | $120,000 |
| Road construction | 6 km | $80,000/km | $480,000 |
| 45 man camp + services etc. | capital cost | | $8,000,000 |
| Sample Preparation and Analysis | 8,500 | 45 | $382,500 |
| Metallurgical Testing | First round of testing | | $75,000 |
| | Batch round of testing | | $300,000 |
| | Variability | | $400,000 |
| Land Acquisition and staking costs | | | $2,500,000 |
| Environmental Studies | Environmental Assessment | | $175,000 |
| | Ongoing baseline studies | | $300,000 |
| | plan of operations | | |
| | Environmental Impact Statement | | |
| | Permitting | | |
| Engineering studies scoping | mill site, tailings site analysis | | $350,000 |
| | Intergoverment Task Force creation | | $50,000 |
| | pre-feasibility | | |
| | feasibility | | |
| | **Yearly Charges** | | |
| Mob-Demobilize | | | $200,000 |
| Road Maintenance\pad construction | | | $150,000 |
| Supervision and Project Management | Exploration Manager | $15,000/mth | $75,000 |
| | Project Geologist | $10,000/mth | $120,000 |
| | Assistant Geologist(2) | $8,000/mth | $192,000 |
| | Technicians (8) | $15/hr | $259,200 |
| Vehicles | 3 vehicles | $1000/mth | $36,000 |
| Accommodation cost of running camp | 30 men | $40/man/day | $360,000 |
| Travel | | $2000/mth | $24,000 |
| Project office and Warehouse | | $2200/mth | $26,400 |
| Land Filing Fees | BLM: $140/claim; County: $8.50 | | $195,000 |
| Consultants | (Mining Metallurgical and Marketing) | | $150,000 |
| Resource Modeling | | | $150,000 |
| Public Relations and Project Presentation | Liaison with county and state officials | | $100,000 |
| **yearly Subtotal** | | | **$23,470,100** |
| Contingency | | | $1,529,900 |
| **Subtotal (2010)** | | | **$25,000,000** |

| YEAR #2 - 2011 | | | |
|---|---|---|---|
| Diamond Drilling | | | |
| Delineation, infill | 23,042 meters (75,600 feet) | $100/ft | $9,072,000 |
| Road construction | | | |
| 45 man camp + services etc. | capital cost | | done |
| Sample Preparation and Analysis | 7,600 | 45 | $342,000 |
| Metallurgical Testing | First round of testing | | done |
| | Batch round of testing | | done |
| | Variability | | done |
| Environmental Studies | Environmental Assessment | | done |
| | Ongoing baseline studies | | $500,000 |
| | plan of operations | | $250,000 |
| | Environmental Impact Statement | | $5,000,000 |
| | Permitting | | |
| Engineering studies scoping | Scoping sizing Study | | done |
| | mill site, tailings site analysis | | done |
| | Inter Agency Task Force creation | | done |
| | pre-feasibility | | $1,500,000 |
| | feasibility | | |
| | **Yearly Charges** | | |
| Mob-Demobilize | | | $200,000 |
| Road Maintenance\pad construction | | | $150,000 |
| Supervision and Project Management | Exploration Manager | $15,000/mth | $75,000 |
| | Project Geologist | $10,000/mth | $120,000 |
| | Assistant Geologist(2) | $8,000/mth | $192,000 |
| | Technicians (4) | $15/hr | $129,600 |
| Vehicles | 3 vehicles | $1000/mth | $36,000 |
| Accommodation cost of running camp | 30 men | $40/man/day | $360,000 |
| Travel | | $2000/mth | $24,000 |
| Project office and Warehouse | | $2200/mth | $26,400 |
| Land Filing Fees | BLM: $140/claim; County: $8.50 | | $195,000 |
| Consultants | (Mining Metallurgical and Marketing) | | $150,000 |
| Resource Modeling | | | $200,000 |
| Public Relations and Project Presentation | Liaison with county and state officials | | $100,000 |
| **yearly Subtotal** | | | **$18,622,000** |
| Contingency | | | $1,378,000 |
| **Subtotal (2011)** | | | **$20,000,000** |

| YEAR #3 - 2012 | | | |
|---|---|---|---|
| Diamond Drilling | | | |
| Delineation, infill, condemnation | 3650 meters(12,000 feet) | $100/ft | $1,200,000 |
| Road construction | | | |
| 45 man camp + services etc. | | | |
| Sample Preparation and Analysis | 1,200 | 45 | $54,000 |
| Metallurgical Testing | First round of testing | | done |
| | Batch round of testing | | done |
| | Variability | | done |
| Environmental Studies | Environmental Assessment | | |
| | Ongoing baseline studies | | $250,000 |
| | plan of operations | | |
| | Environmental Impact Statement | | $15,000,000 |
| | Permitting | | $4,000,000 |
| Engineering studies scoping | Scoping sizing Study | | |
| | mill site, tailings site analysis | | |
| | Intergoverment Task Force creation | | |
| | pre-feasibility | | |
| | feasibility | | $2,500,000 |
| | **Yearly Charges** | | |
| Mob-Demobilize | | | $200,000 |
| Road Maintenance\pad construction | | | $150,000 |
| Supervision and Project Management | Exploration Manager | $15,000/mth | $75,000 |
| | Project Geologist | $10,000/mth | $120,000 |
| | Assistant Geologist(2) | $8,000/mth | $192,000 |
| | Technicians (4) | $15/hr | $129,600 |
| Vehicles | 3 vehicles | $1000/mth | $36,000 |
| Accommodation cost of running camp | 30 men | $40/man/day | $360,000 |
| Travel | | $2000/mth | $24,000 |
| Project office and Warehouse | | $2200/mth | $26,400 |
| Land Filing Fees | BLM: $135/claim; County: $8.50 | | $49,364 |
| Consultants | (Mining Metallurgical and Marketing) | | $150,000 |
| Resource Modeling | | | $50,000 |
| Public Relations and Project Presentation | Liaison with county and state officials | | $100,000 |
| **yearly Subtotal** | | | **$24,666,364** |
| Contingency | | | $2,833,636 |
| **Total** | | | **$27,500,000** |

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## 21 CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE OF AUTHOR

I, Jackie A. Holmgren, Consultant, Roche Jaime Exploration, Mile 12 Rawhide Road, P.O. Box 09, Luning, Nevada, hereby certify:

1. I am a graduate of the University of Oregon (1979) and hold a B.Sc. degree in geology.

2. I am presently self-employed as an independent geological consultant.

3. Since becoming a geologist I have been employed in my profession by various mining companies including Anaconda, Arco, Chevron, and the Stillwater Platinum/Palladium Project, later known as the Stillwater Mining Company. I am presently and have been in the past a geological consultant with my own consulting company.

4. I am a member of the American Institute of Professional Geologists

5. I certify that by reason of my education, past relevant work experience, and affiliation with professional associations, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. This report entitled *"CUMO Property Preliminary Economic Assessment, Throughput Scoping Study Report"*, is based on a study of the data and literature available on the CUMO Property, I am responsible for the sections 1.1, 1.2, 1.3, 1.6, 2, 3, 4, 5, 6, 7, 8, 9, 11, 12 ,13, 15.1.2, 17.5, 19.1, 19.6 and 19.7 of the report.

7. I personally visited and inspected the CUMO property including the core shed, core/core storage, sampling room /technicians, buildings/office, diamond drilling in progress on site, as well as maps and other information freely supplied to me on 11/29/2008 through 12/02/2008 and again on August 22, 2008.

8. I am not aware of any material fact or material change with respect to the technical report that is not reflected in the technical report, the omission to disclose which makes the technical report misleading.

9. I am a consultant for Mosquito Consolidated Gold Mines Ltd., and am independent of Mosquito Mining Corporation according to the test in Section 1.4 of NI 43-101.

10. I have read NI 43-101 and NI 43-10IF1 and the technical report has been prepared in compliance with that instrument and form.

Signed

***"Jackie A Holmgren"***

Jackie A. Holmgren, B. Sc., Geologist
Dated: November 18, 2009

## CERTIFICATE

I, G.H. Giroux, of 982 Broadview Drive, North Vancouver, British Columbia, do hereby certify that:

1) I am a consulting geological engineer with an office at #1215 - 675 West Hastings Street, Vancouver, British Columbia.
2) I am a graduate of the University of British Columbia in 1970 with a B.A. Sc. and in 1984 with a M.A. Sc. both in Geological Engineering.
3) I have practiced my profession continuously since 1970. I have had over 30 years experience calculating mineral resources. I have previously completed resource estimations on a wide variety of molybdenum deposits including the Ajax, Redbird, Davidson, Sphinx and Chu Deposits.
4) I am a member in good standing of the Association of Professional Engineers of the Province of British Columbia.
5) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.
6) This report titled *"CUMO Property Preliminary Economic Assessment, Throughput Scoping Study Report"*, is based on a study of the data and literature available on the CUMO Property. I am responsible for Sections 14 and 16 on data verification and resource estimations completed in Vancouver during 2009. I have not visited the property.
7) I have previously completed a statistical review of this property in 2005 and a resource estimation in 2008.
8) As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
9) I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
10) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this day of November 18, 2009

GIROUX CONSULTANTS LTD.
Signed:

***"G. H. Giroux"***

G. H. Giroux, P.Eng., MASc.

## CERTIFICATE OF QUALIFIED PERSONS

Robert Braun, MAusIMM
Ausenco Minerals Canada Inc.
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| | |
|---|---|
| Telephone: | +(1) 604 453 4800 |
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I, Robert Braun, MAusIMM, certify that I am a Lead Metallurgist for Ausenco Minerals Canada Inc. Suite 605, 375 Water St. Vancouver, British Columbia, V6B 5C6, Canada.

This certificate applies to the Technical Report titled *"CUMO Property Preliminary Economic Assessment, Throughput Scoping Study Report"* dated November 18, 2009.

My qualifications and relevant experiences are that:

1. I graduated with a Bachelors degree in Metallurgical Engineering from the Royal Melbourne Institute of Technology in 1995.
2. I am a member of the AusIMM.
3. I have worked as a Metallurgist for a total of 14 years.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have not visited the Property.
6. I am responsible for the preparation of sections 1.4, 1.5, 15 (except section 15.1.2), 17.6, 17.7, 17.11, 17.9, 17.12.2, 17.12.3, 18, 19.2, 19.5 and 19.8 of the Technical Report.
7. I am independent of the issuer per Section 1.4 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with that instrument.
10. 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: | November, 18 2009 |
| Signature of Qualified Person: | ***"Robert Braun"*** |
| Name of Qualified Person: | Robert Braun |

## CERTIFICATE OF QUALIFIED PERSONS

Richard J Kehmeier
Senior Geologist
Vector Engineering Inc.
1120 Washington Ave., Suite 250
Golden, CO 80401

| | |
|---|---|
| Telephone: | 303 279-7533 |
| Facsimile: | 303 271-0796 |
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I, Richard J. Kehmeier, CPG-10879 certify that I am a Senior Geologist at Vector Engineering Inc. 1120 Washington Ave, Suite 250 Golden, CO 80401 USA.

This certificate applies to the Technical Report titled *"CUMO Property Preliminary Economic Assessment, Throughput Scoping Study Report"* dated November 18, 2009.

My qualifications and relevant experiences are that:

1. I graduated with a B.Sc. Geological Engineering and M.Sc. Geology from the Colorado School of Mines, May 1970 and May 1973 respectively.
2. I am a member of the American Institute of Professional Geologist CPG-10879.
3. I have worked as a Geologist for a total of 39 years.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have not visited the Property.
6. I am responsible for the preparation of sections 10, 17.1, 17.8, 17.12.1, and 19.3 of the Technical Report.
7. I am independent of the issuer per Section 1.4 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is as an exploration geologist in 1972
9. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with that instrument.
10. 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: | November 18, 2009 | |
| Signature of Qualified Person: | ***"Richard J. Kehmeier"*** | "Sealed" |
| Name of Qualified Person: | Richard J. Kehmeier | |

## CERTIFICATE OF QUALIFIED PERSONS

Charles J. Khoury, P.E.
Vector Engineering, Inc.
1120 Washington Ave., Suite 250
Golden, Colorado 80401 USA

| | |
|---|---|
| Telephone: | (303) 279-7533 |
| Facsimile: | (303) 271-0796 |
| E-mail: | khoury@vectoreng.com |


I, Charles Khoury, P.E. certify that I am a Senior Geotechnical Engineer at Vector Engineering, Inc., 1120 Washington Ave., Suite 250, Golden, Colorado 80401, USA.

This certificate applies to the Technical Report titled *"CUMO Property Preliminary Economic Assessment, Throughput Scoping Study Report"* dated November 18, 2009.

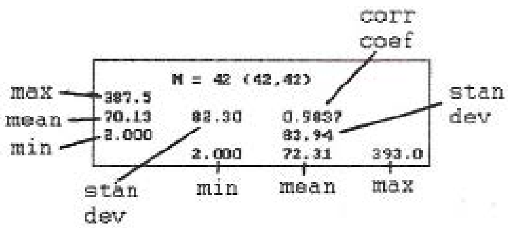
My qualifications and relevant experiences are that:

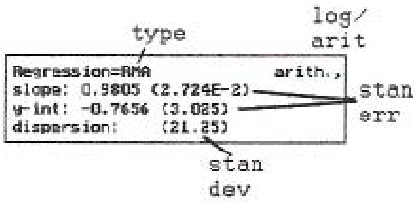
1. I graduated with a Master of Science in Civil Engineering from the University of Kentucky, May 1987.
2. I am a member of the SME and ASCE and a Professional Engineer in the States of Colorado and Nevada.
3. I have worked as a Geotechnical Engineer for a total of 22 years.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I have not visited the Property.
6. I am responsible for the preparation of Sections 17.2, 17.3, 17.4, 17.10 and 19.4 of the Technical Report.
7. I am independent of the issuer per Section 1.4 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with that instrument.
10. 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.

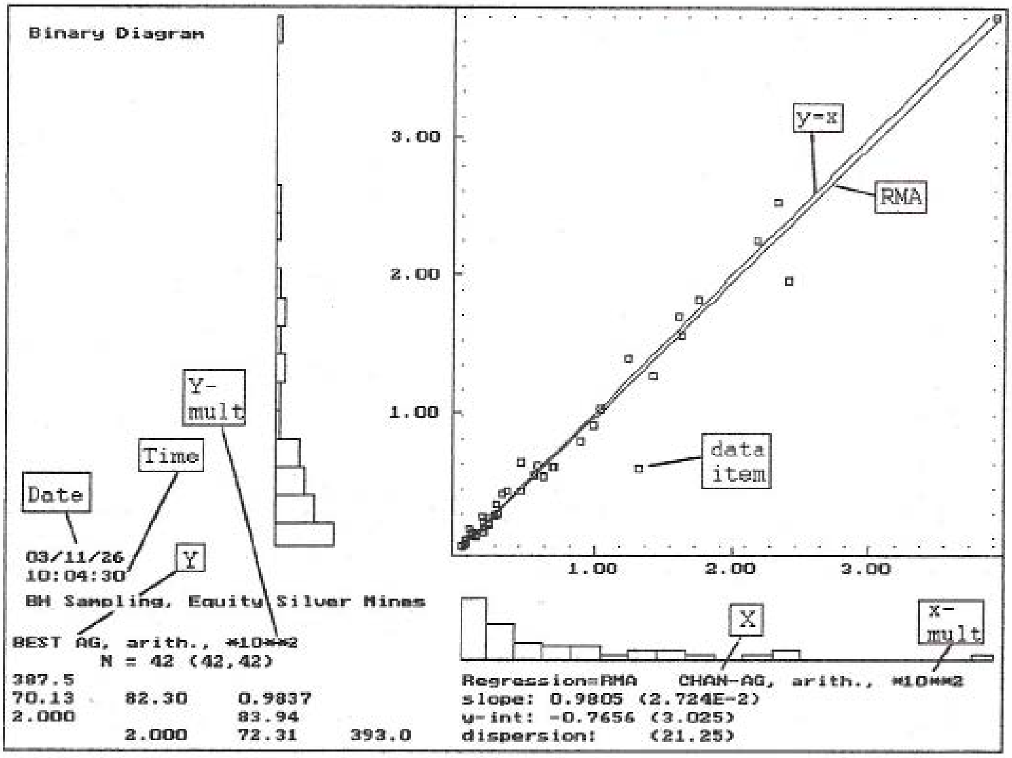
| | |
|---|---|
| Dated: | November 18, 2009 |
| Signature of Qualified Person: | ***"Charles J. Khoury"*** |
| Name of Qualified Person: | Charles J. Khoury |

# APPENDIX 1
# RE-SPLITS OF REJECTS

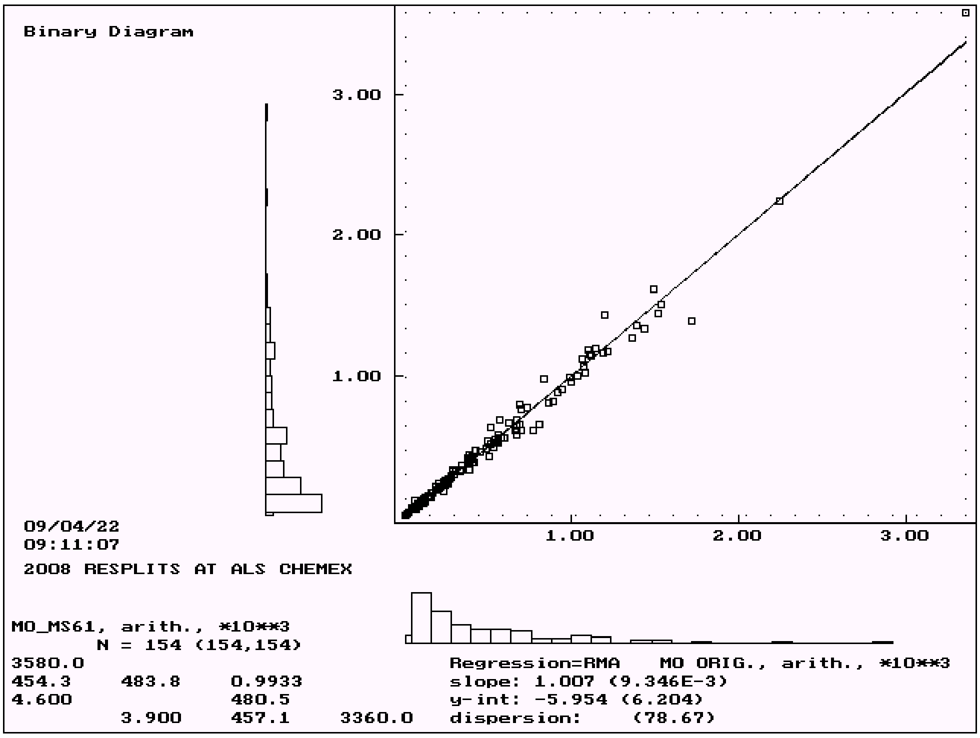
LABELLED DIAGRAMS INDICATING THE VARIOUS STATISTICS APPEARIN
ON DIAGRAMS THROUGHOUT TEXT.



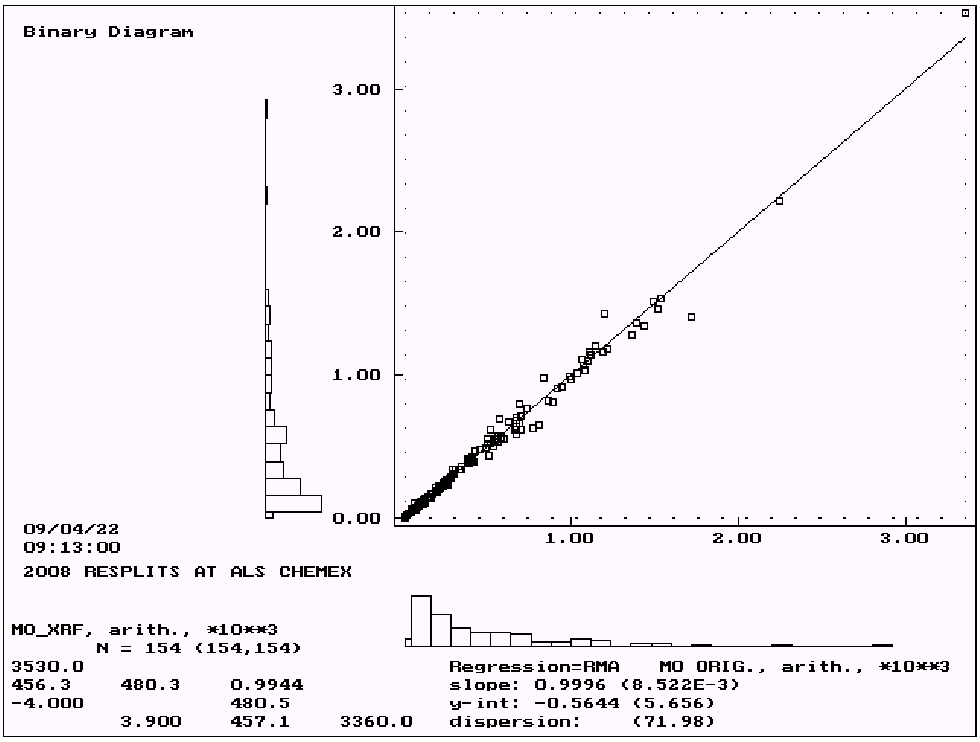



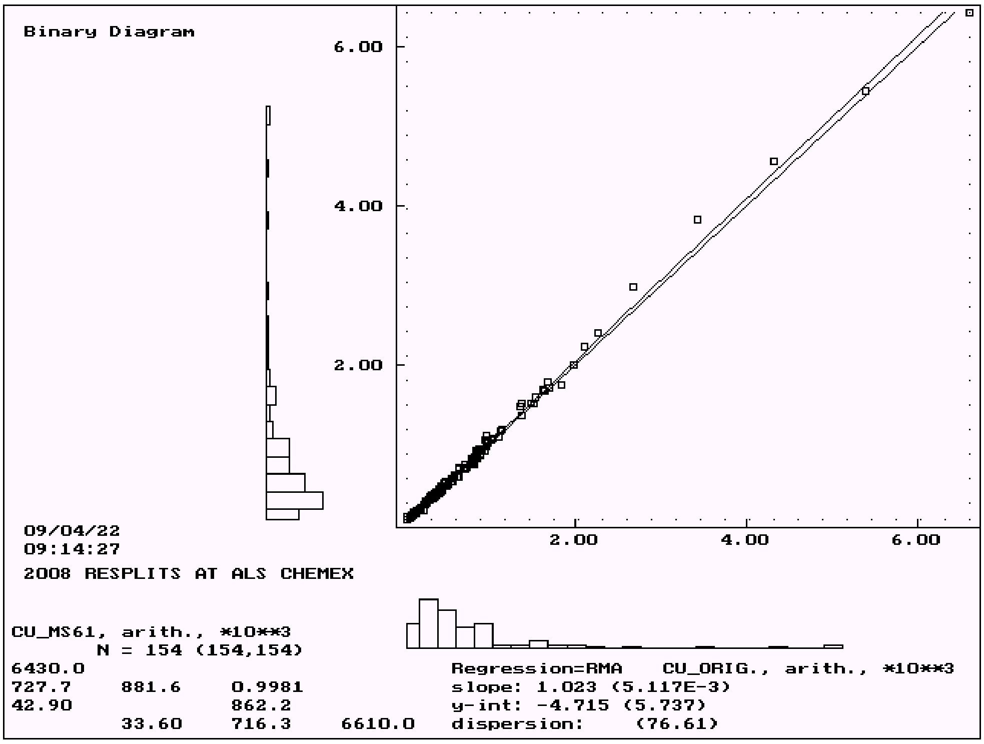

Results for Mo - Chemex - Original vs. ICP Check



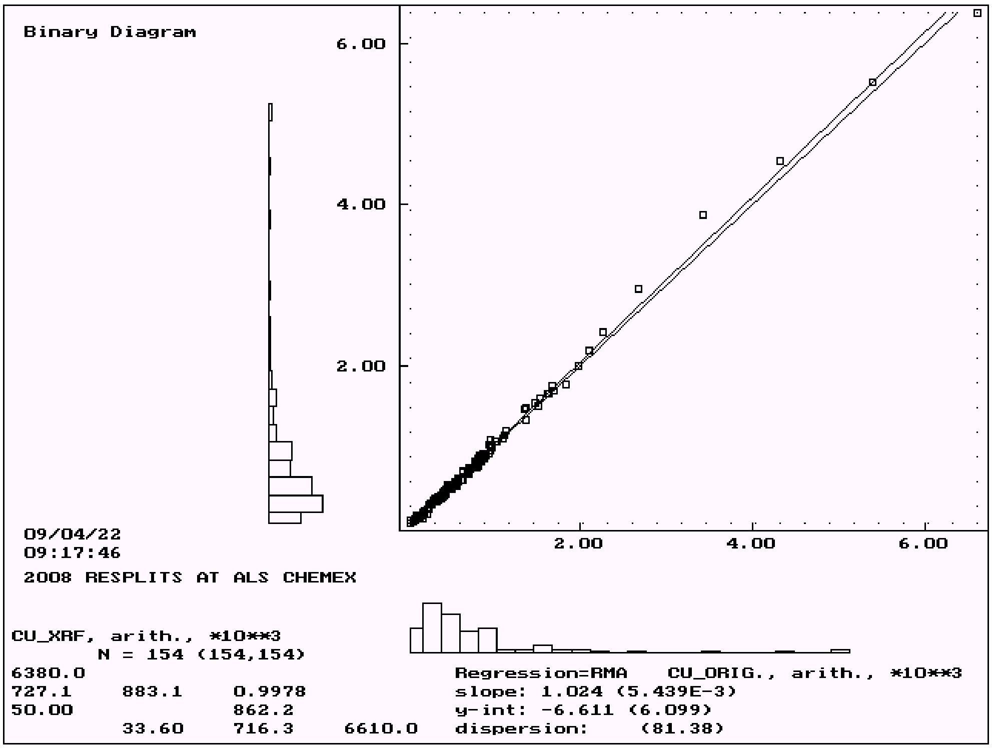

Results for Mo – Chemex – Original vs. XRF Check



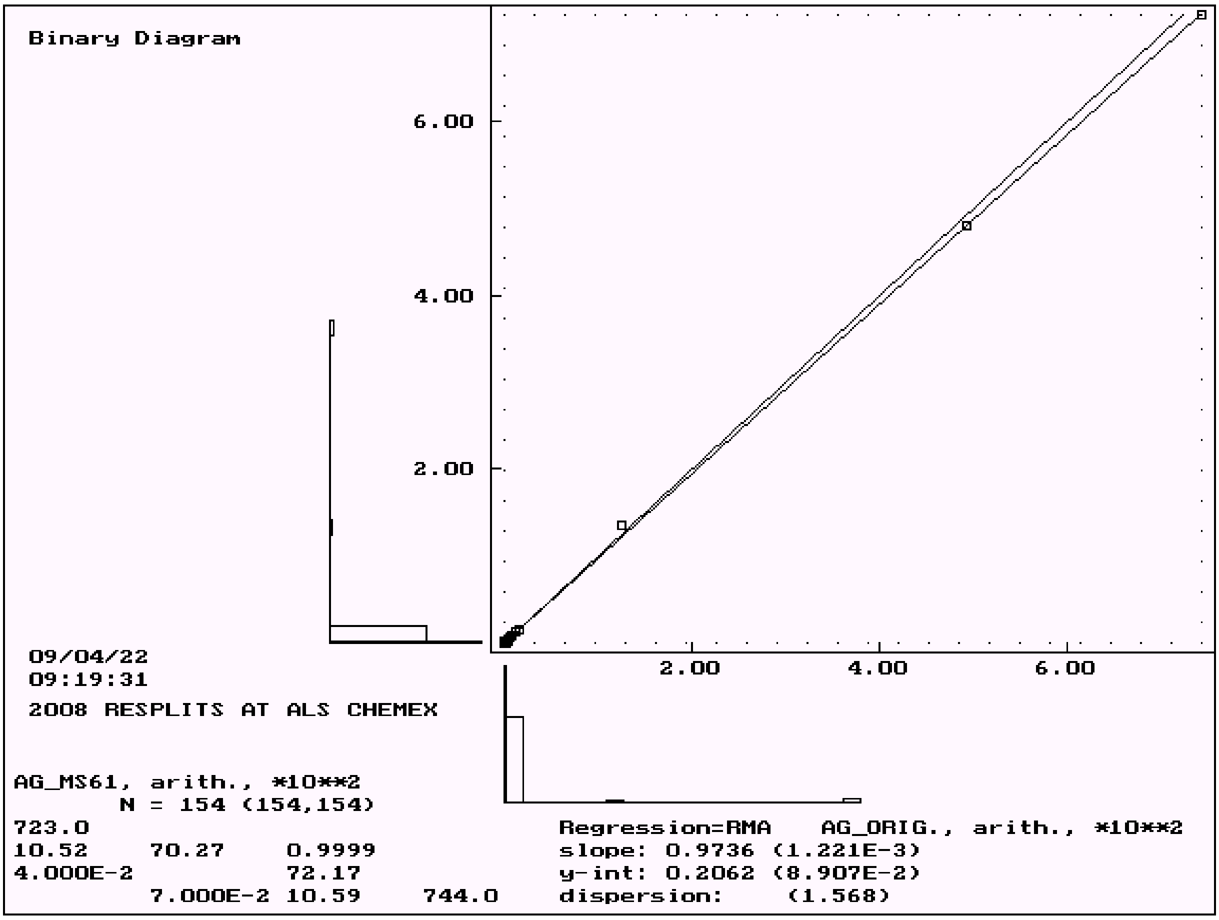

Results for Cu – Chemex – Original vs. ICP Check



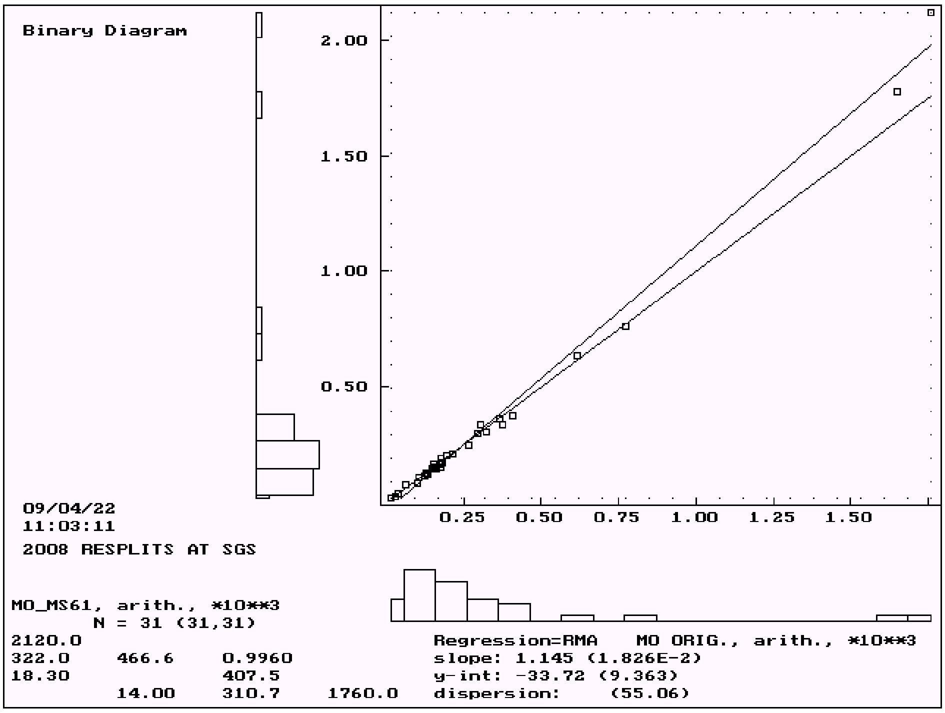

Results for Cu – Chemex – Original vs. XRF Check



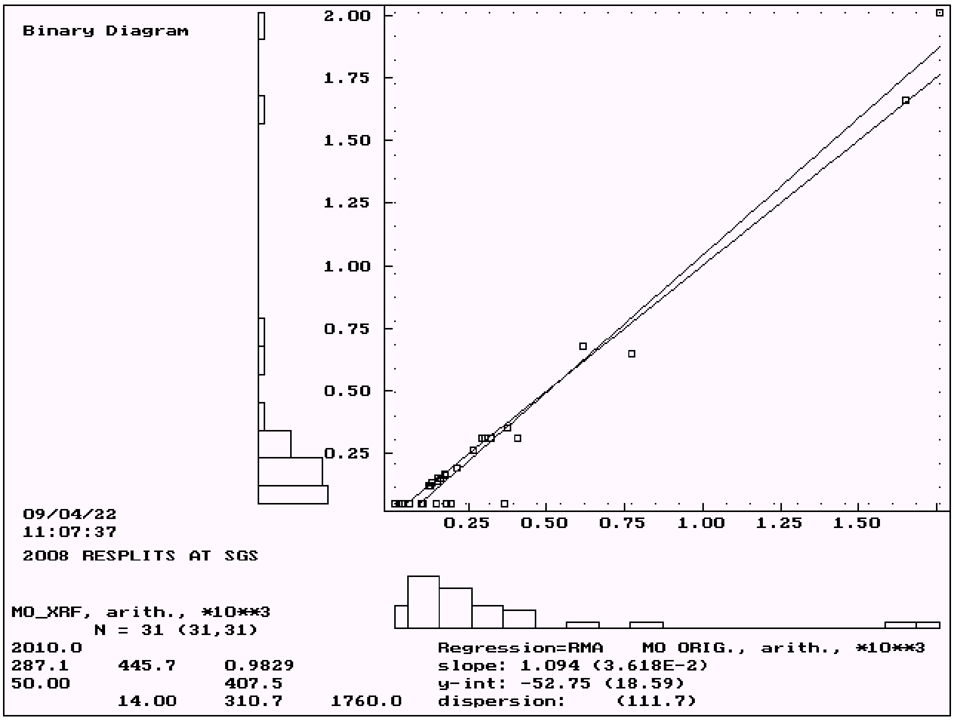

Results for Ag – Chemex Original vs. ICP Check


Binary Diagram
6.00
4.00
2.00
2.00
4.00
6.00
09/04/22
09:19:31
2008 RESPLITS AT ALS CHEMEX
AG_MS61, arith., *10**2
N = 154 (154,154)
723.0
10.52 70.27 0.9999
4.000E-2 72.17
7.000E-2 10.59 744.0
Regression=RMA AG_ORIG., arith., *10**2
slope: 0.9736 (1.221E-3)
y-int: 0.2062 (8.907E-2)
dispersion: (1.568)

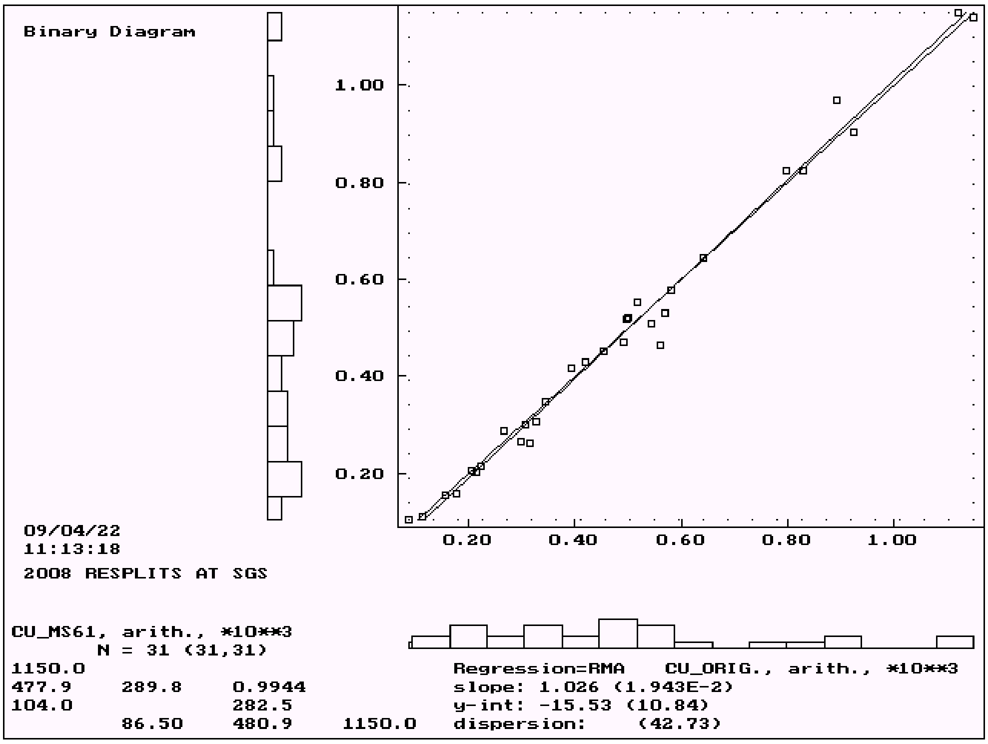
Results for Mo – SGS Original vs. SGS ICP check



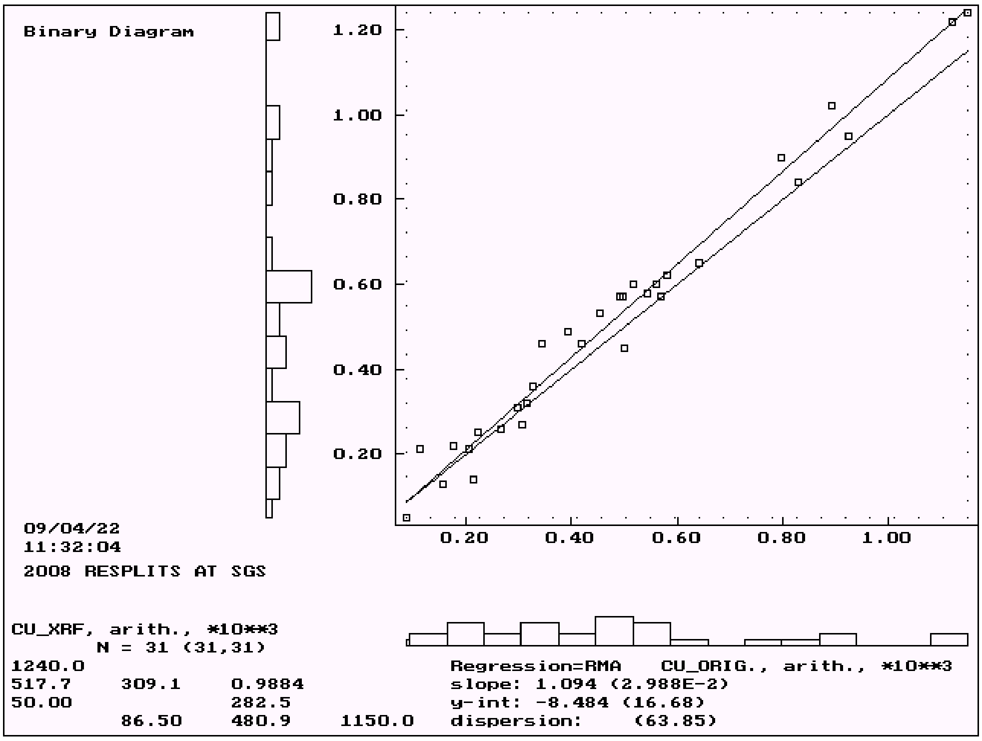

Results for Mo – SGS Original vs SGS XRF Check



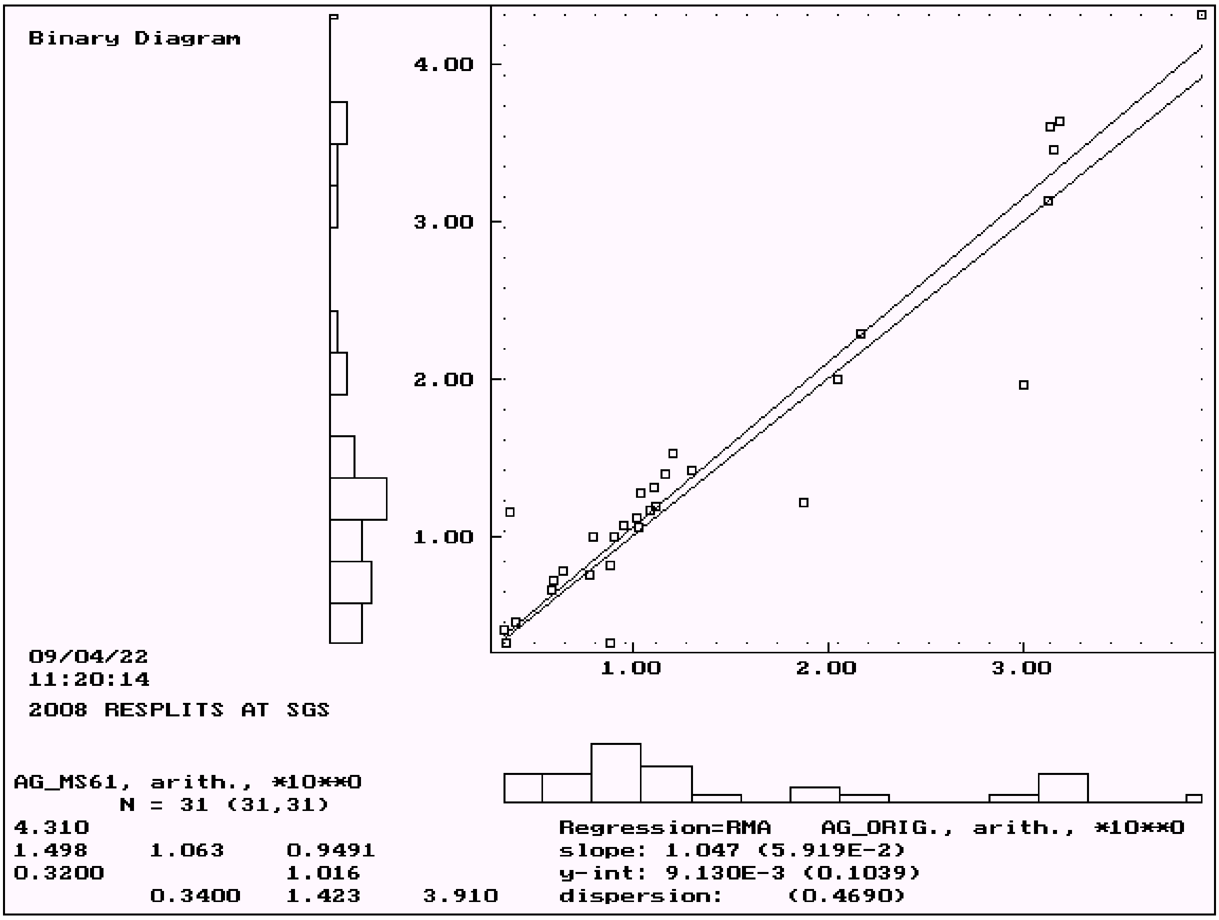
Results for Cu – SGS Original vs. SGS ICP Check




Results for Cu – SGS Original vs. SGS XRF



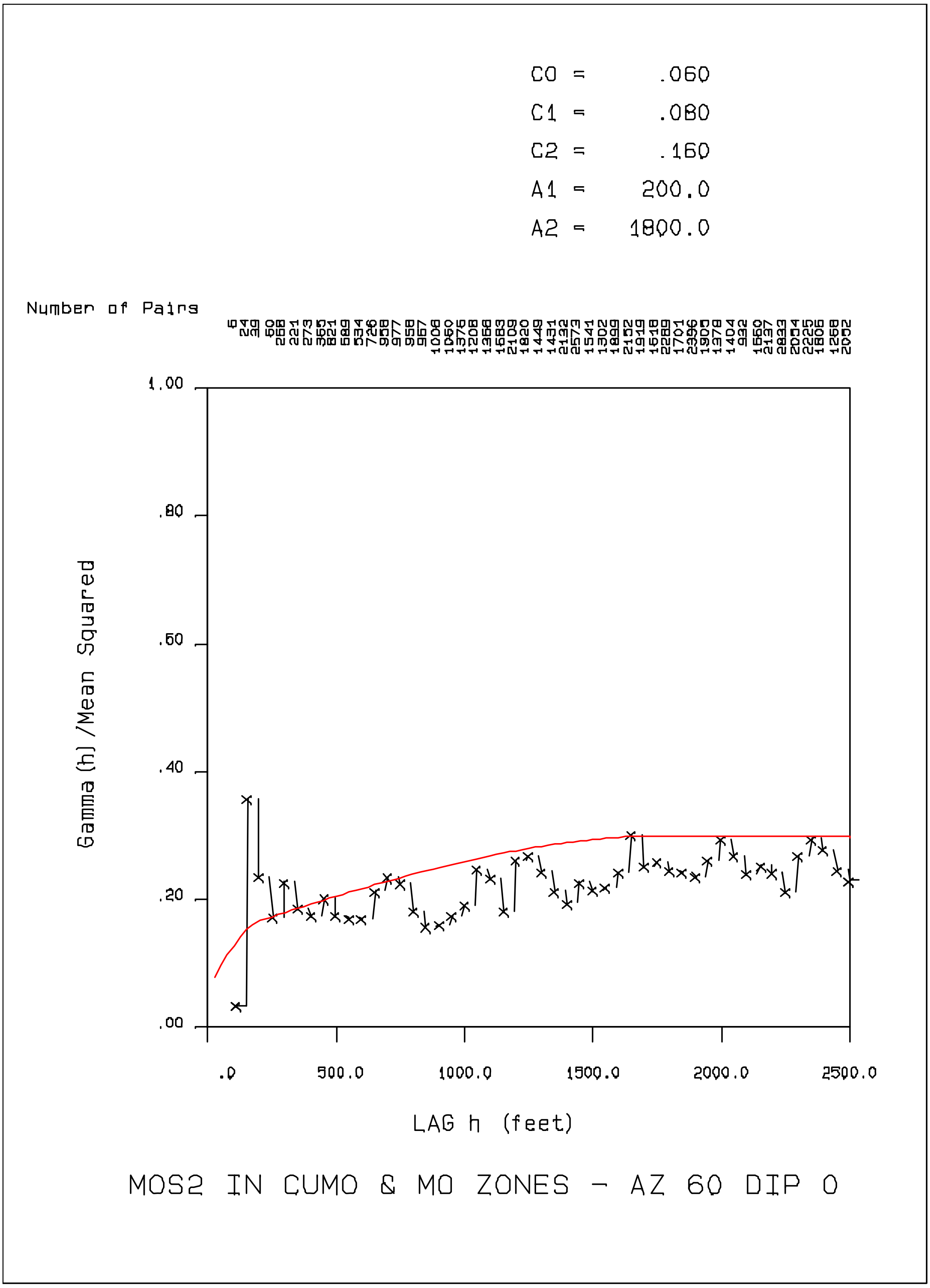
Results for Ag – SGS Original vs. SGS ICP Check

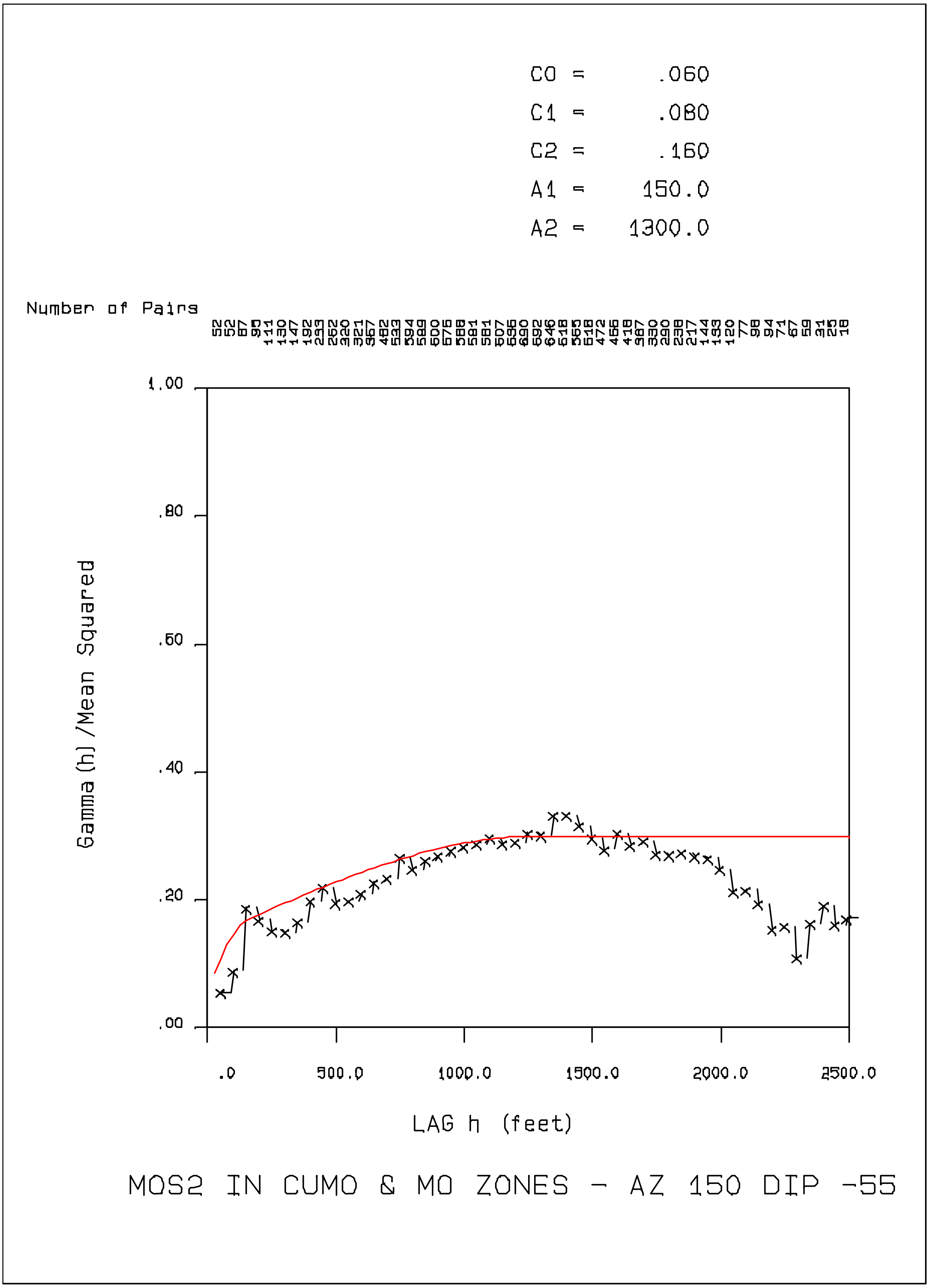


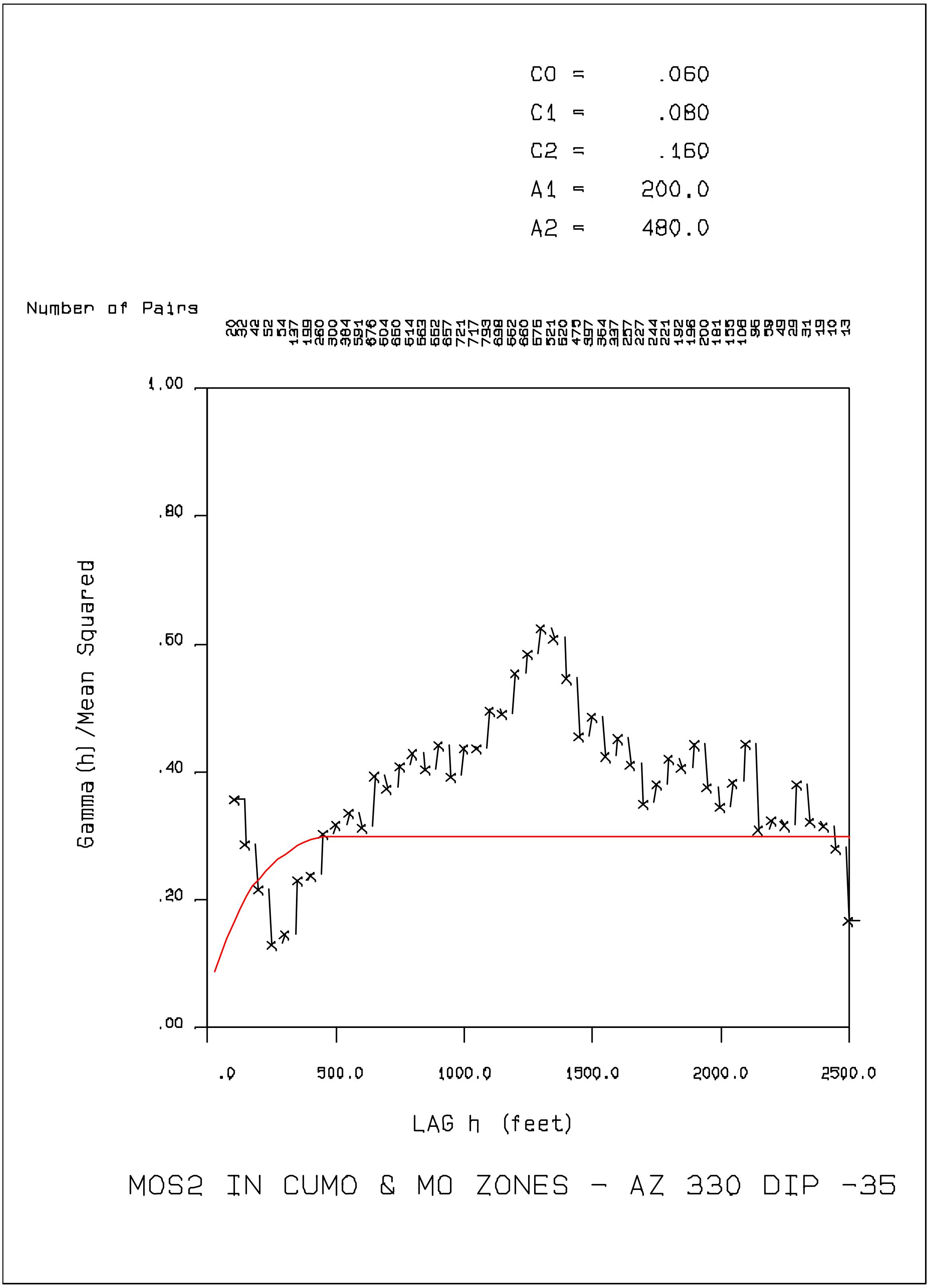
# APPENDIX 2
# DRILL HOLES USED IN RESOURCE ESTIMATE

| HOLE | EASTING | NORTHING | ELEVATION | HLENGTH |
|---|---|---|---|---|
| C-01 | 219904.46 | 120989.86 | 6026.47 | 1884.00 |
| C-02 | 219820.00 | 120575.00 | 6060.00 | 405.00 |
| C-03 | 219905.00 | 120250.00 | 6165.00 | 70.00 |
| C-04 | 219940.00 | 120785.00 | 6045.00 | 113.00 |
| C-05 | 220569.93 | 120524.76 | 6201.69 | 1416.00 |
| C-06 | 219919.00 | 121749.00 | 5902.00 | 663.00 |
| C-07 | 219823.00 | 121491.00 | 5962.00 | 275.00 |
| C-08 | 220025.00 | 118890.00 | 6467.00 | 379.00 |
| C-09 | 220687.00 | 121438.00 | 5890.00 | 804.60 |
| C-10 | 221220.36 | 119755.68 | 6340.99 | 2381.00 |
| C-11 | 221230.17 | 120415.79 | 5995.98 | 3003.00 |
| C-12 | 221432.00 | 120955.00 | 5742.00 | 1340.00 |
| C-13 | 219902.90 | 119471.88 | 6426.28 | 1804.00 |
| C-14 | 221271.28 | 119085.42 | 6613.28 | 2123.80 |
| C-15 | 221950.85 | 119772.14 | 6339.04 | 1933.20 |
| C-16 | 219147.54 | 119209.68 | 6247.86 | 2131.70 |
| C-17 | 219886.62 | 118711.94 | 6544.26 | 2281.50 |
| C-18 | 222649.13 | 119823.48 | 6168.32 | 2361.00 |
| C-19 | 219887.00 | 120178.00 | 6170.00 | 2280.00 |
| C-20 | 220787.00 | 120878.00 | 6105.00 | 2543.00 |
| C-24 | 222009.45 | 120671.11 | 6069.80 | 1000.00 |
| C-25 | 219289.66 | 119889.95 | 6019.00 | 1011.00 |
| C-26 | 221432.92 | 121338.14 | 5767.50 | 1193.00 |
| C06-27 | 220207.88 | 120031.89 | 6351.39 | 1849.00 |
| C06-28 | 220816.79 | 119539.82 | 6321.08 | 1711.00 |
| C07-29 | 221246.65 | 119778.87 | 6343.67 | 2281.70 |
| C07-30 | 219616.75 | 119732.18 | 6213.05 | 2416.50 |
| C07-31 | 221243.31 | 119792.48 | 6342.25 | 2104.00 |
| C07-32 | 220822.61 | 119558.40 | 6323.57 | 2044.00 |
| C07-33 | 221227.04 | 118476.72 | 6796.80 | 2095.00 |
| C07-34 | 220487.36 | 118658.32 | 6534.18 | 1769.00 |
| C08-35 | 220480.40 | 118655.20 | 6533.21 | 2817.00 |
| C08-36 | 219448.70 | 119335.30 | 6274.59 | 2488.00 |
| C08-37 | 221246.80 | 119780.40 | 6341.47 | 2195.00 |
| C08-38 | 220480.40 | 118655.20 | 6533.21 | 2445.00 |
| C08-39 | 220813.20 | 118917.90 | 6575.13 | 2688.00 |
| C08-40 | 220791.40 | 119530.10 | 6321.42 | 2252.00 |
| C08-41 | 218951.00 | 119663.70 | 6219.92 | 3018.00 |
| C08-42 | 219911.00 | 118748.90 | 6549.23 | 2707.00 |
| C08-43C | 220052.80 | 120612.80 | 6173.79 | 1313.00 |
| C08-44 | 221515.90 | 118085.10 | 6739.37 | 3047.00 |
| C08-45 | 218821.40 | 119802.30 | 6183.65 | 1800.00 |
| RC-21 | 220541.00 | 120511.00 | 6202.00 | 1000.00 |
| RC-22 | 220412.00 | 119913.00 | 6239.00 | 670.00 |
| RC-23 | 219420.00 | 120695.00 | 5827.00 | 960.00 |

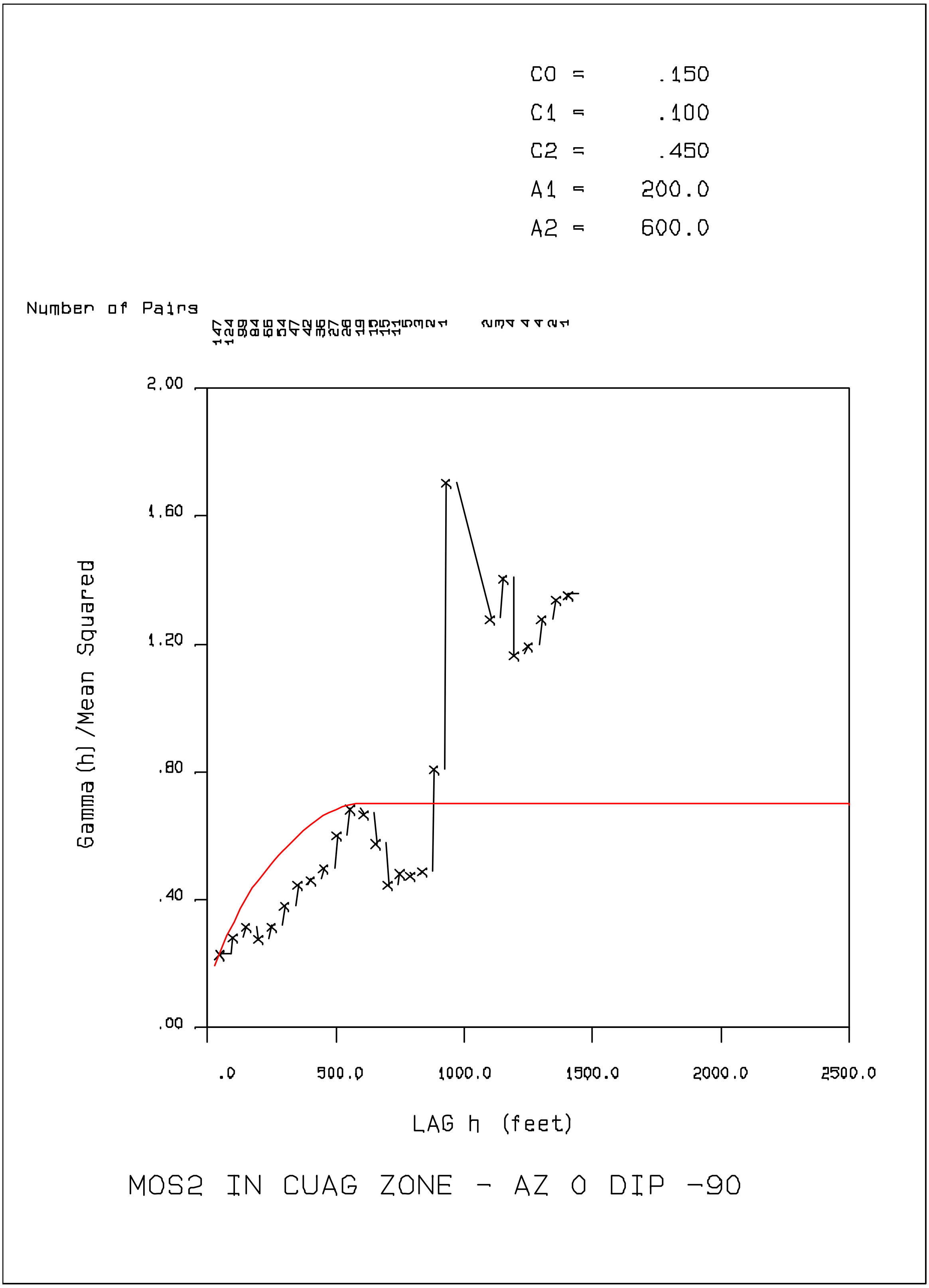
# APPENDIX 3
# SEMIVARIOGRAMS

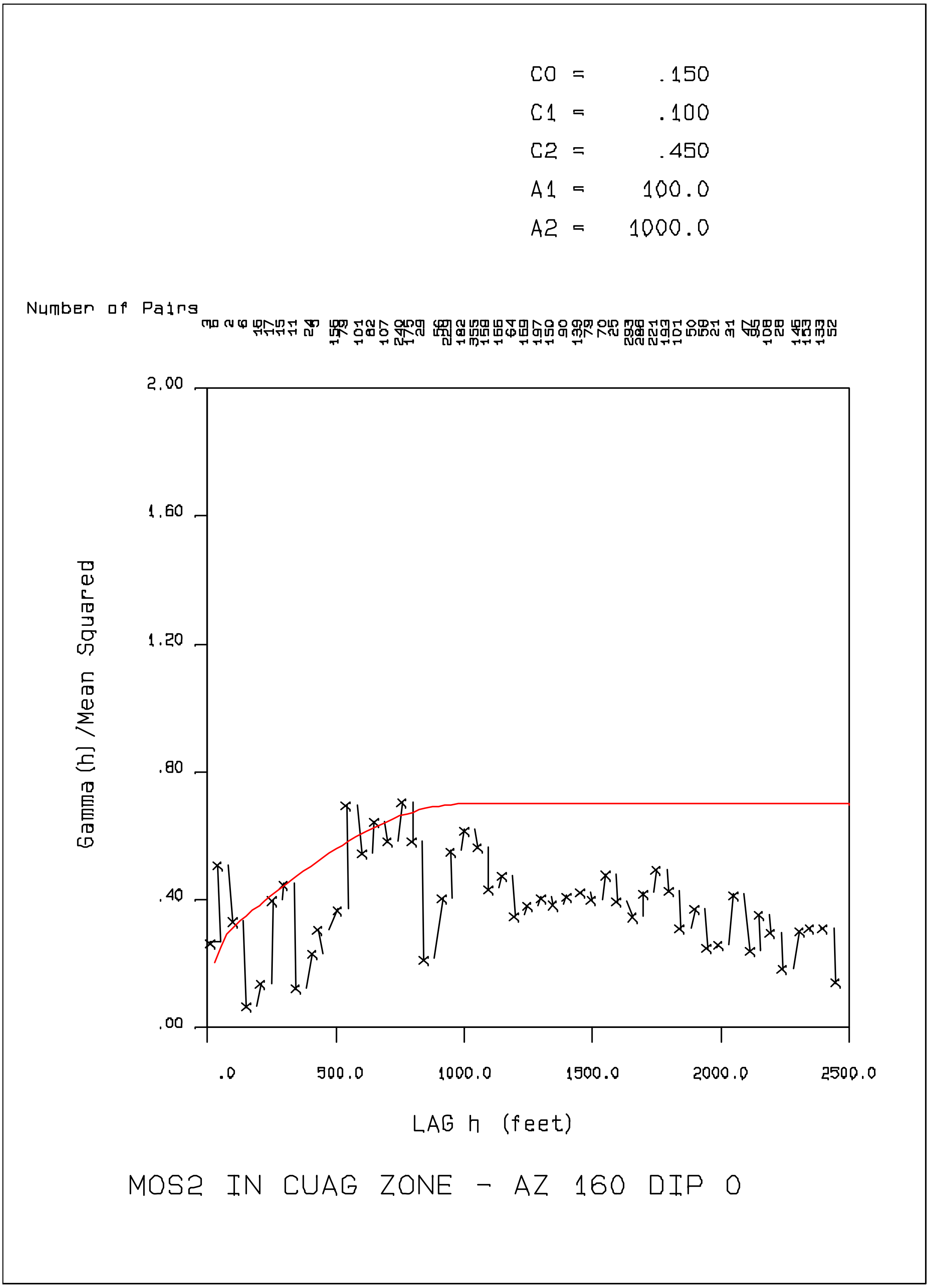


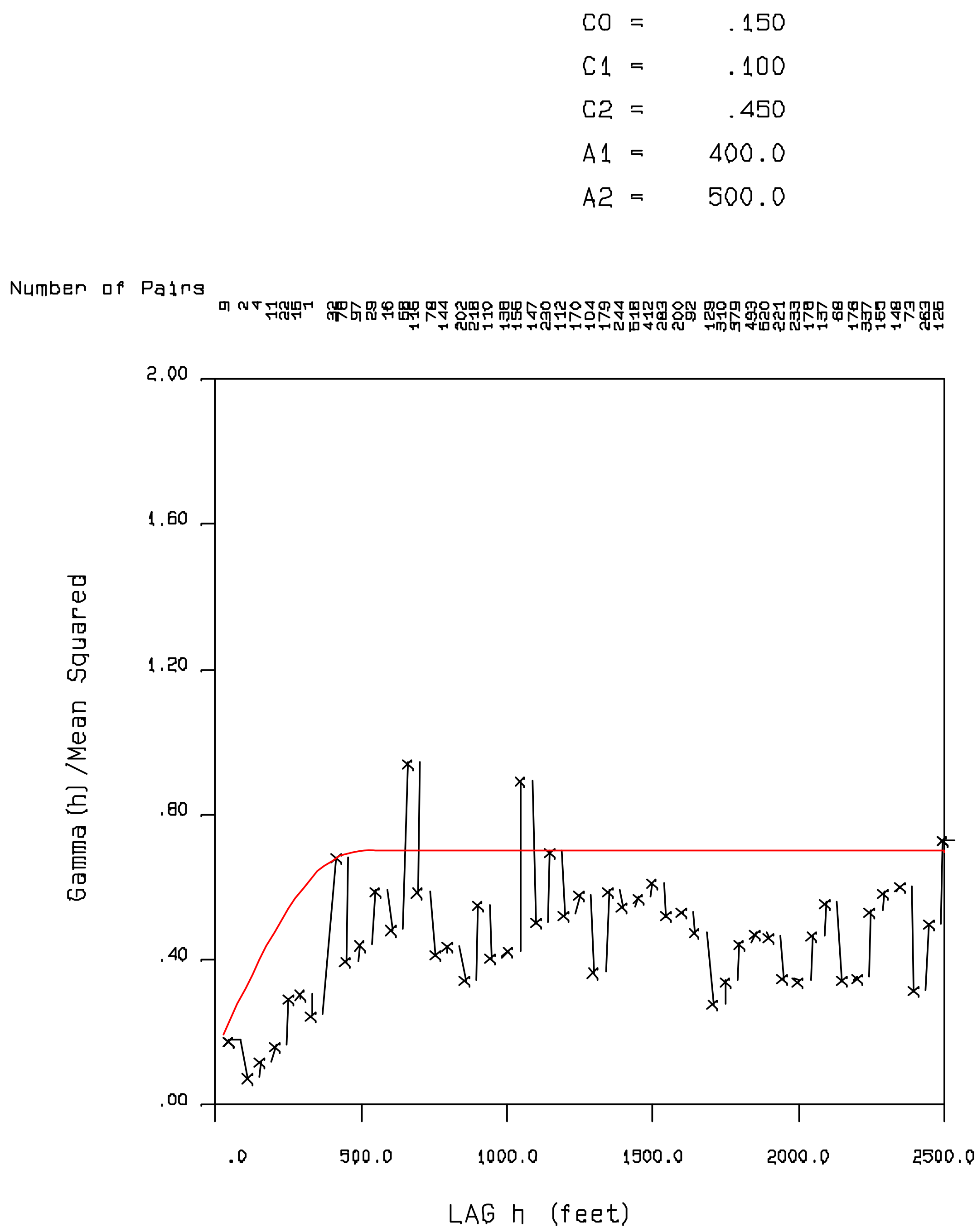

C0 = .060
C1 = .080
C2 = .160
A1 = 150.0
A2 = 1300.0
Number of Pairs
52 52 67 95 111 130 147 192 233 262 320 321 357 482 533 594 589 600 575 588 581 581 507 536 690 692 646 618 565 618 472 456 418 387 330 290 238 217 144 133 120 77 98 94 71 67 59 31 25 18
1.00
.80
.60
.40
.20
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
MOS2 IN CUMO & MO ZONES - AZ 150 DIP -55



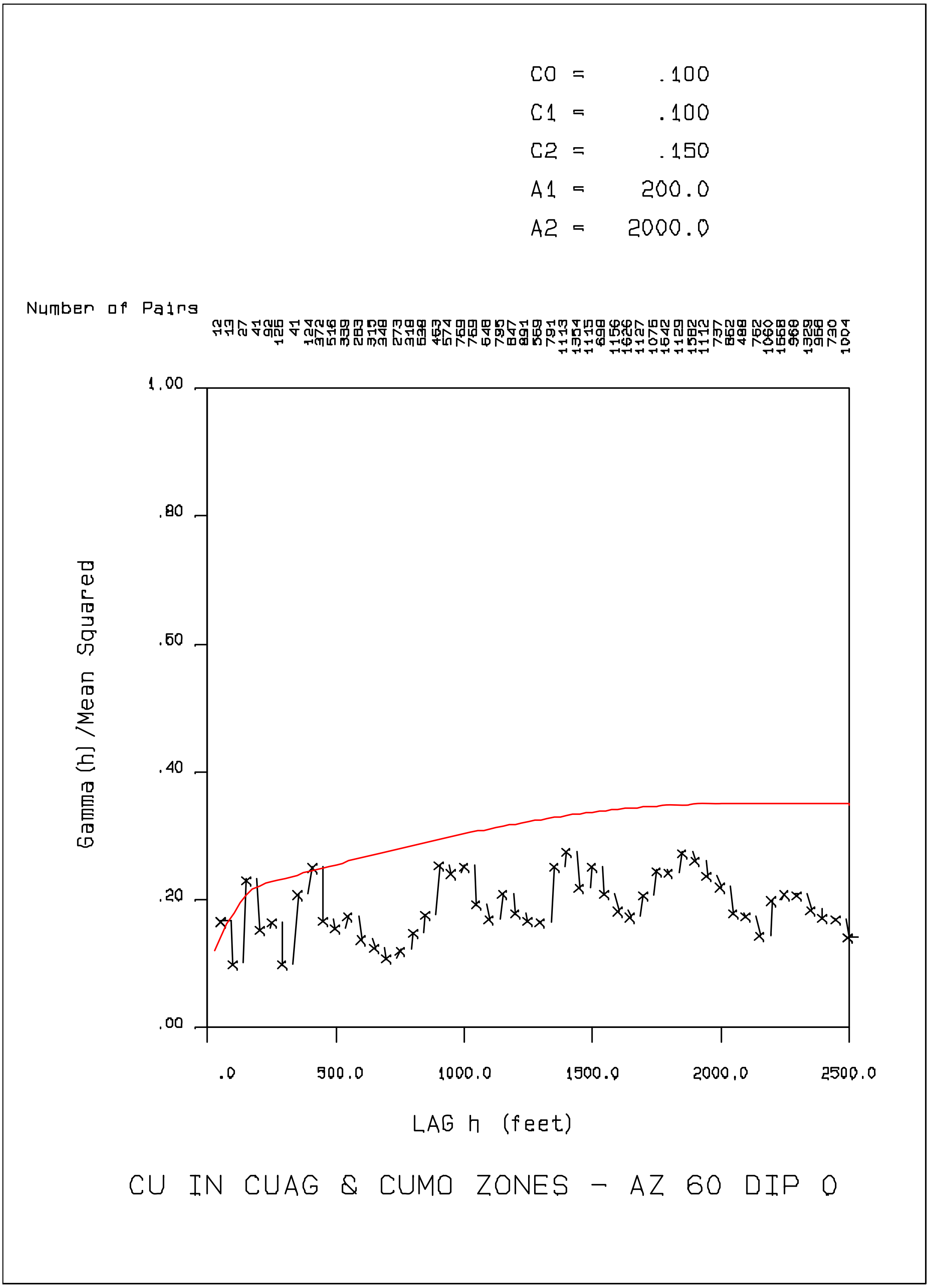
C0 = .060
C1 = .080
C2 = .160
A1 = 200.0
A2 = 480.0
Number of Pairs
20 32 42 52 54 137 199 260 300 384 591 676 604 650 614 563 552 657 721 717 793 698 562 660 575 521 520 475 397 364 337 257 227 244 221 192 196 200 181 155 106 96 59 49 29 31 19 10 13
1.00
.80
.60
.40
.20
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
MOS2 IN CUMO & MO ZONES - AZ 330 DIP -35

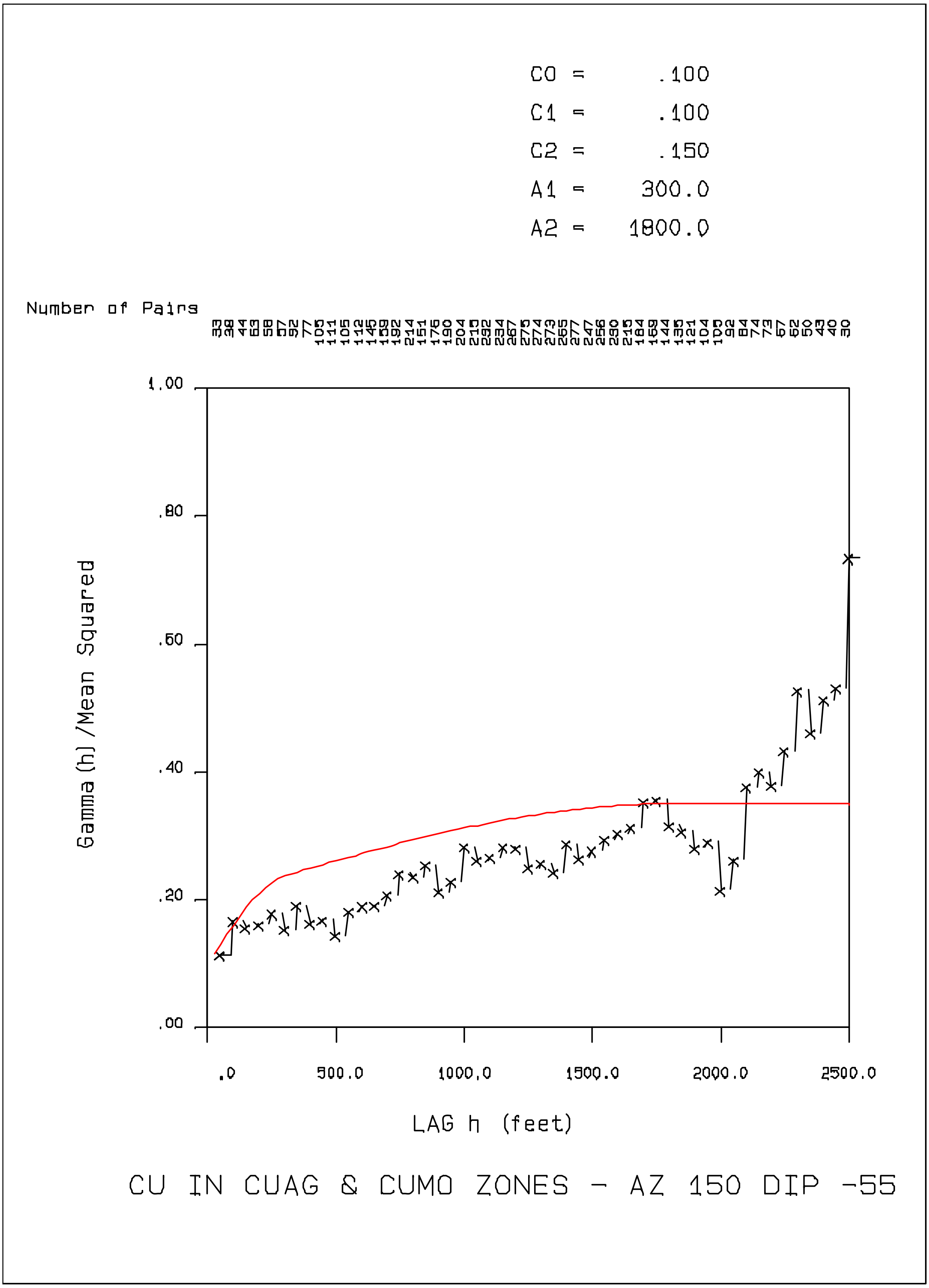

C0 = .150
C1 = .100
C2 = .450
A1 = 200.0
A2 = 600.0
Number of Pairs
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
MOS2 IN CUAG ZONE - AZ 0 DIP -90

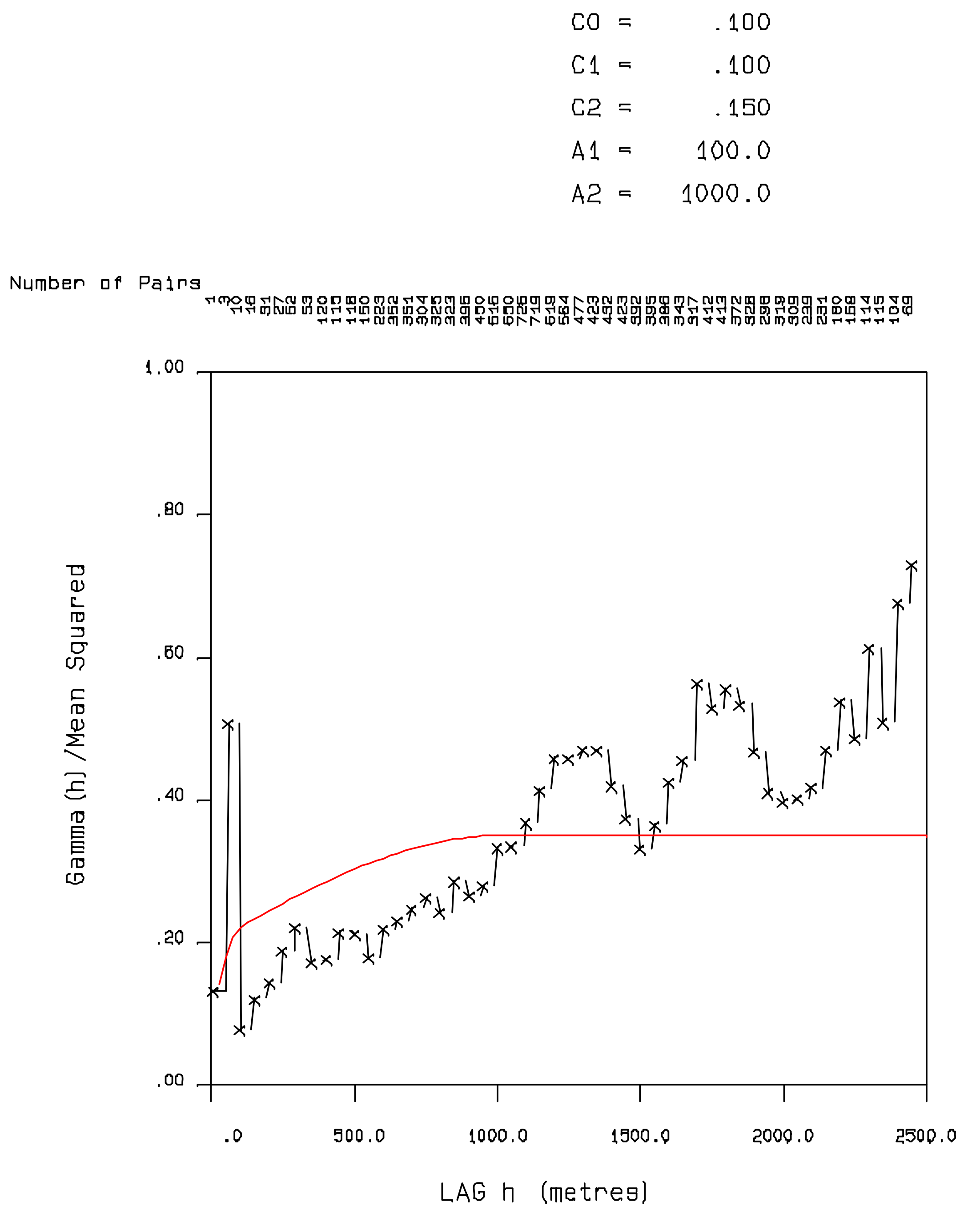

C0 = .150
C1 = .100
C2 = .450
A1 = 100.0
A2 = 1000.0
Number of Pairs
Gamma (h) /Mean Squared
2.00
1.60
1.20
.80
.40
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
MOS2 IN CUAG ZONE - AZ 160 DIP 0



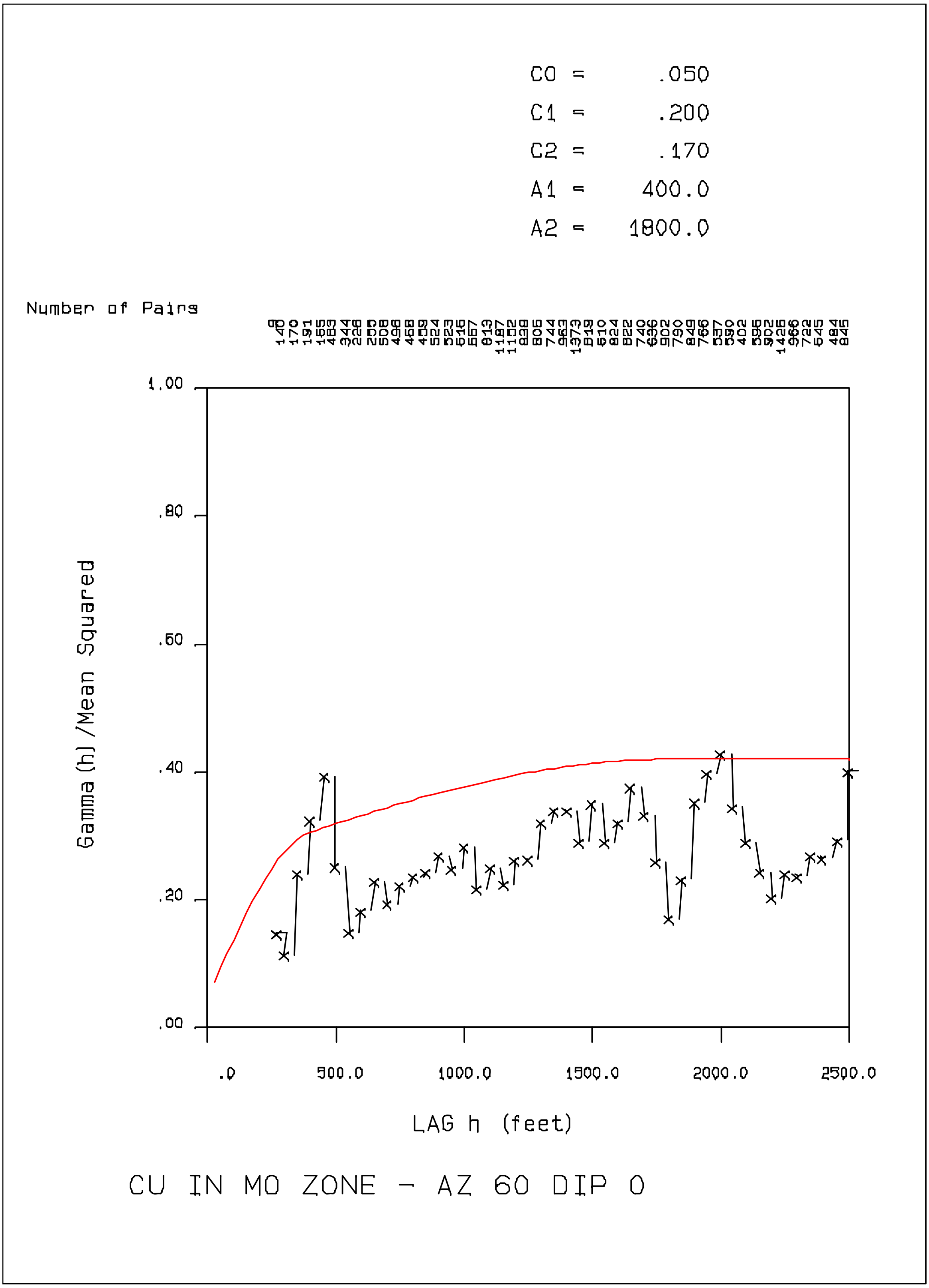

MOS2 IN CUAG ZONE - AZ 70 DIP 0

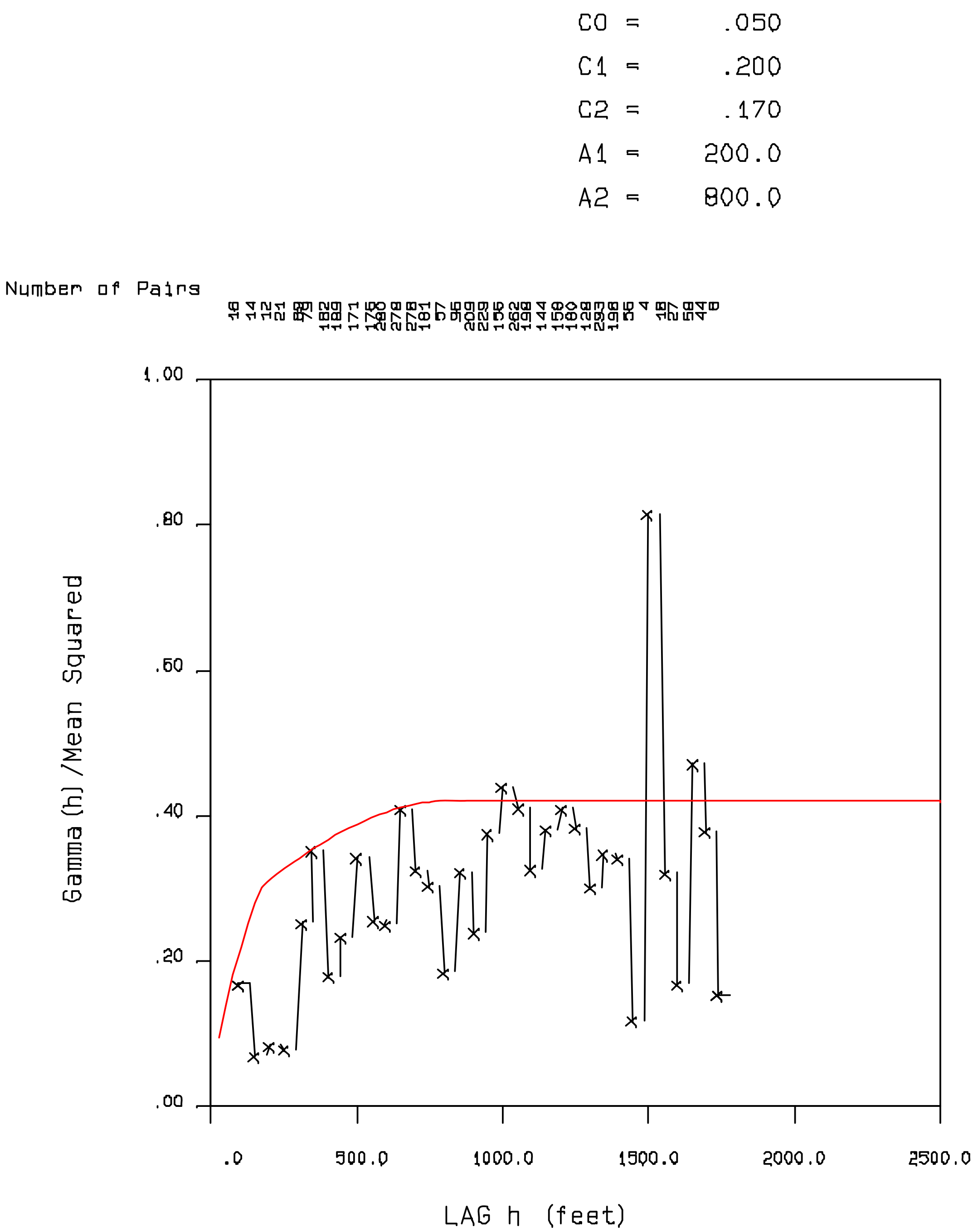

C0 = .100
C1 = .100
C2 = .150
A1 = 200.0
A2 = 2000.0
Number of Pairs
Gamma (h) /Mean Squared
1.00
.80
.60
.40
.20
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CU IN CUAG & CUMO ZONES - AZ 60 DIP 0


C0 = .100
C1 = .100
C2 = .150
A1 = 300.0
A2 = 1800.0
Number of Pairs
33 38 44 63 58 57 92 77 103 111 105 112 145 159 192 214 151 176 190 204 215 232 234 267 275 274 273 265 277 247 256 230 215 184 169 144 135 121 104 105 92 84 74 73 57 62 50 43 40 30
1.00
.80
.60
.40
.20
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CU IN CUAG & CUMO ZONES - AZ 150 DIP -55

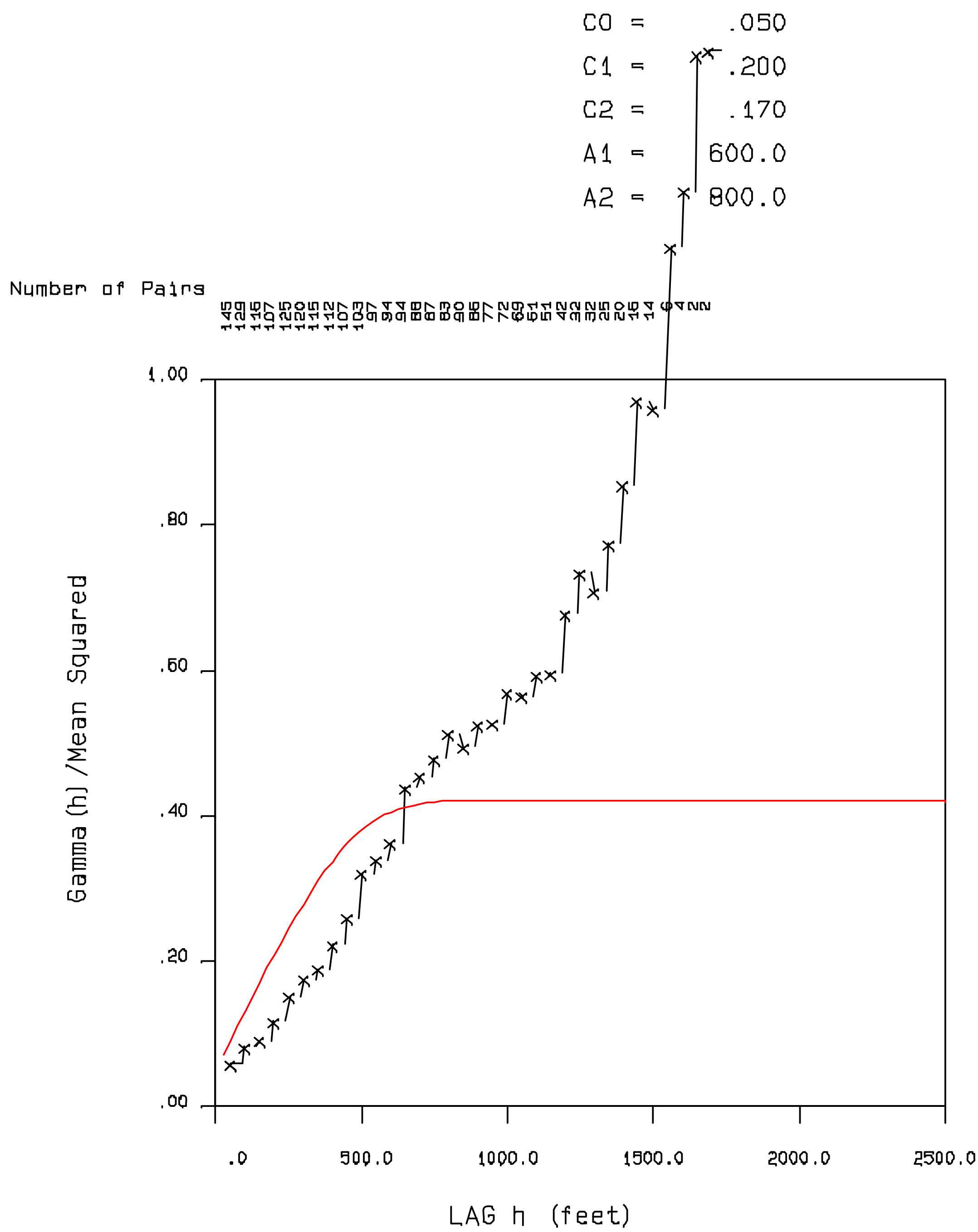



CU IN CUAG & CUMO ZONES - AZ 330 DIP -35


C0 = .050
C1 = .200
C2 = .170
A1 = 400.0
A2 = 1800.0
Number of Pairs
1.00
.80
.60
.40
.20
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CU IN MO ZONE - AZ 60 DIP 0




CU IN MO ZONE - AZ 330 DIP 0




CU IN MO ZONE - AZ 0 DIP -90

# APPENDIX 4
# ECONOMIC MODELLING SUMMARIES

CUMO
PRELIMINARY FEASIBILITY STUDY
ECONOMIC ANALYSIS - 50 kT/d

SUMMARY OF PROJECT RESULTS
BASE CASE METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 3,915.3 |
| INTERNAL RATE OF RETURN | % | | 19.0 |
| PAYBACK PERIOD | years | | 4.9 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 6.1 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 18,000 | 720,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 16,247 | 450,056 |
| - Molybdenum in Concentrate | short tons | 9,778 | 438,730 |
| - Silver Production | oz | 1,104,063 | 32,633,263 |
| - Rhenium | kg | 596 | 26,757 |
| - Sulfuric Acid | short tons | 21,051 | 944,525 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 16.0 | 16.0 |
| COPPER BASE PRICE | $US/lb | 2.10 | 2.10 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 464.8 | 20,852.8 |
| Copper Gross Metal Value | $USM | 65.8 | 1,824.1 |
| Silver Gross Metal Value | $USM | 12.3 | 364.2 |
| Rhenium Gross Metal Value | $USM | 3.9 | 173.9 |
| Sulfuric Acid Gross Value | $USM | 2.8 | 127.5 |
| TOTAL GROSS METAL VALUE | $USM | 549.6 | 23,342.5 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 4.48 | 4.48 |
| PLANT | $US/T (short) ore | 5.04 | 5.04 |
| ROASTER | $US/T (short) ore | 0.81 | 0.91 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.30 | 0.30 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **10.62** | **10.78** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.60** | **0.44** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **11.23** | **11.22** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 191.2 | 7,759.8 |
| TOTAL REALISATION COSTS | $USM | 10.9 | 315.1 |
| **TOTAL OPERATING COSTS** | **$USM** | **202.1** | **8,074.9** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 592.7 |
| ROASTER CAPITAL | $USM | | 120.4 |
| MINING FLEET CAPITAL | $USM | | 99.8 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 748.7 |
| TAILINGS | $USM | | 38.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **1,599.8** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 5.9 | 237.1 |
| ROASTER | $USM | 1.2 | 48.2 |
| MINING | $USM | 0.2 | 305.3 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 188.8 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **7.3** | **779.4** |
| **TOTAL CAPITAL** | **$USM** | **7.3** | **2,379.3** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 340 | 12,888 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 29.9 | 32.0 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 5.88 | 5.63 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.77 | 0.73 |

Economics50ktpd Rev D.xls Summary 08/10/09 1:27 PM Ausenco Confidential

CUMO
PRELIMINARY FEASIBILITY STUDY
ECONOMIC ANALYSIS - 50 kT/d

SUMMARY OF PROJECT RESULTS
LOW METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | (453.9) |
| INTERNAL RATE OF RETURN | % | | 2.6 |
| PAYBACK PERIOD | years | | 25.1 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | No Payback |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 18,000 | 720,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 16,247 | 450,056 |
| - Molybdenum in Concentrate | short tons | 9,778 | 438,730 |
| - Silver Production | oz | 1,104,063 | 32,633,263 |
| - Rhenium | kg | 596 | 26,757 |
| - Sulfuric Acid | short tons | 21,051 | 944,525 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 7.5 | 7.5 |
| COPPER BASE PRICE | $US/lb | 1.50 | 1.50 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 217.9 | 9,774.8 |
| Copper Gross Metal Value | $USM | 47.0 | 1,302.9 |
| Silver Gross Metal Value | $USM | 9.2 | 273.1 |
| Rhenium Gross Metal Value | $USM | 1.5 | 66.9 |
| Sulfuric Acid Gross Value | $USM | 1.8 | 80.3 |
| TOTAL GROSS METAL VALUE | $USM | 277.4 | 11,498.0 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 4.48 | 4.48 |
| PLANT | $US/T (short) ore | 5.04 | 5.04 |
| ROASTER | $US/T (short) ore | 0.81 | 0.91 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.30 | 0.30 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **10.62** | **10.78** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.60** | **0.44** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **11.23** | **11.22** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 191.2 | 7,759.8 |
| TOTAL REALISATION COSTS | $USM | 10.9 | 315.1 |
| **TOTAL OPERATING COSTS** | **$USM** | **202.1** | **8,074.9** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 592.7 |
| ROASTER CAPITAL | $USM | | 120.4 |
| MINING FLEET CAPITAL | $USM | | 99.8 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 748.7 |
| TAILINGS | $USM | | 38.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **1,599.8** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 5.9 | 237.1 |
| ROASTER | $USM | 1.2 | 48.2 |
| MINING | $USM | 0.2 | 305.3 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 188.8 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **7.3** | **779.4** |
| **TOTAL CAPITAL** | **$USM** | **7.3** | **2,379.3** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 68 | 1,044 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 14.8 | 15.5 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 5.46 | 5.27 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 1.09 | 1.05 |

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SUMMARY OF PROJECT RESULTS
HIGH METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 10,232.4 |
| INTERNAL RATE OF RETURN | % | | 35.8 |
| PAYBACK PERIOD | years | | 2.4 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 2.8 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 18,000 | 720,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 16,247 | 450,056 |
| - Molybdenum in Concentrate | short tons | 9,778 | 438,730 |
| - Silver Production | oz | 1,104,063 | 32,633,263 |
| - Rhenium | kg | 596 | 26,757 |
| - Sulfuric Acid | short tons | 21,051 | 944,525 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 28.0 | 28.0 |
| COPPER BASE PRICE | $US/lb | 3.50 | 3.50 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 813.3 | 36,492.4 |
| Copper Gross Metal Value | $USM | 109.7 | 3,040.1 |
| Silver Gross Metal Value | $USM | 15.4 | 455.2 |
| Rhenium Gross Metal Value | $USM | 6.0 | 267.6 |
| Sulfuric Acid Gross Value | $USM | 4.9 | 222.0 |
| TOTAL GROSS METAL VALUE | $USM | 949.4 | 40,477.3 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 4.48 | 4.48 |
| PLANT | $US/T (short) ore | 5.04 | 5.04 |
| ROASTER | $US/T (short) ore | 0.81 | 0.91 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.30 | 0.30 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **10.62** | **10.78** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.69** | **0.50** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **11.32** | **11.28** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 191.2 | 7,759.8 |
| TOTAL REALISATION COSTS | $USM | 12.4 | 358.5 |
| **TOTAL OPERATING COSTS** | **$USM** | **203.7** | **8,118.3** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 592.7 |
| ROASTER CAPITAL | $USM | | 120.4 |
| MINING FLEET CAPITAL | $USM | | 99.8 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 748.7 |
| TAILINGS | $USM | | 38.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **1,599.8** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 5.9 | 237.1 |
| ROASTER | $USM | 1.2 | 48.2 |
| MINING | $USM | 0.2 | 305.3 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 188.8 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **7.3** | **779.4** |
| **TOTAL CAPITAL** | **$USM** | **7.3** | **2,379.3** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 738 | 29,980 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 52.1 | 55.7 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 6.01 | 5.62 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.75 | 0.70 |

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SUMMARY OF PROJECT RESULTS
CYCLICAL METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 5,356.0 |
| INTERNAL RATE OF RETURN | % | | 25.5 |
| PAYBACK PERIOD | years | | 2.8 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 3.3 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 18,000 | 720,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 16,247 | 450,056 |
| - Molybdenum in Concentrate | short tons | 9,778 | 438,730 |
| - Silver Production | oz | 1,104,063 | 32,633,263 |
| - Rhenium | kg | 596 | 26,757 |
| - Sulfuric Acid | short tons | 21,051 | 944,525 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 28.0 | 18.2 |
| COPPER BASE PRICE | $US/lb | 3.50 | 2.69 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 813.3 | 23,697.7 |
| Copper Gross Metal Value | $USM | 109.7 | 2,333.8 |
| Silver Gross Metal Value | $USM | 15.4 | 367.5 |
| Rhenium Gross Metal Value | $USM | 6.0 | 169.3 |
| Sulfuric Acid Gross Value | $USM | 4.9 | 136.5 |
| TOTAL GROSS METAL VALUE | $USM | 949.4 | 26,704.7 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 4.48 | 4.48 |
| PLANT | $US/T (short) ore | 5.04 | 5.04 |
| ROASTER | $US/T (short) ore | 0.81 | 0.91 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.30 | 0.30 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **10.62** | **10.78** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.69** | **0.45** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **11.32** | **11.22** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 191.2 | 7,759.8 |
| TOTAL REALISATION COSTS | $USM | 12.4 | 321.5 |
| **TOTAL OPERATING COSTS** | **$USM** | **203.7** | **8,081.3** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 592.7 |
| ROASTER CAPITAL | $USM | | 120.4 |
| MINING FLEET CAPITAL | $USM | | 99.8 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 748.7 |
| TAILINGS | $USM | | 38.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **1,599.8** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 5.9 | 237.1 |
| ROASTER | $USM | 1.2 | 48.2 |
| MINING | $USM | 0.2 | 305.3 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 188.8 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **7.3** | **779.4** |
| **TOTAL CAPITAL** | **$USM** | **7.3** | **2,379.3** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 738 | 16,244 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 52.1 | 36.6 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 6.01 | 5.50 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.75 | 0.81 |

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SUMMARY OF PROJECT RESULTS
BASE CASE METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 9,760.8 |
| INTERNAL RATE OF RETURN | % | | 29.1 |
| PAYBACK PERIOD | years | | 3.0 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 3.5 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 36,000 | 1,440,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 25,596 | 750,435 |
| - Molybdenum in Concentrate | short tons | 21,684 | 864,263 |
| - Silver Production | oz | 1,889,010 | 57,809,099 |
| - Rhenium | kg | 1,322 | 52,709 |
| - Sulfuric Acid | short tons | 46,682 | 1,860,637 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 16.0 | 16.0 |
| COPPER BASE PRICE | $US/lb | 2.10 | 2.10 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 1,030.6 | 41,078.3 |
| Copper Gross Metal Value | $USM | 103.7 | 3,041.5 |
| Silver Gross Metal Value | $USM | 21.1 | 645.1 |
| Rhenium Gross Metal Value | $USM | 8.6 | 342.6 |
| Sulfuric Acid Gross Value | $USM | 6.3 | 251.2 |
| TOTAL GROSS METAL VALUE | $USM | 1,170.3 | 45,358.8 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| MINING | $US/T (short) ore | 2.34 | 2.34 |
| PLANT | $US/T (short) ore | 4.71 | 4.70 |
| ROASTER | $US/T (short) ore | 0.89 | 0.89 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.20 | 0.20 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **8.14** | **8.18** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.49** | **0.37** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **8.63** | **8.55** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 293.0 | 11,779.4 |
| TOTAL REALISATION COSTS | $USM | 17.7 | 537.1 |
| **TOTAL OPERATING COSTS** | **$USM** | **310.7** | **12,316.4** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,070.9 |
| ROASTER CAPITAL | $USM | | 199.9 |
| MINING FLEET CAPITAL | $USM | | 192.9 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 692.8 |
| TAILINGS | $USM | | 81.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **2,237.7** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 10.7 | 428.3 |
| ROASTER | $USM | 2.0 | 80.0 |
| MINING | $USM | 0.1 | 719.1 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 471.9 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **12.8** | **1,699.2** |
| **TOTAL CAPITAL** | **$USM** | **12.8** | **3,936.9** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 847 | 29,092 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 32.0 | 31.1 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 4.25 | 4.34 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.56 | 0.57 |

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SUMMARY OF PROJECT RESULTS
LOW METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 1,129.6 |
| INTERNAL RATE OF RETURN | % | | 8.8 |
| PAYBACK PERIOD | years | | 9.6 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 15.9 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 36,000 | 1,440,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 25,596 | 750,435 |
| - Molybdenum in Concentrate | short tons | 21,684 | 864,263 |
| - Silver Production | oz | 1,889,010 | 57,809,099 |
| - Rhenium | kg | 1,322 | 52,709 |
| - Sulfuric Acid | short tons | 46,682 | 1,860,637 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 7.5 | 7.5 |
| COPPER BASE PRICE | $US/lb | 1.50 | 1.50 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 483.1 | 19,255.5 |
| Copper Gross Metal Value | $USM | 74.1 | 2,172.5 |
| Silver Gross Metal Value | $USM | 15.8 | 483.9 |
| Rhenium Gross Metal Value | $USM | 3.3 | 131.8 |
| Sulfuric Acid Gross Value | $USM | 4.0 | 158.2 |
| TOTAL GROSS METAL VALUE | $USM | 580.3 | 22,201.8 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| MINING | $US/T (short) ore | 2.34 | 2.34 |
| PLANT | $US/T (short) ore | 4.71 | 4.70 |
| ROASTER | $US/T (short) ore | 0.89 | 0.89 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.20 | 0.20 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **8.14** | **8.18** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.49** | **0.37** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **8.63** | **8.55** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 293.0 | 11,779.4 |
| TOTAL REALISATION COSTS | $USM | 17.7 | 537.1 |
| **TOTAL OPERATING COSTS** | **$USM** | **310.7** | **12,316.4** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,070.9 |
| ROASTER CAPITAL | $USM | | 199.9 |
| MINING FLEET CAPITAL | $USM | | 192.9 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 692.8 |
| TAILINGS | $USM | | 81.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **2,237.7** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 10.7 | 428.3 |
| ROASTER | $USM | 2.0 | 80.0 |
| MINING | $USM | 0.1 | 719.1 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 471.9 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **12.8** | **1,699.2** |
| **TOTAL CAPITAL** | **$USM** | **12.8** | **3,936.9** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 257 | 5,935 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 15.6 | 15.0 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 4.02 | 4.16 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.80 | 0.83 |

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SUMMARY OF PROJECT RESULTS
HIGH METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 22,198.0 |
| INTERNAL RATE OF RETURN | % | | 51.1 |
| PAYBACK PERIOD | years | | 1.6 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 1.8 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 36,000 | 1,440,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 25,596 | 750,435 |
| - Molybdenum in Concentrate | short tons | 21,684 | 864,263 |
| - Silver Production | oz | 1,889,010 | 57,809,099 |
| - Rhenium | kg | 1,322 | 52,709 |
| - Sulfuric Acid | short tons | 46,682 | 1,860,637 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 28.0 | 28.0 |
| COPPER BASE PRICE | $US/lb | 3.50 | 3.50 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 1,803.6 | 71,887.1 |
| Copper Gross Metal Value | $USM | 172.9 | 5,069.2 |
| Silver Gross Metal Value | $USM | 26.4 | 806.4 |
| Rhenium Gross Metal Value | $USM | 13.2 | 527.1 |
| Sulfuric Acid Gross Value | $USM | 11.0 | 437.2 |
| TOTAL GROSS METAL VALUE | $USM | 2,027.0 | 78,727.1 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| MINING | $US/T (short) ore | 2.34 | 2.34 |
| PLANT | $US/T (short) ore | 4.71 | 4.70 |
| ROASTER | $US/T (short) ore | 0.89 | 0.89 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.20 | 0.20 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **8.14** | **8.18** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.56** | **0.42** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **8.70** | **8.60** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 293.0 | 11,779.4 |
| TOTAL REALISATION COSTS | $USM | 20.1 | 609.5 |
| **TOTAL OPERATING COSTS** | **$USM** | **313.1** | **12,388.8** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,070.9 |
| ROASTER CAPITAL | $USM | | 199.9 |
| MINING FLEET CAPITAL | $USM | | 192.9 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 692.8 |
| TAILINGS | $USM | | 81.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **2,237.7** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 10.7 | 428.3 |
| ROASTER | $USM | 2.0 | 80.0 |
| MINING | $USM | 0.1 | 719.1 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 471.9 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **12.8** | **1,699.2** |
| **TOTAL CAPITAL** | **$USM** | **12.8** | **3,936.9** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 1,701 | 62,388 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 55.7 | 54.2 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 4.33 | 4.41 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.54 | 0.55 |

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SUMMARY OF PROJECT RESULTS
CYCLICAL METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 12,481.2 |
| INTERNAL RATE OF RETURN | % | | 39.3 |
| PAYBACK PERIOD | years | | 1.9 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 2.2 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 36,000 | 1,440,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 25,596 | 750,435 |
| - Molybdenum in Concentrate | short tons | 21,684 | 864,263 |
| - Silver Production | oz | 1,889,010 | 57,809,099 |
| - Rhenium | kg | 1,322 | 52,709 |
| - Sulfuric Acid | short tons | 46,682 | 1,860,637 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 28.0 | 18.1 |
| COPPER BASE PRICE | $US/lb | 3.50 | 2.72 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 1,803.6 | 46,440.5 |
| Copper Gross Metal Value | $USM | 172.9 | 3,942.2 |
| Silver Gross Metal Value | $USM | 26.4 | 653.8 |
| Rhenium Gross Metal Value | $USM | 13.2 | 332.5 |
| Sulfuric Acid Gross Value | $USM | 11.0 | 268.2 |
| TOTAL GROSS METAL VALUE | $USM | 2,027.0 | 51,637.2 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| MINING | $US/T (short) ore | 2.34 | 2.34 |
| PLANT | $US/T (short) ore | 4.71 | 4.70 |
| ROASTER | $US/T (short) ore | 0.89 | 0.89 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.20 | 0.20 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **8.14** | **8.18** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.56** | **0.38** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **8.70** | **8.56** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 293.0 | 11,779.4 |
| TOTAL REALISATION COSTS | $USM | 20.1 | 548.8 |
| **TOTAL OPERATING COSTS** | **$USM** | **313.1** | **12,328.1** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,070.9 |
| ROASTER CAPITAL | $USM | | 199.9 |
| MINING FLEET CAPITAL | $USM | | 192.9 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 692.8 |
| TAILINGS | $USM | | 81.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **2,237.7** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 10.7 | 428.3 |
| ROASTER | $USM | 2.0 | 80.0 |
| MINING | $USM | 0.1 | 719.1 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 471.9 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **12.8** | **1,699.2** |
| **TOTAL CAPITAL** | **$USM** | **12.8** | **3,936.9** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 1,701 | 35,359 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 55.7 | 35.5 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 4.33 | 4.32 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.54 | 0.65 |

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SUMMARY OF PROJECT RESULTS
BASE CASE METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 15,717.6 |
| INTERNAL RATE OF RETURN | % | | 35.7 |
| PAYBACK PERIOD | years | | 2.3 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 2.7 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 54,000 | 2,160,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 39,168 | 1,078,795 |
| - Molybdenum in Concentrate | short tons | 33,858 | 1,282,505 |
| - Silver Production | oz | 2,730,733 | 84,718,784 |
| - Rhenium | kg | 2,065 | 78,217 |
| - Sulfuric Acid | short tons | 72,892 | 2,761,052 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 16.0 | 16.0 |
| COPPER BASE PRICE | $US/lb | 2.10 | 2.10 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 1,609.3 | 60,957.3 |
| Copper Gross Metal Value | $USM | 158.7 | 4,372.4 |
| Silver Gross Metal Value | $USM | 30.5 | 945.5 |
| Rhenium Gross Metal Value | $USM | 13.4 | 508.4 |
| Sulfuric Acid Gross Value | $USM | 9.8 | 372.7 |
| TOTAL GROSS METAL VALUE | $USM | 1,821.8 | 67,156.3 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 1.50 | 1.50 |
| PLANT | $US/T (short) ore | 4.65 | 4.65 |
| ROASTER | $US/T (short) ore | 0.93 | 0.88 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.16 | 0.16 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **7.24** | **7.25** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.50** | **0.36** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **7.74** | **7.61** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 391.0 | 15,652.0 |
| TOTAL REALISATION COSTS | $USM | 27.0 | 775.8 |
| **TOTAL OPERATING COSTS** | **$USM** | **418.0** | **16,427.8** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,536.1 |
| ROASTER CAPITAL | $USM | | 274.4 |
| MINING FLEET CAPITAL | $USM | | 271.6 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 644.2 |
| TAILINGS | $USM | | 81.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **2,807.5** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 15.4 | 614.4 |
| ROASTER | $USM | 2.7 | 109.8 |
| MINING | $USM | 0.1 | 1,019.8 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 710.5 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **18.2** | **2,454.5** |
| **TOTAL CAPITAL** | **$USM** | **18.2** | **5,261.9** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 1,385 | 45,467 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 33.2 | 30.7 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 3.67 | 3.91 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.48 | 0.51 |

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SUMMARY OF PROJECT RESULTS
LOW METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 2,870.5 |
| INTERNAL RATE OF RETURN | % | | 12.4 |
| PAYBACK PERIOD | years | | 6.4 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 10.1 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 54,000 | 2,160,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 39,168 | 1,078,795 |
| - Molybdenum in Concentrate | short tons | 33,858 | 1,282,505 |
| - Silver Production | oz | 2,730,733 | 84,718,784 |
| - Rhenium | kg | 2,065 | 78,217 |
| - Sulfuric Acid | short tons | 72,892 | 2,761,052 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 7.5 | 7.5 |
| COPPER BASE PRICE | $US/lb | 1.50 | 1.50 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 754.4 | 28,573.7 |
| Copper Gross Metal Value | $USM | 113.4 | 3,123.1 |
| Silver Gross Metal Value | $USM | 22.9 | 709.1 |
| Rhenium Gross Metal Value | $USM | 5.2 | 195.5 |
| Sulfuric Acid Gross Value | $USM | 6.2 | 234.7 |
| TOTAL GROSS METAL VALUE | $USM | 902.0 | 32,836.2 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 1.50 | 1.50 |
| PLANT | $US/T (short) ore | 4.65 | 4.65 |
| ROASTER | $US/T (short) ore | 0.93 | 0.88 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.16 | 0.16 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **7.24** | **7.25** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.50** | **0.36** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **7.74** | **7.61** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 391.0 | 15,652.0 |
| TOTAL REALISATION COSTS | $USM | 27.0 | 775.8 |
| **TOTAL OPERATING COSTS** | **$USM** | **418.0** | **16,427.8** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,536.1 |
| ROASTER CAPITAL | $USM | | 274.4 |
| MINING FLEET CAPITAL | $USM | | 271.6 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 644.2 |
| TAILINGS | $USM | | 81.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **2,807.5** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 15.4 | 614.4 |
| ROASTER | $USM | 2.7 | 109.8 |
| MINING | $USM | 0.1 | 1,019.8 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 710.5 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **18.2** | **2,454.5** |
| **TOTAL CAPITAL** | **$USM** | **18.2** | **5,261.9** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 466 | 11,146 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 16.2 | 14.8 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 3.48 | 3.75 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.70 | 0.75 |

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SUMMARY OF PROJECT RESULTS
HIGH METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 34,783.6 |
| INTERNAL RATE OF RETURN | % | | 60.6 |
| PAYBACK PERIOD | years | | 1.2 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 1.4 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 54,000 | 2,160,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 39,168 | 1,078,795 |
| - Molybdenum in Concentrate | short tons | 33,858 | 1,282,505 |
| - Silver Production | oz | 2,730,733 | 84,718,784 |
| - Rhenium | kg | 2,065 | 78,217 |
| - Sulfuric Acid | short tons | 72,892 | 2,761,052 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 28.0 | 28.0 |
| COPPER BASE PRICE | $US/lb | 3.50 | 3.50 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 2,816.2 | 106,675.3 |
| Copper Gross Metal Value | $USM | 264.6 | 7,287.3 |
| Silver Gross Metal Value | $USM | 38.1 | 1,181.8 |
| Rhenium Gross Metal Value | $USM | 20.6 | 782.2 |
| Sulfuric Acid Gross Value | $USM | 17.1 | 648.8 |
| TOTAL GROSS METAL VALUE | $USM | 3,156.7 | 116,575.4 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 0.78 | 0.78 |
| PLANT | $US/T (short) ore | 4.65 | 4.65 |
| ROASTER | $US/T (short) ore | 0.93 | 0.88 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.16 | 0.16 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **6.52** | **6.53** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.57** | **0.41** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **7.09** | **6.93** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 352.1 | 14,094.6 |
| TOTAL REALISATION COSTS | $USM | 30.8 | 879.9 |
| **TOTAL OPERATING COSTS** | **$USM** | **382.9** | **14,974.5** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,536.1 |
| ROASTER CAPITAL | $USM | | 274.4 |
| MINING FLEET CAPITAL | $USM | | 271.6 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 644.2 |
| TAILINGS | $USM | | 81.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **2,807.5** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 15.4 | 614.4 |
| ROASTER | $USM | 2.7 | 109.8 |
| MINING | $USM | 0.1 | 1,019.8 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 710.5 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **18.2** | **2,454.5** |
| **TOTAL CAPITAL** | **$USM** | **18.2** | **5,261.9** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 2,756 | 96,339 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 57.9 | 53.6 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 3.40 | 3.60 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.42 | 0.45 |

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SUMMARY OF PROJECT RESULTS
CYCLICAL METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 20,739.0 |
| INTERNAL RATE OF RETURN | % | | 49.1 |
| PAYBACK PERIOD | years | | 1.5 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 1.7 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 54,000 | 2,160,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 39,168 | 1,078,795 |
| - Molybdenum in Concentrate | short tons | 33,858 | 1,282,505 |
| - Silver Production | oz | 2,730,733 | 84,718,784 |
| - Rhenium | kg | 2,065 | 78,217 |
| - Sulfuric Acid | short tons | 72,892 | 2,761,052 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 28.0 | 18.3 |
| COPPER BASE PRICE | $US/lb | 3.50 | 2.68 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 2,816.2 | 69,752.0 |
| Copper Gross Metal Value | $USM | 264.6 | 5,578.1 |
| Silver Gross Metal Value | $USM | 38.1 | 951.4 |
| Rhenium Gross Metal Value | $USM | 20.6 | 497.4 |
| Sulfuric Acid Gross Value | $USM | 17.1 | 401.1 |
| TOTAL GROSS METAL VALUE | $USM | 3,156.7 | 77,180.1 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 0.78 | 0.78 |
| PLANT | $US/T (short) ore | 4.65 | 4.65 |
| ROASTER | $US/T (short) ore | 0.93 | 0.88 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.16 | 0.16 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **6.52** | **6.53** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.57** | **0.37** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **7.09** | **6.89** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 352.1 | 14,094.6 |
| TOTAL REALISATION COSTS | $USM | 30.8 | 790.7 |
| **TOTAL OPERATING COSTS** | **$USM** | **382.9** | **14,885.3** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,536.1 |
| ROASTER CAPITAL | $USM | | 274.4 |
| MINING FLEET CAPITAL | $USM | | 271.6 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 644.2 |
| TAILINGS | $USM | | 81.2 |
| **TOTAL INITIAL CAPITAL** | $USM | | **2,807.5** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 15.4 | 614.4 |
| ROASTER | $USM | 2.7 | 109.8 |
| MINING | $USM | 0.1 | 1,019.8 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 710.5 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **18.2** | **2,454.5** |
| **TOTAL CAPITAL** | **$USM** | **18.2** | **5,261.9** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 2,756 | 57,033 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 57.9 | 35.4 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 3.40 | 3.53 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.42 | 0.52 |

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ECONOMIC ANALYSIS - 200 kT/d

SUMMARY OF PROJECT RESULTS
BASE CASE METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 20,887.2 |
| INTERNAL RATE OF RETURN | % | | 39.6 |
| PAYBACK PERIOD | years | | 2.0 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 2.3 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 72,000 | 2,880,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 41,068 | 1,463,043 |
| - Molybdenum in Concentrate | short tons | 48,180 | 1,610,763 |
| - Silver Production | oz | 2,843,233 | 115,719,431 |
| - Rhenium | kg | 2,938 | 98,236 |
| - Sulfuric Acid | short tons | 103,725 | 3,467,746 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 16.0 | 16.0 |
| COPPER BASE PRICE | $US/lb | 2.10 | 2.10 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 2,290.0 | 76,559.4 |
| Copper Gross Metal Value | $USM | 166.5 | 5,929.7 |
| Silver Gross Metal Value | $USM | 31.7 | 1,291.4 |
| Rhenium Gross Metal Value | $USM | 19.1 | 638.5 |
| Sulfuric Acid Gross Value | $USM | 14.0 | 468.1 |
| TOTAL GROSS METAL VALUE | $USM | 2,521.3 | 84,887.2 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 1.11 | 1.11 |
| PLANT | $US/T (short) ore | 4.59 | 4.59 |
| ROASTER | $US/T (short) ore | 0.99 | 0.83 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.13 | 0.13 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **6.83** | **6.72** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.41** | **0.36** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **7.24** | **7.08** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 491.7 | 19,356.4 |
| TOTAL REALISATION COSTS | $USM | 29.3 | 1,042.4 |
| **TOTAL OPERATING COSTS** | **$USM** | **521.0** | **20,398.8** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,970.2 |
| ROASTER CAPITAL | $USM | | 346.2 |
| MINING FLEET CAPITAL | $USM | | 270.7 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 658.3 |
| TAILINGS | $USM | | 162.3 |
| **TOTAL INITIAL CAPITAL** | $USM | | **3,407.6** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 19.7 | 788.1 |
| ROASTER | $USM | 3.5 | 138.5 |
| MINING | $USM | 0.1 | 939.6 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 683.1 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **23.3** | **2,549.2** |
| **TOTAL CAPITAL** | **$USM** | **23.3** | **5,956.8** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 1,977 | 58,532 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 34.6 | 29.1 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 3.31 | 3.84 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.43 | 0.50 |

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SUMMARY OF PROJECT RESULTS
LOW METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 4,366.9 |
| INTERNAL RATE OF RETURN | % | | 14.5 |
| PAYBACK PERIOD | years | | 5.6 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 7.7 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 72,000 | 2,880,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 41,068 | 1,463,043 |
| - Molybdenum in Concentrate | short tons | 48,180 | 1,610,763 |
| - Silver Production | oz | 2,843,233 | 115,719,431 |
| - Rhenium | kg | 2,938 | 98,236 |
| - Sulfuric Acid | short tons | 103,725 | 3,467,746 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 7.5 | 7.5 |
| COPPER BASE PRICE | $US/lb | 1.50 | 1.50 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 1,073.4 | 35,887.2 |
| Copper Gross Metal Value | $USM | 118.9 | 4,235.5 |
| Silver Gross Metal Value | $USM | 23.8 | 968.6 |
| Rhenium Gross Metal Value | $USM | 7.3 | 245.6 |
| Sulfuric Acid Gross Value | $USM | 8.8 | 294.8 |
| TOTAL GROSS METAL VALUE | $USM | 1,232.3 | 41,631.6 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 1.11 | 1.11 |
| PLANT | $US/T (short) ore | 4.59 | 4.59 |
| ROASTER | $US/T (short) ore | 0.99 | 0.83 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.13 | 0.13 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **6.83** | **6.72** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.41** | **0.36** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **7.24** | **7.08** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 491.7 | 19,356.4 |
| TOTAL REALISATION COSTS | $USM | 29.3 | 1,042.4 |
| **TOTAL OPERATING COSTS** | **$USM** | **521.0** | **20,398.8** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,970.2 |
| ROASTER CAPITAL | $USM | | 346.2 |
| MINING FLEET CAPITAL | $USM | | 270.7 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 658.3 |
| TAILINGS | $USM | | 162.3 |
| **TOTAL INITIAL CAPITAL** | $USM | | **3,407.6** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 19.7 | 788.1 |
| ROASTER | $USM | 3.5 | 0.0 |
| MINING | $USM | 0.1 | 0.0 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 0.0 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **23.3** | **2,549.2** |
| **TOTAL CAPITAL** | **$USM** | **23.3** | **5,956.8** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 688 | 15,276 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 16.7 | 14.1 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 3.17 | 3.67 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.63 | 0.73 |

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SUMMARY OF PROJECT RESULTS
HIGH METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 45,121.3 |
| INTERNAL RATE OF RETURN | % | | 66.1 |
| PAYBACK PERIOD | years | | 1.1 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 1.2 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 72,000 | 2,880,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 41,068 | 1,463,043 |
| - Molybdenum in Concentrate | short tons | 48,180 | 1,610,763 |
| - Silver Production | oz | 2,843,233 | 115,719,431 |
| - Rhenium | kg | 2,938 | 98,236 |
| - Sulfuric Acid | short tons | 103,725 | 3,467,746 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 28.0 | 28.0 |
| COPPER BASE PRICE | $US/lb | 3.50 | 3.50 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 4,007.5 | 133,978.9 |
| Copper Gross Metal Value | $USM | 277.4 | 9,882.9 |
| Silver Gross Metal Value | $USM | 39.7 | 1,614.3 |
| Rhenium Gross Metal Value | $USM | 29.4 | 982.4 |
| Sulfuric Acid Gross Value | $USM | 24.4 | 814.9 |
| TOTAL GROSS METAL VALUE | $USM | 4,378.3 | 147,273.4 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 0.69 | 0.69 |
| PLANT | $US/T (short) ore | 4.59 | 4.59 |
| ROASTER | $US/T (short) ore | 0.99 | 0.83 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.13 | 0.13 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **6.41** | **6.30** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.46** | **0.41** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **6.87** | **6.71** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 461.6 | 18,153.3 |
| TOTAL REALISATION COSTS | $USM | 33.3 | 1,183.5 |
| **TOTAL OPERATING COSTS** | **$USM** | **494.9** | **19,336.9** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,970.2 |
| ROASTER CAPITAL | $USM | | 346.2 |
| MINING FLEET CAPITAL | $USM | | 270.7 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 658.3 |
| TAILINGS | $USM | | 162.3 |
| **TOTAL INITIAL CAPITAL** | $USM | | **3,407.6** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 19.7 | 788.1 |
| ROASTER | $USM | 3.5 | 0.0 |
| MINING | $USM | 0.1 | 0.0 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 0.0 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **23.3** | **2,549.2** |
| **TOTAL CAPITAL** | **$USM** | **23.3** | **5,956.8** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 3,860 | 121,980 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 60.3 | 50.7 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 3.17 | 3.68 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.40 | 0.46 |

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ECONOMIC ANALYSIS - 200 kT/d

SUMMARY OF PROJECT RESULTS
CYCLICAL METALS PRICES

| ITEM | UNIT | Year 3 (Typical) | TOTAL |
|---|---|---|---|
| **PROJECT ECONOMICS** | | | |
| **BASED ON EBITD&A** | | | |
| NPV AT A DISCOUNT RATE OF 5.0 % | $USM | | 27,445.0 |
| INTERNAL RATE OF RETURN | % | | 54.6 |
| PAYBACK PERIOD | years | | 1.3 |
| DISCOUNTED PAYBACK PERIOD @ 5.0 % | years | | 1.5 |
| **PRODUCTION** | | | |
| QUANTITY ORE TREATED | 000 short tons | 72,000 | 2,880,000 |
| METAL PRODUCED - Molybdenum and Copper in Concentrate | | | |
| - Copper in Concentrate | short tons | 41,068 | 1,463,043 |
| - Molybdenum in Concentrate | short tons | 48,180 | 1,610,763 |
| - Silver Production | oz | 2,843,233 | 115,719,431 |
| - Rhenium | kg | 2,938 | 98,236 |
| - Sulfuric Acid | short tons | 103,725 | 3,467,746 |
| **REVENUE** | | | |
| TRI MOLYBDENUM OXIDE BASE PRICE | $US/lb | 28.0 | 18.6 |
| COPPER BASE PRICE | $US/lb | 3.50 | 2.56 |
| COPPER CONCENTRATE | | | |
| Molybdenum Oxide Gross Metal Value | $USM | 4,007.5 | 88,927.7 |
| Copper Gross Metal Value | $USM | 277.4 | 7,220.2 |
| Silver Gross Metal Value | $USM | 39.7 | 1,274.2 |
| Rhenium Gross Metal Value | $USM | 29.4 | 633.4 |
| Sulfuric Acid Gross Value | $USM | 24.4 | 508.4 |
| TOTAL GROSS METAL VALUE | $USM | 4,378.3 | 98,563.9 |
| **OPERATING COSTS** | | | |
| **AVERAGE UNIT OPERATING COSTS** | | | |
| **ON SITE COSTS PER TON ORE TREATED** | | | |
| MINING | $US/T (short) ore | 0.69 | 0.69 |
| PLANT | $US/T (short) ore | 4.59 | 4.59 |
| ROASTER | $US/T (short) ore | 0.99 | 0.83 |
| CLOSURE & RECLAMATION ALLOWANCE | $US/T (short) ore | 0.00 | 0.06 |
| GENERAL & ADMINISTRATION | $US/T (short) ore | 0.13 | 0.13 |
| **TOTAL SITE OPERATING COSTS/TON ORE** | **$US/T (short) ore** | **6.41** | **6.30** |
| **REALISATION COSTS PER TON ORE TREATED** | **$US/T (short) ore** | **0.46** | **0.37** |
| **TOTAL UNIT OPERATING COSTS** | **$US/T (short) ore** | **6.87** | **6.67** |
| **TOTAL OPERATING COSTS** | | | |
| TOTAL SITE OPERATING COSTS | $USM | 461.6 | 18,153.3 |
| TOTAL REALISATION COSTS | $USM | 33.3 | 1,057.1 |
| **TOTAL OPERATING COSTS** | **$USM** | **494.9** | **19,210.5** |
| **CAPITAL COSTS** | | | |
| PLANT CAPITAL | $USM | | 1,970.2 |
| ROASTER CAPITAL | $USM | | 346.2 |
| MINING FLEET CAPITAL | $USM | | 270.7 |
| MINE PRE-DEVELOPMENT (INC. PRESTRIP) | $USM | | 658.3 |
| TAILINGS | $USM | | 162.3 |
| **TOTAL INITIAL CAPITAL** | $USM | | **3,407.6** |
| **DEFERRED/SUSTAINING CAPITAL** | | | |
| PROCESS PLANT | $USM | 19.7 | 788.1 |
| ROASTER | $USM | 3.5 | 0.0 |
| MINING | $USM | 0.1 | 0.0 |
| TAILINGS STORAGE FACILITIES & PUMPING | $USM | 0.0 | 0.0 |
| **TOTAL DEFERRED/SUSTAINING CAPITAL** | **$USM** | **23.3** | **2,549.2** |
| **TOTAL CAPITAL** | **$USM** | **23.3** | **5,956.8** |
| **TOTAL PROJECT CASHFLOWS** | | | |
| PROJECT PRETAX CASHFLOW | $USM | 3,860 | 73,397 |
| PROFIT DISTRIBUTION TO EMPLOYEES | $USM | 0.0 | 0.0 |
| **PRODUCTION STATISTICS** | | | |
| NET REVENUE /TON ORE TREATED | $US/T (short) ore | 60.3 | 33.9 |
| TOTAL OPERATING COSTS / lb Molybdenum Oxide EQUIVALENT | $US/lb Molybdenum Oxide | 3.17 | 3.62 |
| TOTAL OPERATING COSTS / lb COPPER EQUIVALENT | $US/lb Cu | 0.40 | 0.50 |

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