# Summary Report on the CuMo Property, Boise County, Idaho (Amended and Restated)

A National Instrument 43-101 report

Prepared For: American CuMo Mining Corp.

Prepared by: Gary Giroux, P. Eng., Shaun M Dykes P.Geo and James H Place, P. Geo.

Dated November 5, 2015

# Table of Contents

# List of Figures

List of Tables

# 1.0 Executive Summary

The CUMO deposit is a molybdenum-copper deposit situated 37 miles (60 km) northeast of Boise, Idaho, USA. Situated in a historic lode gold camp with recorded production of 2.8 million ounces, molybdenite mineralization was not discovered in this area until 1963 by Amax Exploration. After conducting surface sampling in 1964, Amax dropped the property. It was subsequently explored by Curwood Mining Company, Midwest Oil Corporation (later Amoco Minerals Company), Amax (a second time), 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 million tonnes at 0.092% Mo (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., (now American CuMo Mining Corp. ("CuMoCo") in 2004. Kobex Resources Ltd optioned the property from CuMoCo in 2005 and commenced drilling in 2006. In late 2006, CuMoCo resumed control and has since completed the 2006, 2007 and 2008 exploration drilling program. CuMoCo has completed 14,729 meters (44,188 feet) of drilling in 19 diamond drill holes. During 2009 to 2012 CuMoCo drilled 23 more drill holes (22,968 feet), for improving the resource categorization and better understand the 3D extent of the deposits.

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.

CuMoCo'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.

A resource estimate update was completed at the request of CuMoCo based on a total of 65 diamond drill holes totaling 120,685 feet (36,784.9 meters). Nine (9) of the sixty-five (65) diamond drill holes were completed in 2012 since the previous resource calculation.

A geologic model separating the CUMO Deposit into four domains with an oxidized layer on top was produced by CuMoCo geologists. In addition major fault blocks were identified both by assay data and by marker beds.

Assays were tagged as one of four geologic domains: a near surface Cu-Ag zone, a deeper Cu-Mo zone and a still deeper Mo zone and an underlying potassic-silica zone (MSI). 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. Semi-variograms 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 Measured, Indicated or Inferred based on their location relative to drill-hole composites. To take into account the four main economic minerals estimated, a form of metal equivalent or Recoverable Value (RCV) 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, Mo zone and MSI zone.

The resource is summarized below for Recovered Metal Value (RCV) cutoffs. The RCV is based on: $MoS_2$ - Molybdenum is sold as molybdenum trioxide ($MoO_3$), which has higher Mo content.

A molybdenum long term medium price of $15 Mo ($10 $MoO_3$) was used in the calculation. This is higher than current prices. However at least 42 % of the world's production of Molybdenum is at a cost well above current metal prices, and the low prices are not sustainable over the long term. Mines have already begun to shut down and more will as their long term metal contracts run out. For example, the Chinese have stated that they will not be selling their Mo for less than $15/lb. due to their production costs in the area of $12 to $13 per pound. Numerous forecasts, Roskill, Platts are for Mo prices to rise in the short term, including a rise to $15 ($10 $MoO_3$) in 2016 and to $20 ($16.7 Mo O3) in 2020 (CPM group, Feb.2015). Finally, the 5 year average price of molybdenum is just under $13 per pound and the 10 year average price is just under $19 per pound Mo, a $15 price per pound falls within the two averages. Overall the long term price of Molybdenum of $15 Mo is reasonable and sustainable.

$MoO_3$ is calculated from $MoS_2$ by the following: Pounds Mo = $MoS_2$ * 20 / 1.6681 and then Pounds $MoO_3$ = Pounds Mo * 1.5

Three sets of prices are used in the study Low, medium and high

| Zone | Low Price | Medium Price | High Price |
|---|---|---|---|
| Copper (Cu) | $2.50 | $3.00 | $3.50 |
| Molybdenum oxide(MoO3) | $7.50 | $10.00 | $15.00 |
| Molybdenum Metal(Mo) | $11.25 | $15.00 | $22.50 |
| Silver (Ag) | $12.50 | $12.50 | $12.50 |
| Tungsten (W) | $15.00 | $15.00 | $15.00 |

Note: silver and tungsten are kept constant as they are only minor constituents in the overall analysis and it's the prices of copper and molybdenum that are of interest.

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

| Zone | Cu Recovery | $MoS_2$ Recovery | Ag Recovery | W Recovery |
|---|---|---|---|---|
| oxide | 60 | 80 | 65 | 0 |
| cuag | 68 | 86 | 75 | 35 |
| cumo | 85 | 92 | 78 | 35 |
| mo | 72 | 95 | 55 | 35 |
| msi | 72 | 95 | 55 | 35 |

A value in non-oxide material of $2.50 US has been highlighted as a possible open pit cutoff based on similar size mines at a feasibility or production stage.

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

Since the infill drill holes completed in 2011-12 were all within this conceptual pit this resource update uses the Snowden 2012 optimum pit shell to constrain the estimate.

## Using Medium Prices of $3.00 for Cu and $10 for $MoO_3$

**Table 1-1: Measured Resource within Pit Shell using Medium Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **308.4** | **0.079** | **0.074** | **2.09** | **48.6** | **17.83** | **292.1** | **438.2** | **456.5** | **18.8** | **30.0** |
| 5.00 | 297.2 | 0.081 | 0.076 | 2.09 | 49.6 | 18.35 | 288.6 | 432.9 | 451.7 | 18.1 | 29.5 |
| 7.50 | 282.0 | 0.085 | 0.076 | 2.06 | 50.6 | 19.01 | 287.4 | 431.1 | 428.7 | 16.9 | 28.5 |
| 12.50 | 227.9 | 0.097 | 0.075 | 2.00 | 51.8 | 21.04 | 265.0 | 397.5 | 341.8 | 13.3 | 23.6 |
| 15.00 | 195.4 | 0.105 | 0.072 | 1.90 | 52.0 | 22.26 | 246.0 | 368.9 | 281.3 | 10.8 | 20.3 |
| 17.50 | 159.7 | 0.115 | 0.067 | 1.80 | 51.6 | 23.58 | 220.1 | 330.2 | 213.9 | 8.4 | 16.5 |
| 20.00 | 122.9 | 0.125 | 0.063 | 1.70 | 51.7 | 25.04 | 184.1 | 276.2 | 154.8 | 6.1 | 12.7 |

**Table 1-2: Indicated Resource within Pit Shell using Medium Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2216.1** | **0.049** | **0.079** | **2.48** | **37.6** | **12.71** | **1301.9** | **1952.9** | **3501.4** | **160.3** | **166.6** |
| 5.00 | 1972.3 | 0.053 | 0.085 | 2.57 | 39.6 | 13.82 | 1253.3 | 1880.0 | 3352.9 | 147.8 | 156.2 |
| 7.50 | 1708.3 | 0.059 | 0.088 | 2.59 | 41.1 | 14.98 | 1208.4 | 1812.6 | 3006.5 | 129.0 | 140.4 |
| 12.50 | 1050.6 | 0.076 | 0.090 | 2.55 | 44.2 | 18.13 | 957.4 | 1436.0 | 1891.1 | 78.1 | 92.9 |
| 15.00 | 798.5 | 0.083 | 0.090 | 2.56 | 45.6 | 19.54 | 794.6 | 1191.9 | 1437.2 | 59.6 | 72.8 |
| 17.50 | 541.6 | 0.093 | 0.088 | 2.49 | 46.4 | 21.09 | 603.9 | 905.8 | 953.2 | 39.3 | 50.3 |
| 20.00 | 301.3 | 0.106 | 0.082 | 2.36 | 47.7 | 22.99 | 383.0 | 574.5 | 494.2 | 20.7 | 28.7 |

**Table 1-3: Inferred Resource within Pit Shell using Medium Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **3373.6** | **0.040** | **0.057** | **1.93** | **32.1** | **9.89** | **1617.9** | **2426.9** | **3845.9** | **189.9** | **216.6** |
| 5.00 | 2556.6 | 0.048 | 0.067 | 2.13 | 34.7 | 11.84 | 1471.4 | 2207.0 | 3425.9 | 158.8 | 177.4 |
| 7.50 | 1996.0 | 0.056 | 0.070 | 2.23 | 35.1 | 13.44 | 1340.1 | 2010.2 | 2794.4 | 129.8 | 140.1 |
| 12.50 | 996.4 | 0.078 | 0.064 | 1.98 | 37.6 | 17.13 | 931.8 | 1397.7 | 1275.4 | 57.5 | 74.9 |
| 15.00 | 637.0 | 0.086 | 0.074 | 2.16 | 39.8 | 19.05 | 656.8 | 985.2 | 942.7 | 40.1 | 50.7 |
| 17.50 | 384.8 | 0.094 | 0.084 | 2.34 | 41.5 | 20.93 | 433.7 | 650.5 | 646.4 | 26.3 | 31.9 |
| 20.00 | 190.2 | 0.109 | 0.078 | 2.37 | 41.9 | 23.24 | 248.6 | 372.9 | 296.8 | 13.1 | 15.9 |

**Table 1-4: Measured and Indicated Resource within Pit Shell using Medium Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2524.5** | **0.053** | **0.079** | **2.43** | **39.0** | **13.34** | **1594.1** | **2391.1** | **3957.9** | **179.1** | **196.6** |
| 5.00 | 2269.6 | 0.057 | 0.084 | 2.50 | 40.9 | 14.41 | **1541.9** | **2312.9** | **3804.6** | **165.9** | **185.7** |
| 7.50 | 1990.4 | 0.063 | 0.086 | 2.51 | 42.4 | 15.55 | **1495.8** | **2243.7** | **3435.2** | **145.9** | **168.9** |
| 12.50 | 1278.6 | 0.079 | 0.087 | 2.46 | 45.5 | 18.65 | **1222.4** | **1833.5** | **2232.9** | **91.4** | **116.5** |
| 15.00 | 993.9 | 0.088 | 0.087 | 2.43 | 46.8 | 20.07 | **1040.6** | **1560.8** | **1718.5** | **70.4** | **93.1** |
| 17.50 | 701.4 | 0.098 | 0.083 | 2.33 | 47.6 | 21.66 | **824** | **1236** | **1167.1** | **47.7** | **66.8** |
| 20.00 | 424.3 | 0.112 | 0.077 | 2.17 | 48.9 | 23.58 | **567.1** | **850.7** | **649** | **26.8** | **41.4** |

## Using Low Prices of \$2.50 for Cu and \$7.50 for $MoO_3$

### Table 1-5: Measured Resource within Pit Shell using Low Prices

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **307.0** | **0.079** | **0.075** | **2.09** | **48.7** | **13.96** | **290.8** | **436.2** | **460.5** | **18.7** | **29.9** |
| 5.00 | 290.1 | 0.083 | 0.076 | 2.08 | 50.1 | 14.56 | 288.7 | 433.0 | 440.9 | 17.6 | 29.1 |
| 7.50 | 270.4 | 0.087 | 0.076 | 2.06 | 51.0 | 15.16 | 282.0 | 423.0 | 410.9 | 16.2 | 27.6 |
| 12.50 | 185.8 | 0.107 | 0.072 | 1.90 | 51.9 | 17.49 | 238.3 | 357.5 | 267.5 | 10.3 | 19.3 |
| 15.00 | 134.0 | 0.121 | 0.066 | 1.76 | 52.0 | 18.92 | 194.3 | 291.5 | 176.8 | 6.9 | 13.9 |
| 17.50 | 86.0 | 0.134 | 0.062 | 1.69 | 53.1 | 20.43 | 138.1 | 207.2 | 106.6 | 4.2 | 9.1 |
| 20.00 | 41.8 | 0.151 | 0.056 | 1.56 | 53.5 | 22.27 | 75.7 | 113.5 | 46.8 | 1.9 | 4.5 |

### Table 1-6: Indicated Resource within Pit Shell using Low Prices

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2159.5** | **0.050** | **0.081** | **2.51** | **38.2** | **10.29** | **1294.6** | **1941.9** | **3498.4** | **158.1** | **165.0** |
| 5.00 | 1870.9 | 0.056 | 0.087 | 2.59 | 40.2 | 11.30 | 1256.2 | 1884.2 | 3255.3 | 141.3 | 150.4 |
| 7.50 | 1464.8 | 0.064 | 0.090 | 2.62 | 42.4 | 12.68 | 1124.0 | 1686.0 | 2636.6 | 111.9 | 124.2 |
| 12.50 | 720.7 | 0.085 | 0.092 | 2.61 | 46.0 | 15.64 | 734.4 | 1101.7 | 1326.0 | 54.9 | 66.3 |
| 15.00 | 382.9 | 0.099 | 0.090 | 2.55 | 47.7 | 17.32 | 454.5 | 681.8 | 689.2 | 28.5 | 36.5 |
| 17.50 | 136.2 | 0.120 | 0.080 | 2.33 | 49.5 | 19.51 | 196.0 | 294.0 | 218.0 | 9.3 | 13.5 |
| 20.00 | 41.9 | 0.143 | 0.067 | 1.99 | 49.5 | 21.79 | 71.8 | 107.7 | 56.1 | 2.4 | 4.1 |

### Table 1-7: Inferred Resource within Pit Shell using Low Prices

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **3214.8** | **0.041** | **0.059** | **1.97** | **32.9** | **8.10** | **1580.3** | **2370.5** | **3793.5** | **184.7** | **211.5** |
| 5.00 | 2257.7 | 0.052 | 0.070 | 2.21 | 34.9 | 9.99 | 1407.6 | 2111.4 | 3160.8 | 145.5 | 157.6 |
| 7.50 | 1591.6 | 0.063 | 0.070 | 2.21 | 35.9 | 11.53 | 1202.2 | 1803.4 | 2228.3 | 102.6 | 114.3 |
| 12.50 | 519.6 | 0.089 | 0.080 | 2.27 | 40.7 | 15.46 | 554.5 | 831.7 | 831.4 | 34.4 | 42.3 |
| 15.00 | 249.8 | 0.101 | 0.087 | 2.47 | 42.4 | 17.39 | 302.5 | 453.7 | 434.6 | 18.0 | 21.2 |
| 17.50 | 94.7 | 0.122 | 0.076 | 2.46 | 41.9 | 19.62 | 138.6 | 207.9 | 144.0 | 6.8 | 7.9 |
| 20.00 | 30.6 | 0.137 | 0.081 | 2.70 | 43.3 | 21.69 | 50.2 | 75.3 | 49.5 | 2.4 | 2.6 |

**Table 1-8: Measured and Indicated Resource within Pit Shell using Low Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2466.6** | **0.054** | **0.080** | **2.45** | **39.5** | **10.75** | **1585.4** | **2378.1** | **3958.9** | **176.8** | **194.9** |
| 5.00 | 2161.0 | 0.059 | 0.085 | 2.52 | 41.6 | 11.74 | **1544.9** | **2317.2** | **3696.2** | **158.9** | **179.5** |
| 7.50 | 1735.2 | 0.068 | 0.088 | 2.53 | 43.7 | 13.07 | **1406** | **2109** | **3047.5** | **128.1** | **151.8** |
| 12.50 | 906.5 | 0.090 | 0.088 | 2.47 | 47.2 | 16.01 | **972.7** | **1459.2** | **1593.5** | **65.2** | **85.6** |
| 15.00 | 516.9 | 0.105 | 0.084 | 2.35 | 48.8 | 17.73 | **648.8** | **973.3** | **866** | **35.4** | **50.4** |
| 17.50 | 222.2 | 0.126 | 0.073 | 2.08 | 50.9 | 19.87 | **334.1** | **501.2** | **324.6** | **13.5** | **22.6** |
| 20.00 | 83.7 | 0.147 | 0.062 | 1.78 | 51.5 | 22.03 | **147.5** | **221.2** | **102.9** | **4.3** | **8.6** |

## Using High Prices of $3.50 for Cu and $15.00 for $MoO_3$

**Table 1-9: Measured Resource within Pit Shell using High Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **308.6** | **0.079** | **0.074** | **2.09** | **48.6** | **25.06** | **292.3** | **438.4** | **456.7** | **18.8** | **30.0** |
| 5.00 | 303.3 | 0.080 | 0.075 | 2.09 | 49.0 | 25.42 | 290.9 | 436.4 | 455.0 | 18.5 | 29.7 |
| 7.50 | 291.1 | 0.083 | 0.076 | 2.07 | 50.0 | 26.22 | 289.7 | 434.5 | 442.5 | 17.6 | 29.1 |
| 12.50 | 268.2 | 0.088 | 0.075 | 2.04 | 50.8 | 27.59 | 283.0 | 424.5 | 402.4 | 16.0 | 27.3 |
| 15.00 | 245.9 | 0.093 | 0.075 | 2.01 | 51.3 | 28.84 | 274.2 | 411.4 | 368.9 | 14.4 | 25.2 |
| 17.50 | 219.7 | 0.100 | 0.073 | 1.93 | 51.9 | 30.36 | 263.4 | 395.1 | 320.7 | 12.4 | 22.8 |
| 20.00 | 199.2 | 0.105 | 0.071 | 1.89 | 51.9 | 31.55 | 250.8 | 376.2 | 282.9 | 11.0 | 20.7 |

**Table 1-10: Indicated Resource within Pit Shell using High Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2288.6** | **0.048** | **0.078** | **2.44** | **36.8** | **16.93** | **1317.1** | **1975.6** | **3570.1** | **162.9** | **168.4** |
| 5.00 | 2086.0 | 0.051 | 0.083 | 2.52 | 38.7 | 18.22 | 1275.5 | 1913.3 | 3462.7 | 153.3 | 161.5 |
| 7.50 | 1894.9 | 0.055 | 0.086 | 2.56 | 40.1 | 19.42 | 1249.6 | 1874.4 | 3259.3 | 141.5 | 152.0 |
| 12.50 | 1444.2 | 0.066 | 0.087 | 2.53 | 42.0 | 22.34 | 1142.8 | 1714.2 | 2512.9 | 106.6 | 121.3 |
| 15.00 | 1202.6 | 0.072 | 0.087 | 2.49 | 43.1 | 24.07 | 1038.1 | 1557.2 | 2092.5 | 87.3 | 103.7 |
| 17.50 | 1008.2 | 0.078 | 0.086 | 2.46 | 44.1 | 25.59 | 942.9 | 1414.3 | 1734.2 | 72.3 | 88.9 |
| 20.00 | 830.0 | 0.083 | 0.087 | 2.47 | 45.2 | 27.06 | 825.9 | 1238.9 | 1444.2 | 59.8 | 75.0 |

**Table 1-11: Inferred Resource within Pit Shell using High Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **3567.2** | **0.038** | **0.055** | **1.90** | **31.0** | **13.00** | **1625.2** | **2437.9** | **3923.9** | **197.7** | **221.2** |
| 5.00 | 2988.6 | 0.043 | 0.061 | 2.00 | 33.7 | 14.80 | 1540.8 | 2311.2 | 3646.1 | 174.3 | 201.4 |
| 7.50 | 2348.0 | 0.051 | 0.068 | 2.16 | 34.8 | 17.14 | 1435.7 | 2153.6 | 3193.2 | 147.9 | 163.4 |
| 12.50 | 1543.7 | 0.065 | 0.065 | 2.06 | 35.9 | 20.83 | 1203.0 | 1804.5 | 2006.8 | 92.7 | 110.8 |
| 15.00 | 1199.7 | 0.074 | 0.060 | 1.90 | 36.7 | 22.89 | 1064.4 | 1596.6 | 1439.6 | 66.5 | 88.1 |
| 17.50 | 1005.6 | 0.078 | 0.061 | 1.92 | 37.3 | 24.18 | 940.4 | 1410.6 | 1226.8 | 56.3 | 75.0 |
| 20.00 | 767.9 | 0.084 | 0.067 | 2.02 | 38.5 | 25.83 | 773.4 | 1160.1 | 1029.0 | 45.2 | 59.1 |

**Table 1-12: Measured and Indicated Resource within Pit Shell using High Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2597.3** | **0.051** | **0.077** | **2.40** | **38.2** | **17.89** | **1588.2** | **2382.3** | **3999.9** | **181.8** | **198.4** |
| 5.00 | 2389.5 | 0.055 | 0.082 | 2.47 | 40.1 | 19.13 | 1575.7 | 2363.6 | 3918.8 | 172.1 | 191.6 |
| 7.50 | 2186.3 | 0.059 | 0.085 | 2.50 | 41.4 | 20.33 | 1546.5 | 2319.8 | 3716.7 | 159.4 | 181.0 |
| 12.50 | 1712.6 | 0.069 | 0.085 | 2.45 | 43.4 | 23.16 | 1416.8 | 2125.3 | 2911.5 | 122.4 | 148.7 |
| 15.00 | 1448.7 | 0.076 | 0.085 | 2.41 | 44.5 | 24.88 | 1320.1 | 1980.2 | 2462.9 | 101.8 | 128.9 |
| 17.50 | 1228.1 | 0.082 | 0.084 | 2.37 | 45.5 | 26.44 | 1207.4 | 1811.1 | 2063.2 | 84.9 | 111.8 |
| 20.00 | 1029.4 | 0.087 | 0.084 | 2.36 | 46.5 | 27.92 | 1073.8 | 1610.7 | 1729.4 | 70.9 | 95.7 |

Author Mr.Dykes, a professional qualified, has performed a detailed review the Preliminary Economic Analysis (PEA) and states the following, Whilst this part of the report was prepared in 2009 and re-verified by Snowden in 2012, Mr. Dykes, after a detailed examination of current long term metal prices trends, markets and analysis of operating and capital costs, considers it to remain relevant and valid as there have been no significant changes to the assumptions, and the grade of the mineralization remains similar to the 2009 estimates. The one change that has been made is that the size of the deposit and the confidence has increased, and it is Mr. Dykes's opinion, as the person responsible for the section, that this can only improve the confidence in the results of the PEA.

Mr Dykes also reviewed and verified the original PEA, last verified by Snowdon in June 2012, and made the following conclusions:

**Operating costs:** overall operating costs in mining within the USA are slightly down due to improvement in efficiency, lower fuel costs and technological advances. Several mines have responded to lower metal price environment by becoming more efficient. So operating costs provided in the original PEA are still valid within the realms of the accuracy of a PEA level report.

**Capital costs**: overall capital costs on the individual items are varied with certain equipment prices higher others lower when compared to the values used in the PEA. Lower equipment costs are occurring as the result from the need to maintain market share and their skilled labor force. Large truck prices have actually dropped by 3% since the PEA for example. So as with operating costs the capital costs used in the original PEA easily fall within the parameters of the PEA.

**Metal Prices**: Molybdenum and copper prices have had a wide range in values over the past years. Copper ranging from a low of $2.20 to a high of $4.50, while Molybdenum has arranged from $6 to $43 per pound. This reflects the cyclical nature of metal prices. Copper is traded on a regular basis, however molybdenum is sold mainly by long term contracts, contracts are usually signed during favorable price times and last through the lower prices, The PEA reports numerous different price scenarios which are summarized in the section. The base case is toward the high end of the expert predictions and prices for long term metal prices, but still within the accuracy level of the report. However Mr. Dykes has reduced the base case metal prices to reflect the same prices used in this resource report. (see section 16.13.1). The best scenario is probably the cyclical price scenario which better fits the market better and reflects the long term contract side of the Molybdenum market. Thompson Creek Mine was able to keep producing for about 18 months as they were receiving prices that were significantly higher than the spot market prices. So overall the price scenarios are reasonable and still valid and cover the full range of possibilities.

In response to the change in June 2011 to the 43-101 requirement that Preliminary Economic Analysis reports should contain after tax values. Wherever appropriate Mr. Dykes has provided after tax values, based on the taxes and royalties paid by Thompson Creek Mine, a mine located 60 miles from CuMo within the same tax system. These tax values were checked against a pre-feasibility study produced in June 2015 by M3-engineering for the Stibnite project in Idaho for additional confirmation and found to match very well.

The updated base case using the same metal prices that are included in the resource calculation, namely $15 Molybdenum metal ($10 Molybdenum oxide) and $3 copper. These prices fall within the range of prices used in the original PEA report. The results are

**Updated Base Case Economic Analysis (pre-tax)**

| Economic parameters (EBITD&A) | Throughput Option | | | |
|---|---|---|---|---|
| | 50 kt/d | 100 kt/d | 150 kt/d | 200 kt/d |
| NPV (US$B@5%) | 2 | 6 | 9 | 12 |
| IRR% | 13 | 20 | 25 | 27 |
| Simple payback Period (years) | 7.5 | 4.6 | 3.5 | 3.0 |
| Discounted Payback period (years@5%) | 8.8 | 5.2 | 3.9 | 3.3 |
| Total Operating costs per lb of molybdenum Oxide equivalent | 5.5 | 4.3 | 3.9 | 3.8 |

**Updated Base Case Economic Analysis (after-tax)**

| Economic parameters After Tax | Throughput Option | | | |
|---|---|---|---|---|
| | 50 kt/d | 100 kt/d | 150 kt/d | 200 kt/d |
| NPV (US$B@5%) | 1.8 | 4.5 | 7.2 | 9.4 |

| | | | | |
|---|---|---|---|---|
| IRR% | 11.5 | 17.3 | 21.3 | 23.0 |
| Simple payback Period (years) | 7.7 | 4.9 | 3.8 | 3.4 |
| Discounted Payback period (years@5%) | 9.0 | 5.6 | 4.2 | 3.6 |
| Total Operating costs per lb of molybdenum Oxide equivalent | 5.5 | 4.3 | 3.9 | 3.8 |

Section 16 has the details.

Based on the resources defined to date, The authors recommend that the CUMO project be advanced to feasibility stage. The recommended program is proposed to be carried out over a minimum time frame of three years at an estimated cost of $100,000,000 (US$).

# 2.0 Introduction

The authors of this 43-101 compliant Technical Report were asked by CuMoCo to update the previously filed 43-101 Report (Snowden et.al, 2005) and to produce a resource estimate on the CUMO Property in Boise County, Idaho.

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 CuMoCo's information. Reports and data were obtained from all parties. The authors have relied heavily on historical Climax Molybdenum Company (Amax) information presented by CuMoCo, and in particular a report titled "The CUMO Molybdenite System, Boise, Idaho, A Comprehensive Summary" complied by Donald Baker, Climax Molybdenum Company dated April 1983. This immediate area of Idaho is poorly documented in the professional literature and there are very few pertinent papers available for review. Independent qualified author Gary Giroux, responsible for the resource calculation, visited the site between June 1 and June 3 2015 while co-author James H Place has not visited the site. Shaun M Dykes, MSc. (Eng), P.Geo ,a non-independent fully qualified person, has also reviewed the work done by co-author James H Place (non-qualified person) and in his professional opinion found it to be sound. Mr. Dykes takes responsibility for all of the sections produced by Mr. Place, as well sections for which he is responsible. Mr. Dykes has visited the property on numerous occasions over the past 10 years, with the latest visit between June 1 and June 3, 2015.

# 3.0 Reliance on Other Experts

The preparation of this report has relied upon public and private information provided by CuMoCo regarding the property. The authors assume and believe that the information provided and relied upon for preparation of this report is accurate and that interpretations and opinions expressed in them are reasonable and based on current understanding of mineralization processes and the host geologic setting.

# 4.0 Property Description and Location

### *4.1 General*

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

### *4.2 Mineral Tenure*

The property consists of 163 unpatented and un-surveyed contiguous mining lode claims covering an area of approximately 3,260 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 4-2 and the claim information is listed in Appendix A.

In Idaho, staked claims expire annually on September 1. Therefore, the annual fee of $155/claim must be paid to the BLM prior to Aug 31, 2015 or all claims will expire on Sept 1, 2015. At $155/claim, the company must make annual payments to the BLM of US$25,265 to keep all claims in good standing.

### *4.3 Ownership Agreements*

On October 13, 2004, CuMoCo 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 CuMoCo agreement, all claims acquired within 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.

On January 21, 2005, CuMoCo entered into an option agreement with Kobex Resources Ltd. ("Kobex"), whereby Kobex could acquire a 100% interest in the CUMO Molybdenum Property and another property in Australia. Under the terms of the Agreement, Kobex would earn a 100% undivided interest in these properties in consideration of cash payment of $5,000,000, 12,500,000 treasury shares and $10,000,000 of work expenditure commitment.

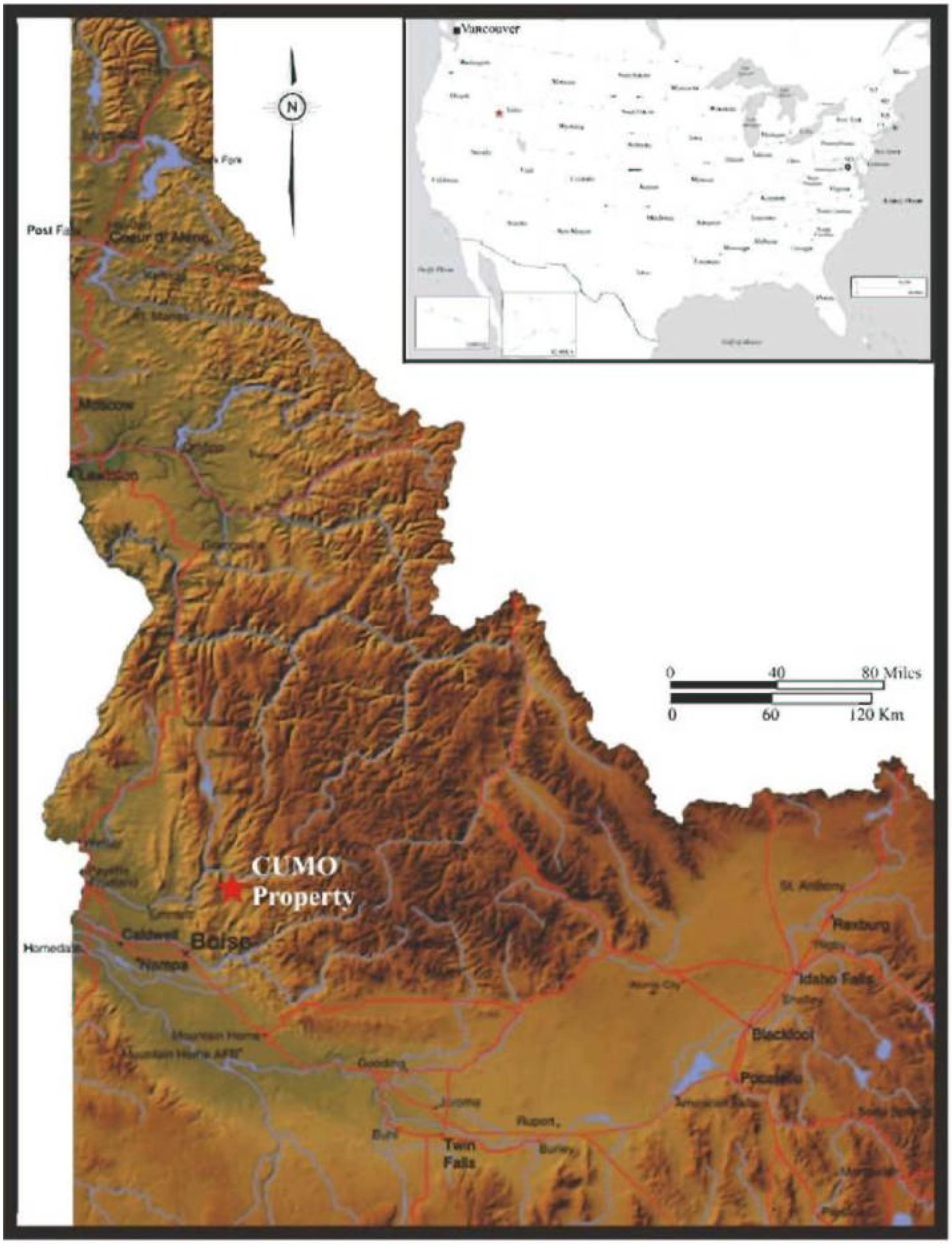
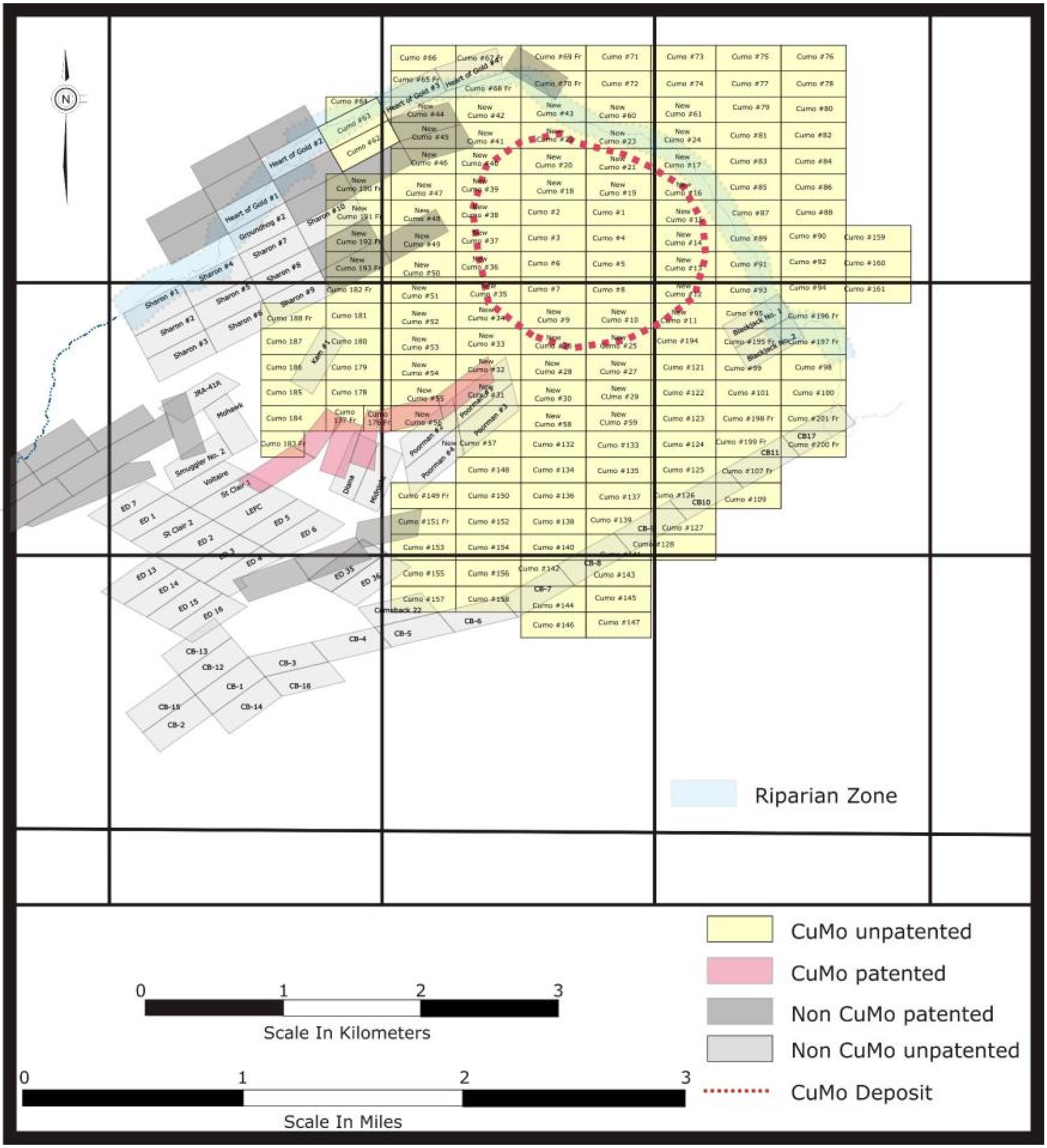



**Figure 4-1: CUMO Property Location Map.**

**Figure 4-2: Claim location map for the CUMO property.**



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

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

***4.4 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. The authors understand that track mounted auger/rotary drilling with no new road clearing would fit in this category according to United States Forest Service (USFS) personnel.

- Environmental assessment (EA) requires an in-depth study with 30 days for public comment, plus additional time for appeal. The authors understand that drilling with an RC rig using water, new road construction, etc., would require this level of permit. USFS personnel suggest that one year may be required to receive a permit. Spot Studies on archaeology and sensitive plant species would be required prior to disturbance.

- Environmental Impact Statement (EIS) is the highest permit level and would be required for mine development~~.~~

Approval for a diamond drilling program has been obtained from the USFS, to be carried out from the existing network of drill access roads and is currently permitted under an existing Categorical Exclusion (CE) permit. An application for a Water Use Permit for 2008 was filed with the Idaho Department of Water Resources. The water is used for diamond drilling purposes

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

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

On May 17 2011, the USFS denied the appeal allowing CuMoCo to begin work under the new exploration permit following a mandatory 15 day stay period which ended on June 7, 2011. The permit covers all exploration work required to produce the information necessary to produce a feasibility Study and lasts for up to 5 years.

The Idaho conservation league filed a challenge in the "United States District Court for the District of Idaho" on December 15th 2011: "Plaintiffs Idaho Conservation League, Idaho Rivers United, and Golden Eagle Audubon Society seek summary judgment reversing and remanding the Forest

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

On August 29, 2012 the judge in the case dismissed four of the five claims by the opponents but remanded the section on groundwater over for further study. As a result on February 7 2013 the USFS initiated a Supplemental Environmental Assessment in order to address the Judge's concerns. This worked culminated on April 13 2015 with the re-issuance of a draft Finding of No Significant Impact (FONSI) and at the current writing of this report the process is in the appeal period with the final decision notice expected in September 2015.

# 5.0 Accessibility, Physiography, Climate, and Infrastructure

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 kilometers (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 kilometers (10 miles). One mile east of Garden Valley is a secondary road heading south across the Payette River. Following this secondary road, the western most edge of the CUMO claim block is approximately 16 kilometers (10 miles) from Garden Valley.

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

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

The climate is defined by summer temperatures to a maximum of 100° F (38°C) and cold, windy winters with lows to -10° F (-23°C). Precipitation is moderately light with an average rainfall of 30 inches (<1 meter) and an average snowfall of approximately 140 inches (3.6 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 20 road miles (32 kilometers) to the west of the property.

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

Exploration and mining at the property can be conducted year-round, due to the established road system and its proximity to other infrastructure. The property is large enough to accommodate all future exploration or mining operations including facilities and potential waste disposal areas.

# 6.0 History

The Boise Basin was first explored following the discovery of placer gold deposits in 1862. Several lode gold deposits were discovered and developed immediately following the initial alluvial gold rush, with significant production occurring in the late 1800's and early 1900's. There are a number of lode prospects within approximately two miles of the CUMO property, some of which have recorded minor past production of base and precious metals.

The first interest in the CUMO property was shown during aerial reconnaissance by AMAX Exploration in 1963. Follow-up geochemical rock and soil sampling indicated anomalous molybdenum and copper values. Forty claims were then staked and three previously existing claims were optioned. A 2.5 mile (4 kilometer) 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 5-1), surface geological mapping, re-logging of the core, road construction, an aerial topographic survey, and age dating. In 1980, AMAX Exploration Inc. transferred its interest to Climax Molybdenum Company, also a subsidiary of AMAX Inc. In 1982, Climax collected more than 300 soil geochemical samples from 3 different grids.

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 5-2).

**Table 5-1: 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 | |

**Table 5-2: CUMO Historical Resource, 1982 AMAX Block Model**

| Cutoff Grade (% $MoS_2$) | Million Tons | Average Grade (% $MoS_2$) |
|---|---|---|
| 0.02 | 2,100 | 0.072 |
| 0.03 | 1,900 | 0.078 |
| 0.04 | 1,600 | 0.084 |
| 0.05 | 1,500 | 0.092 |
| 0.06 | 1,100 | 0.097 |
| 0.08 | 730 | 0.116 |
| 0.1 | 470 | 0.131 |
| 0.12 | 280 | 0.145 |
| 0.14 | 140 | 0.170 |

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

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

# 7.0 Geologic Setting and Mineralization

### *7.1 Regional Geology*

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.

The regional tectonic setting consists of a basement of amalgamated Archean and Paleoproterozoic crystalline terrains that were joined during the Paleoproterozoic Trans-Montana orogeny, and are overlain discontinuously by sedimentary rocks of Mesoproterozoic, Neoproterozoic, and Paleozoic ages; and volcanic and sedimentary rocks of Eocene and Miocene ages. Voluminous tonalite to granite bodies of the Idaho batholith and later granitic plutons of Eocene age intrude the older rocks. Major deformational episodes 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 4-3) 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 4-4), 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 fore-deep assemblages overlying an Archean basement that is juxtaposed with accreted conjoined terrains. The juncture is the linear deformed belt between the Great Falls and Dillon shear zones. The fold-and-thrust belt of the Trans-Montana orogeny coincides in part with the Great Falls tectonic zone.

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

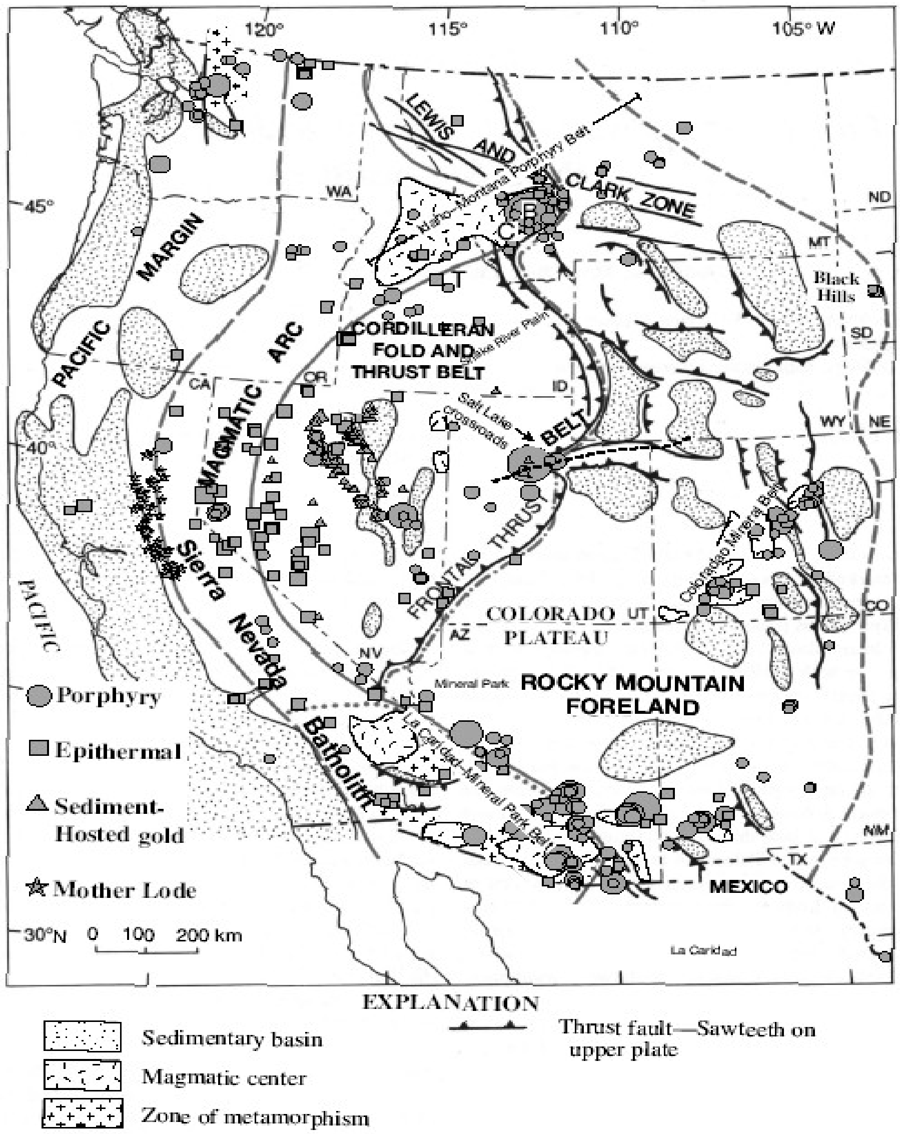



FIG. 1. Map of the western United States cordillera showing ore deposits superimposed on major tectonic elements and Laramide igneous zones, sedimentary basins, and metamorphic belts. The western United States is divided into four generalized geologic provinces (boundaries shown as heavy solid and dashed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

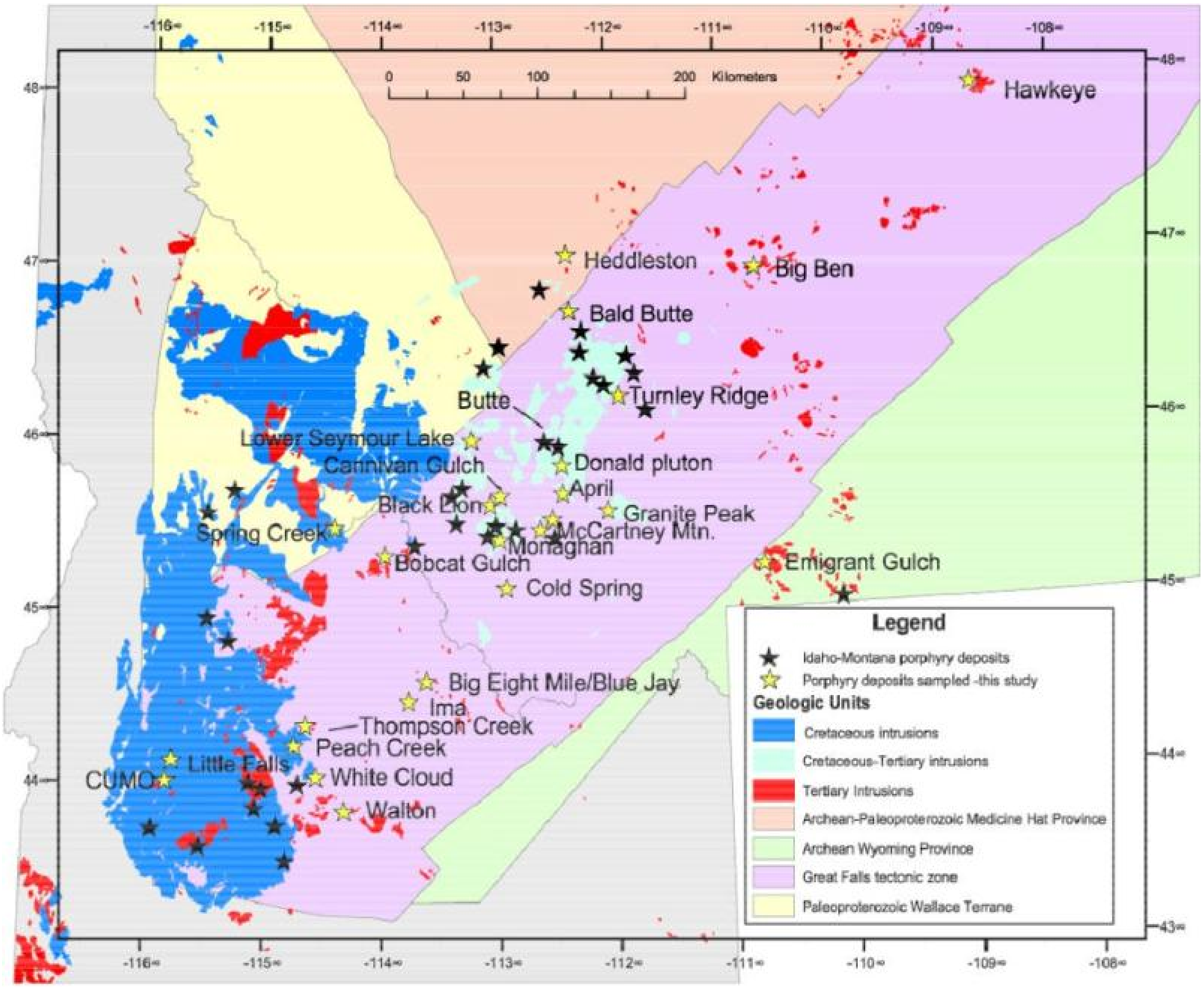



**Figure 7-1: Tectonic map of the western United States (Hildenbrand and others, 2000)**

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 and 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).

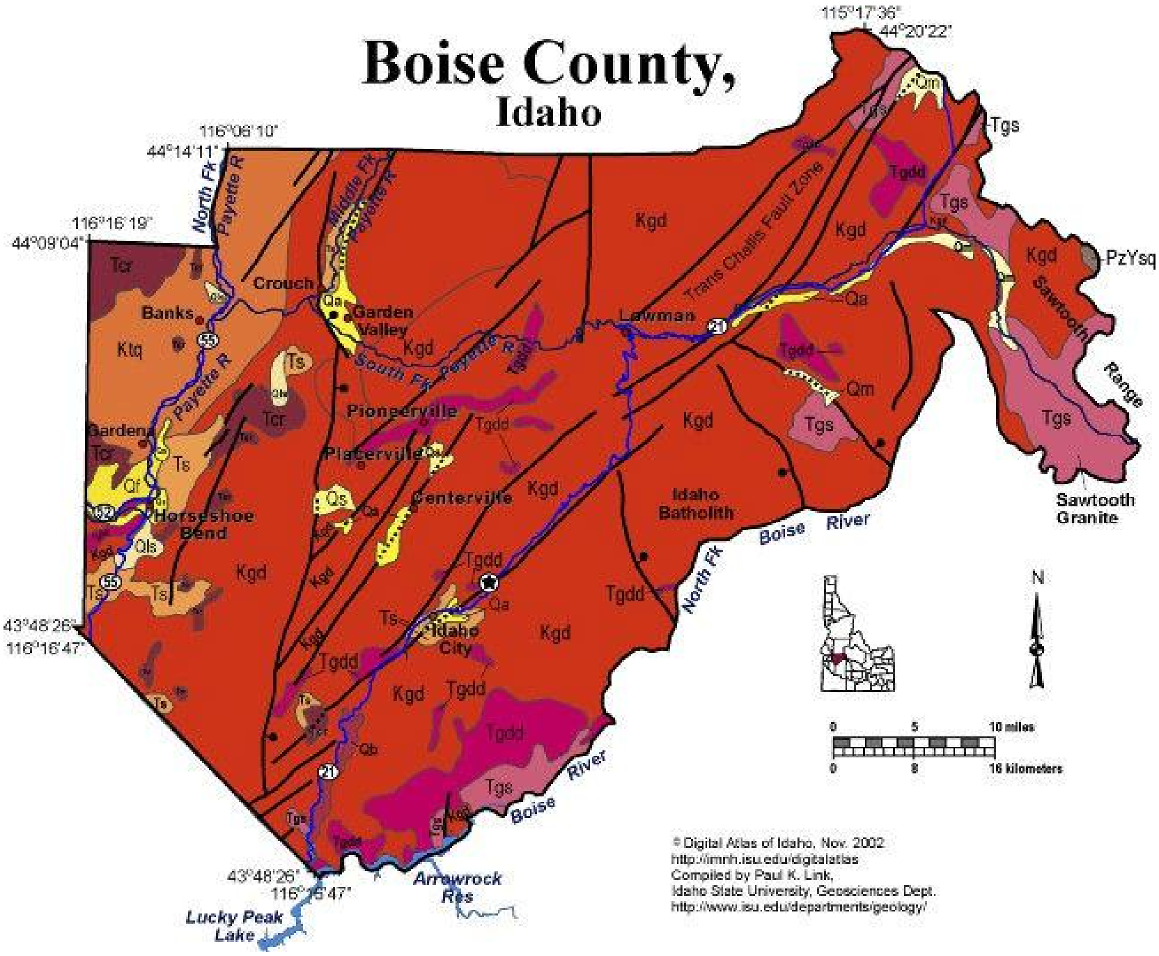
The CUMO deposit is situated within the Atlanta Lobe of the Idaho batholith. The western margin of the Atlanta lobe is strongly folded and metamorphosed into gneissic rocks, which are well exposed near McCall. The western side is composed of tonalite, 95 to 85 million years old. The batholith core is biotite granodiorite; and the eastern side 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 7-3).

The CUMO area is underlain by biotite granodiorite, the most common rock type of the Atlanta lobe of the Idaho batholith (unit Kgd of Killsgaard 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 7-3: 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 7-3, a north-trending Basin and Range fault, down on the east, bounds the system of northeast-striking trans-Challis faults to the west of CUMO (Link, 2002).

### *7.3 Property Geology*

Amax completed detailed bedrock mapping on the CUMO property between 1964 and 1981. Earlier periods of mapping outlined five general rock types, including quartz monzonite of the Idaho Batholith, rhyolite porphyry, lamprophyre, dacite and diabase dykes. Subsequent mapping through to 1982 resulted in subdivision of those five units into 17 separate units as follows:

**Table 7-1: Summary of Rock units at CUMO**

| UNIT | AGE | ROCK TYPE | TEXTURE | Grain Size (groundmass) |
|---|---|---|---|---|
| Tl | Tertiary | lamprophyre | porphyritic | fine |
| Td | Tertiary | diabase | massive, amygdaloidal | aphanitic |
| Tr | Tertiary | rhyolite | massive to flow-banded | aphanitic to fine |
| TpE | Tertiary | biotite quartz monzonite porphyry | porphyritic | fine |
| Tbx | Tertiary | intrusion to intrusive breccia | breccia | aphanitic to fine |
| Trp | Tertiary | biotite quartz monzonite porphyry | porphyritic | aphanitic to fine |
| TpF | Tertiary | biotite quartz latite to rhyolite porphyry | porphyritic | aphanitic |
| TpB | Tertiary | biotite quartz latite to rhyolite porphyry | porphyritic | aphanitic |
| TpA | Tertiary | biotite quartz latite to quartz monzonite porphyry | porphyritic | aphanitic to fine |
| TpD | Tertiary | biotite quartz monzonite to quartz latite porphyry | porphyritic | aphanitic to fine |
| TpC | Tertiary | biotite quartz latite to quartz monzonite porphyry | porphyritic | aphanitic to fine |
| Tbhqmp | Tertiary | biotite hornblende quartz monzonite porphyry | porphyritic | fine |
| Tbdp | Tertiary | biotite dacite porphyry | porphyritic | aphanitic |
| Tgd | Tertiary | granodiorite | equigranular | fine-medium |
| Ta | Tertiary | andesite | porphyritic | aphanitic |
| Kg | Cretaceous | gabbro | Equigranular - diabasic | fine |
| Kqm | Cretaceous | biotite-quartz monzonite | Equigranular to porphyritic | coarse-medium |

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

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 7-4. The quartz monzonite porphyry (unit Tbqmp) varies considerably in proportion and size of phenocrysts, with at least four varieties recognized (Figure 7-4). The first and possibly earliest phase (Tbqmp Type I) is dark to medium grey, with 10-15%, <7mm feldspar phenocrysts, 1-2% fine-grained biotite, and <5% quartz set in a fine-grained groundmass. The second phase (Tbqmp Type II) is medium to light grey, with 30% feldspar phenocrysts and minor biotite set in a medium-grained groundmass. The third phase (Tbqmp Type III) is similar to Type II but contains K-feldspar megacrysts. The fourth phase and possibly most recent is a crowded porphyry variant of Type III containing >30% feldspar phenocrysts set in a medium-grained groundmass. Type I through IV phases may correlate with Amax units TpD, TpB, TpA and TpC, respectively, and appear to follow a general pattern of early, phenocryst poor phases intruded by later phenocryst-rich phases, which is opposite to the general progression observed by previous workers.

Structure may be an important factor on the distribution of mineralization at the CUMO property. A strong northeast to east-northeast structural trend, characteristic of the trans-Challis fault system, is evident in the area of the property. The Tertiary dyke system trends in this same orientation with steep to moderate dips to the south. Faults and mineralized structures identified to date dominantly trend to the northeast as well. These include numerous small base and precious metal occurrences that occur in the area and surrounding the CUMO deposit with most of the major lodes striking east-northeast (N70E) whereas subordinate lodes are oriented northeasterly (N35E, N10-20E and N30-60E). Several fault zones, marked by sections of broken core, were logged in 2006, which appear to offset the interpreted mineral zones. The full significance of these fault structures to the deposit geometry remains to be determined.

**Figure 7-4: Core photographs of Felsic Porphyry Types recognized in the Drill programs.**

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)

### *7.4 Mineralization*

This section is reproduced in total for completeness from "Resource Estimate Update, Technical Report" dated May 13, 2012 and filed on SEDAR on May 14, 2012.

#### 7.4.1 Description of Mineralized Zones

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

Anderson (1947) recognized two groups that he referred to as early Tertiary and early Miocene. The first group consists of gold-quartz veins containing minor sulphides that occur within the Idaho batholith and are associated with weak wall rock alteration. Associated sulphides include pyrite, arsenopyrite, sphalerite, tetrahedrite, chalcopyrite, galena and stibnite. The second group of deposits occurs within porphyry dikes and stocks as well as in the batholith, and is characterized by relatively abundant sulphides, subordinate quartz and widespread wall rock alteration. Base metal mineralization consists of pyrite, sphalerite, galena, tetrahedrite, chalcopyrite, minor quartz and siderite with local occurrences of pyrrhotite and enargite. The gold-quartz veins generally occur relatively distal to CUMO (within 4 to 6 miles/6 to 10 kilometers), 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 sulphide matrix breccia developed in quartz monzonite porphyry, with no conspicuous fault control.

Molybdenum mineralization was discovered at CUMO in 1963. The only other molybdenum showing in Boise County is the Little Falls molybdenum prospect, which is situated just to the northeast of CUMO. Little Falls was extensively drilled between 1978 and 1981, where mineralization occurs within a rhyolite dike that is part of a swarm of dikes that extends northeast from CUMO. An age of 29±3 Ma was obtained by fission-track dating of a zircon from one of the mineralized dikes (Killsgaard et al, 1989).

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

### 7.4.2 Property Mineralization

Mineralization on the CUMO property occurs in veins and veinlets developed within various intrusive bodies. Molybdenite (MoS2) occurs within quartz veins, veinlets and vein stockworks. Individual veinlets vary in size from tiny fractures to veinlets five centimeters in width, with an overall thickness averaging 0.3-0.4 cm. Pyrite and/or chalcopyrite are commonly associated with molybdenite although molybdenite can occur alone without other metallic mineralization.

Chalcopyrite occurs in quartz-pyrite + molybdenite veinlets, in magnetite + pyrite as well as in pyrite-biotite +quartz +magnetite veins with secondary biotite halos. Scheelite is common on the property and closely parallels the distribution of molybdenite (Baker, 1983). Figure 7-5 and Figure 7-6 show examples of mineralization at CUMO from the recent drill holes.

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

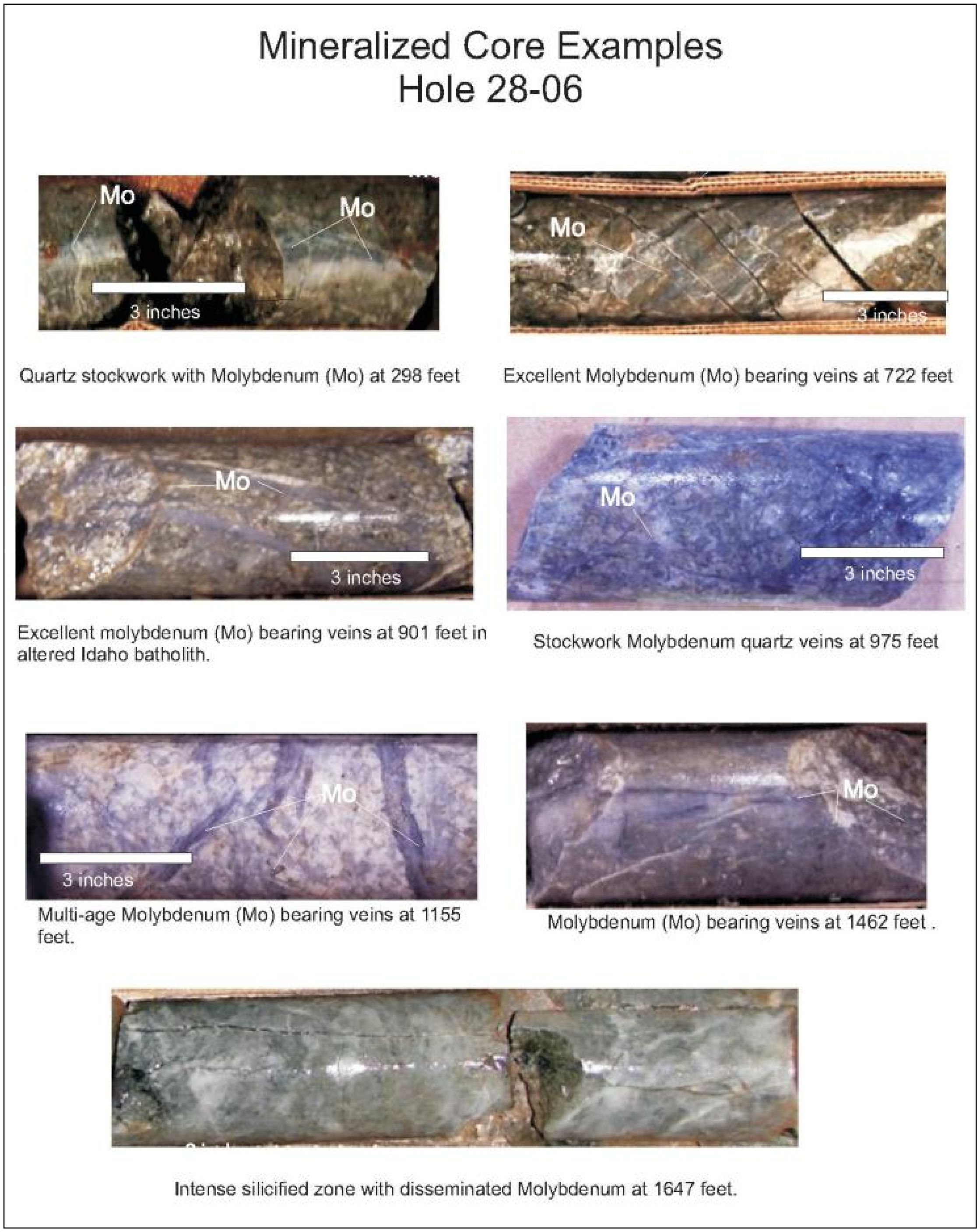


**Figure 7-6: Photographs of molybdenite mineralization in 2008 drill core**

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

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

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

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

Compilation of Amax data on the frequency of veins mapped on surface as well as their mineral constituents was presented by Giroux et al (2005) and is shown in Figure 7-7. A concentric pattern is clearly evident, which is also shown by the distribution of anomalous Mo and Cu rock geochemical results (Figure 7-8a and Figure 7-8b). 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 7-7: Surface distribution of quartz and epidote veinlets and metal zonation**

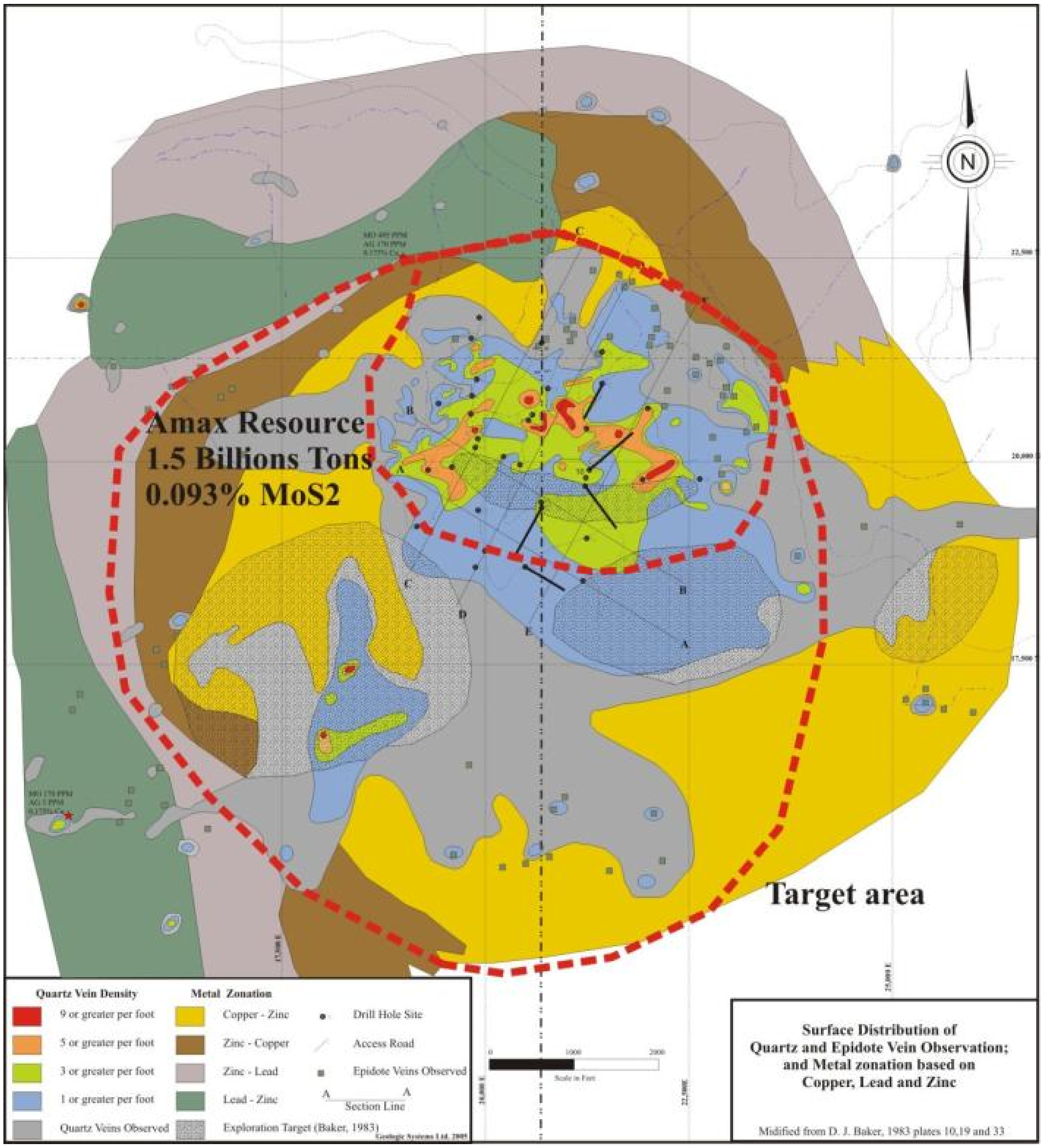


**Figure 7-8a: Geochemical distribution of Mo in surface rock chip samples**

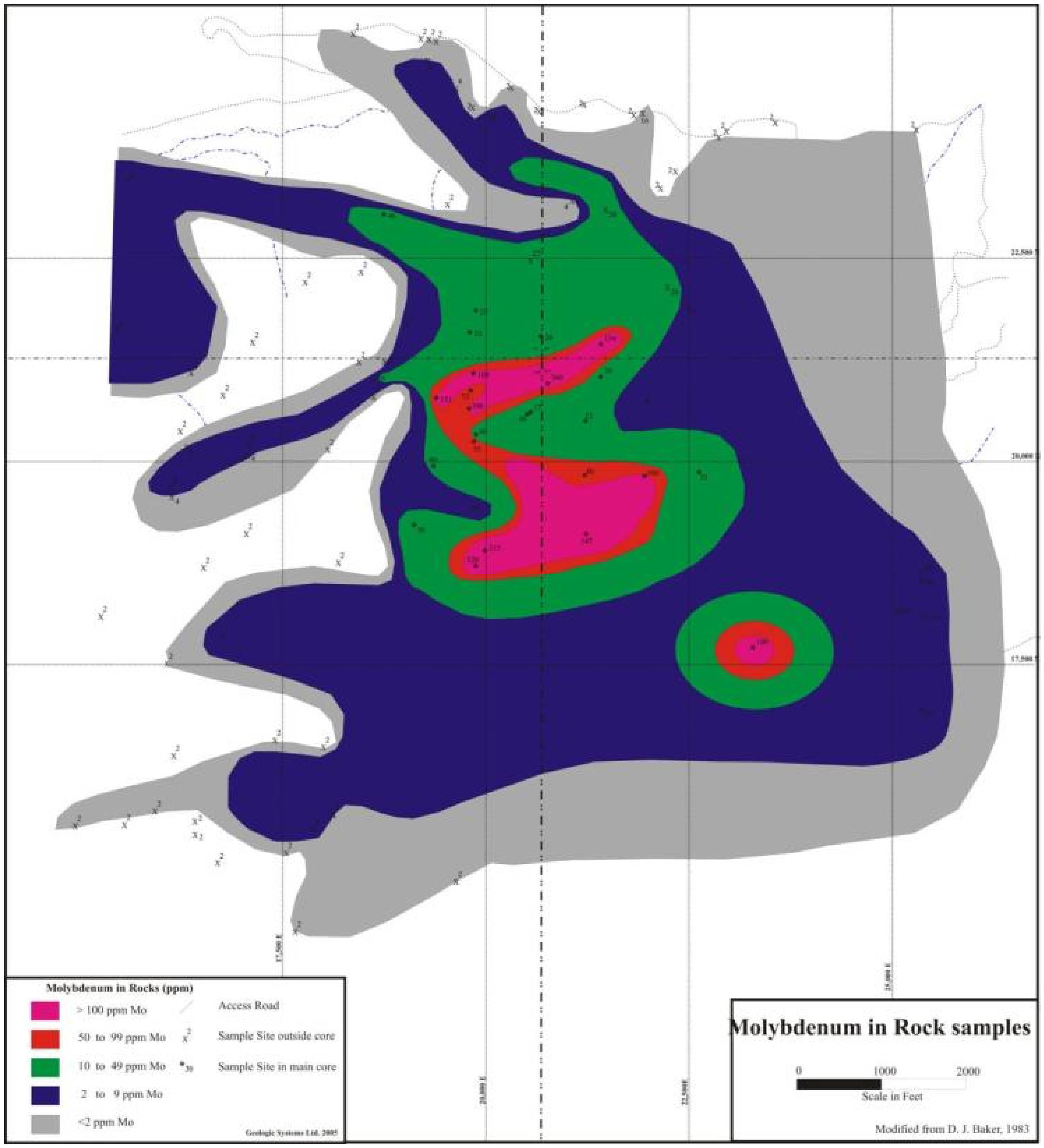
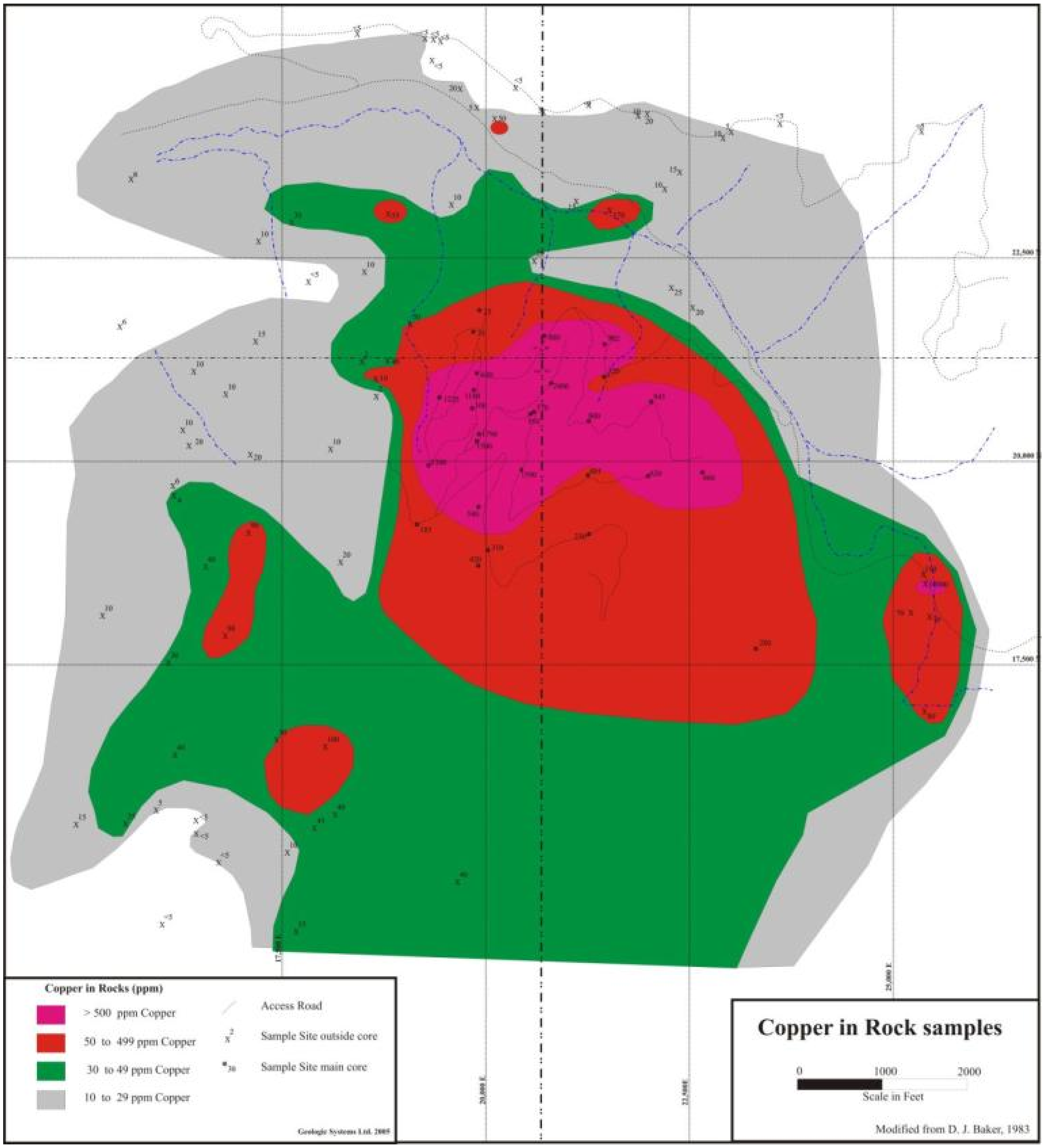


**Figure 7-8b: Geochemical distribution of Cu in surface rock chip samples**




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).
CuMoCo 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/CuMoCo).

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 currently drilled area is located on the north side of a potentially large mineralized system, of which only a small part has been drilled to date.

In 2007 and 2008 CuMoCo 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.

# 8.0 Deposit Types

The CUMO deposit is a porphyry type deposit and has been classified as a porphyry copper molybdenum deposit (Klein, 2004; Spanski, 2004), or as a porphyry molybdenum-copper (low fluorine type) deposit (Mutchler 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 MT at 0.085% MoS2 and very few deposits exceed 500 MT) compared to the average porphyry Cu-Mo (500 Mt with 0.41 % Cu, 0.016 % Mo, 0.012 g/t Au and 1.2 g/t Ag) in which tonnages can range up to over 2 billion tonnes.

The CUMO deposit is primarily of economic interest for its Mo content but contains significant values of Cu and Ag. According to Carten 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 first in terms of total potential contained molybdenum, based on the historical CuMo resource (Table 8-1).

**Table 8-1: Ranking of Open Pit Resources under Exploration or Development (2015).**

| Deposit | Meas.+Ind. | Inferred | Total | Cu | Mo | Au | Ag | Re | Cu Eq. | Gross Value | lbs MoS2 | lbs Mo | Total Value |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | tons (millions) | tons (millions) | tons (millions) | % | % | g/t | g/t | g/t | % | $/ton | (millions) | (millions) | $ (millions) |
| Cumo - Total | 2,501.8 | 3,404.5 | 5,906.3 | 0.07 | 0.027 | | 2.13 | | 0.37 | $15.47 | 5,290.1 | 3171.4 | $91,398 |
| Cumo $7.50 Cut-off | 2,011.5 | 2097.1 | 4,108.6 | 0.08 | 0.035 | | 2.34 | | 0.47 | $19.76 | 4,827.1 | 2,893.7 | $81,202 |
| Cumo $10 Cut-off | 1,746.4 | 1,656.5 | 4,108.6 | 0.07 | 0.039 | | 2.23 | | 0.52 | $21.88 | 3,361.0 | 2,014.9 | $55,825 |
| Jinduicheng | 910 | | 910 | 0.03 | 0.102 | 0.00 | 0.00 | | 1.56 | $46.80 | 3,096.7 | 1,856.4 | $42,588 |
| Mt Toleman | 1,565 | 340 | 1905.0 | 0.09 | 0.047 | 0.00 | 0.00 | | 0.80 | $23.85 | 2,987.1 | 1,790.7 | $45,434 |
| Cumo Amax Historic | | 1,500 | 1,500 | 0.07 | 0.056 | | 0.06 | | 0.91 | $27.44 | 2,802.4 | 1,680.0 | $41,162 |
| Mt Hope | 966 | 191 | 1,157 | | 0.068 | | | | 1.02 | $30.60 | 2,624.8 | 1,573.5 | $35,404 |
| 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 |
| 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 | $40,737 |
| Toquepala | 1,161 | | 1161.0 | 0.67 | 0.040 | | | | 1.27 | $38.04 | 1,549.3 | 928.8 | $44,165 |
| Chuquicamata (remaining) | 700 | | 700 | 1.53 | 0.065 | 0.01 | 5.00 | | 2.57 | $77.13 | 1,518.0 | 910.0 | $53,994 |
| Spinifex ridge | 652.3 | 399 | 1051.3 | 0.07 | 0.042 | | 1.21 | | 0.55 | $23.08 | 1,479.1 | 886.7 | $24,261 |
| 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 |
| Climax (remaining) | 150 | 25 | 175 | | 0.167 | | | | 2.51 | $75.15 | 975.0 | 584.5 | $13,151 |
| Cajone | 1,261 | | 1261.3 | 0.61 | 0.020 | | | | 0.91 | $27.30 | 841.6 | 504.5 | $34,435 |
| Thompson Creek | 372 | | 372 | | 0.063 | | | | 0.95 | $28.35 | 781.0 | 468.2 | $10,535 |
| Mineral Park | 520 | | 520 | 0.13 | 0.039 | | 2.74 | | 0.75 | $22.41 | 677.0 | 405.9 | $11,660 |
| Bingham (remaining) | 557 | | 557 | 0.54 | 0.033 | 0.27 | 2.52 | | 1.23 | $36.79 | 613.2 | 367.6 | $20,494 |
| Endako | 368 | | 368 | | 0.050 | | | | 0.75 | $22.50 | 613.0 | 367.5 | $8,269 |
| Bagdad | 1,600 | | 1,600 | 0.40 | 0.010 | 0.00 | 0.97 | 0.000 | 0.56 | $16.86 | 533.8 | 320.0 | $26,975 |
| Sonora | 94 | 93 | 187 | 0.05 | 0.081 | | | | 1.27 | $37.95 | 504.3 | 302.3 | $7,083 |
| Atlin | 213 | | 213 | | 0.063 | | | | 0.95 | $28.35 | 447.7 | 268.4 | $6,039 |
| Quellaveco | 947 | | 947.0 | 0.94 | 0.014 | | | | 1.15 | $34.50 | 442.3 | 265.2 | $32,672 |
| Magistral | 196 | 55 | 251 | 0.52 | 0.041 | | | | 1.14 | $34.05 | 343.2 | 205.7 | $8,543 |
| Gibralter | 965 | | 965 | 0.32 | 0.010 | 0.07 | 0.90 | 0.000 | 0.52 | $15.68 | 321.9 | 193.0 | $15,127 |
| Island copper | 377 | | 377 | 0.41 | 0.017 | 0.19 | 1.40 | 0.032 | 0.80 | $23.86 | 213.8 | 128.2 | $8,996 |
| Max | 43 | | 43 | | 0.120 | | | | 1.80 | $54.00 | 171.7 | 103.0 | $2,317 |
| Lucky Ship | 45 | 17 | 62 | | 0.068 | | | | 1.02 | $30.60 | 139.5 | 83.6 | $1,882 |
| Poplar | 116 | | 116 | 0.32 | 0.009 | 0.10 | | | 0.52 | $15.45 | 34.8 | 20.9 | $1,792 |

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 sulphide 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.

IDENTIFICATION SYNONYM: Calc-alkaline porphyry Cu, Cu-Mo, Cu-Au.

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

EXAMPLES (British Columbia - Canada/International):

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

Classic deposits (Cu + Mo * Au) - Brenda (092HNE047), Berg (093E046), Huckleberry (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).

### *8.1 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 sulphide minerals are present, generally in subordinate amounts. The mineralization is spatially, temporally and genetically associated with hydrothermal alteration of the host rock intrusions and wall rocks.

**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 distinct deposit type (see model L03).

**DEPOSIT FORM**: Large zones of hydrothermally altered rock contain quartz veins and stockworks, sulphide-bearing veinlets; fractures and lesser disseminations in areas up to 10 km2 in size, commonly coincident wholly or in part with hydrothermal or intrusion breccias and dike swarms. Deposit boundaries are determined by economic factors that outline ore zones within larger areas of low-grade, concentrically zoned mineralization. Cordilleran deposits are commonly subdivided according to their morphology into three classes - classic, volcanic and plutonic (see Sutherland Brown, 1976; McMillan and Panteleyev, 1988): Volcanic type deposits (e.g. Island Copper) are associated with multiple intrusions in sub volcanic settings of small stocks, sills, dikes and diverse types of intrusive breccias. Reconstruction of volcanic landforms, structures, vent-proximal extrusive deposits and sub volcanic intrusive centers 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 host rocks with high primary permeability. Propyllitic 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 sulphide minerals from a weakly mineralized potassic/propyllitic core is usual. Surrounding ore zones with potassic (commonly biotite-rich) or phyllic alteration contain molybdenite * chalcopyrite, then chalcopyrite and a generally widespread propyllitic, 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. Host rocks are phaneritic coarse-grained to porphyritic. The intrusions can display internal compositional differences as a result of differentiation with gradational to sharp boundaries between the different phases of magma emplacement. Local swarms of dikes, many with associated breccias, and fault zones are sites of mineralization. Orebodies around silicified alteration zones tend to occur as diffuse vein stockworks carrying chalcopyrite, bornite and minor pyrite in intensely fractured rocks but, overall, sulphide minerals are sparse. Much of the early potassic and phyllic alteration in central parts of orebodies is restricted to the margins of mineralized fractures as selvages. Later phyllic-argillic alteration forms envelopes on the veins and fractures and is more pervasive and widespread. Propyllitic 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-sulphide and sulphide veinlets and stockworks; sulphide grains in fractures and fracture selvages. Minor disseminated sulphides commonly replacing primary mafic minerals. Quartz phenocrysts can be partially resorbed and overgrown by silica.

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

**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 host rocks by biotite-rich rocks that grade outward into propyllitic 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 over-printed 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 sulphate minerals. Oxidized and leached zones at surface are marked by ferruginous 'cappings' with supergene clay minerals, limonite (goethite, hematite and jarosite) and residual quartz.

**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 sulphidation type (H05) or epithermal Cu-Au-Ag as high-sulphidation type enargite-bearing veins (L01), replacements and stockworks; auriferous and polymetallic base metal quartz and quartz-carbonate veins (I01, I05), Au-Ag and base metal sulphide mantles 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 calc-alkaline 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 calc-alkaline quartz-bearing porphyry copper deposits and the alkalic (silica under saturated) class. The alkalic porphyry copper deposits are described in a separate model - L03.

### *8.2 Exploration Guides*

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

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

**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).

### *8.3 Economic Factors*

**TYPICAL GRADE AND TONNAGE:**
Worldwide, according to 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.

# 9.0 Exploration

In 2006, diamond drilling was done by Kettle Drilling Inc. of Coeur d'Alene on behalf of Kobex Resources Ltd. and CuMoCo Resources Corp. Kobex commenced drilling in August, 2006 and completed one hole. On October 6, 2006, Kobex Resources Ltd delivered a notice of termination in respect of the CUMO Property. The option on the project was terminated when the second hole was at a depth of 600 feet, and the action was taken before any assays were received. Idaho CuMo Mining Corp. (wholly owned US subsidiary of CuMoCo.) 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.

Between 2007 and 2011, diamond drilling was done by Kirkness Drilling of Carson City, Nevada. Kirkness drilled thirty-three (33) diamond drill holes. Table 9-1 provides details of the drilling undertaken from 2006 to 2011.

**Table 9-1: Summary of 2006 to 2011 Diamond Drilling at CUMO.**

| Hole | Northing | Easting | Elevation | Dip | Azimuth | Length (feet) |
|---|---|---|---|---|---|---|
| 27-06 | 120,016.7 | 220,160.3 | 7105 | -90 | 000 | 1849 completed |
| 28-06 | 119,531.6 | 120,796.4 | 7170 | -90 | 000 | 1711 completed |
| 29-07 | 120,016.7 | 220,160.3 | 6305 | -70 | 140 | 2281.7 completed |
| 30-07 | 119,531.6 | 220,796.4 | 6206 | -90 | 000 | 2416.5 completed |
| 31-07 | 120,016.7 | 220,160.3 | 6305 | -70 | 045 | 2104 completed |
| 32-07 | 119,480.0 | 220,720.3 | 6316 | -70 | 190 | 2044 completed |
| 33-07 | 118,585.3 | 221,268.9 | 6798 | -90 | 000 | 2095 stopped |
| 34-07 | 118,530.5 | 220,343.8 | 6512 | -70 | 095 | 1769 stopped |
| 34-07 | 118,530.5 | 220,343.8 | 6512 | -70 | 095 | 1769 stopped |
| 36-08 | 119,266.8 | 219,322.9 | 6457 | -90 | 000 | 2488 completed |
| 37-08 | 119,755.7 | 221,220.4 | 6341 | -70 | 335 | 2195 completed |
| 38-08 | 118,658.3 | 220,487.4 | 6534 | -70 | 180 | 2441 completed |
| 39-08 | 118,872.7 | 220,777.6 | 6466 | -90 | 000 | 2688 completed |
| 40-08 | 119,539.8 | 220,816.8 | 6321 | -70 | 225 | 2252 completed |
| 41-08 | 119,545.7 | 219,005.8 | 6247 | -90 | 000 | 3018 completed |
| 42-08 | 118,711.9 | 219,886.6 | 6544 | -70 | 270 | 2707 stopped (winter) |
| 43-08 | 120,515.6 | 220,178.6 | 6198 | -80 | 040 | 1308 stopped by fault |
| 44-08 | 118,068.1 | 221,448.9 | 6733 | -65 | 075 | 3047 completed |
| 45-08 | 119,802.3 | 218,821.4 | 6183 | -80 | 330 | 1796 stopped (winter) |
| 46-09 | 220,811.3 | 118,913.9 | 6575.1 | -75 | 110 | 959 stopped |
| 47-09 | 219,421.7 | 120,686.7 | 5832.6 | -90 | 000 | 2530 completed |
| 48-09 | 120,741.3 | 219,432.5 | 5827 | -70 | 305 | 2576 completed |
| 49-09 | 118,881.6 | 221,719.8 | 6668 | -90 | 000 | 2847 completed |
| 50-09 | 121,752.9 | 219,929.4 | 5885 | -75 | 270 | 1826 completed |
| 51-09 | 121,752.9 | 219,929.4 | 5885 | -90 | 000 | 1583.5 completed |
| 52-09 | 118,585.3 | 221,268.9 | 6798 | -75 | 020 | 2772 completed |
| 53-09 | 119,802.3 | 218,821.4 | 6183 | -75 | 015 | 2461 completed |
| 54-09 | 119,802.3 | 218,821.4 | 6183 | -75 | 015 | 2471 completed |
| 55-10 | 117,559.6 | 218,422.4 | 6724.2 | -65 | 0 | 2479 completed |
| 56-10 | 117,559.9 | 218,421.8 | 6724.2 | -65 | 305 | 1294 completed |
| 57-10 | 117,559.3 | 218,422.2 | 6724.2 | -90 | 000 | 534 stopped (winter) |
| 58-11 | 219,970.3 | 119,095.6 | 6451.3 | -90 | 000 | 1885 completed |
| 59-11 | 221,745.9 | 117,559.9 | 6645.3 | -75 | 000 | 1910 completed |

All CuMoCo drilling programs were supervised by onsite geology staff located in Garden Valley, Idaho. All holes were surveyed down the hole at regular intervals using a Reflex survey instrument. Figure 9-1 shows the locations of all holes drilled to date in the deposit. Table 9-2 summarizes the drilling undertaken to date by CuMoCo on the CUMO property prior to this report.

**Table 9-2 Summary of drilling undertaken by American CuMo prior to 2012**

| Company | Year | Holes | Footage | Meters | Comments |
|---|---|---|---|---|---|
| **Kobex** | 2006 | 2 | 3,560 | 1,085.10 | Kobex drilled 1.5 holes only, completed by CuMoCo |
| **CuMoCo (former company name)** | 2007 | 6 | 12,710 | 3,874.20 | vertical and angle holes |
| | 2008 | 11 | 26,770 | 8,159.70 | vertical and angle holes |
| | 2009 | 9 | 18,661 | 5,687.80 | vertical and angle holes |
| | 2010 | 3 | 4,307 | 1,312.80 | vertical and angle holes |
| **CuMoCo** | 2011 | 2 | 3,795 | 1,156.70 | vertical and angle holes |
| | **Total** | **35** | **69,803** | **21,275.90** | |

Mr. Shaun M. Dykes, M.Sc. (Eng), P.Geo. President and Chief Executive Officer of CuMoCo is the designated qualified person for the CUMO Project, and prepared the technical information on the 2006-2011 results.

A summary of significant intersections for all the CUMO drilling undertaken by CuMoCo is given in Table 9-3. Potential economic metals include copper, molybdenum, silver, tungsten, rhenium and gallium. The presence of the by-product elements silver, tungsten, rhenium and gallium is significant in terms of the economic development of the property. The description of the calculation and formulas used for producing the metal equivalents and the recovered metal value for the intersections is covered in section 10.2 on page 46.

**Figure 9-1: Map showing the location of completed and proposed drill holes**

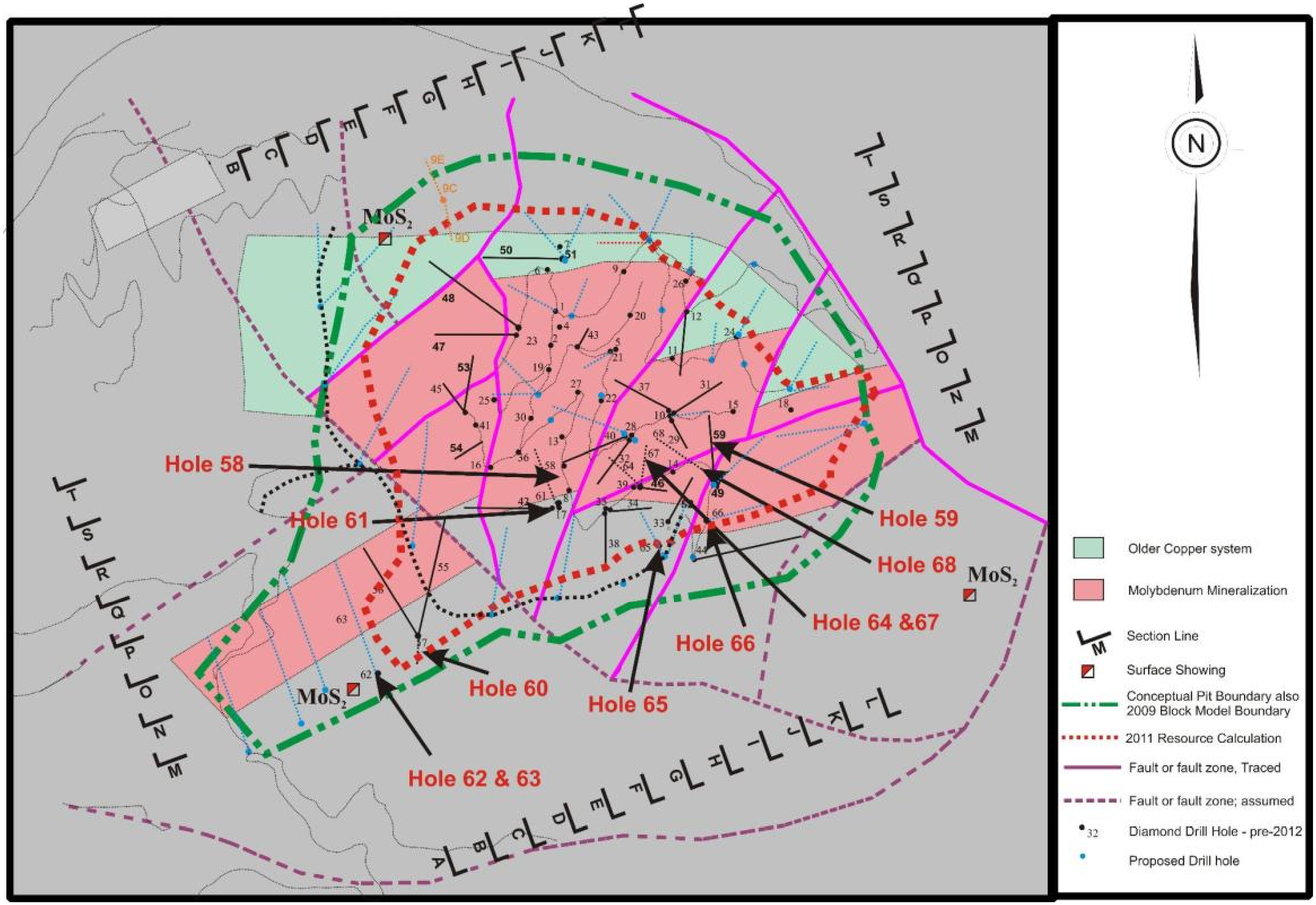


**Table 9-3: Significant Intersections from CUMO Drilling**

| 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 Gms/T | Re ppm | W ppm | Recovered Metal value US$/ton |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| C71-01 | 231.0 | 1884.0 | 1653.0 | 70.4 | 574.2 | 503.8 | main | 0.38 | 0.088 | 1.59 | 0.059 | 0.12 | 2.59 | 0.00 | 46 | 15.94 |
| C71-01 | 390.0 | 470.0 | 80.0 | 118.9 | 143.3 | 24.4 | sub | 0.53 | 0.122 | 2.20 | 0.099 | 0.14 | 2.56 | 0.00 | 44 | 22.12 |
| C71-01 | 1700.0 | 1884.0 | 184.0 | 518.2 | 574.2 | 56.1 | sub | 0.49 | 0.114 | 2.06 | 0.100 | 0.08 | 1.21 | 0.00 | 54 | 20.67 |
| C72-05 | 450.0 | 1416.0 | 966.0 | 137.2 | 431.6 | 294.4 | main | 0.43 | 0.099 | 1.78 | 0.060 | 0.13 | 4.46 | 0.00 | 75 | 17.88 |
| C74-09 | 460.0 | 804.6 | 344.6 | 140.2 | 245.2 | 105.0 | main | 0.54 | 0.126 | 2.26 | 0.077 | 0.12 | 7.16 | 0.00 | 71 | 22.67 |
| C75-10 | 220.0 | 2160.0 | 1940.0 | 67.1 | 658.4 | 591.3 | main | 0.47 | 0.109 | 1.96 | 0.099 | 0.05 | 1.43 | 0.00 | 48 | 19.73 |
| C76-11 | 140.0 | 2428.3 | 2288.3 | 42.7 | 740.1 | 697.5 | main | 0.36 | 0.084 | 1.52 | 0.074 | 0.05 | 1.55 | 0.00 | 36 | 15.24 |
| C76-11 | 1300.0 | 1960.0 | 660.0 | 396.2 | 597.4 | 201.2 | sub | 0.55 | 0.128 | 2.30 | 0.127 | 0.03 | 0.77 | 0.00 | 58 | 23.13 |
| C76-12 | 98.3 | 1430.0 | 1331.7 | 29.9 | 435.9 | 405.9 | main | 0.25 | 0.058 | 1.04 | 0.041 | 0.06 | 1.66 | 0.00 | 45 | 10.40 |
| C77-13 | 680.0 | 1804.0 | 1124.0 | 207.3 | 549.9 | 342.6 | main | 0.51 | 0.119 | 2.14 | 0.111 | 0.05 | 1.98 | 0.00 | 49 | 21.53 |
| C77-14 | 780.0 | 2123.8 | 1343.8 | 237.7 | 647.3 | 409.6 | main | 0.53 | 0.124 | 2.22 | 0.114 | 0.06 | 1.84 | 0.00 | 65 | 22.31 |
| C77-14 | 1200.0 | 1960.0 | 760.0 | 365.8 | 597.4 | 231.6 | sub | 0.68 | 0.158 | 2.84 | 0.151 | 0.06 | 1.91 | 0.00 | 74 | 28.52 |
| C77-15 | 600.0 | 1933.2 | 1333.2 | 182.9 | 589.2 | 406.4 | main | 0.53 | 0.123 | 2.21 | 0.113 | 0.06 | 1.73 | 0.00 | 57 | 22.20 |
| C77-15 | 1260.0 | 1880.0 | 620.0 | 384.0 | 573.0 | 189.0 | sub | 0.64 | 0.150 | 2.69 | 0.153 | 0.02 | 0.75 | 0.00 | 69 | 27.02 |
| C78-16 | 1000.0 | 2131.7 | 1131.7 | 304.8 | 649.7 | 344.9 | main | 0.44 | 0.102 | 1.84 | 0.093 | 0.04 | 1.86 | 0.00 | 32 | 18.51 |
| C78-17 | 1160.0 | 2281.5 | 1121.5 | 353.6 | 695.4 | 341.8 | main | 0.37 | 0.086 | 1.55 | 0.064 | 0.08 | 2.55 | 0.00 | 40 | 15.61 |
| C78-18 | 1400.0 | 2361.0 | 961.0 | 426.7 | 719.6 | 292.9 | main | 0.62 | 0.144 | 2.59 | 0.129 | 0.08 | 2.71 | 0.00 | 41 | 26.01 |
| C79-19 | 120.0 | 2280.0 | 2160.0 | 36.6 | 694.9 | 658.4 | main | 0.51 | 0.118 | 2.11 | 0.101 | 0.08 | 2.27 | 0.00 | 49 | 21.22 |
| C79-20 | 165.0 | 1800.0 | 1635.0 | 50.3 | 548.6 | 498.3 | main | 0.43 | 0.099 | 1.78 | 0.069 | 0.11 | 3.83 | 0.00 | 52 | 17.90 |
| C81-25 | 190.0 | 1011.0 | 821.0 | 57.9 | 308.2 | 250.2 | main | 0.43 | 0.101 | 1.82 | 0.070 | 0.13 | 2.42 | 0.00 | 58 | 18.27 |
| C81-25 | 740.0 | 1011.0 | 271.0 | 225.6 | 308.2 | 82.6 | sub | 0.53 | 0.124 | 2.23 | 0.090 | 0.14 | 2.98 | 0.00 | 84 | 22.41 |
| C81-26 | 30.0 | 750.0 | 720.0 | 9.1 | 228.6 | 219.5 | main | 0.41 | 0.094 | 1.70 | 0.034 | 0.18 | 7.58 | 0.00 | 28 | 17.03 |
| C06-27 | 120.0 | 1849.0 | 1729.0 | 36.6 | 563.6 | 527.0 | main | 0.42 | 0.097 | 1.75 | 0.084 | 0.06 | 1.60 | 0.02 | 49 | 17.54 |
| C06-27 | 1080.0 | 1849.0 | 769.0 | 329.2 | 563.6 | 234.4 | sub | 0.58 | 0.136 | 2.44 | 0.133 | 0.04 | 0.99 | 0.04 | 59 | 24.55 |
| C06-28 | 50.0 | 1690.0 | 1640.0 | 15.2 | 515.1 | 499.9 | main | 0.47 | 0.110 | 1.98 | 0.097 | 0.07 | 1.92 | 0.05 | 54 | 19.88 |
| C06-28 | 840.0 | 1240.0 | 400.0 | 256.0 | 378.0 | 121.9 | sub | 0.70 | 0.162 | 2.92 | 0.162 | 0.03 | 0.98 | 0.09 | 68 | 29.33 |
| C07-29 | 190.0 | 2230.0 | 2040.0 | 57.9 | 679.7 | 621.8 | main | 0.52 | 0.121 | 2.18 | 0.103 | 0.08 | 2.13 | 0.05 | 53 | 21.91 |

Table 9-3: Significant Intersections from CUMO Drilling (cont'd)

| Hole | From | To | Length | From | To | Length | Zone | recv | recv | MoO3 | MoS2 | Cu | Ag | Re | W | Recovered |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Name | feet | feet | feet | meters | meters | meters | | Cu | MoS2 | Equiv | % | % | Gms/T | ppm | ppm | Metal value |
| | | | | | | | | Equiv. | equiv. | lbs | | | | | | US$/T |
| C07-29 | 1180.0 | 1790.0 | 610.0 | 359.7 | 545.6 | 185.9 | sub | 0.74 | 0.171 | 3.08 | 0.169 | 0.04 | 1.2 | 0.08 | 37 | 30.89 |
| C07-30 | 40.0 | 2386.0 | 2346.0 | 12.2 | 727.3 | 715.1 | main | 0.52 | 0.122 | 2.19 | 0.108 | 0.06 | 2.05 | 0.04 | 41 | 22.02 |
| C07-30 | 1180.0 | 1988.0 | 808.0 | 359.7 | 605.9 | 246.3 | sub | 0.80 | 0.187 | 3.36 | 0.185 | 0.04 | 1.46 | 0.07 | 37 | 33.74 |
| C07-31 | 22.0 | 2104.0 | 2082.0 | 6.7 | 641.3 | 634.6 | main | 0.34 | 0.079 | 1.42 | 0.064 | 0.07 | 1.76 | 0.02 | 43 | 14.26 |
| C07-31 | 780.0 | 1540.0 | 760.0 | 237.7 | 469.4 | 231.6 | sub | 0.40 | 0.092 | 1.66 | 0.081 | 0.05 | 1.45 | 0.03 | 45 | 16.69 |
| C07-32 | 22.0 | 2104.0 | 2082.0 | 6.7 | 641.3 | 634.6 | main | 0.55 | 0.129 | 2.31 | 0.109 | 0.09 | 2.26 | 0.04 | 61 | 23.22 |
| C07-32 | 780.0 | 1540.0 | 760.0 | 237.7 | 469.4 | 231.6 | sub | 0.65 | 0.151 | 2.71 | 0.129 | 0.10 | 2.62 | 0.05 | 77 | 27.26 |
| C07-33 | 721.8 | 2094.0 | 1372.2 | 220.0 | 638.3 | 418.2 | main | 0.20 | 0.048 | 0.86 | 0.026 | 0.07 | 2.01 | 0.01 | 48 | 8.60 |
| C07-33 | 1980.0 | 2094.0 | 114.0 | 603.5 | 638.3 | 34.7 | sub | 0.48 | 0.111 | 2.00 | 0.084 | 0.10 | 2.68 | 0.03 | 67 | 20.05 |
| C07-34 | 140.0 | 1769.0 | 1629.0 | 42.7 | 539.2 | 496.5 | main | 0.25 | 0.058 | 1.05 | 0.034 | 0.08 | 2.30 | 0.01 | 53 | 10.53 |
| C07-34 | 1550.0 | 1769.0 | 219.0 | 472.4 | 539.2 | 66.8 | sub | 0.41 | 0.096 | 1.73 | 0.074 | 0.09 | 2.36 | 0.02 | 67 | 17.34 |
| C08-35 | 120.0 | 2640.0 | 2520.0 | 36.6 | 804.7 | 768.1 | main | 0.31 | 0.072 | 1.30 | 0.057 | 0.06 | 1.73 | 0.02 | 37 | 13.08 |
| C08-35 | 420.0 | 2640.0 | 2220.0 | 128.0 | 804.7 | 676.7 | sub | 0.33 | 0.077 | 1.38 | 0.062 | 0.07 | 1.69 | 0.02 | 39 | 13.90 |
| C08-35 | 1730.0 | 2640.0 | 910.0 | 527.3 | 804.7 | 277.4 | sub | 0.43 | 0.100 | 1.80 | 0.089 | 0.05 | 1.37 | 0.03 | 35 | 18.05 |
| C08-36 | 560.0 | 2488.0 | 1928.0 | 170.7 | 758.3 | 587.7 | main | 0.39 | 0.090 | 1.63 | 0.088 | 0.05 | 1.42 | 0.03 | 34 | 16.33 |
| C08-36 | 920.0 | 2488.0 | 1568.0 | 280.4 | 758.3 | 477.9 | sub | 0.43 | 0.100 | 1.80 | 0.103 | 0.04 | 1.04 | 0.03 | 33 | 18.06 |
| C08-37 | 60.0 | 2195.0 | 2135.0 | 18.3 | 669.0 | 650.7 | main | 0.40 | 0.094 | 1.69 | 0.084 | 0.05 | 1.67 | 0.03 | 42 | 16.98 |
| C08-37 | 780.0 | 2130.0 | 1350.0 | 237.7 | 649.2 | 411.5 | sub | 0.46 | 0.106 | 1.90 | 0.104 | 0.02 | 1.17 | 0.04 | 41 | 19.13 |
| C08-38 | 170.0 | 2441.0 | 2271.0 | 51.8 | 744.0 | 692.2 | main | 0.24 | 0.056 | 1.00 | 0.029 | 0.06 | 4.40 | 0.00 | 32 | 10.08 |
| C08-39 | 310.0 | 2688.0 | 2378.0 | 94.5 | 819.3 | 724.8 | main | 0.47 | 0.109 | 1.95 | 0.099 | 0.06 | 1.38 | 0.03 | 52 | 19.60 |
| C08-39 | 900.0 | 2390.0 | 1490.0 | 274.3 | 728.5 | 454.2 | sub | 0.54 | 0.127 | 2.28 | 0.122 | 0.04 | 1.09 | 0.04 | 57 | 22.87 |
| C08-40 | 60.0 | 2252.0 | 2192.0 | 18.3 | 686.4 | 668.1 | main | 0.57 | 0.133 | 2.40 | 0.115 | 0.06 | 3.79 | 0.04 | 46 | 24.06 |
| C08-40 | 390.0 | 2080.0 | 1690.0 | 118.9 | 634.0 | 515.1 | sub | 0.64 | 0.150 | 2.69 | 0.129 | 0.06 | 4.27 | 0.05 | 45 | 27.01 |
| C08-40 | 1110.0 | 1820.0 | 710.0 | 338.3 | 554.7 | 216.4 | sub | 0.75 | 0.173 | 3.12 | 0.142 | 0.04 | 7.78 | 0.06 | 45 | 31.32 |
| C08-41 | 850.0 | 2830.0 | 1980.0 | 259.1 | 862.6 | 603.5 | main | 0.38 | 0.088 | 1.58 | 0.067 | 0.08 | 2.23 | 0.02 | 43 | 15.87 |
| C08-41 | 1490.0 | 2030.0 | 540.0 | 454.2 | 618.7 | 164.6 | sub | 0.56 | 0.129 | 2.32 | 0.107 | 0.08 | 2.99 | 0.03 | 38 | 23.32 |
| C08-41 | 2490.0 | 2830.0 | 340.0 | 759.0 | 862.6 | 103.6 | sub | 0.38 | 0.089 | 1.60 | 0.077 | 0.06 | 1.53 | 0.03 | 34 | 16.09 |

Table 9-3: Significant Intersections from CUMO Drilling (cont'd)

| 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 Gms/T | Re ppm | W ppm | Recovered Metal value US$/T |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| C08-42 | 550.0 | 2707.0 | 2157.0 | 167.6 | 825.1 | 657.5 | main | 0.33 | 0.077 | 1.38 | 0.044 | 0.06 | 5.81 | 0.01 | 25 | 13.87 |
| C08-42 | 950.0 | 2707.0 | 1757.0 | 289.6 | 825.1 | 535.5 | sub | 0.36 | 0.084 | 1.51 | 0.047 | 0.07 | 6.78 | 0.01 | 27 | 15.21 |
| C08-42 | 1970.0 | 2707.0 | 737.0 | 600.5 | 825.1 | 224.6 | sub | 0.32 | 0.075 | 1.35 | 0.063 | 0.05 | 1.61 | 0.01 | 21 | 13.58 |
| C08-43 | 165.0 | 1303.0 | 1138.0 | 50.3 | 397.2 | 346.9 | main | 0.48 | 0.053 | 0.95 | 0.044 | 0.09 | 4.23 | 0.02 | 52 | 14.34 |
| C08-43 | 660.0 | 820.0 | 160.0 | 201.2 | 249.9 | 48.8 | sub | 0.71 | 0.078 | 1.41 | 0.07 | 0.11 | 3.14 | 0.03 | 45 | 21.24 |
| C08-44 | 1125.0 | 2840.0 | 1715.0 | 342.9 | 865.6 | 522.7 | main | 0.15 | 0.035 | 0.63 | 0.03 | 0.02 | 0.89 | 0.01 | 29 | 7.98 |
| C08-44 | 2560.0 | 2690.0 | 130.0 | 780.3 | 819.9 | 39.6 | sub | 0.27 | 0.062 | 1.11 | 0.06 | 0.02 | 1.47 | 0.01 | 20 | 14.76 |
| C08-45 | 170.0 | 1796.0 | 1626.0 | 51.8 | 547.4 | 495.6 | main | 0.27 | 0.062 | 1.12 | 0.02 | 0.15 | 3.08 | 0.00 | 42 | 9.97 |
| C08-45 | 1010.0 | 1796.0 | 786.0 | 307.8 | 547.4 | 239.6 | sub | 0.33 | 0.077 | 1.38 | 0.03 | 0.18 | 3.05 | 0.00 | 40 | 13.13 |
| C09-46 | 300.0 | 959.0 | 659.0 | 91.4 | 292.3 | 200.9 | main | 0.27 | 0.062 | 1.11 | 0.03 | 0.09 | 2.61 | 0.01 | 55 | 11.17 |
| C09-47 | 290.0 | 1736.5 | 1446.5 | 88.4 | 529.3 | 440.9 | main | 0.36 | 0.084 | 1.50 | 0.07 | 0.18 | 4.29 | 0.02 | 20 | 16.92 |
| C09-47 | 960.0 | 2840.0 | 1880.0 | 292.6 | 865.6 | 573.0 | main | 0.42 | 0.097 | 1.74 | 0.05 | 0.18 | 5.03 | 0.02 | 20 | 15.77 |
| C09-48 | 1520.0 | 2420.0 | 900.0 | 463.3 | 737.6 | 274.3 | sub | 0.40 | 0.094 | 1.69 | 0.08 | 0.05 | 1.70 | 0.03 | 17 | 20.27 |
| C09-49 | 810.0 | 1524.5 | 714.5 | 246.9 | 464.7 | 217.8 | main | 0.38 | 0.087 | 1.57 | 0.11 | 0.06 | 1.91 | 0.04 | 17 | 12.99 |
| C09-49 | 520.0 | 1570.0 | 1050.0 | 158.5 | 478.5 | 320.0 | main | 0.48 | 0.112 | 2.02 | 0.03 | 0.15 | 5.29 | 0.01 | 20 | 14.42 |
| C09-50 | 890.0 | 2700.0 | 1810.0 | 271.3 | 823.0 | 551.7 | main | 0.31 | 0.072 | 1.29 | 0.04 | 0.15 | 4.86 | 0.02 | 19 | 18.06 |
| C09-51 | 1790.0 | 2640.0 | 850.0 | 545.6 | 804.7 | 259.1 | sub | 0.34 | 0.080 | 1.44 | 0.09 | 0.07 | 1.69 | 0.03 | 18 | 26.63 |
| C09-52 | 800.0 | 2471.0 | 1671.0 | 243.8 | 753.2 | 509.3 | main | 0.43 | 0.100 | 1.80 | 0.14 | 0.05 | 1.29 | 0.06 | 17 | 17.71 |
| C09-52 | 1510.0 | 2471.0 | 961.0 | 460.2 | 753.2 | 292.9 | sub | 0.63 | 0.147 | 2.65 | 0.09 | 0.19 | 4.07 | 0.02 | 18 | 20.43 |
| C09-53 | 589.0 | 1096.0 | 507.0 | 179.5 | 334.1 | 154.5 | main | 0.42 | 0.098 | 1.76 | 0.12 | 0.15 | 3.68 | 0.03 | 19 | 8.32 |
| C09-53 | 230.0 | 420.0 | 190.0 | 70.1 | 128.0 | 57.9 | main | 0.49 | 0.113 | 2.03 | 0.11 | 0.05 | 1.69 | 0.03 | 17 | 10.32 |
| C09-54 | 1190.0 | 1200.0 | 10.0 | 362.7 | 365.8 | 3.0 | sub | 0.20 | 0.046 | 0.83 | 0.03 | 0.07 | 35.44 | 0.00 | 21 | 165.08 |
| C10-55 | 220.0 | 500.0 | 280.0 | 67.1 | 152.4 | 85.3 | main | 0.25 | 0.057 | 1.03 | 0.04 | 0.01 | 0.42 | 0.01 | 21 | 11.72 |
| C10-55* | 300.0 | 490.0 | 190.0 | 91.4 | 149.4 | 57.9 | main | 0.49 | 0.071 | 1.27 | 0.07 | 0.02 | 3.80 | 0.02 | 21 | 20.37 |
| C10-56 | 220.0 | 500.0 | 280.0 | 67.1 | 152.4 | 85.3 | main | 0.15 | 0.035 | 0.64 | 0.03 | 0.01 | 0.01 | 0.01 | 25 | 6.40 |
| C10-57 | 300.0 | 490.0 | 190.0 | 91.4 | 149.4 | 57.9 | main | 0.35 | 0.082 | 1.47 | 0.07 | 0.02 | 0.02 | 0.02 | 18 | 14.81 |

The 2006 - 2011 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 - 2011 drilling data supports the presence of three distinct metal zones within the deposit. Amax previously interpreted these zones as distinct ore shells that were produced by separate intrusions. Re-interpretation of the geology, alteration and down-hole histograms for Cu, Ag and Mo have confirmed the metal zones are a part of a single, large, concentrically zoned system with an upper copper-silver zone (cuag), underlain by a transitional copper-molybdenum zone (cumo), in turn underlain by a lower molybdenum-rich zone (mo) (Figure 9-2).

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 9-3).

# 10.0 Drilling

## *10.1 General*

### 10.1.1 Procedures

In 2012, a total of 15,463 feet in 9 holes were completed. The holes were sited to infill gaps in the existing drilling coverage and were drilled along existing tracks and roads. All holes were surveyed down the hole at regular intervals using a Reflex survey instrument.

Figure 9-1 shows the locations of all holes drilled to date in the deposit. Mr. Shaun M. Dykes, M.Sc. (Eng), P.Geo., President and Chief executive Officer and Director of CuMoCo, is the designated qualified person for the CUMO Project, and prepared the technical information on the 2006 to 2012 results for the news releases. Co-ordinates, elevations and lengths are in feet.

**Table 10-1 Summary of 2012 diamond drilling**

| HOLE | Year | EASTING | NORTHING | ELEVATION | DIP | Azimuth | LENGTH | Comment |
|---|---|---|---|---|---|---|---|---|
| 12-60 | 2012 | 218421.86 | 117559.92 | 6724.20 | -50.00 | 180.00 | 1455.00 | completed |
| 12-61 | 2012 | 219911.00 | 118748.90 | 6549.23 | -75.00 | 335.00 | 1318.00 | Stopped |
| 12-62 | 2012 | 218040.50 | 116866.10 | 6628.70 | -50.00 | 135.00 | 1484.00 | completed |
| 12-63 | 2012 | 218041.50 | 116866.80 | 6628.70 | -60.00 | 330.00 | 807.00 | completed |
| 12-64 | 2012 | 220811.30 | 118913.90 | 6575.10 | -75.00 | 25.00 | 2139.00 | completed |
| 12-65 | 2012 | 221117.50 | 118148.80 | 6785.70 | -80.00 | 315.00 | 1908.00 | completed |
| 12-66 | 2012 | 221687.80 | 118674.00 | 6689.70 | -90.00 | 0.00 | 2241.00 | completed |
| 12-67 | 2012 | 220811.30 | 118913.90 | 6575.10 | -70.00 | 340.00 | 1978.00 | completed |
| 12-68 | 2012 | 221745.90 | 119095.60 | 6645.30 | -70.00 | 310.00 | 2133.50 | completed |

### *10.2 Results*

A summary of significant intersections for all the CUMO drilling are given in Table 10-3. 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 involved:
Copper equivalent (Cu. Equiv.) and Molybdenite equivalent (MoS2 Equiv.) are based on the following metal prices (all in US$): Copper $2.50/lb, Molybdenum Oxide ($10/lb), Silver $0.35/gram and Tungsten $0.22/gram.($7.00 per lb). Other factors include 1% = 20 pounds/t or 22.04 lbs/T; 1 ppm = 1 gm/T; 1000 ppb =1 ppm = 1 gm/T.

Molybdenum is sold as either ferro-molybdenite or molybdenum oxide.
The price used in these estimates is $10 per pound Molybdenum oxide or $15 per pound Molybdenum metal (Mo)

To obtain the amount of Molybdenum oxide that can be produced from MoS2, the following is required: convert MoS2 to Mo by dividing MoS2 by 1.6681 then convert to MoO3 (Molybdenum Oxide) by multiplying by 1.5. Therefore the amount of molybdenum oxide is pounds MoS2 times 1.5 / 1.6681.
Estimated metallurgical recoveries used in the calculations are as follows for each metal zone. Recoveries are slightly lower than those currently reported by SGS in their recent metallurgical study, as they have been adjusted by Ausenco to reflect losses during the cleaning and roasting stages.

**Table 10-2 Metallurgical recoveries for equivalency calculations**

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

Recoveries take into account not only mill recoveries but smelting recoveries and payables. Recovery (recv) for a metal is taken from the above table for each assay/block in a particular zone and is applied in the following formula:
RCV = Cu x 20 x $(Cu) x recv(Cu) + MoS2 x 20 x (1.5/1.6681) x $(MoO3) x recv(MoS2) + Ag x $(Ag) x recv(Ag) + W x $(W) x recv(W)
Then,
Cu Equiv = RCV / ($(Cu) x recv(Cu) x 20)
Mo Equiv = RCV / ($(MoO3) x recv(MoS2) x 20 x 1.5/1.6681)

**Table 10-3 Significant Intersections from 2011 - 2012 CUMO Drilling**

| 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 Gms/T | Re ppm | W ppm | Recovered Metal value US$ |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| C11-58 | 700.0 | 1885.0 | 1185.0 | 213.4 | 574.5 | 361.2 | main | 0.43 | 0.101 | 1.82 | 0.08 | 0.07 | 0.07 | 0.03 | 41 | $18.26 |
| C11-59 | 500.0 | 1910.0 | 1410.0 | 152.4 | 582.2 | 429.8 | main | 0.56 | 0.129 | 2.33 | 0.07 | 0.13 | 0.13 | 0.02 | 109 | $23.37 |
| C12-60 | 230.0 | 390.0 | 160.0 | 70.1 | 118.9 | 48.8 | main | 0.29 | 0.068 | 1.22 | 0.05 | 0.02 | 0.02 | 0.00 | 7 | $12.21 |
| C12-61 | 400.0 | 1317.0 | 917.0 | 121.9 | 401.4 | 279.5 | main | 0.27 | 0.062 | 1.11 | 0.03 | 0.11 | 0.11 | 0.01 | 28 | $11.14 |
| C12-62 | No significant intersections hole drilled away from deposit | | | | | | | | | | | | | | | |
| C12-63 | 605.0 | 620.0 | 15.0 | 184.4 | 189.0 | 4.6 | main | 2.39 | 0.556 | 10.00 | 0.00 | 0.21 | 130.60 | 0.00 | 7 | $100.45 |
| C12-64 | 300.0 | 2190.0 | 1890.0 | 91.4 | 667.5 | 576.1 | main | 0.42 | 0.098 | 1.76 | 0.08 | 0.07 | 1.77 | 0.03 | 47 | $17.67 |
| C12-64 | 990.0 | 1880.0 | 890.0 | 301.8 | 573.0 | 271.3 | sub | 0.57 | 0.131 | 2.36 | 0.12 | 0.07 | 1.60 | 0.04 | 59 | $23.73 |
| C12-65 | 550.0 | 1570.0 | 1020.0 | 167.6 | 478.5 | 310.9 | main | 0.14 | 0.032 | 0.57 | 0.02 | 0.05 | 1.23 | 0.01 | 44 | $5.73 |
| C12-66 | 400.0 | 1317.0 | 917.0 | 121.9 | 401.4 | 279.5 | main | 0.14 | 0.032 | 0.57 | 0.02 | 0.06 | 1.58 | 0.00 | 40 | $5.75 |
| C12-66 | 535.0 | 1317.0 | 782.0 | 163.1 | 401.4 | 238.4 | sub | 0.15 | 0.035 | 0.63 | 0.02 | 0.07 | 1.69 | 0.00 | 45 | $6.32 |
| C12-67 | 570.0 | 1970.0 | 1400.0 | 173.7 | 600.5 | 426.7 | main | 0.52 | 0.120 | 2.16 | 0.10 | 0.09 | 2.11 | 0.04 | 56 | $21.72 |
| C12-67 | 910.0 | 1970.0 | 1060.0 | 277.4 | 600.5 | 323.1 | sub | 0.57 | 0.131 | 2.36 | 0.12 | 0.08 | 1.66 | 0.05 | 61 | $23.74 |
| C12-68 | 910.0 | 1800.0 | 890.0 | 277.4 | 548.6 | 271.3 | main | 0.49 | 0.113 | 2.04 | 0.10 | 0.08 | 1.85 | 0.04 | 73 | $20.47 |
| C12-68 | 1320.0 | 1800.0 | 480.0 | 402.3 | 548.6 | 146.3 | sub | 0.62 | 0.144 | 2.59 | 0.13 | 0.07 | 1.77 | 0.06 | 65 | $25.99 |

# 11.0 Sample Preparation, Analyses and Security

## *11.1 General sampling*

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

At the time of drilling, each core box is clearly labeled by the driller's helper with the DDH hole number, core box number, and "to" and "from" drill core footages. Wooden core boxes are used at all times, and 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 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 permanent 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 retained core is replaced in their original core boxes which are sealed with a plywood cover and stacked upon a standard pallet. Each plywood cover is clearly labelled with the core's 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 are selected and inserted by the geologist-in-charge.

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

The standards were prepared and packaged by CDN Labs of Surrey, British Columbia. Each bulk sample was pulverized in a large rod mill, screened through 200 mesh using an electric sieve, and homogenized in a large rotating mixer. Each standard was sealed in plastic to prevent gravity separation and oxidation. The standards were certified by Smee & Associates Consulting Ltd. of North Vancouver, British Columbia, based on round-robin analysis at five laboratories using a four-acid digestion and ICP-ES finish (Table 11 -1).

**Table 11-1 Certified standards prepared for CUMO project**

| Standard | Element | Certified Mean | Standard Deviation (between lab) |
|---|---|---|---|
| Standard 1 | Tot. Cu | 1138 ppm | 65 ppm |
| Standard 1 | Tot. Mo | 367 ppm | 19 ppm |
| Standard 2 | Tot. Cu | 151 ppm | 8 ppm |
| Standard 2 | Tot. Mo | 995 ppm | 41 ppm |
| Standard 3 | Tot. Cu | 840 ppm | 35 ppm |
| Standard 3 | Tot. Mo | 54.0 ppm | 3.7 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 by |CuMoCo personnel to ALS-Chemex in Elko, Nevada for preparation and analysis.

### *11.2 Density Determinations*

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

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

The digital scale has a function to record density of solids. The scale must be elevated on a stand that allows for the hook underneath the scale to hang about 1 foot, with an additional foot or so beneath the hook. Next, the scale must be leveled using the self-leveling knobs that double as the scale's feet. Once leveled, the calibration weight must be applied to insure that measurements will be accurate. The scale has an internal calibration function as well. Once the scale is calibrated, the density function is selected. The sample is weighed in both air and distilled water using the hook to hold the core. The trick is to have the bucket of water handy and simply submerge the sample by elevating the bucket and placing bricks beneath it to hold it steady. Once the weight in air and water are recorded, the scale will calculate the density and the next sample can be processed.

The density calculations are very straightforward as follows:

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

Therefore, the following data is recorded on the excel spreadsheet:

| Hole | Sample | Dl | Mg | Ml | Ds | Diameter | Scanner Max | Scanner Avg | code |
|---|---|---|---|---|---|---|---|---|---|
| C08-41 | 95.5 | 1 | 396.53 | 240.82 | 2.55 | 16 | 0.101 | 0.048 | cuag |

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

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

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

Table outlines the density values for each of the different grade domains.

| | DENSITY | COUNT |
|---|---|---|
| OX | 2.5 | 578 |
| CUAG | 2.58 | 1496 |
| CUMO | 2.58 | 1458 |
| MO | 2.57 | 638 |
| MSI | 2.57 | 91 |
| DYKE | 2.52 | 78 |

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 CuMoCo 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-principlesanalyticalmethodologies.htm#Inductively%20Coupled%20Plasma%20Mass%20Spectroscopy%20(ICP-MS)

Samples submitted by CuMoCo 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.

### *11.3 Security*

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

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

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

## 12.0 Data Verification

### *12.1 Historical Checks*

As reported in the June 2005 report (Cavey et. al. 2005) there were six data sets available to verify the original Skyline MoS2 assay data base (pre-CuMoCo involvement in project). The original Skyline assays were re-assayed by Skyline at three stages of the sampling procedure; from core duplicate samples, from splits of rejects and from splits from pulps. Three inter lab sets of duplicates are also available to compare with the Skyline original assays; a pulp sent to Amax Lab in Climax from diamond drill hole assays, a second split at the drill of reverse circulation drill cuttings and a selected set of samples sent to Hazen Laboratory. The results from all comparisons are presented in the 2005 report. In general, the results showed good correlation and high sampling variability for MoS2.

During the CuMoCo 2007-2012 drill campaign blanks and 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.

### *12.2 Blanks*

During CuMoCo's diamond drill programs blank samples were inserted in the sample stream at or about a 1 in 20 frequency. A total of 431 were analyzed for MoS2, Cu, Ag, Re, Ga, W, Fe and S. The results were very good with no anomalies produced. The graphs for MoS2 and Cu are shown below.

**Figure 12-1: MoS2 in Blank Samples from CuMoCo Drill Programs at Cumo**

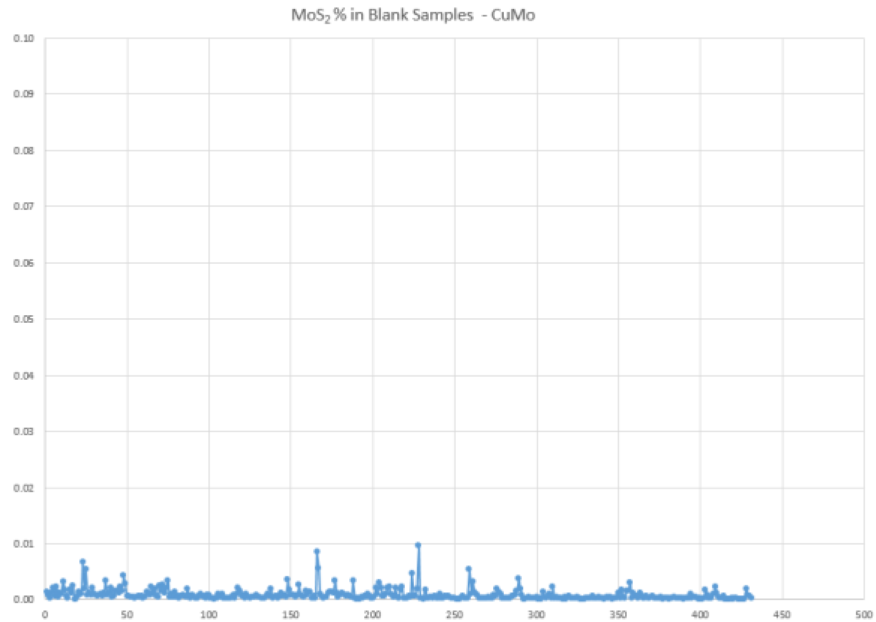
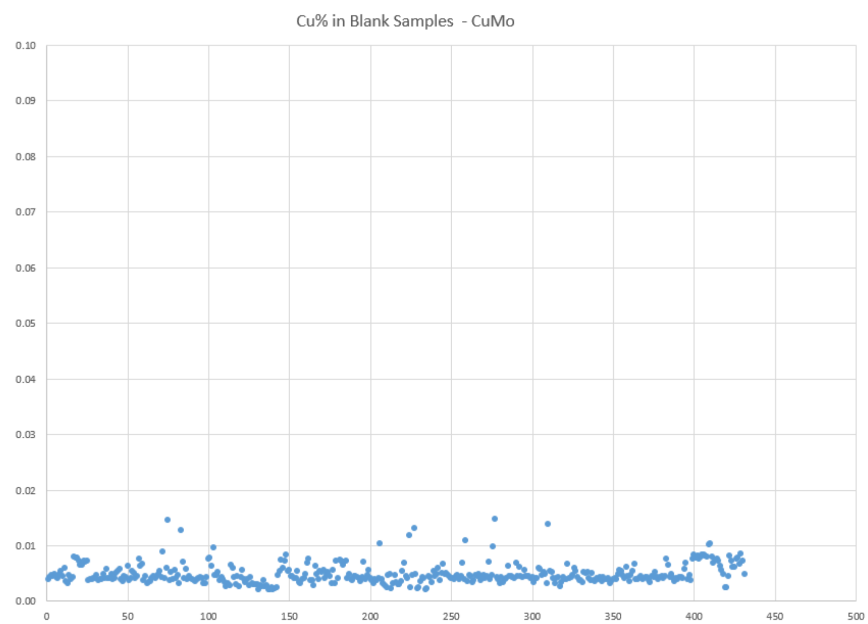



**Figure 12-2: Cu in Blank Samples from 2008 Drill Program Cumo**




### *12.3 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.

### *12.4 Internal Pulp Checks*

ALS Chemex also routinely runs duplicate checks on sample pulps. Over the 2007-2012 drill program a total of 143 check samples were run for MoS2. Figure 12-3 and 12-4 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.

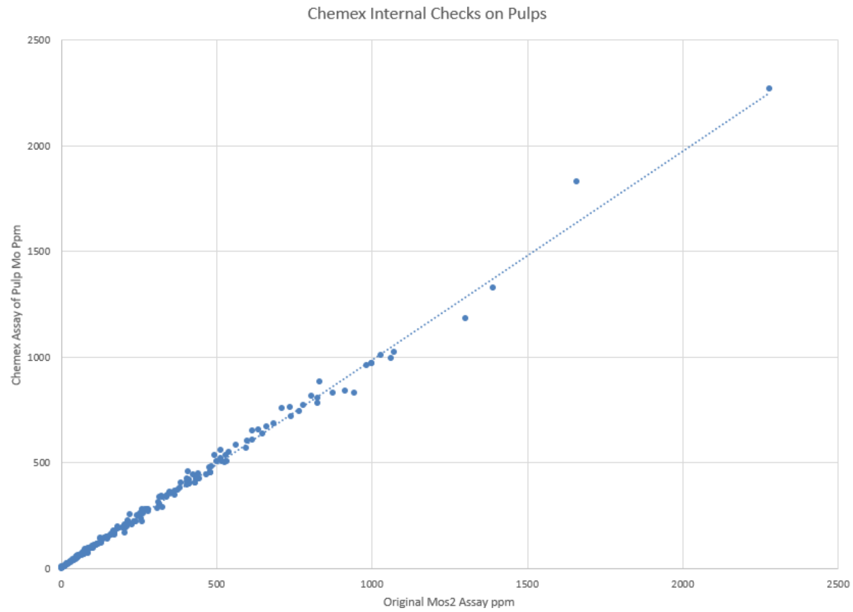



**Figure 12-3: Scatter plot of Chemex Internal Duplicates for Mo ppm**

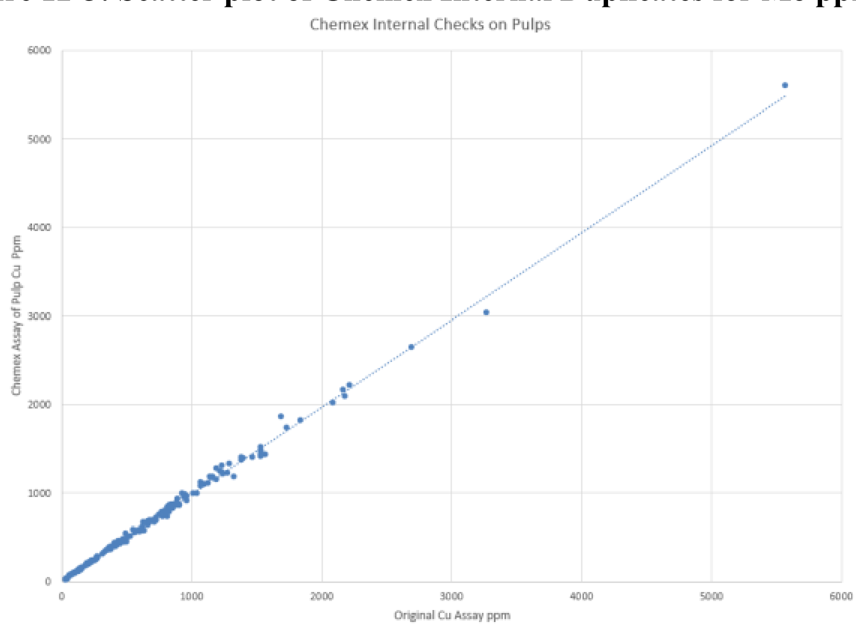
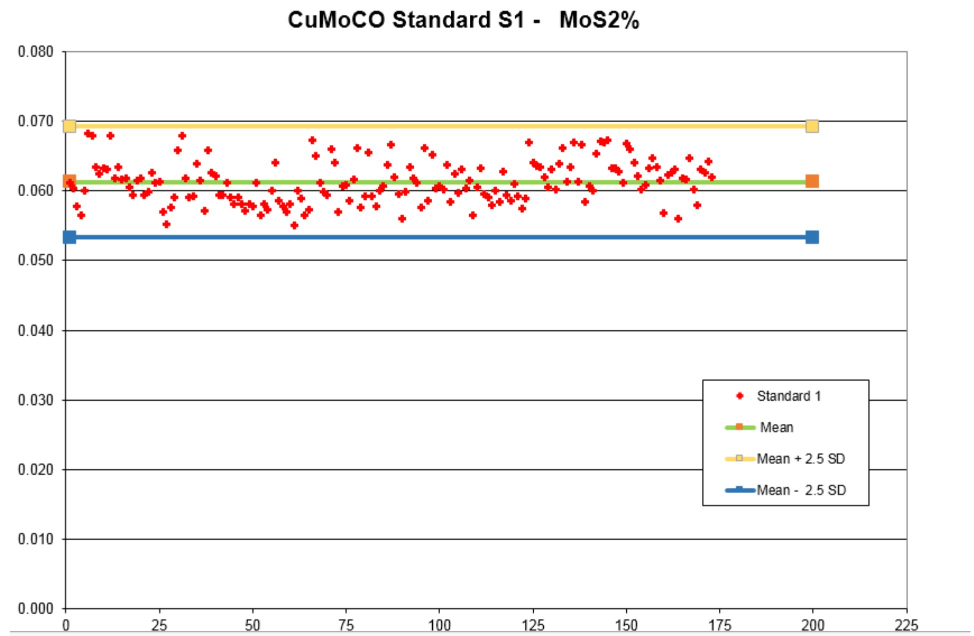



**Figure 12-4: Scatter plot of Chemex Internal Duplicates for Cu ppm**

### *12.5 CuMoCo Standards*

As explained in Section 12 CDN Labs prepared a set of Standards using drill core from the Cumo property. Results for Standard 1 (Figure 12-5), the medium grade standard for Mo and highest grade for Cu, show results are reasonable with most falling between the mean ± 2.5 standard deviations.



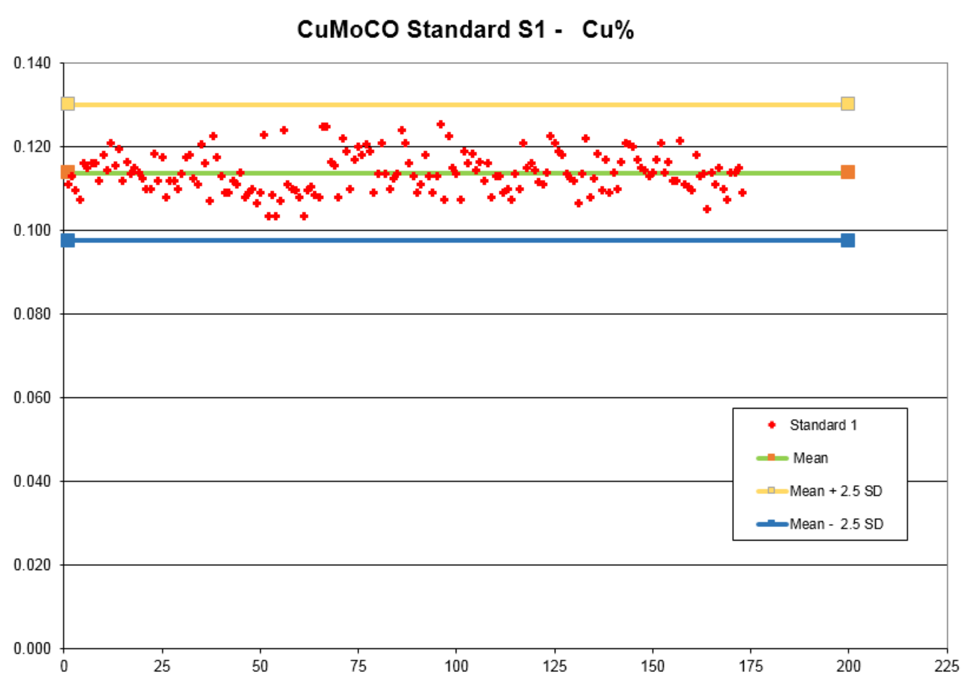
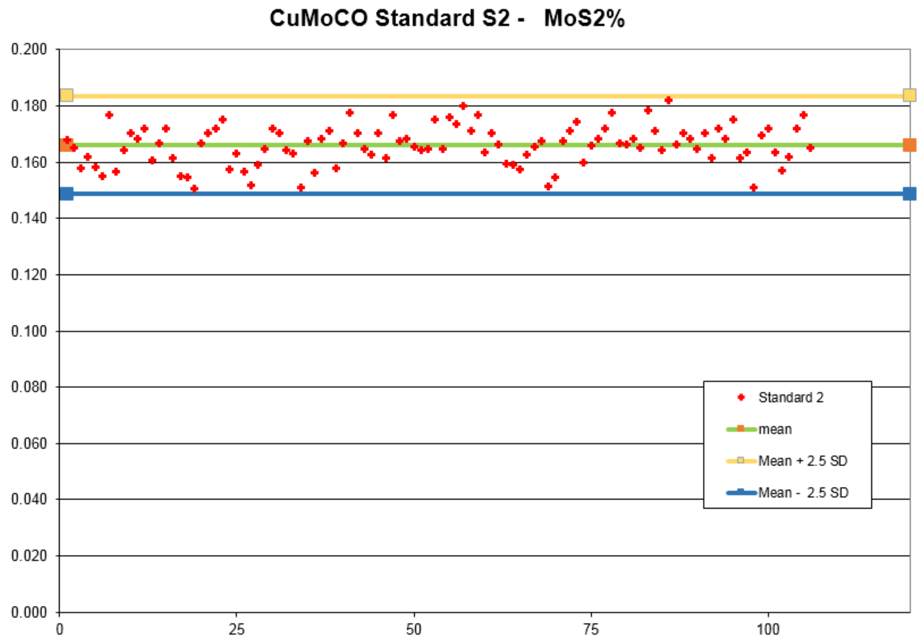




**Figure 12-5: Results for Standard S1**

Results for Standard S2, a higher grade Mo and low grade Cu standard, show reasonable results for Cu Mo assays (see Figure 12-6) with all falling between the mean ± 2.5 standard deviations.



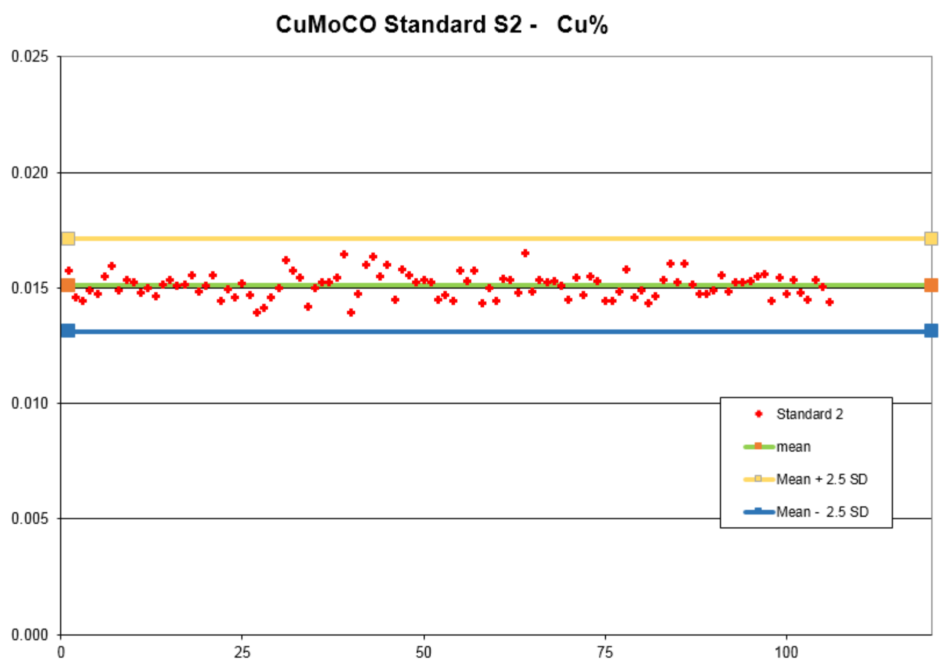
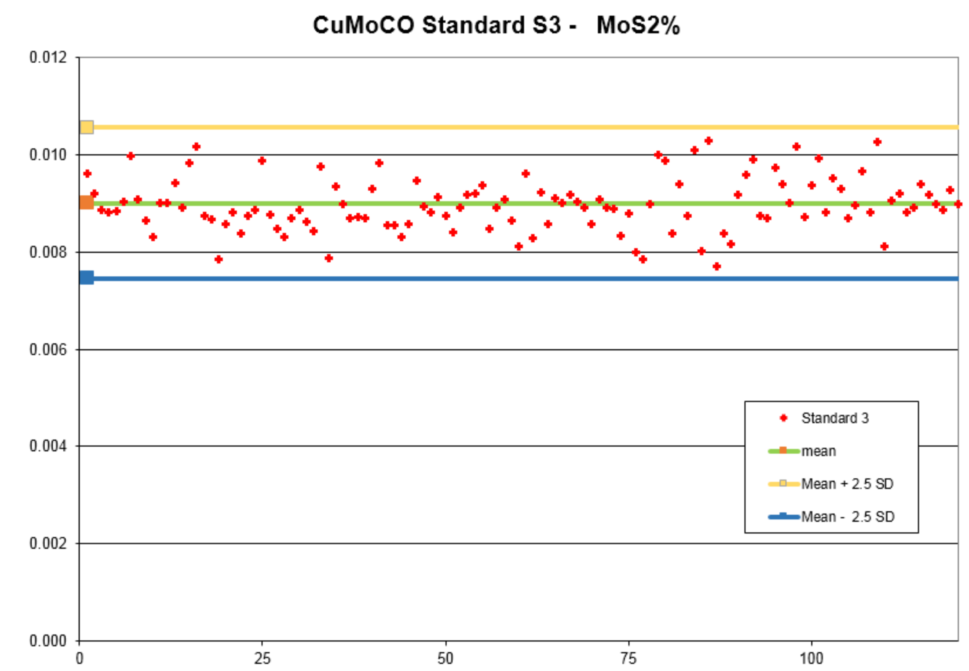




**Figure 12-6: Results for Standard S2**

The results for Standard S3 are also reasonable with more noise in the analysis, due to the low grade values encountered, but no large variations (see Figure 12-7).



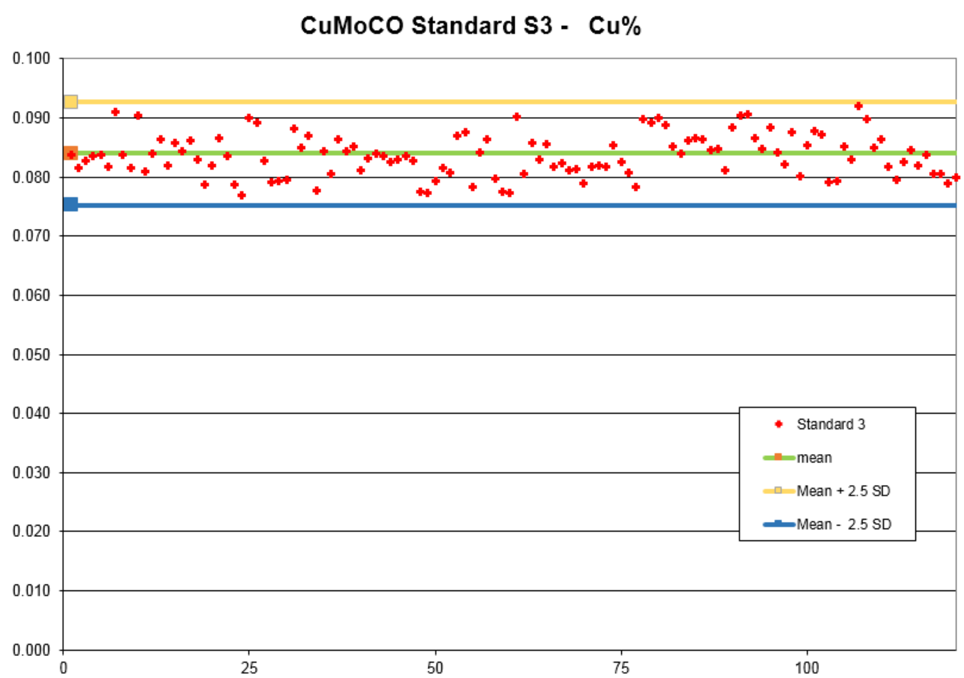




**Figure 12-7: Results for Standard S3**

### *12.6 Coarse Reject Duplicates*

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

708 duplicate samples were submitted in between 2008 and 2012, for a submission frequency rate of 1 in 20 samples.

The results are presented as a series of scatter plots with all variables reported in ppm and are shown in Appendix 2.

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

### *12.7 Statements regarding verification*

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

## 13.0 Metallurgical testing

This section is reproduced in total for completeness from “CUMO 2009 PEA Technical Report, November 18, 2009”.

### *13.1 Metallurgical testing*

#### 13.1.1 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.

### 13.1.2 Sample selection

CuMoCo 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 mineralized zones. Technicians supervised by geological staff collected the samples and prepared them for shipment. A bonded carrier took the samples from Garden Valley, Idaho to Vancouver, British Columbia. The samples were taken to SGS Canada, Kent Corporate Center, Kent Avenue N., Vancouver, British Columbia, for the metallurgical study. The test work results are detailed in an independent 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".

### 13.1.3 Test work program

The metallurgical test work program used as the basis for this report consisted of comminution and flotation test work on three separate metallurgical composites; copper/silver, copper/molybdenum and molybdenum, that were assembled to represent the three known ore types in the CUMO deposit. The test work results are reported in "An Investigation into the Recovery of Molybdenum, Copper and Silver from CUMO samples prepared for Mosquito Consolidated Gold Mines Ltd Project 50004-001" (SGS, 2009).

Two main phases of metallurgical testing were undertaken on the CUMO ore body samples:

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

**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 13-1 summarizes the outcomes of the comminution laboratory test work undertaken for this study, the table also shows the selected design case, which typically corresponds to copper/silver 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 13-1: 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 |

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.

### 13.1.4 Conceptual Study Flotation Test Work

The Conceptual Study flotation test work program was divided into three phases: rougher flotation; open circuit cleaner flotation; and 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 summarized in Table 13-2.

**Table 13-2: Baseline flotation results for CUMO composite samples**

| ore | test | Feed | | Concentrate Grade | | | Concentrate Recovery | | |
|---|---|---|---|---|---|---|---|---|---|
| type | No. | %Cu | g/t Mo | %Cu | g/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 |
| | VF1-2 | 0.16 | 179 | 1.71 | 0.27 | 53 | 58.7 | 81.6 | 70.3 |
| Cu-Mo | VF2-1 | 0.12 | 435 | 2.11 | 0.79 | 42 | 89.7 | 92.4 | 74 |
| | VF2-2 | 0.11 | 398 | 1.54 | 0.61 | 36 | 89.3 | 92.9 | 74.5 |
| Mo | VF3-1 | 0.03 | 1135 | 0.47 | 1.99 | 13 | 77 | 94.4 | 64.4 |
| | VF3-2 | 0.03 | 1135 | 0.44 | 1.75 | 12 | 83.1 | 96.9 | 71.8 |

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 for all the ore types will be required to ensure good recovery. Additional grind sensitivity test work should be included in subsequent testing to optimize the mineral recovery with grind size.

### 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 optimization tests are summarized in Table 13-3.

**Table 13-3: Cleaner flotation results for CUMO composite samples**

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

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

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

### c) Locked Cycle Test Work at Design Conditions

Flotation results from the optimization test work highlighted the benefit of fine regrinding and multiple stages of concentrate cleaning on improving concentrate grade. A flow sheet incorporating rougher concentrate regrinding and multiple stages of cleaning, similar to that from the open circuit cleaner testing was selected for the Conceptual Study. To test the flow sheet performance on all 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 summarized in Table 13-4.

**Table 13-4: Locked cycle test results**

| ore type | test No. | Feed %Cu | Feed g/t Mo | Concentrate Grade %Cu | Concentrate Grade g/t Mo | Concentrate Grade g/t Ag | Concentrate Recovery %Cu | Concentrate Recovery % Mo | Concentrate Recovery %Ag |
|---|---|---|---|---|---|---|---|---|---|
| Cu-Ag | VF1-LCT1 | 0.16 | 190 | 13 | 2 | 357 | 62.5 | 82.00% | 71.70% |
| Cu-Mo | VF2-LCT1 | 0.12 | 401 | 16.4 | 5.66 | 324 | 90.7 | 93.80% | 80.00% |
| Mo | VF3-LCT1 | 0.04 | 1065 | 5.1 | 21.6 | 122 | 71.6 | 99.60% | 59.30% |

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

### 13.1.5 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 oxidized and non-oxidized material.

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

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 optimize 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 13-5. These figures include bulk concentrate recovery, copper/molybdenum flotation separation and ferric chloride leach recovery.

**Table 13-5: Grade/recovery predictions for CUMO ores**

| Ore Type | Concentrate | Concentrate Grade | | Concentrate Recovery | | |
|---|---|---|---|---|---|---|
| | | % Cu | % Mo | % Cu | % Mo | % Ag |
| Cu-Ag | Molybdenum | 0.1 | 52 | 0.02 | 83 | |
| | Copper | 19 | 0.1 | 64 | 2.4 | 70 |
| Cu-Mo | Molybdenum | 0.1 | 51 | 0.04 | 92 | |
| | Copper | 22 | 0.1 | 85 | 0.7 | 78 |
| Mo | Molybdenum | 0.02 | 49 | 0.1 | 95 | |
| | Copper | 20 | 0.8 | 72 | 1.0 | 55 |

## *13.2 Mineral Processing*

### 13.2.1 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.

### *13.2.2 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 13-6.

**Table 13-6: Summary of the process plant design criteria.**

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

### 13.2.3 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), and 136 kt/d (metric tonnes) and181 kt/d (metric tonnes);
- Design availability of 92% (after ramp-up), being 8,059 operating hours per year, with standby equipment in critical areas, and
- Sufficient plant design flexibility for treatment of all ore types at design throughput.

The selection of these parameters is discussed in detail below.

### 13.2.4 Throughput and Availability

Four different throughput scenarios were nominated by CuMoCo 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.

### 13.2.5 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 CuMoCo have been used in the process design criteria.

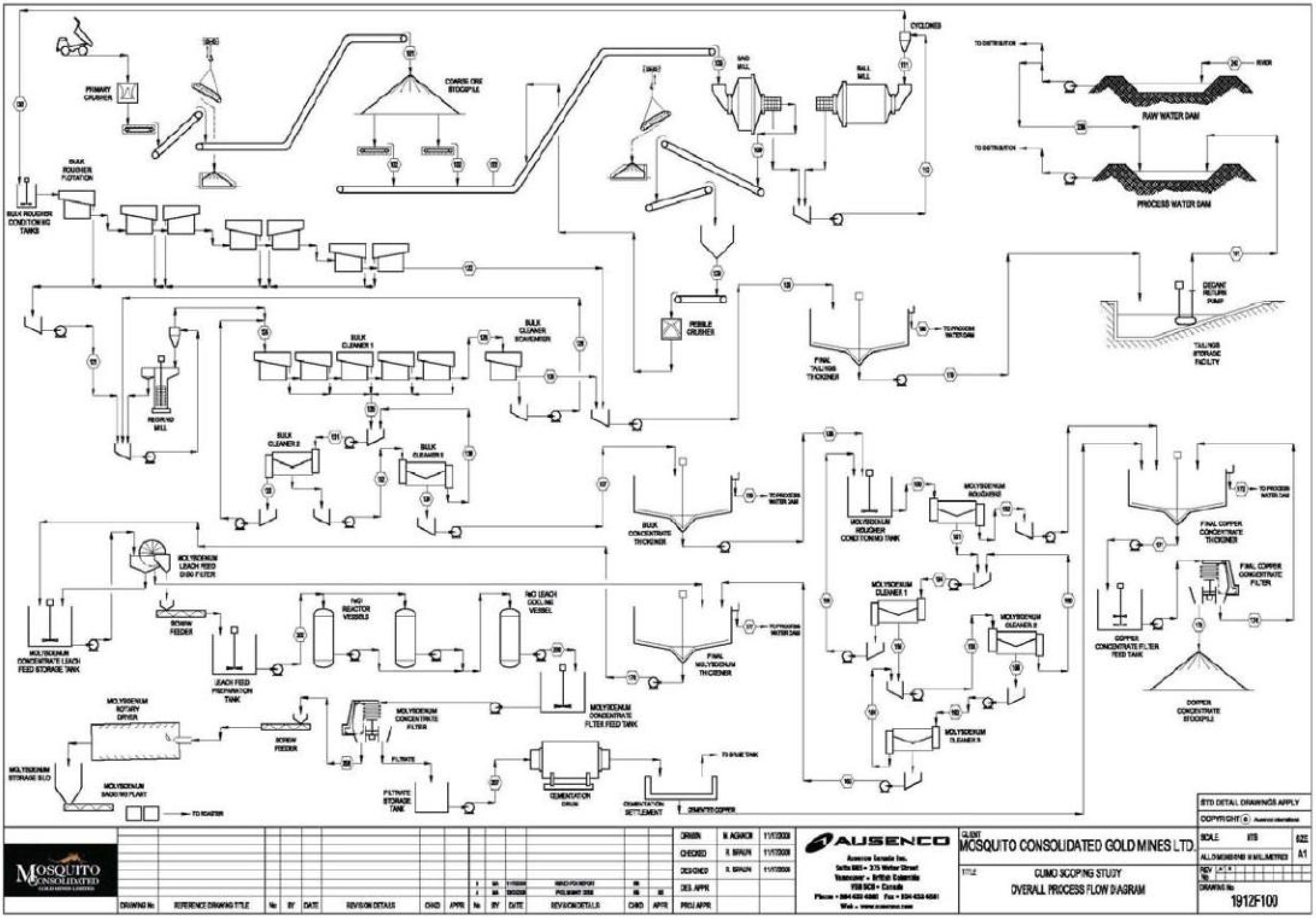
### 13.2.6 Head Grade

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% $MoS_2$).

## *13.3 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 13-1 shows a process schematic for the CUMO plant.
Details of the flow sheet design and selection of major equipment for the various options are discussed in the sections below.

**Figure 13-1: CUMO Process Plant Process Schematic**



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

- Ore from the open pit is crushed using a primary gyratory crusher to a crushed product size of nominally 80% passing (P80) 120 mm and fed onto the stockpile feed conveyor
- Conical stockpile of crushed ore with a live capacity of 18 h, with two apron feeders per grinding train, each capable of feeding 120% of the full mill throughput
- A 22 MW SAG mill, 11.58 m diameter with 7.60 m EGL, in closed circuit with pebble crushing
- Pebble crushing will be comprised of 2 MP800's per grinding train, crushing to a product size of nominally 80% passing (P80) 12 mm
- Three 13 MW ball mills per grinding train, 7.32 m diameter with 12.19 m EGL, in closed circuit with hydrocyclones, grinding to a product size of nominally 80% passing (P80) 63 µm
- Bulk rougher flotation consisting of 200 m3 forced air tank flotation cells to provide a total of 28 minutes of retention time
- Rougher concentrate regrinding in 3 off 1.0 MW vertical stirred mills per grinding train to a P80 of 10 µm
- Bulk cleaner 1 and cleaner scavenger flotation consisting of 20 m3 forced air tank flotation cells to provide a total of 13 minutes of retention time
- Bulk cleaner 2 flotation cells consisting of 8 m3 trough shaped flotation cells to provide a total of 10 minutes of retention time
- Bulk cleaner 3 flotation cells consisting of 8 m3 trough shaped flotation cells to provide a total of 5 minutes retention time
- Bulk concentrate thickening in 11 m diameter high rate thickeners
- Molybdenum rougher flotation consisting of 8 m3 trough shaped flotation cells to provide a total of 35 minutes of retention time
- Molybdenum cleaner 1 consisting of 1.5 m3 trough shaped flotation cells to provide a total of 25 minutes of retention time

- Molybdenum cleaner 2 flotation cells consisting of 1.5 m3 trough shaped flotation cells to provide a total of 25 minutes of retention time

- Molybdenum cleaner 3 flotation cells consisting of 1.5 m3 trough shaped flotation cells to provide a total of 25 minutes retention time

- Copper concentrate thickening in a high rate thickener and filtration in a horizontal plate and frame pressure filter

- Molybdenum concentrate thickening in a high rate thickener

- Molybdenum ferric chloride leach in 4 000 U.S. gallon, glass lined steel leach reactors followed by drying and storage in bulk 1 ton bags

- Tailings thickening in a high rate thickener to an underflow density of 65% solids

- TSF for process tailings in a conventional dam

- Raw process plant water supply from site water storage facility reticulated throughout the plant as required. (Harvesting and storage of raw water sufficient to allow continued water supply throughout the year is excluded from the study scope)

- Process water dam and distribution system for reticulation of process water throughout the plant as required. Process water is supplied from water reclaimed from the TSF, from process operations and site run-off with raw water used as make-up water as required

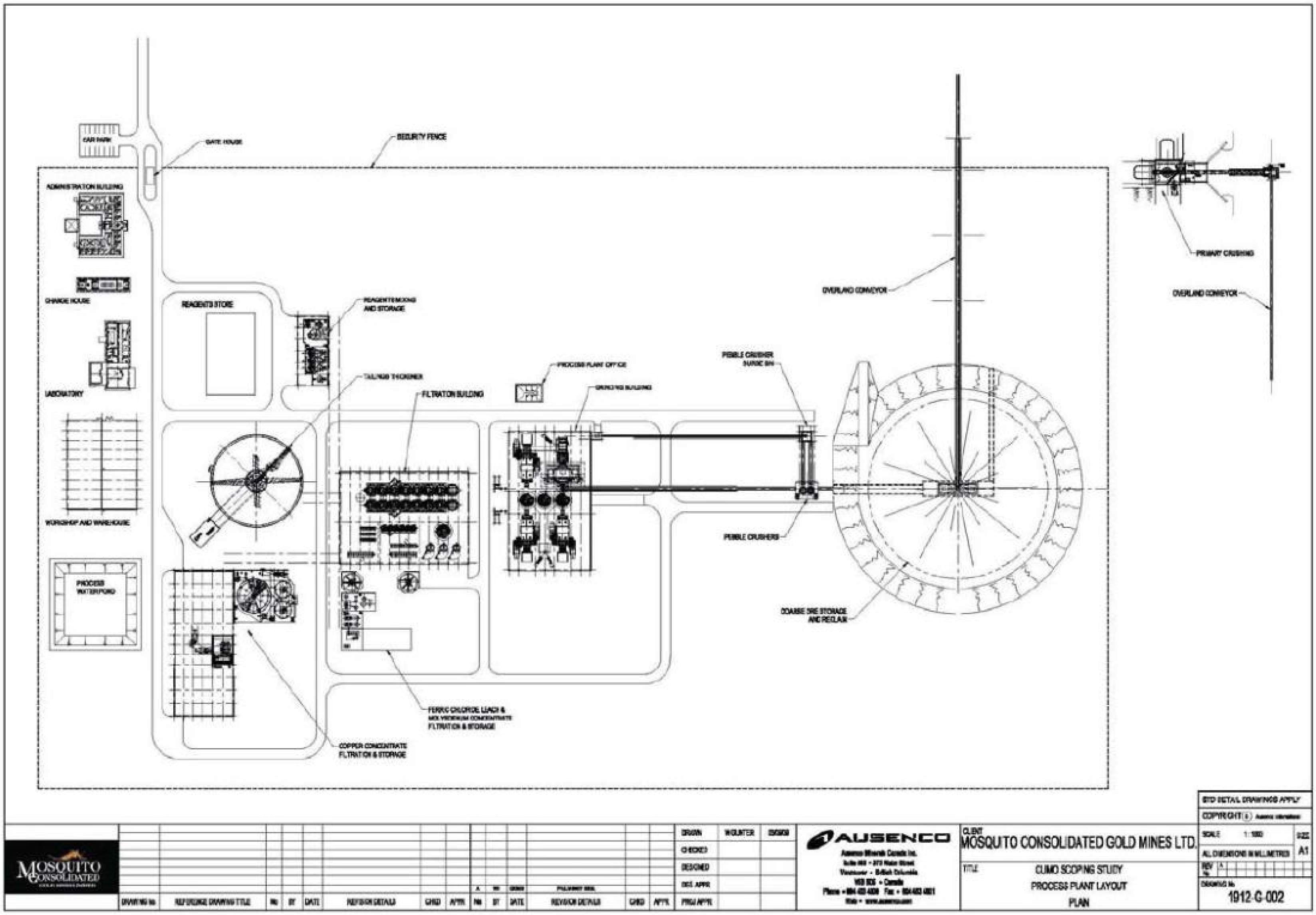
- Potable water is generated by treatment of raw water in a reverse osmosis (RO) unit at the process plant. Potable water is distributed to the plant, and for miscellaneous purposes around the site

- Plant, instrument and flotation air services and associated infrastructure.

### 13.3.2 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 13-2.
The circuit layout has taken cognizance of the site topography and worked within the bounds imposed by preliminary locations of the pit, stockpiles and waste dumps.

**Figure 13-2 CUMO Process Plant Layout**



# 14.0 Mineral Resource Estimates

At the request of Shaun Dykes, CEO of American CuMo Mining Corporation, ("CUMOCO") Giroux Consultants Ltd. was retained to produce a resource estimate on the CuMo Project in Southern Idaho. A total of 68 drill holes covering the various mineralized zones were provided. The effective date for this Estimate is April 28, 2015, the day the data was received.

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

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

As reported by Snowden in 2012 there appears to be no issues or factors that could materially affect the Mineral Resource Estimate. This includes no issue involved with environmental permitting, legal, title, taxation, socio-economic, marketing, political, mining, metallurgical or infrastructure.

Since the property is in the United States all units are in Imperial.

## *14.1 Data Analysis*

A total of 65 diamond drill holes and 3 reverse circulation drill holes, over a combined total of 121,280 ft., were provided with 1,001 downhole surveys and 10,456 assays for $MoS_2$ and Cu. For this resource estimation the 3 reverse circulation holes were not used (see Appendix 3 for a list of drill holes used in the Estimate).

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

The basic assay statistics for diamond drill holes are presented below in Table 14-1.

**Table 14-1: Summary of Assay Statistics**

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

The molybdenum and copper mineralization at CuMo lies in four distinct mineral zones with an oxidized layer on top. More or less from top to bottom there occurs in most drill holes an Oxide Zone, Cu-Ag zone, a Cu-Mo zone and a Mo zone. Within one fault block the Cu-Ag Zone is missing and the oxide sits on top of the CuMo Zone. These zones are underlain by a potassic-silica zone with lower grade copper and molybdenum grades called the MSI zone. While the oxide zone has been modeled for metallurgical reasons it has been combined with the Cu-Ag zone or in a few cases the CuMo Zone for estimation purposes. Contact plots for each variable (Figures 14.1) show there is no difference in average grade across the Oxide – Cu-Ag Zone contact. There are also several post mineral dykes that are large enough and continuous enough to be modeled. The Cu and $MoS_2$ grades statistics are shown in Table 14-2 sorted by Zone. Silver and tungsten assays are shown in Table 14-3 for the same mineral zones. Values for $MoS_2$ and Cu reported as 0.000 were assigned values of 0.0005% and 0.001 % respectively. Silver values reported as 0.000 were set to 0.01 g/t while tungsten values reported as 0.000 were set to 0.1 ppm.

**Table 14-2: Summary of Assay Statistics for Cu and $MoS_2$ Sorted by Zone**

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

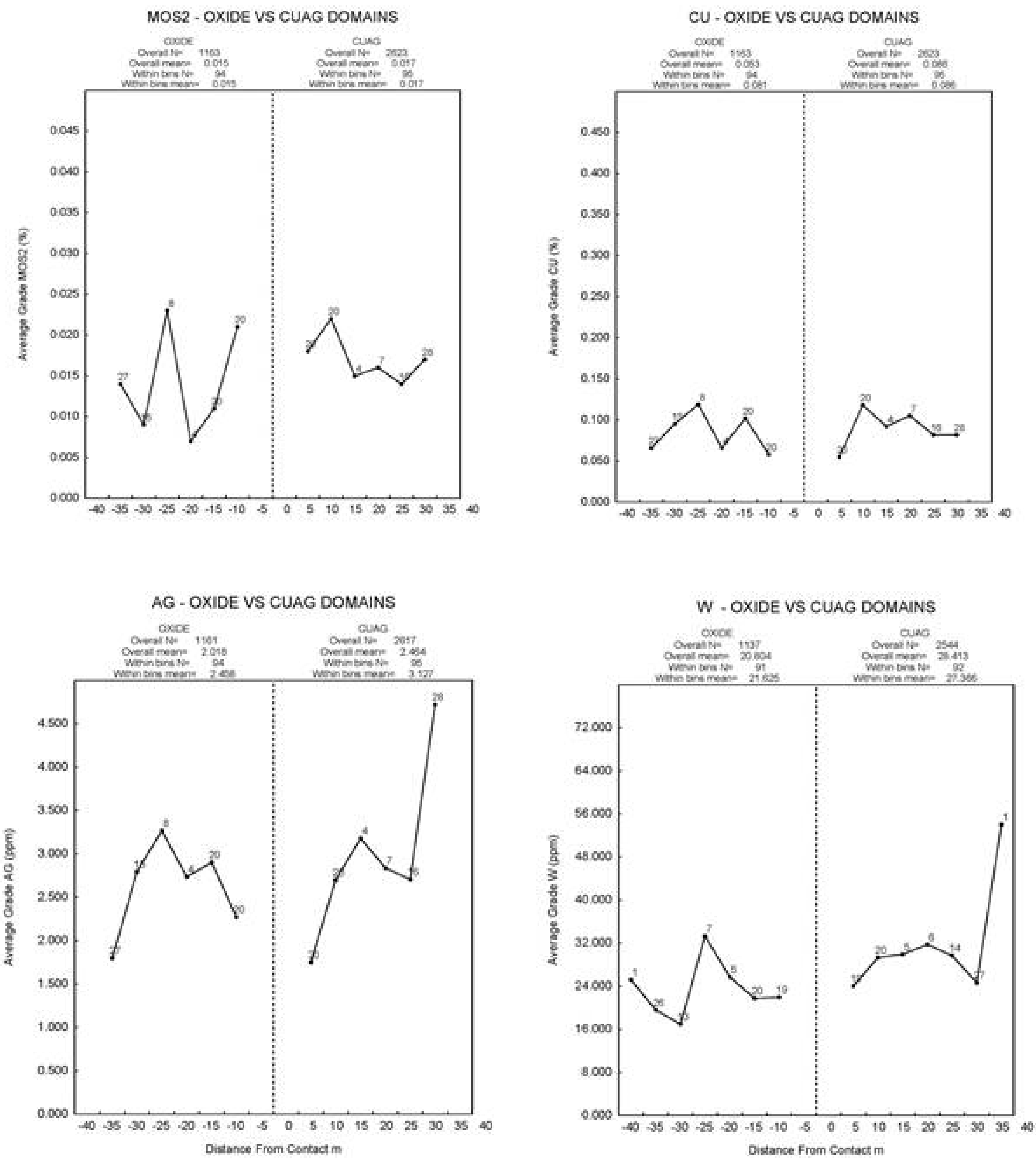



**Figure 14.1 – Contact plots for Oxide-CuAg Domain contact**

**Table 14-3: Summary of Assay Statistics for Ag and W Sorted by Zone**

| | Cu–Ag Zone | | Cu-Mo Zone | | Mo Zone | | MSI Zone | | Dykes | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Ag (g/t) | W (ppm) | Ag (g/t) | W (ppm) | Ag (g/t) | W (ppm) | Ag (g/t) | W (ppm) | Ag (g/t) | W (ppm) |
| Number | 3,806 | 3,791 | 3,492 | 3,497 | 2,653 | 2,654 | 327 | 330 | 128 | 121 |
| Mean | 2.88 | 32.3 | 3.07 | 46.7 | 1.78 | 45.9 | 1.65 | 37.1 | 0.62 | 9.8 |
| Standard Deviation | 16.28 | 108.9 | 13.35 | 33.8 | 9.81 | 38.3 | 10.39 | 109.3 | 1.23 | 11.9 |
| Minimum | 0.01 | 0.1 | 0.01 | 0.1 | 0.01 | 0.1 | 0.01 | 3.3 | 0.01 | 0.1 |
| Maximum | 838.0 | 5400.0 | 744.0 | 470.0 | 494.0 | 890.0 | 182.0 | 1980.0 | 8.6 | 65.0 |
| Coefficient of Variation | 5.65 | 3.37 | 4.35 | 0.72 | 5.51 | 0.83 | 6.28 | 2.95 | 1.99 | 1.21 |

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

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

**Table 14-4: Summary of Capping levels by Domain**

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

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

**Table 14-5: Summary of Capped Assay Statistics for Cu and $MoS_2$ Sorted by Zone**

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

**Table 14-6: Summary of Capped Assay Statistics for Ag and W Sorted by Zone**

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

## *14.2 50 Foot Composites*

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

**Table 14-7: Summary of 50 ft. Composite Statistics**

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

### *14.3 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. Nested spherical models were fit to all directions with the model parameters tabulated below and the models shown in Appendix 4.

**Table 14-8: Parameters for semivariogram models at CuMo**

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

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

### *14.4 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:

| | | |
|---|---|---|
| Lower Left Corner | | |
| 214,600 E | Column Size – 50 ft. | 207 Columns |
| 114,250 N | Row Size – 50 ft. | 179 Rows |
| Top of Model | | |
| 7075 Elevation | Level Size – 50 ft. | 76 Levels |

### *14.5 Grade Interpolation*

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

$MoS_2$ - Estimated for Cu-Ag Domain using only composites from Cu-Ag and Oxide Domains
- 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, Cu-Mo and Oxide 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, Cu-Mo and Oxide Domains

W - Estimated for Cu-Ag Domain using only composites from Cu-Ag and Oxide Domains
- 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 ranges. Pass 1 required a minimum of 4 composites within a search ellipsoid with dimensions equal to ¼ of the semivariogram range for each direction. 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. Pass 3 expanded the search ellipse to the entire range and a final 4$^{th}$ pass used double the range. In all cases the maximum number of composites from a single hole was set to 3 to insure that a minimum of two drill holes were used in each estimate. The maximum number of composites used was set to 16 and if more than 16 composites were found the closest 16 were used. The search parameters for each run are listed below in Table 14-9. Pass 4 for Ag and W used larger search ellipses to produce a value for all blocks estimated for $MoS_2$ and Cu. This was due to the under-sampling of Ag and W relative to $MoS_2$ and Cu.

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

**Table 14-9: 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 | 4,614 | | 300 | | 100 | | 200 |
| | | 2 | 26,207 | 0 /0 | 600 | 270 / 0 | 200 | 0 / -90 | 400 |
| | | 3 | 83,342 | | 1,200 | | 400 | | 800 |
| | | 4 | 252,646 | | 2,400 | | 800 | | 1,600 |
| Cu-Mo | $MoS_2$ | 1 | 35,447 | | 450 | | 325 | | 125 |
| & | | 2 | 110,887 | 60 / 0 | 900 | 150 / -55 | 650 | 330 / -35 | 250 |
| Mo | | 3 | 121,147 | | 1,800 | | 1,300 | | 500 |
| | | 4 | 59,784 | | 3,600 | | 2,600 | | 1,000 |
| Cu-Ag | Cu | 1 | 50,852 | | 400 | | 175 | | 400 |
| & | | 2 | 128,958 | 60 / 0 | 800 | 150 / -55 | 350 | 330 / -35 | 800 |
| Cu-Mo | | 3 | 235,739 | | 1,600 | | 700 | | 1,600 |
| | | 4 | 139,891 | | 3,200 | | 1,400 | | 3,200 |
| Mo | Cu | 1 | 1,789 | | 300 | | 100 | | 125 |
| | | 2 | 22,307 | 60 / 0 | 600 | 330 / 0 | 200 | 0 / -90 | 250 |
| | | 3 | 58,857 | | 1,200 | | 400 | | 500 |
| | | 4 | 80,068 | | 2,400 | | 800 | | 1,000 |
| Cu-Ag | Ag | 1 | 1,859 | | 250 | | 50 | | 125 |
| & | | 2 | 18,305 | 70 / 0 | 500 | 340 / 0 | 100 | 0 / -90 | 250 |
| Cu-Mo | | 3 | 94,108 | | 1,000 | | 200 | | 500 |
| | | 4 | 441,174 | | 2000 | | 400 | | 1,000 |
| Mo | Ag | 1 | 3,067 | | 300 | | 125 | | 175 |
| | | 2 | 31,146 | | 600 | | 250 | | 350 |
| | | 3 | 63,317 | 60 / 0 | 1,200 | 330 / 0 | 500 | 0 / -90 | 700 |
| | | 4 | 65,491 | | 2,400 | | 1,000 | | 1,400 |
| Cu-Ag | W | 1 | 14,288 | | 275 | | 300 | | 100 |
| | | 2 | 51,953 | 30 / 0 | 550 | 300 / 0 | 600 | 0 / -90 | 200 |
| | | 3 | 122,F65 | | 1,100 | | 1,200 | | 400 |
| | | 4 | 179,224 | | 2,200 | | 2,400 | | 800 |
| Cu-Mo | W | 1 | 4,799 | | 250 | | 125 | | 200 |
| & | | 2 | 59,057 | 0 / 0 | 500 | 270 / 0 | 250 | 0 / -90 | 400 |
| Mo | | 3 | 130,570 | | 1,000 | | 500 | | 800 |
| | | 4 | 144,312 | | 2000 | | 1,000 | | 1,600 |

## *14.6 Bulk Density*

A total of 4,539 specific gravity determinations were made for CuMo in all grade Domains. This total includes 4,339 determinations made during the 2011 drill program. The measurements were made using the weight in air/weight in water procedure. The results are summarized below.

**Table 14-10: Summary of Density Parameters for each Domain**

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

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

## *14.7 Classification*

### 14.7.1 Introduction

Based on the study herein reported, delineated mineralization of the CuMo Project is classified as a resource according to the following definitions from National Instrument 43-101 and from CIM (2014):

*"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 Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council on May 10, 2014, as those definitions may be amended."*

**Mineral Resource**

*"Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.*

*A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.*

*Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal and industrial minerals.*

*The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase "reasonable prospects for economic extraction" implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual* *economic extraction. Assumptions should include estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.*

*Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time."*

The terms Measured, Indicated and Inferred are defined by CIM (2014) as follows:

### Inferred Mineral Resource

*"An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.*

*An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*

*An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.*

*There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource."*

### Indicated Mineral Resource

*"An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.*

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

*An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.*

*Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions."*

## Measured Mineral Resource

*"A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.*

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

*A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.*

*Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit."*

## Modifying Factors

*"Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors."*

### 14.7.2 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 domains. 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.

Contiguous blocks estimated in Pass 1 (using ¼ of the semivariogram range) for both $MoS_2$ and Cu were classified as Measured. For the Mo and MSI zones where Cu, Ag and W provide little of the economic benefit contiguous blocks estimated in Pass 1 for $MoS_2$ were classified as Measured. Unclassified blocks estimated for Cu or $MoS_2$ in Pass 1 or 2 using search ellipses up to a maximum of ½ the semivariogram range were classified as Indicated. All other blocks were classified as Inferred.

Figure 14.2 shows isometric plan views of the measured, indicated and inferred blocks at CuMo.

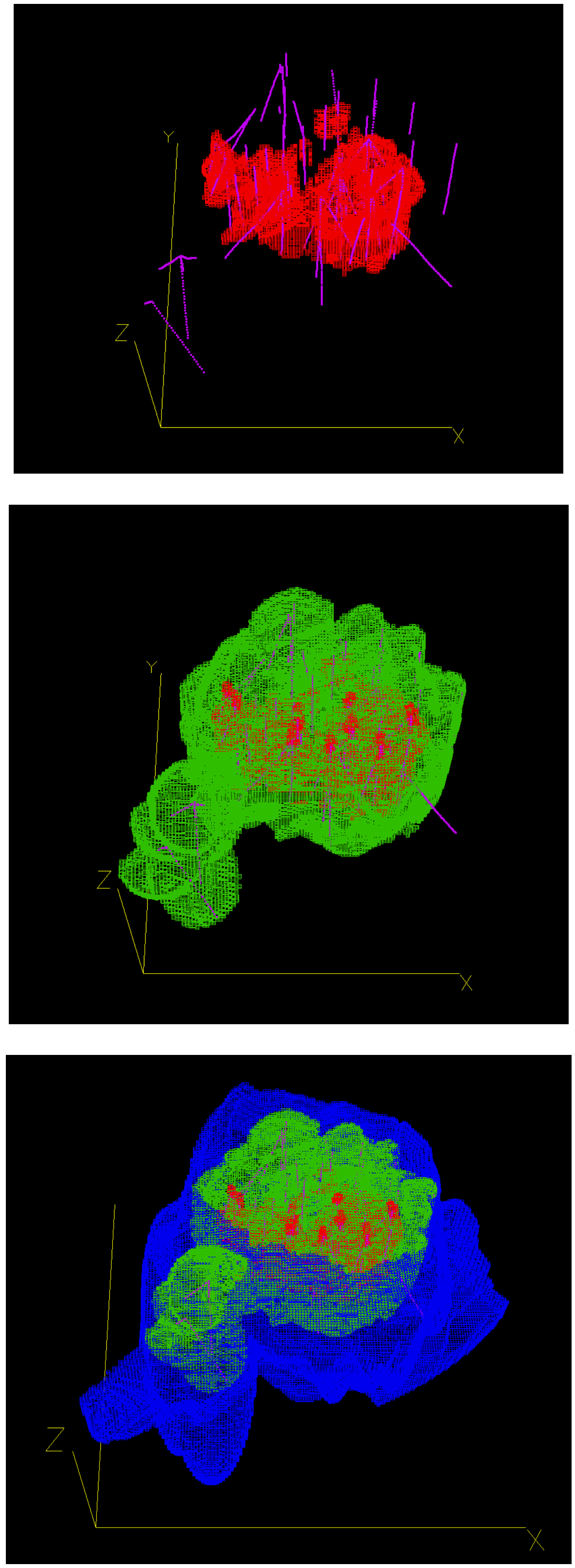



**Figure 14.2: Plan views showing Measured blocks in red, Indicated blocks in green, Inferred blocks in blue and composites in magenta**

To properly evaluate the CuMo Deposit with 4 metals occurring in different zones, a form of metal equivalent or Recoverable Value (RCV) 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. The price used in this study for $MoO_3$ is $10/lb. $MoO_3$ is calculated from $MoS_2$ by the following: Pounds Mo = $MoS_2$ * 20 / 1.6681 and then Pounds $MoO_3$ = Pounds Mo * 1.5
Cu – A copper price of $3.00 / lb was used
Ag – A silver price of $12.50 / oz was used
W – A tungsten price of $15.00 / lb was used

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

**Table 14-11: Metal recoveries sorted by Domain**

| | %Recoveries in Oxides | %Recoveries in Cu-Ag Domain | %Recoveries in Cu-Mo Domain | %Recoveries in Mo & MSI Domains |
|---|---|---|---|---|
| Cu | 60.0 | 68.0 | 85.0 | 72.0 |
| Ag | 65.0 | 75.0 | 78.0 | 55.0 |
| W | 0.0 | 35.0 | 35.0 | 35.0 |
| $MoS_2$ | 80.0 | 86.0 | 92.0 | 95.0 |

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

Factors to use in RCV equation were as follows:

$$MoS_2\ \text{Factor (\$/ton)} = \frac{MoS_2\ \%}{100\%} * \frac{\text{Recovery }\%}{100\ \%} * \frac{2000\ \text{lb}}{\text{ton}} * \frac{\text{Price for } MoO_3\ \$}{\text{lb}} * \frac{1.5}{1.6681}$$

$$\text{Cu Factor (\$/ton)} = \frac{\text{Cu }\%}{100\ \%} * \frac{\text{Recovery }\%}{100\ \%} * \frac{\text{Price for Cu }\$}{\text{lb}} * \frac{2000\ \text{lb}}{\text{ton}}$$

$$\text{Ag Factor (\$/ton)} = \frac{\text{Ag gms}}{\text{Tonne}} * \frac{1\ \text{oz}}{31.1035\ \text{gms}} * \frac{1\ \text{Tonne}}{1.1023\ \text{ton}} * \frac{\text{Recovery }\%}{100\%} * \frac{\text{Price for Ag }\$}{\text{oz}}$$

$$\text{W Factor (\$/ton)} = \frac{\text{W ppm} * 1\%}{10000\ \text{ppm}} * \frac{\text{Recovery }\%}{100\ \%} * \frac{\text{Price for W }\$}{\text{lb}} * \frac{2000\ \text{lb}}{\text{ton}}$$

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

RCV (oxides) = (Cu% * 36.0) + (Ag(g/t) * 0.24) + ($MoS_2$% * 143.88)
RCV (Cu-Ag) = (Cu% * 40.8) + (Ag(g/t) * 0.27) + (W(ppm) * 0.0105) + ($MoS_2$% * 154.67)
RCV (Cu-Mo) = (Cu% * 51.0) + (Ag(g/t) * 0.28) + (W(ppm) * 0.0105) + ($MoS_2$% * 165.46)
RCV (Mo) = (Cu% * 43.2) + (Ag(g/t) * 0.20) + (W(ppm) * 0.0105) + ($MoS_2$% * 170.85)
RCV (MSI) = (Cu% * 43.2) + (Ag(g/t) * 0.20) + (W(ppm) * 0.0105) + ($MoS_2$% * 170.85)

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

A value in non-oxide material of $2.50 US has been highlighted as a possible open pit cutoff based on similar size mines at a feasibility or production stage.

In 2012 Snowden used a Whittle pit optimizer to determine a constraining open pit for the CuMo deposit. Optimization parameters were from Thompson Creek mine (a comparable open pit molybdenum project located in Idaho). The optimization parameters included ore mining and processing costs of $7.52 per processed ton, overall pit slope angles of 45 degrees, metallurgical recoveries as shown in Table 14-11 and appropriate dilution and offsite costs and royalties. The metal prices used in 2012 by Snowden for pit optimization were Mo at $25/lb, Cu at $3/lb, Ag at $20/oz and W at $10/lb.

Since the infill drill holes completed in 2011-12 were all within this conceptual pit this resource update uses the Snowden 2012 optimum pit shell to constrain the estimate.

The results are shown as three separate sets of tables based on a low, medium and high set of prices defined as follows:

| Tables | 14-12 to 14-15 | 14-16 to 14-19 | 14-20 to 14-23 |
|---|---|---|---|
| Zone | Low Price | Medium Price | High Price |
| Copper (Cu)/lb | $2.50 | $3.00 | $3.50 |
| Molybdenum oxide(MoO3)/lb | $7.50 | $10.00 | $15.00 |
| Molybdenum Metal(Mo)/lb | $11.25 | $15.00 | $22.50 |
| Silver (Ag)/ounce | $12.50 | $12.50 | $12.50 |
| Tungsten (W)/lb | $15.00 | $15.00 | $15.00 |

**Table 14-12: Measured Resource within Pit Shell using Medium Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **308.4** | **0.079** | **0.074** | **2.09** | **48.6** | **17.83** | **292.1** | **438.2** | **456.5** | **18.8** | **30.0** |
| 5.00 | 297.2 | 0.081 | 0.076 | 2.09 | 49.6 | 18.35 | 288.6 | 432.9 | 451.7 | 18.1 | 29.5 |
| 7.50 | 282.0 | 0.085 | 0.076 | 2.06 | 50.6 | 19.01 | 287.4 | 431.1 | 428.7 | 16.9 | 28.5 |
| 12.50 | 227.9 | 0.097 | 0.075 | 2.00 | 51.8 | 21.04 | 265.0 | 397.5 | 341.8 | 13.3 | 23.6 |
| 15.00 | 195.4 | 0.105 | 0.072 | 1.90 | 52.0 | 22.26 | 246.0 | 368.9 | 281.3 | 10.8 | 20.3 |
| 17.50 | 159.7 | 0.115 | 0.067 | 1.80 | 51.6 | 23.58 | 220.1 | 330.2 | 213.9 | 8.4 | 16.5 |
| 20.00 | 122.9 | 0.125 | 0.063 | 1.70 | 51.7 | 25.04 | 184.1 | 276.2 | 154.8 | 6.1 | 12.7 |

**Table 14-13: Indicated Resource within Pit Shell using Medium Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2216.1** | **0.049** | **0.079** | **2.48** | **37.6** | **12.71** | **1301.9** | **1952.9** | **3501.4** | **160.3** | **166.6** |
| 5.00 | 1972.3 | 0.053 | 0.085 | 2.57 | 39.6 | 13.82 | 1253.3 | 1880.0 | 3352.9 | 147.8 | 156.2 |
| 7.50 | 1708.3 | 0.059 | 0.088 | 2.59 | 41.1 | 14.98 | 1208.4 | 1812.6 | 3006.5 | 129.0 | 140.4 |
| 12.50 | 1050.6 | 0.076 | 0.090 | 2.55 | 44.2 | 18.13 | 957.4 | 1436.0 | 1891.1 | 78.1 | 92.9 |
| 15.00 | 798.5 | 0.083 | 0.090 | 2.56 | 45.6 | 19.54 | 794.6 | 1191.9 | 1437.2 | 59.6 | 72.8 |
| 17.50 | 541.6 | 0.093 | 0.088 | 2.49 | 46.4 | 21.09 | 603.9 | 905.8 | 953.2 | 39.3 | 50.3 |
| 20.00 | 301.3 | 0.106 | 0.082 | 2.36 | 47.7 | 22.99 | 383.0 | 574.5 | 494.2 | 20.7 | 28.7 |

**Table 14-14: Inferred Resource within Pit Shell using Medium Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **3373.6** | **0.040** | **0.057** | **1.93** | **32.1** | **9.89** | **1617.9** | **2426.9** | **3845.9** | **189.9** | **216.6** |
| 5.00 | 2556.6 | 0.048 | 0.067 | 2.13 | 34.7 | 11.84 | 1471.4 | 2207.0 | 3425.9 | 158.8 | 177.4 |
| 7.50 | 1996.0 | 0.056 | 0.070 | 2.23 | 35.1 | 13.44 | 1340.1 | 2010.2 | 2794.4 | 129.8 | 140.1 |
| 12.50 | 996.4 | 0.078 | 0.064 | 1.98 | 37.6 | 17.13 | 931.8 | 1397.7 | 1275.4 | 57.5 | 74.9 |
| 15.00 | 637.0 | 0.086 | 0.074 | 2.16 | 39.8 | 19.05 | 656.8 | 985.2 | 942.7 | 40.1 | 50.7 |
| 17.50 | 384.8 | 0.094 | 0.084 | 2.34 | 41.5 | 20.93 | 433.7 | 650.5 | 646.4 | 26.3 | 31.9 |
| 20.00 | 190.2 | 0.109 | 0.078 | 2.37 | 41.9 | 23.24 | 248.6 | 372.9 | 296.8 | 13.1 | 15.9 |

**Table 14-15: Measured and Indicated Resource within Pit Shell using Medium Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2524.5** | **0.053** | **0.079** | **2.43** | **39.0** | **13.34** | **1594.1** | **2391.1** | **3957.9** | **179.1** | **196.6** |
| 5.00 | 2269.6 | 0.057 | 0.084 | 2.50 | 40.9 | 14.41 | **1541.9** | **2312.9** | **3804.6** | **165.9** | **185.7** |
| 7.50 | 1990.4 | 0.063 | 0.086 | 2.51 | 42.4 | 15.55 | **1495.8** | **2243.7** | **3435.2** | **145.9** | **168.9** |
| 12.50 | 1278.6 | 0.079 | 0.087 | 2.46 | 45.5 | 18.65 | **1222.4** | **1833.5** | **2232.9** | **91.4** | **116.5** |
| 15.00 | 993.9 | 0.088 | 0.087 | 2.43 | 46.8 | 20.07 | **1040.6** | **1560.8** | **1718.5** | **70.4** | **93.1** |
| 17.50 | 701.4 | 0.098 | 0.083 | 2.33 | 47.6 | 21.66 | **824** | **1236** | **1167.1** | **47.7** | **66.8** |
| 20.00 | 424.3 | 0.112 | 0.077 | 2.17 | 48.9 | 23.58 | **567.1** | **850.7** | **649** | **26.8** | **41.4** |

**Table 14-16: Measured Resource within Pit Shell using Low Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **307.0** | **0.079** | **0.075** | **2.09** | **48.7** | **13.96** | **290.8** | **436.2** | **460.5** | **18.7** | **29.9** |
| 5.00 | 290.1 | 0.083 | 0.076 | 2.08 | 50.1 | 14.56 | 288.7 | 433.0 | 440.9 | 17.6 | 29.1 |
| 7.50 | 270.4 | 0.087 | 0.076 | 2.06 | 51.0 | 15.16 | 282.0 | 423.0 | 410.9 | 16.2 | 27.6 |
| 12.50 | 185.8 | 0.107 | 0.072 | 1.90 | 51.9 | 17.49 | 238.3 | 357.5 | 267.5 | 10.3 | 19.3 |
| 15.00 | 134.0 | 0.121 | 0.066 | 1.76 | 52.0 | 18.92 | 194.3 | 291.5 | 176.8 | 6.9 | 13.9 |
| 17.50 | 86.0 | 0.134 | 0.062 | 1.69 | 53.1 | 20.43 | 138.1 | 207.2 | 106.6 | 4.2 | 9.1 |
| 20.00 | 41.8 | 0.151 | 0.056 | 1.56 | 53.5 | 22.27 | 75.7 | 113.5 | 46.8 | 1.9 | 4.5 |

**Table 14-17: Indicated Resource within Pit Shell using Low Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2159.5** | **0.050** | **0.081** | **2.51** | **38.2** | **10.29** | **1294.6** | **1941.9** | **3498.4** | **158.1** | **165.0** |
| 5.00 | 1870.9 | 0.056 | 0.087 | 2.59 | 40.2 | 11.30 | 1256.2 | 1884.2 | 3255.3 | 141.3 | 150.4 |
| 7.50 | 1464.8 | 0.064 | 0.090 | 2.62 | 42.4 | 12.68 | 1124.0 | 1686.0 | 2636.6 | 111.9 | 124.2 |
| 12.50 | 720.7 | 0.085 | 0.092 | 2.61 | 46.0 | 15.64 | 734.4 | 1101.7 | 1326.0 | 54.9 | 66.3 |
| 15.00 | 382.9 | 0.099 | 0.090 | 2.55 | 47.7 | 17.32 | 454.5 | 681.8 | 689.2 | 28.5 | 36.5 |
| 17.50 | 136.2 | 0.120 | 0.080 | 2.33 | 49.5 | 19.51 | 196.0 | 294.0 | 218.0 | 9.3 | 13.5 |
| 20.00 | 41.9 | 0.143 | 0.067 | 1.99 | 49.5 | 21.79 | 71.8 | 107.7 | 56.1 | 2.4 | 4.1 |

**Table 14-18: Inferred Resource within Pit Shell using Low Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **3214.8** | **0.041** | **0.059** | **1.97** | **32.9** | **8.10** | **1580.3** | **2370.5** | **3793.5** | **184.7** | **211.5** |
| 5.00 | 2257.7 | 0.052 | 0.070 | 2.21 | 34.9 | 9.99 | 1407.6 | 2111.4 | 3160.8 | 145.5 | 157.6 |
| 7.50 | 1591.6 | 0.063 | 0.070 | 2.21 | 35.9 | 11.53 | 1202.2 | 1803.4 | 2228.3 | 102.6 | 114.3 |
| 12.50 | 519.6 | 0.089 | 0.080 | 2.27 | 40.7 | 15.46 | 554.5 | 831.7 | 831.4 | 34.4 | 42.3 |
| 15.00 | 249.8 | 0.101 | 0.087 | 2.47 | 42.4 | 17.39 | 302.5 | 453.7 | 434.6 | 18.0 | 21.2 |
| 17.50 | 94.7 | 0.122 | 0.076 | 2.46 | 41.9 | 19.62 | 138.6 | 207.9 | 144.0 | 6.8 | 7.9 |
| 20.00 | 30.6 | 0.137 | 0.081 | 2.70 | 43.3 | 21.69 | 50.2 | 75.3 | 49.5 | 2.4 | 2.6 |

**Table 14-19: Measured and Indicated Resource within Pit Shell using Low Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2466.6** | **0.054** | **0.080** | **2.45** | **39.5** | **10.75** | **1585.4** | **2378.1** | **3958.9** | **176.8** | **194.9** |
| 5.00 | 2161.0 | 0.059 | 0.085 | 2.52 | 41.6 | 11.74 | **1544.9** | **2317.2** | **3696.2** | **158.9** | **179.5** |
| 7.50 | 1735.2 | 0.068 | 0.088 | 2.53 | 43.7 | 13.07 | **1406** | **2109** | **3047.5** | **128.1** | **151.8** |
| 12.50 | 906.5 | 0.090 | 0.088 | 2.47 | 47.2 | 16.01 | **972.7** | **1459.2** | **1593.5** | **65.2** | **85.6** |
| 15.00 | 516.9 | 0.105 | 0.084 | 2.35 | 48.8 | 17.73 | **648.8** | **973.3** | **866** | **35.4** | **50.4** |
| 17.50 | 222.2 | 0.126 | 0.073 | 2.08 | 50.9 | 19.87 | **334.1** | **501.2** | **324.6** | **13.5** | **22.6** |
| 20.00 | 83.7 | 0.147 | 0.062 | 1.78 | 51.5 | 22.03 | **147.5** | **221.2** | **102.9** | **4.3** | **8.6** |

**Table 14-20: Measured Resource within Pit Shell using High Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **308.6** | **0.079** | **0.074** | **2.09** | **48.6** | **25.06** | **292.3** | **438.4** | **456.7** | **18.8** | **30.0** |
| 5.00 | 303.3 | 0.080 | 0.075 | 2.09 | 49.0 | 25.42 | 290.9 | 436.4 | 455.0 | 18.5 | 29.7 |
| 7.50 | 291.1 | 0.083 | 0.076 | 2.07 | 50.0 | 26.22 | 289.7 | 434.5 | 442.5 | 17.6 | 29.1 |
| 12.50 | 268.2 | 0.088 | 0.075 | 2.04 | 50.8 | 27.59 | 283.0 | 424.5 | 402.4 | 16.0 | 27.3 |
| 15.00 | 245.9 | 0.093 | 0.075 | 2.01 | 51.3 | 28.84 | 274.2 | 411.4 | 368.9 | 14.4 | 25.2 |
| 17.50 | 219.7 | 0.100 | 0.073 | 1.93 | 51.9 | 30.36 | 263.4 | 395.1 | 320.7 | 12.4 | 22.8 |
| 20.00 | 199.2 | 0.105 | 0.071 | 1.89 | 51.9 | 31.55 | 250.8 | 376.2 | 282.9 | 11.0 | 20.7 |

**Table 14-21: Indicated Resource within Pit Shell using High Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2288.6** | **0.048** | **0.078** | **2.44** | **36.8** | **16.93** | **1317.1** | **1975.6** | **3570.1** | **162.9** | **168.4** |
| 5.00 | 2086.0 | 0.051 | 0.083 | 2.52 | 38.7 | 18.22 | 1275.5 | 1913.3 | 3462.7 | 153.3 | 161.5 |
| 7.50 | 1894.9 | 0.055 | 0.086 | 2.56 | 40.1 | 19.42 | 1249.6 | 1874.4 | 3259.3 | 141.5 | 152.0 |
| 12.50 | 1444.2 | 0.066 | 0.087 | 2.53 | 42.0 | 22.34 | 1142.8 | 1714.2 | 2512.9 | 106.6 | 121.3 |
| 15.00 | 1202.6 | 0.072 | 0.087 | 2.49 | 43.1 | 24.07 | 1038.1 | 1557.2 | 2092.5 | 87.3 | 103.7 |
| 17.50 | 1008.2 | 0.078 | 0.086 | 2.46 | 44.1 | 25.59 | 942.9 | 1414.3 | 1734.2 | 72.3 | 88.9 |
| 20.00 | 830.0 | 0.083 | 0.087 | 2.47 | 45.2 | 27.06 | 825.9 | 1238.9 | 1444.2 | 59.8 | 75.0 |

**Table 14-22: Inferred Resource within Pit Shell using High Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **3567.2** | **0.038** | **0.055** | **1.90** | **31.0** | **13.00** | **1625.2** | **2437.9** | **3923.9** | **197.7** | **221.2** |
| 5.00 | 2988.6 | 0.043 | 0.061 | 2.00 | 33.7 | 14.80 | 1540.8 | 2311.2 | 3646.1 | 174.3 | 201.4 |
| 7.50 | 2348.0 | 0.051 | 0.068 | 2.16 | 34.8 | 17.14 | 1435.7 | 2153.6 | 3193.2 | 147.9 | 163.4 |
| 12.50 | 1543.7 | 0.065 | 0.065 | 2.06 | 35.9 | 20.83 | 1203.0 | 1804.5 | 2006.8 | 92.7 | 110.8 |
| 15.00 | 1199.7 | 0.074 | 0.060 | 1.90 | 36.7 | 22.89 | 1064.4 | 1596.6 | 1439.6 | 66.5 | 88.1 |
| 17.50 | 1005.6 | 0.078 | 0.061 | 1.92 | 37.3 | 24.18 | 940.4 | 1410.6 | 1226.8 | 56.3 | 75.0 |
| 20.00 | 767.9 | 0.084 | 0.067 | 2.02 | 38.5 | 25.83 | 773.4 | 1160.1 | 1029.0 | 45.2 | 59.1 |

**Table 14-23: Measured and Indicated Resource within Pit Shell using High Prices**

| Cut-off RCV $US | Million tons | Grade > Cut-off | | | | | Contained Metal | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | MoS2 (%) | Cu (%) | Ag (g/t) | W (ppm) | RCV US$ | Million lbs. Mo | Million lbs MoO3 | Million lbs Cu | Million oz Ag | Million lbs W |
| **2.50** | **2597.3** | **0.051** | **0.077** | **2.40** | **38.2** | **17.89** | **1588.2** | **2382.3** | **3999.9** | **181.8** | **198.4** |
| 5.00 | 2389.5 | 0.055 | 0.082 | 2.47 | 40.1 | 19.13 | 1575.7 | 2363.6 | 3918.8 | 172.1 | 191.6 |
| 7.50 | 2186.3 | 0.059 | 0.085 | 2.50 | 41.4 | 20.33 | 1546.5 | 2319.8 | 3716.7 | 159.4 | 181.0 |
| 12.50 | 1712.6 | 0.069 | 0.085 | 2.45 | 43.4 | 23.16 | 1416.8 | 2125.3 | 2911.5 | 122.4 | 148.7 |
| 15.00 | 1448.7 | 0.076 | 0.085 | 2.41 | 44.5 | 24.88 | 1320.1 | 1980.2 | 2462.9 | 101.8 | 128.9 |
| 17.50 | 1228.1 | 0.082 | 0.084 | 2.37 | 45.5 | 26.44 | 1207.4 | 1811.1 | 2063.2 | 84.9 | 111.8 |
| 20.00 | 1029.4 | 0.087 | 0.084 | 2.36 | 46.5 | 27.92 | 1073.8 | 1610.7 | 1729.4 | 70.9 | 95.7 |

# 15.0 Adjacent properties

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

# 16.0 Other relevant data and information

This section is reproduced in total for completeness from "CUMO 2009 PEA Technical Report, November 18, 2009" (PEA). The report was produced for/by Mosquito Consolidated Gold Mines (former name of CuMoCo). All quoted costs are based on 2009 prices and are in US dollars.

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.

Mr. Shaun Dykes, a professional qualified person is responsible for this section, has reviewed this section of the report and states the following, Whilst this part of the report was prepared in 2009 and re-verified by Snowden in 2012, Mr. Dykes, after a detailed examination of current long term metal prices trends, markets and analysis of operating and capital costs, considers it to remain relevant and valid as there have been no significant changes to the assumptions, and the grade of the mineralization remains similar to the 2009 estimates. The one change that has been made is that the size of the deposit and the confidence has increased, and it is Mr. Dykes's opinion, as the person responsible for this section, that this can only improve the confidence in the results of the PEA.

The examination and verification of the original PEA made the following conclusions:

**Operating costs:** overall operating costs in mining within the USA are slightly down due to improvement in efficiency, lower fuel costs and technological advances. Several mines have responded to lower metal price environment by becoming more efficient. So operating costs provided in the original PEA are still valid within the realms of the accuracy of a PEA level report.

**Capital costs**: overall capital costs on the individual items are varied with certain equipment prices higher others lower when compared to the values used in the PEA. Lower equipment costs are occurring as the result in the need to maintain market share and their skilled labor force. Large truck prices have actually dropped by 3% since the PEA for example. So as with operating costs the capital costs used in the original PEA easily fall within the parameters of the PEA.

**Metal Prices**: Molybdenum and copper prices have had a wide range in values over the past years. Copper ranging from a low of $2.20 to a high of $4.50, while Molybdenum has arranged from $6 to $43 per pound. This reflects the cyclical nature of metal prices. Copper is traded on a regular basis, however molybdenum is sold mainly by long term contracts, contracts are usually signed during favorable price times and last through the lower prices, The PEA reports numerous different price scenarios which are summarized in the section. The base case is toward the high end of the expert predictions and prices for long term metal prices, but still within the accuracy level of the report. However Mr. Dykes has reduced the base case metal prices to reflect the same prices used in this resource report. (see section 16.13.1). The best scenario is probably the cyclical price scenario which better fits the market and reflects the long term contract side of the Molybdenum market. Thompson Creek Mine was able to keep producing for about 18 months as they were receiving prices that were significantly higher than the spot market prices. So overall the price scenarios are reasonable and still valid and cover the full range of possibilities.

In response to the change in June 2011 to the 43-101 requirement that Preliminary Economic Analysis reports should contain after tax values. Wherever appropriate Mr. Dykes has provided after tax values, based on the taxes and royalties paid by Thompson Creek Mine, a mine located 60 miles from CuMo within the same tax system. These tax values were checked against a pre-feasibility study produced in June 2015 by M3-engineering for the Stibnite project in Idaho for additional confirmation and found to match very well.

## *16.1 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.

### 16.1.1 Equipment Specifications

Equipment specifications were determined for the various production scenarios using information published by InfoMine in CostMine (2009) for a series of typical open pit mine models. The equipment was selected based on total tons moved per day for all categories of mined material, ore, stockpile material, and waste (Table 18-1). These were compared to the equipment selections in the CostMine mine models for similar sized operations. This information was then compared to the equipment selections detailed for similar sized operations to verify that the equipment selections reflected industry standards.

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.

**Table 16-1: Mining Rates for Equipment Specifications**

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

### 16.1.2 Pit Design

The pit designs are conceptual and were provided to Vector by CuMoCo (under previous company name 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 18-2 is a summary of the cutoff grades for each category for each production scenario.

**Table 16-2 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.

### *16.2 Tailings Design (TSF)*

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.

### *16.3 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.

### *16.4 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.

### *16.5 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.

### *16.6 Taxes and Royalties*

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

Note: Author Shaun dykes has updated the section with after tax values to the appropriate areas. Income for tax purposes is defined as Metal revenues minus operating expenditures, royalty, property and severance taxes, reclamation, and closure expense, depreciation and depletion. Depreciation is calculated using the Modified Accelerated Cost Recovery System (MACRS) which is the current depreciation system in the United States. Tax rates consist of: State is 7.4% and federal base rate is 32.4% (35% base *(100%-state rate)). In addition a 1% Idaho Mine License tax (royalty) is included in the calculation to the state of Idaho it is based on taxable income and is deductible from federal income tax.

### *16.7 Capital Cost Estimate*

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

The capital costs for development of the project increase as the design throughput increases. The capital cost for development of the mine (pre-strip cost), is relatively insensitive to the size of the operation and the other capital items; mining fleet, concentrator; tailings storage facilities, roaster and site ancillary buildings do allow some reduction in capital intensity (cost per unit throughput) to be achieved i.e. economies of scale. The ±35% accuracy total project capital cost with a base date of July 2009 for each throughput option are summarized below in Table 18-3 and discussed in detail in sections below.

**Table 16-3: 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** |

### *16.8 Mining Capital Costs*

#### 16.8.1 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.

### 16.8.2 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;
   d) 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;
   e) Return times were calculated at an assumed speed of 15 mph;

f) 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
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 16-4 to Table 16-7 show the initial 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 16-4: 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 16-5: 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 16-6: 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 16-7: 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** |

### 16.8.3 Non-Equipment Capital Costs

Table 18-8 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 Cost Mine (2009) mine models. There has been no estimation of additional sustaining capital for the mine, other than that estimated for equipment replacement.

**Table 16-8: 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 pre-strip 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 18-9 shows the amount of pre-strip material for each production scenario along with the total tons to be moved by the owner and also by the contractor. In addition Table 18-9 shows the cost of the pre-strip operations.

**Table 16-9: 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 16-9, the tonnages moved by the owner increases while the contractor's tonnage stays relatively constant. It also shows overall pre-stripping 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.

### *16.9 Process Plant Capital Costs*

A summary of the estimated capital cost for the processing plant and on-site ancillary facilities is provided in Table 16-10 and Table 16-11 for the roaster and ancillary facilities, which exclude any escalation or foreign currency fluctuations and are current day costs only (3Q 2009). Indirect costs, including project contingency have been provided for in the capital cost estimates. Indirect costs have been estimated based on a factor of the total direct costs established from previous projects.

**Table 16 -10: Summary of Plant Capital Cost Estimate**

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

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

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.

**Table 16-11: 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** |

### 16.9.1 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 are 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.

**c) 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.

### 16.9.2 Contingency

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

### 16.9.3 Owner's Costs

Owner's costs have been excluded from this estimate.

### 16.9.4 Project Fee

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

### 16.9.5 Escalation

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

## *16.10 Tailings Capital Costs*

The TSF capital cost estimate was based on conceptual-level material quantity and cost estimates. The estimates for construction of the TSFs for the four plant throughput options for the LOM are presented in Table 18-12.

The majority of the unit rates was based on experience with similar projects and is to ±35% accuracy (Q3 2009). Costs for some items were assumed for this level of design and should suffice for the required level of accuracy. Other assumptions are noted below including that material shrinkage or bulking was not considered in calculating the site grading earthwork quantities. The cost estimates assume that liquefiable foundation soils will be removed from the valley bottoms within the tailings dam footprints and replaced with rock fill. The presence of unsuitable foundation soils and the soils areal extent and depth will be evaluated in the Feasibility Study by geotechnical site investigations. The cost estimates will be adjusted based on the results of the investigations.

The cost estimates in the Table 18-12 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.

**Table 16-12: TSF Capital Cost Summary LOM**

| Description | Unit | 50 kt/d | | 100 kt/d | | 150 kt/d | | 200 kt/d | |
|---|---|---|---|---|---|---|---|---|---|
| | | 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** |

## *16.11 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 sterilization costs.
- Water compensation.
- Bore field 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.

### *16.12 Operating Cost Estimate*

The total project operating costs for the different throughput options are summarized in Table 18-13. The costs are presented as Life of Mine (LOM) averages per short ton of ore processed.

**Table 16-13: 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 realization costs associated with sale of final products.

### 16.12.1 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 16-14 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 16-15 shows the amount of material moved for each scenario for the LOM. Based on a 40 year mine life with 360 work days per year the total tons moved per day were calculated. Using the average amount of material moved per day for the LOM and using the analysis of the costs for similar large open pit operations, a base case cost per ton moved was calculated for each production scenario without regard to site specific layout or equipment selection.

**Table 16-14: 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 16-15: 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 16-16 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 16-16: 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.

### 16.12.2 Mining Operating Cost Comparison

Overall average annual mining operating costs are approximately US$80M (Table 18-16), 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 re-handling 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.

### 16.12.3 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 16-17. The costs have been divided into the key cost centers. All figures have been based on the study estimates applying as of the third quarter 2009 (calendar year).

**Table 16 -17: 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) Labor**

Site labor 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.

### *16.13 Economic Analysis*

Variability analyses were conducted using different metal prices, and varying capital and operating costs to determine the effect of these variables on the project economics. These analyses were conducted on the basis of the assumptions as listed below in Table 16-18.

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 16-18: Base Assumptions for Economic Analysis**

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

[5] Average grades vary with throughput option

| 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 |

Note: Original assumptions included no interest, taxation, depreciation and amortization. Mr. Dykes, the qualified person, has used the taxation of the Thompson Creek Mine located near to CuMo to produce a second set of tables and figures that are considered after tax numbers. The tax information is applied to the detailed economic analyses. The results take into account all levels of taxation but do not include tax reduction grants and other federal state and county incentives that are provide to new producing mines by state and federal agencies and thus the results are considered conservative. As mentioned previously MACRS is used for depreciation schedules

### 16.13.1 Economic Analysis (Base Case)

The original base case economic analysis, is based on the estimates of capital and operating costs and assumptions as listed in Table 16-19 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. Two sets of results are presented: values are calculated based on Earnings Before Interest Tax Depreciation and Amortization (EBITD&A) and a second set based on after tax.
It is the author’s opinion that the original base case economic analysis is out of date, due to lower metal prices. All other values and numbers are still valid. It was therefore decided to present an

updated base case using the same metal prices that are included in the resource calculation, namely $15 Molybdenum metal ($10 Molybdenum oxide) and $3 copper. These prices fall within the range of prices used in the original PEA report (Table 16-20).

**Table 16-19a: Updated Base Case Economic Analysis (pre-tax)**

| Economic parameters (EBITD&A) | Throughput Option | | | |
|---|---|---|---|---|
| | 50 kt/d | 100 kt/d | 150 kt/d | 200 kt/d |
| NPV (US$B@5%) | 2 | 6 | 9 | 12 |
| IRR% | 13 | 20 | 25 | 27 |
| Simple payback Period (years) | 7.5 | 4.6 | 3.5 | 3.0 |
| Discounted Payback period (years@5%) | 8.8 | 5.2 | 3.9 | 3.3 |
| Total Operating costs per lb of molybdenum Oxide equivalent | 5.5 | 4.3 | 3.9 | 3.8 |

**Table 16-19b: Updated Base Case Economic Analysis (after-tax)**

| Economic parameters After Tax | Throughput Option | | | |
|---|---|---|---|---|
| | 50 kt/d | 100 kt/d | 150 kt/d | 200 kt/d |
| NPV (US$B@5%) | 1.8 | 4.5 | 7.2 | 9.4 |
| IRR% | 11.5 | 17.3 | 21.3 | 23.0 |
| Simple payback Period (years) | 7.7 | 4.9 | 3.8 | 3.4 |
| Discounted Payback period (years@5%) | 9.0 | 5.6 | 4.2 | 3.6 |
| Total Operating costs per lb of molybdenum Oxide equivalent | 5.5 | 4.3 | 3.9 | 3.8 |

### 16.13.2 Sensitivity analysis (Metal Prices)

A basic sensitivity analysis was conducted on the economic effects of various metal price scenarios. The following Table 16-20 shows a matrix of the various metal prices used in the scenarios analyzed. A further sensitivity analysis was conducted on the basis of cyclical metal prices, with average prices similar to the medium prices shown in Table 18-20, but assuming that the operation commences production on the commencement of the upturn in metal prices.

**Table 16-20: 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 |

**Table 16-21: 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 16-22.

**Table 16-22a: Pre-tax IRR Sensitivity to Metal Pricing**

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

**Table 16-22b: After Tax IRR Sensitivity to Metal Pricing**

| Metal Price Scenario | %IRR Sensitivity (After Tax basis) | | | |
|---|---|---|---|---|
| | 50 kt/d | 100 | 150 | 200 |
| High | 30.8 | 42.6 | 49.4 | 53.8 |
| Cyclical | 22.6 | 33.2 | 41.1 | 44.9 |
| Medium | 16.7 | 25.1 | 30.8 | 33.2 |
| Low | 2.7 | 8.0 | 10.6 | 13.2 |

A matrix of the Project NPV5 for the four throughput options for each of the metal pricing scenarios is shown below in Table 16-23.

**Table 16-23a: Pre-Tax NPV5 Sensitivity to Metal Pricing**

| Metal Price Scenario | NPV5 Sensitivity us $B (EBITD&A | | | |
|---|---|---|---|---|
| | 50 kt/d | 100 | 150 | 200 |
| High | 10 | 22 | 35 | 45 |
| Cyclical | 5.2 | 12 | 21 | 27 |
| Medium | 3.8 | 9.7 | 16 | 21 |
| Low | -0.5 | 1.1 | 2.9 | 4.4 |

**Table 16-23b: After-Tax NPV5 Sensitivity to Metal Pricing**

| Metal Price Scenario | NPV5 Sensitivity us $B (After Tax basis) | | | |
|---|---|---|---|---|
| | 50 kt/d | 100 | 150 | 200 |
| High | 7.9 | 16.8 | 25.9 | 32.5 |
| Cyclical | 4.7 | 10.6 | 18.4 | 23.4 |
| Medium | 3.4 | 8.6 | 14.1 | 18.4 |
| Low | -0.5 | 1.0 | 2.6 | 3.9 |

### 16.13.3 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 16-24:

**Table 16-24: 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%.. In this report the new base case was used instead of the original Ausenco base to bring the values to current. 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 16-1a and b and Figure 16-2a and b for the 50 kt/d (short ton) throughput option.

**Figure 16-1a: Pre-Tax 50 kt/d Throughput IRR Sensitivity**

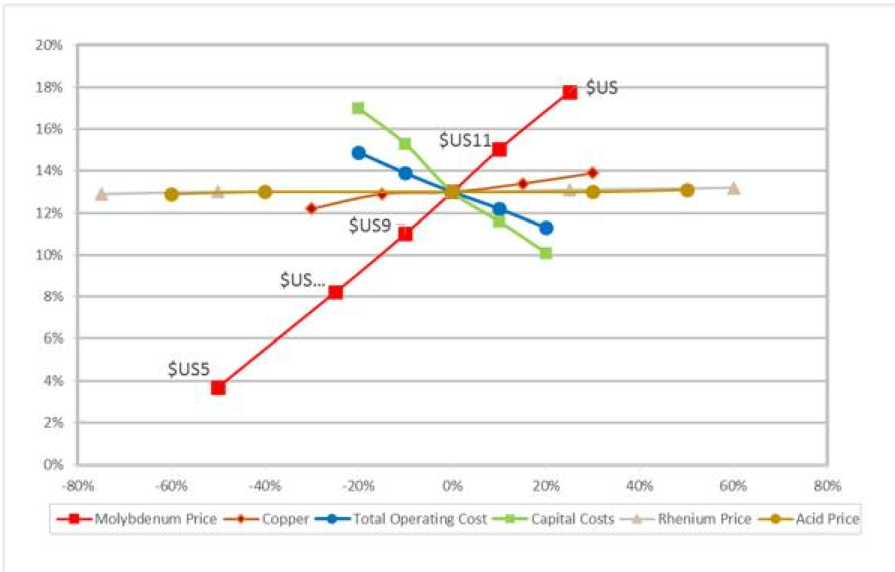



**Figure 16-1b: After-Tax 50 kt/d Throughput IRR Sensitivity**

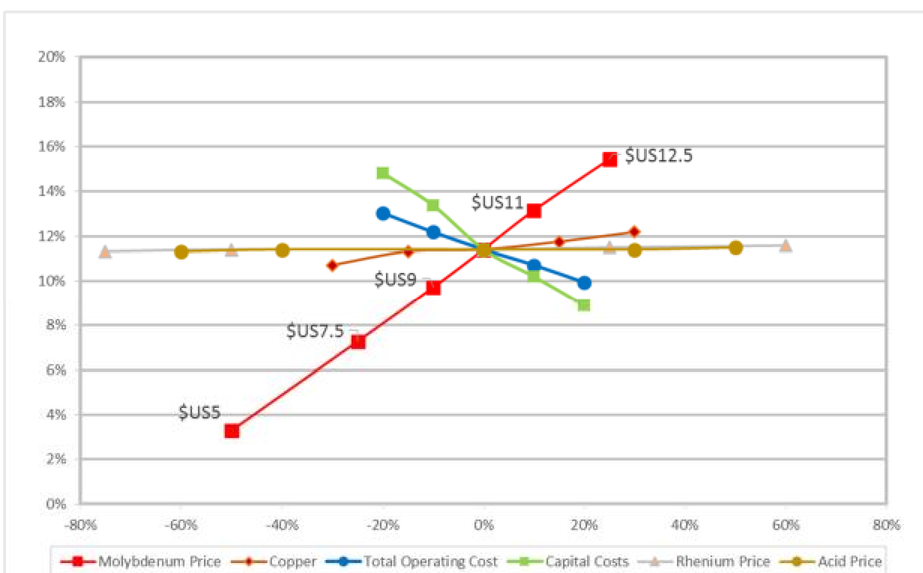


**Figure 16-2a: Pre-Tax 50 kt/d Throughput NPV Sensitivity**

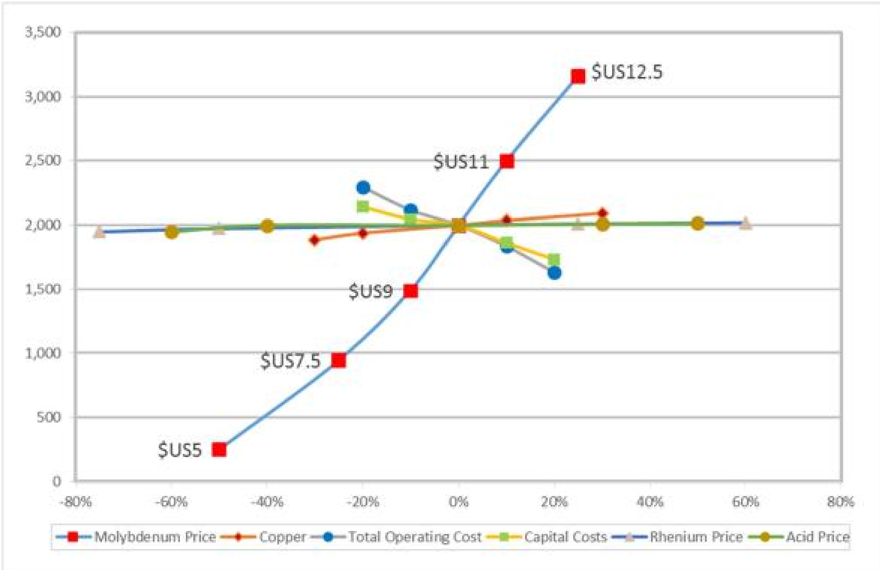
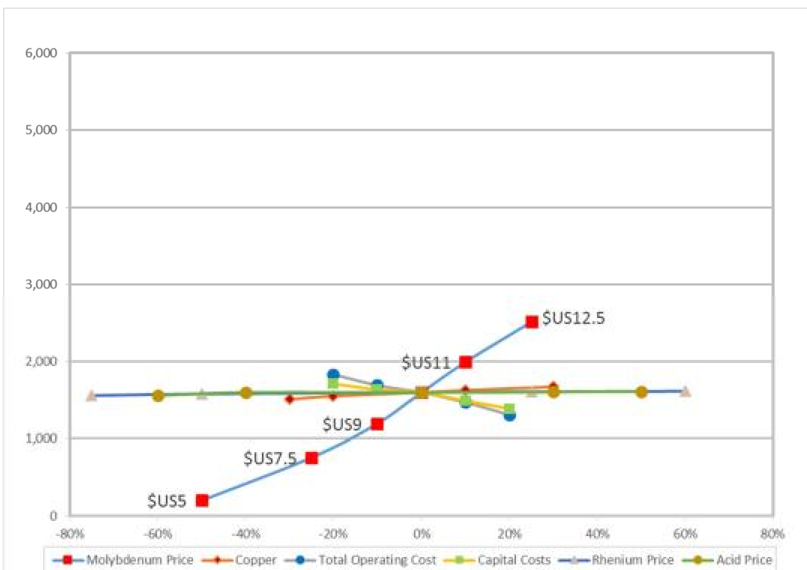



**Figure 16-2b: After-Tax 50 kt/d Throughput NPV Sensitivity**




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 16-3 through to Figure 16-8). The relative sensitivities for variations in the parameters tested are very similar for all throughput options.

**Figure 16-3a: Pre-tax 100 kt/d Throughput IRR Sensitivity**

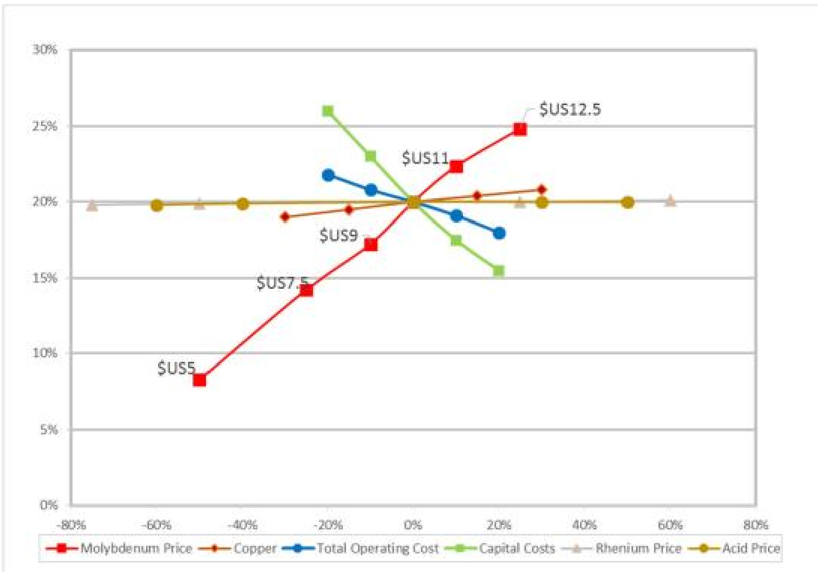



**Figure 16-3b: After-tax 100 kt/d Throughput IRR Sensitivity**

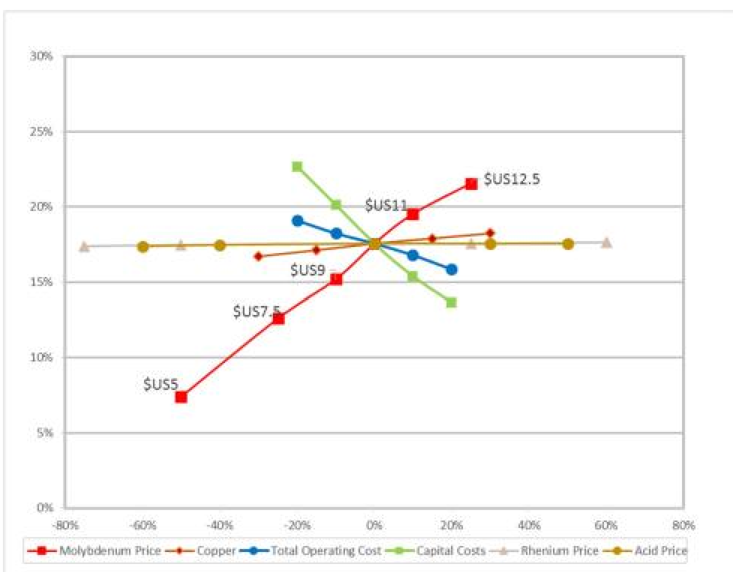


**Figure 16-4a: Pre-Tax 100 kt/d Throughput NPV Sensitivity**


**Figure 16-4b: After-Tax 100 kt/d Throughput NPV Sensitivity**

**Figure 16-5a: Pre-Tax 150 kt/d Throughput IRR Sensitivity**

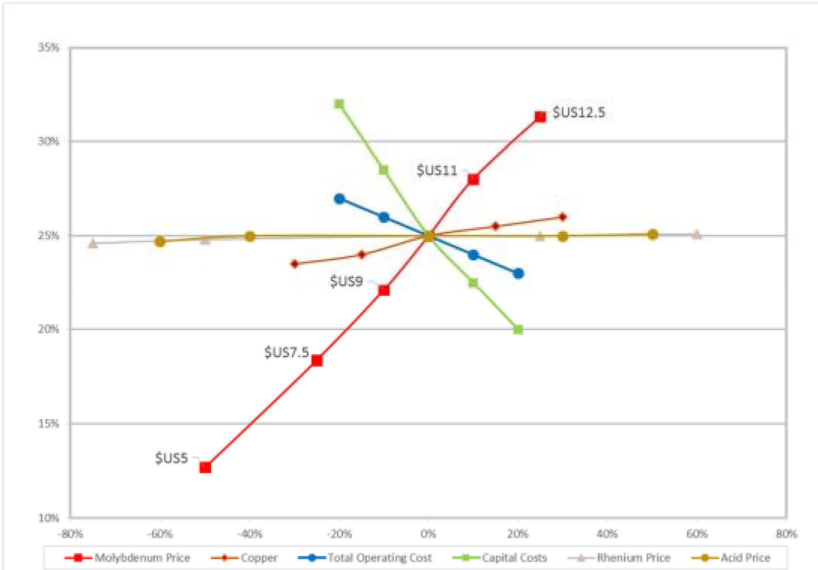



**Figure 16-5b: After-Tax 150 kt/d Throughput IRR Sensitivity**

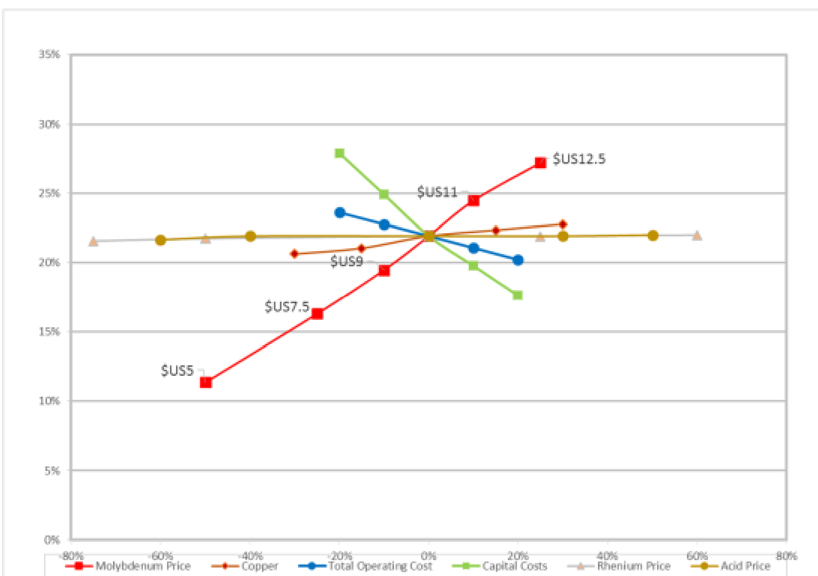


**Figure 16-6a: Pre-Tax 150 kt/d Throughput NPV Sensitivity**

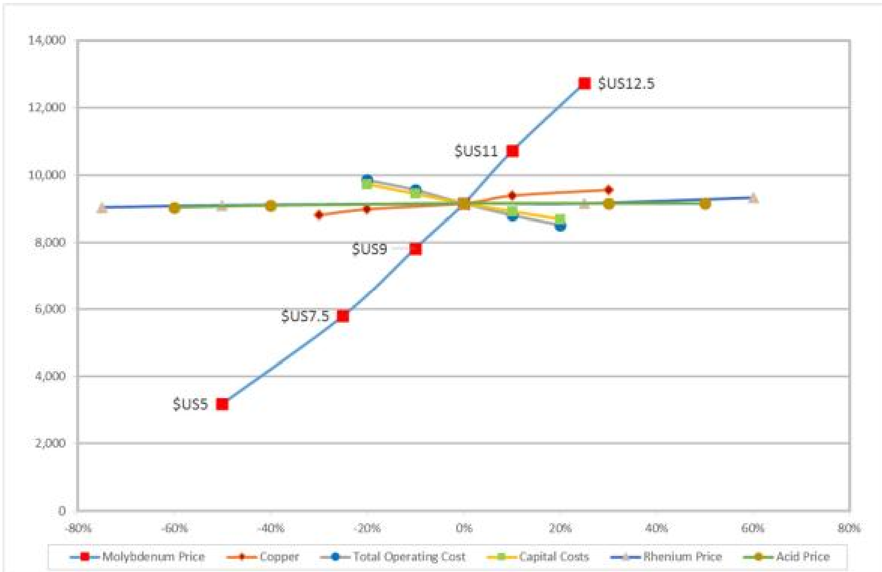



**Figure 16-6b: After Tax 150 kt/d Throughput NPV Sensitivity**

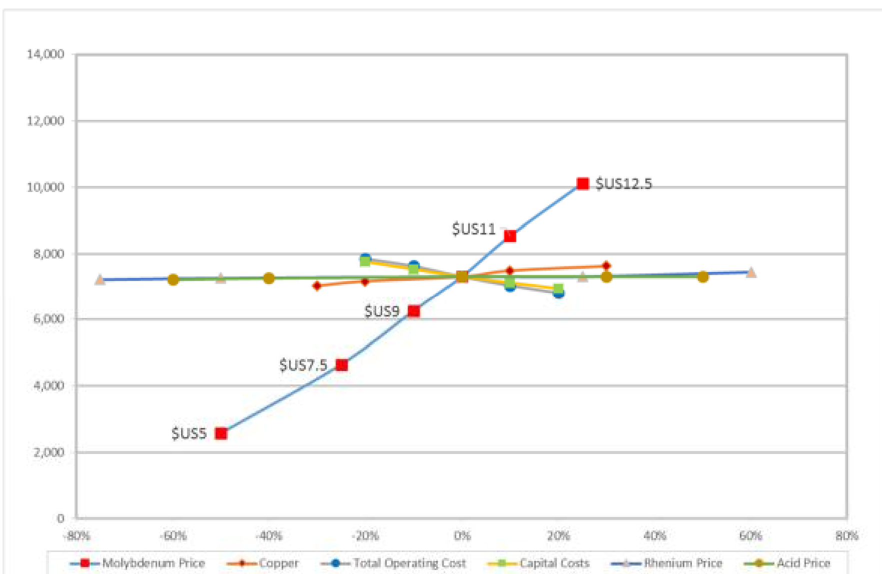


**Figure 16-7a: Pre-Tax 200 kt/d Throughput IRR Sensitivity**

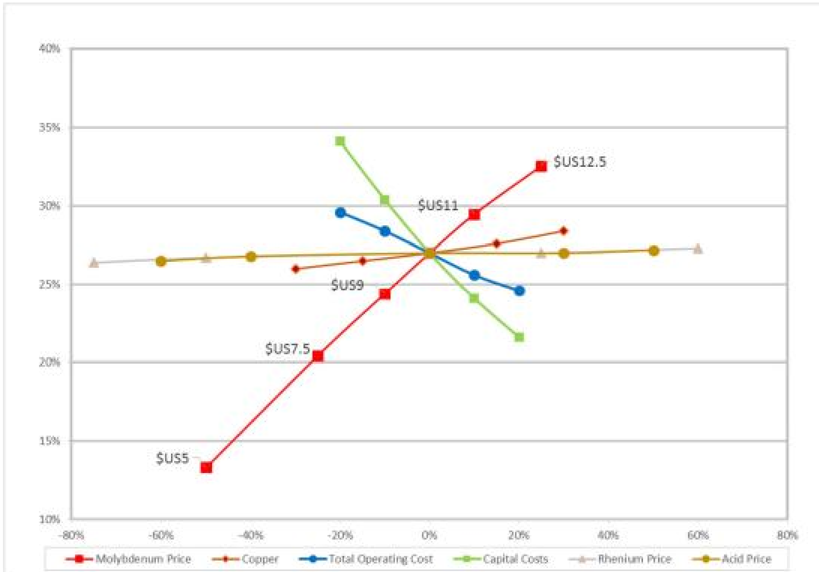



**Figure 16-7b: After Tax 200 kt/d Throughput IRR Sensitivity**

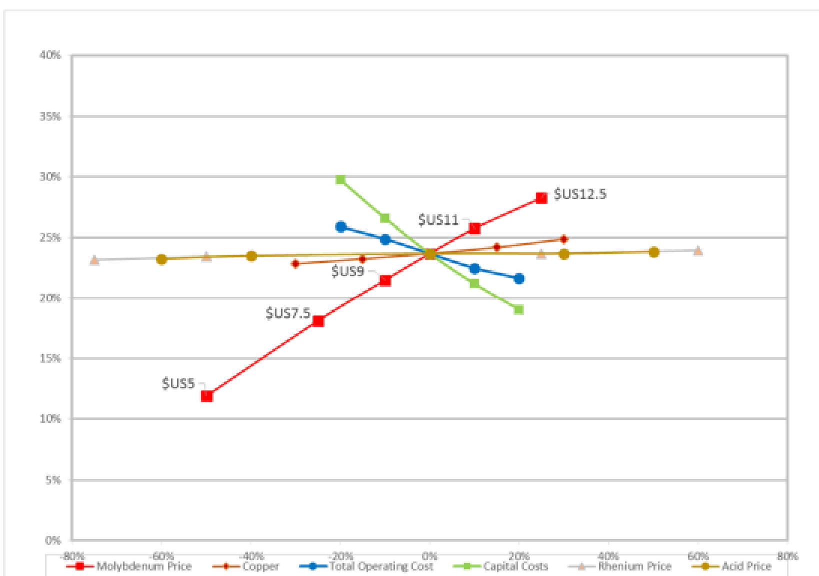


**Figure 16-8a: Pre-Tax 200 kt/d Throughput NPV Sensitivity**

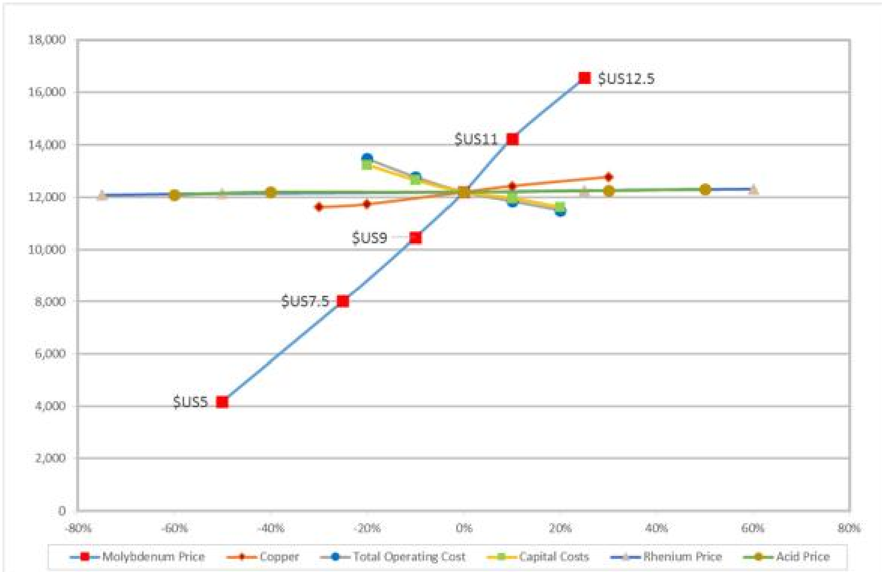
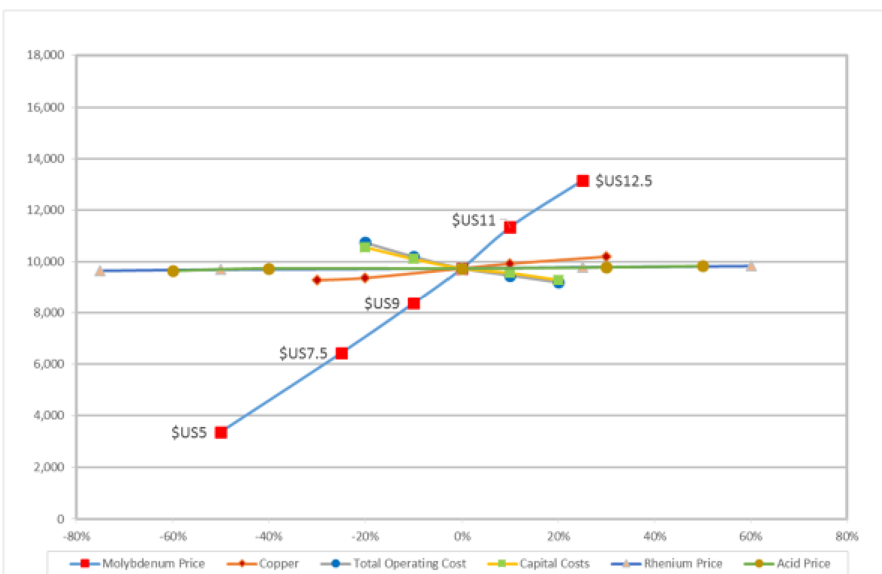



**Figure 16-8b: After Tax 200 kt/d Throughput NPV Sensitivity**



# 17.0 Interpretation and Conclusions

This resource represents an update of the CUMO resource estimate completed after the 2012 drill program, utilizing the additional 9 diamond drill holes totalling 22,968 ft. For this estimate, variography was conducted after the major post mineral fault blocks were rotated back into their pre fault positions based on marker horizons. This allowed for evaluation of data on either side of major faults and resulted in the determination of more realistic anisotropic grade continuity ranges. The additional data and the longer continuity ranges have allowed for a significant portion of this resource to be classified as Measured and Indicated.

# 18.0 Recommendations

The following recommendations are based on the review of the work done to date. The Company believes and authors concur, the next stage in the development of the project is to produce a bankable feasibility study, which would require environmental baseline work and associated legal and public relations costs.
The work is for a three year program to reach the next decision point on the property development that would be decision to proceed or not proceed to production.

## *18.1 Drilling*

Exploration work consisting mainly of drilling is required to reach feasibility. It is estimated that a total of 33 additional holes for 71,000 feet plus an additional 5 geotechnical holes for 12,000 feet on the deposit plus additional 74,800 feet allocated to condemnation drilling of waste dump, mill site and tailings pond areas, making a total of 157,8000 feet of drilling budgeted. This drilling is broken into the following categories.

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

The shortest time to complete this work will be two seasons using 4 drill rigs each season.

## *18.2 Engineering*

### 18.2.1 Feasibility Study

Given the excellent results from the Preliminary Economic Analysis (Ausenco, 2009) it is recommended that the project go directly to a feasibility study to obtain more detailed information regarding the costs and economics of the CuMo Project. The study will require additional drilling and metallurgical work to supply the information required. At the same time work can begin on the environmental baseline studies required to obtain permitting and that form part of the feasibility document.

### 18.2.2 Site Selection and Preliminary Mine Design

Several sites need to be examined and selected in order to prepare an environmental study plan. These include mill, tailings and waste impoundment sites, potential low-impact hydroelectric generating sites, housing and social structure sites, and finally mine and road access sites. Each selection should be narrowed to one or two choices.

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

### *18.3 Metallurgical work*

Metallurgical aspects to be studied were highlighted in the recent preliminary metallurgical analysis, some of which require larger samples to finalize the detailed flow sheet and determine how many cleaning stages will be required. One important part of the analysis is a grinding versus recoverability study, as in the previous study only two grinding sizes were studied: coarse and fine. The fine grind proved to be more profitable despite the increase in costs. Further study with multiple grinding size options is required to determine an optimum grinding system.
Work will consist of collecting and analyzing a large 2+ tonne bulk sample to determine the optimum flow sheet for the deposit; and a variability study to analyze variations within the deposit. A total of 100 to 150 twenty (20) kilogram samples will be used for the variability study.

### *18.4 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 communicate with each other in a timely and cooperative manner.

### *18.5 Public Relations*

CuMoCo has initiated a community relations program to establish the company as a good corporate citizen and disseminate positive information about the potential of this project. This includes discussions with local communities to minimize future issues related to on-going exploration and development. This is a necessary part of the process as it is required in order to obtain the permits necessary to perform the work.

### *18.6 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 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 construction, it is 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 (Table 18-1). 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 to 7 years.

The budget to achieve feasibility in 3 years is summarized in the tables on the following pages.

> The actual engineering portion of the feasibility study is estimated to require about $35 million, while the environmental and permitting costs are estimated at $55 million. A contingency of $10 million is also included in the budget.

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.

**Table 18-1 Three Year Project Cost Estimates**

| | | | |
|---|---|---|---|
| Diamond Drilling | | | |
| Delineation, infill, metallurgy | 48,097 meters (157,800 feet) | 100 | $15,780,000 |
| | | | |
| Road construction | 2 km | $50,000/km | $100,000 |
| | | | |
| Sample Preparation and Analysis | 8,800 | 60 | $528,000 |
| Metallurgical Testing | Sample collection etc | | $125,000 |
| | Batch round of testing | | $1,000,000 |
| | Variability | | $1,200,000 |
| Land Acquisition and staking costs | | | $8,000,000 |
| Environmental Studies | Environmental Assessment | | $712,500 |
| | baseline studies startup | | $12,500,000 |
| | plan of operations | | $800,000 |
| | Environmental Impact Statement | | $23,500,000 |
| | Permitting | | $3,000,000 |
| Engineering studies scoping | mill site, tailings site analysis | | $550,000 |
| | Intergoverment Task Force creation | | $500,000 |
| | Plan of operations | | $1,200,000 |
| | feasibility | | $5,500,000 |
| **Yearly Charges** | | | |
| Mob-Demobilize | | | $427,000 |
| Road Maintenance\pad construction | | | $325,000 |
| Supervision and Project Management | supervision | $7,500/mth | $225,000 |
| | corporate Manager | $15,000/mth | $360,000 |
| | Project Manager | $8,000/mth | $240,000 |
| | Assistant Geologist(2) | $5,000/mth | $364,000 |
| | Technicians (12) | $15/hr | $1,174,000 |
| Vehicles | 5 vehicles | $1000/mth | $150,000 |
| Accommodation and food | 30 men | | $760,000 |
| Travel | | $1000/mth | $42,000 |
| Project office and Warehouse | | | $1,225,000 |
| Land Filing Fees | BLM: $140/claim/year; County: $8.50 | | $342,500 |
| Consultants | (Mining ,Metallurgical and Marketing) | | $575,000 |
| Resource Modeling | | | $1,650,000 |
| Public Relations and Project | Public relations and legal etc | | $2,550,000 |
| Presentation | Liaison county and state officials | | $1,250,000 |
| **yearly Subtotal** | | | **$86,655,000** |
| Contingency | | | $13,345,000 |
| **Total** | | | **$100,000,000** |

## 19.0 References

Anderson, A.L., 1947, Geology and Ore Deposits of the Boise Basin, Idaho, USGS Bull 944C.

Armstrong, R.L., Taubeneck, W.H., Hales, P.O., 1977. Rb–Sr and K–Ar geochronometry of Mesozoic granitic rocks and their Sr isotopic composition, Oregon, Washington, and Idaho. Geological Society of America Bulletin 88, 397–411.

Baker, D.J., 1985, Geology of the CUMO Molybdenum-Copper System, Boise County, Idaho, Geological Society of America, Abstracts with Programs 1985, Rocky Mountain Section, No. 70043, p 205.

Baker, D.J., 1983, The CUMO Molybdenite System, Boise, Idaho, A Comprehensive Summary", Climax Molybdenum Company, April 1983, unpublished.

Bennett, E.H., 1986, Relationship of the trans-Challis fault system in central Idaho to Eocene and Basin and Range extensions, Geology, v. 14, p. 481-484.

Carten, R.B., White, W.H. and Stein, H.J., 1993, High-Grade Granite-Related Molybdenite Systems: Classification and Origin, in Kirkham, R.V., Sinclair, W.D., Thorpe, R.I. and Duke, J.M., eds., Mineral Deposit Modeling; Geological Association of Canada, Special Paper 40, p. 521-544.

Digital Atlas of Idaho, website: http://imnh.isu.edu/digitalatlas/index.htm

Ausenco Canada Inc., 2009, CUMO Property Preliminary Economic Assessment, Boise County, Idaho, for Kobex Resources LTD., submitted in Sedar.

Giroux, G. Cavey, G. and Gunning, D., 2005, Summary Report on the CUMO Molybdenum Property, Boise County, Idaho, for Kobex Resources LTD., unpublished.

Hildenbrand, T.G., Berger, B. and Jachens, R.C., 2000, Regional Crustal Structures and Their Relationship to the Distribution of Ore Deposits in the Western United States, Based on Magnetic and Gravity Data, Econ. Geol. v.95, p. 1583-1603.

Holmgren, J. and Giroux, G. 2008. Summary Report on the CUMO Property, Boise County, Idaho. NI 43-101 Technical Report for Mosquito Consolidated Gold Mines Ltd., posted at www.sedar.com
Johnson, B. R. and Raines G. L., 1996, Digital representation of the Idaho state geologic map: a contribution to the Interior Columbia River Basin Ecosystem Management Project; USGS Open File Report 95-690.

Killsgaard, T.H, Stanford, L.R. and Lewis, R.S., 2006, Geologic Map of the Deadwood River 30 x 60 Minute Quadrangle, Idaho; Idaho Geological Survey, Geologic Map 45.

Killsgaard, T.H, Stanford, L.R. and Lewis, R.S., 2001, Geologic Map of the Idaho City 30 x 60 Minute Quadrangle, Idaho; Idaho Geological Survey, Geologic Map 29.

Killsgaard, T.H, Fisher, F.S. and Bennet, E.H., 1989, Gold-Silver Deposits Associated with the Trans-Challis Fault System, Idaho; USGS Bull 1857-B, p. B22-B44

Killsgaard, T.H, and Lewis, R.S., 1985, Plutonic Rocks of Cretaceous Age and Faults in the Atlanta Lobe of the Idaho Batholith, Challis Quadrangle; USGS Bull 1658 A-S, p. 29-42.

Klein, T.L., 2004, Mineral deposit data for epigenetic base-and precious-metal and uranium-thorium deposits in south-central and southwestern Montana and southern and central Idaho, USGS Open File Report 2004-1005.

Link, P.K., 2002, Geological Map of Boise County, Idaho; in Digital Atlas of Idaho, website: http://imnh.isu.edu/digitalatlas/counties/geomaps/geomap.htm

Lund, K., Klein, T.L, O'Neill and J.M., Sims, P.K., 2005, Influence of structure and composition of basement on mineral deposits across Montana and Idaho; EarthScope in the Northern Rockies Workshop, Program, Session III; website: http://serc.carleton.edu/earthscoperockies/abstracts3.html.
Mutchler, F.E., Ludington, S. and Bookstrom, A.A., 1999, Giant porphyry-related metal camps of the world — a database, USGS Open File Report 99-556.

O'Neill, J.M., and Lopez, D.A., 1985, Character and regional significance of the Great Falls tectonic zone, east-central Idaho and west-central Montana: American Association of Petroleum Geologists Bulletin, v. 69, p. 437–477.

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

Rostad, O.H., 1978, K-Ar dates for mineralization in the White Cloud-Cannivan porphyry molybdenum belt of Idaho-Montana: A discussion: Econ. Geol. v. 73, p. 1366–1367.

Sims, P.K. Lund, K. and Anderson, E., 2005, Precambrian Crystalline Basement Map of Idaho – An Interpretation of Aeromagnetic Anomalies; USGS, Scientific Investigations Map 2884.

Singer, D.A, Berger, V.I., and Moring, B.C., 2005, Porphyry Copper Deposits of the World: Database, Map, and Grade and Tonnage Models, USGS Open File Report 2005-1060.

Spanski, G.T., 2004, Inventory of Significant Mineral Deposit Occurrences in the Headwaters Project Area in Idaho, Western Montana, and Extreme Eastern Oregon and Washington, USGS Open File Report 2004-1038.

SGS, 2009, An Investigation into the Recovery of Molybdenum, Copper and Silver from CUMO samples prepared for Mosquito Consolidated Gold Mines Ltd Project 50004-001

# 20.0 Signature Page

**CERTIFICATE of QUALIFICATION**

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 of 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 and Geoscientists of the Province of British Columbia.
5) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.
6) This report titled "**Updated Summary Report on the CuMo Property, Boise County, Idaho**" dated November 10, 2015, is based on a study of the data and literature available on the CUMO Property. I am responsible for Sections 12 and 14 on data verification and resource estimations completed in Vancouver during 2015. I have visited the property on June 1-3, 2015.
7) I have previously completed a statistical review of this property in 2005 and a resource estimation in 2008.
8) As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
9) I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
10) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this day of November 5, 2015

Gary Giroux

GIROUX CONSULTANTS LTD.
G. H. Giroux, P.Eng. MASc.

**CERTIFICATE of QUALIFICATION**

I, Shaun M Dykes, resident of New Westminster, Province of British Columbia, hereby certify as follows:

1) I am a consulting geologist with an office located at 514 East Columbia St., New Westminster, British Columbia.
2) I graduated with a degree of Bachelor of Science (engineering) in geology from Queen's University in 1976 and with a Master of Science (engineering) in geology from Queen's University in 1979 and have practiced my profession for 7 years on a seasonal and 36 years on a continuous basis and I am a "Qualified Person" under the terms and policies of National Instrument 43-101.
3) I have practiced my profession continuously since 1979. I have over 35 years' experience in geology and engineering having worked on a wide variety of gold, copper, molybdenum, silver, lead and zinc deposit throughout the world. I was major team member in the exploration construction and production at the Premier Gold Mine, British Columbia supplying geological expertise, resource and reserve calculations, and production management. I have evaluated and produced economic analyses and opinions for numerous deposits for a variety of companies including Voisey Bay, Red Chris, Tulsequah Chief, Petaqullia, Barun Holbinsky(Russia), Cariboo Quartz Gold Mine, Mosquito Gold Mine, Blackpoint Gold, and several other gold mines and Prospects.
4) I am registered as Professional Geoscientist ( No. 123245) by the Association of Professional Engineers and Geoscientists 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 and affiliation with a professional association, I meet the requirements of a qualified person as defined in National Instrument 43-101.
6) Although a qualified person, I am not independent of American CuMo Mining Corp. as I am a director, President and Chief Executive officer of the company.
7) This report titled "**Updated Summary Report on the CuMo Property, Boise County, Idaho**" dated October 31, 2015, is based on a study of the data and literature available on the CUMO Property. I am responsible for section 16 of the report and also reviewed, approved and taken responsibility for the sections produced by James H Place, which comprise all sections not including 12, 14 and 16. I have visited the property on numerous occasions with the last visit, June 1-3, 2015.
8) I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclosure, which makes the technical report misleading.
9) The author has read National Instrument 43-101, "Standards Of Disclosure For Mineral Projects " and Form 43-101F1, and this report has been prepared in compliance with 43-101 and Form 43-101F.

Dated at New Westminster, Province of British Columbia, this 5th day of November, 2015

Signed

*Shaun M. Dykes*

Shaun M Dykes, M.Sc. (Eng), P. Geo

# 21.0 Appendix 1: Claims List

**Cumo Claim List 2014**

| Item | Claim Name/Number | BLM Serial No. | County Instrument Number | Loc Dt |
|---|---|---|---|---|
| 1 | CUMO #1 | 188031 | 201255 | Mar-05 |
| 2 | CUMO #2 | 188032 | 201256 | Mar-05 |
| 3 | CUMO #3 | 188033 | 201257 | Mar-05 |
| 4 | CUMO #4 | 188034 | 201258 | Mar-05 |
| 5 | CUMO #5 | 188035 | 201259 | Mar-05 |
| 6 | CUMO #6 | 188036 | 201260 | Mar-05 |
| 7 | CUMO #7 | 188037 | 201261 | Mar-05 |
| 8 | CUMO #8 | 188038 | 201262 | Mar-05 |
| 9 | NEW CUMO #9 | 187938 | 199561 | Nov-04 |
| 10 | NEW CUMO #10 | 187939 | 199562 | Nov-04 |
| 11 | NEW CUMO #11 | 187940 | 199563 | Nov-04 |
| 12 | NEW CUMO #12 | 187941 | 199564 | Nov-04 |
| 13 | NEW CUMO #13 | 187942 | 199565 | Oct-04 |
| 14 | NEW CUMO #14 | 187943 | 199566 | Oct-04 |
| 15 | NEW CUMO #15 | 187944 | 199567 | Oct-04 |
| 16 | NEW CUMO #16 | 187945 | 199568 | Oct-04 |
| 17 | NEW CUMO #17 | 187946 | 199569 | Oct-04 |
| 18 | NEW CUMO #18 | 187947 | 199570 | Oct-04 |
| 19 | NEW CUMO #19 | 187948 | 199571 | Oct-04 |
| 20 | NEW CUMO #20 | 187949 | 199572 | Oct-04 |
| 21 | NEW CUMO #21 | 187950 | 199573 | Oct-04 |
| 22 | NEW CUMO #22 | 187951 | 199574 | Nov-04 |
| 23 | NEW CUMO #23 | 187952 | 199774 | Nov-04 |
| 24 | NEW CUMO #24 | 187953 | 199775 | Nov-04 |
| 25 | NEW CUMO #25 | 187954 | 199575 | Nov-04 |
| 26 | NEW CUMO #26 | 187955 | 199576 | Nov-04 |
| 27 | NEW CUMO #27 | 187956 | 199577 | Nov-04 |
| 28 | NEW CUMO #28 | 187957 | 199578 | Nov-04 |
| 29 | NEW CUMO #29 | 187958 | 199579 | Nov-04 |
| 30 | NEW CUMO #30 | 187959 | 199580 | Nov-04 |
| 31 | NEW CUMO #31 | 187960 | 199581 | Nov-04 |
| 32 | NEW CUMO #32 | 187961 | 199582 | Nov-04 |
| 33 | NEW CUMO #33 | 187962 | 199583 | Nov-04 |
| 34 | NEW CUMO #34 | 187963 | 199584 | Nov-04 |
| 35 | NEW CUMO #35 | 187964 | 199585 | Nov-04 |
| 36 | NEW CUMO #36 | 187965 | 199586 | Nov-04 |
| 37 | NEW CUMO #37 | 187966 | 199587 | Nov-04 |
| 38 | NEW CUMO #38 | 187967 | 199588 | Nov-04 |
| 39 | NEW CUMO #39 | 187968 | 199589 | Nov-04 |
| 40 | NEW CUMO #40 | 187969 | 199590 | Nov-04 |
| 41 | NEW CUMO #41 | 187970 | 199591 | Nov-04 |
| 42 | NEW CUMO #42 | 187971 | 199592 | Nov-04 |
| 43 | NEW CUMO #43 | 187972 | 199593 | Nov-04 |
| 44 | NEW CUMO #44 | 187973 | 199594 | Nov-04 |
| 45 | NEW CUMO #45 | 187974 | 199595 | Nov-04 |

## Cumo Claim List 2014 - Page 2

| Item | Claim Name/Number | BLM Serial No. | County Instrument Number | Loc Dt |
|---|---|---|---|---|
| 46 | NEW CUMO #46 | 187975 | 199596 | Nov-04 |
| 47 | NEW CUMO #47 | 187976 | 199597 | Nov-04 |
| 48 | NEW CUMO #48 | 187977 | 199598 | Nov-04 |
| 49 | NEW CUMO #49 | 187978 | 199599 | Nov-04 |
| 50 | NEW CUMO #50 | 187979 | 199600 | Nov-04 |
| 51 | NEW CUMO #51 | 187980 | 199601 | Nov-04 |
| 52 | NEW CUMO #52 | 187981 | 199602 | Nov-04 |
| 53 | NEW CUMO #53 | 187982 | 199603 | Nov-04 |
| 54 | NEW CUMO #54 | 187983 | 199604 | Nov-04 |
| 55 | NEW CUMO #55 | 187984 | 199605 | Nov-04 |
| 56 | NEW CUMO #56 | 187985 | 199606 | Nov-04 |
| 57 | NEW CUMO #57 | 187986 | 199607 | Nov-04 |
| 58 | NEW CUMO #58 | 187987 | 199608 | Nov-04 |
| 59 | NEW CUMO #59 | 187988 | 199609 | Nov-04 |
| 60 | NEW CUMO #60 | 187989 | 199776 | Nov-04 |
| 61 | NEW CUMO #61 | 187990 | 199777 | Nov-04 |
| 62 | CUMO #62 | 188205 | 202147 | May-05 |
| 63 | CUMO #63 | 188206 | 202148 | May-05 |
| 64 | CUMO #64 | 188207 | 202149 | May-05 |
| 65 | CUMO #65 FRACT. | 188208 | 202150 | May-05 |
| 66 | CUMO #66 | 188209 | 202151 | May-05 |
| 67 | CUMO #67 FRACTION | 188210 | 202152 | May-05 |
| 68 | CUMO #68 FRACT. | 188211 | 202153 | May-05 |
| 69 | CUMO #69 FR. | 188212 | 202154 | May-05 |
| 70 | CUMO #70 FRACT. | 188213 | 202155 | May-05 |
| 71 | CUMO #71 | 188214 | 202156 | May-05 |
| 72 | CUMO #72 | 188215 | 202157 | May-05 |
| 73 | CUMO #73 | 188216 | 202158 | May-05 |
| 74 | CUMO #74 | 188217 | 202159 | May-05 |
| 75 | CUMO #75 | 188218 | 202160 | May-05 |
| 76 | CUMO #76 | 188219 | 202161 | May-05 |
| 77 | CUMO #77 | 188220 | 202162 | May-05 |
| 78 | CUMO #78 | 188221 | 202163 | May-05 |
| 79 | CUMO #79 | 188222 | 202164 | May-05 |
| 80 | CUMO #80 | 188223 | 202165 | May-05 |
| 81 | CUMO #81 | 188224 | 202166 | May-05 |
| 82 | CUMO #82 | 188225 | 202167 | May-05 |
| 83 | CUMO #83 | 188226 | 202168 | May-05 |
| 84 | CUMO #84 | 188227 | 202169 | May-05 |
| 85 | CUMO #85 | 188228 | 202271 | May-05 |
| 86 | CUMO #86 | 188229 | 202272 | May-05 |
| 87 | CUMO #87 | 188230 | 202273 | May-05 |
| 88 | CUMO #88 | 188231 | 202274 | May-05 |
| 89 | CUMO #89 | 188232 | 202275 | May-05 |
| 90 | CUMO #90 | 188233 | 202276 | May-05 |

## Cumo Claim List 2014 - Page 3

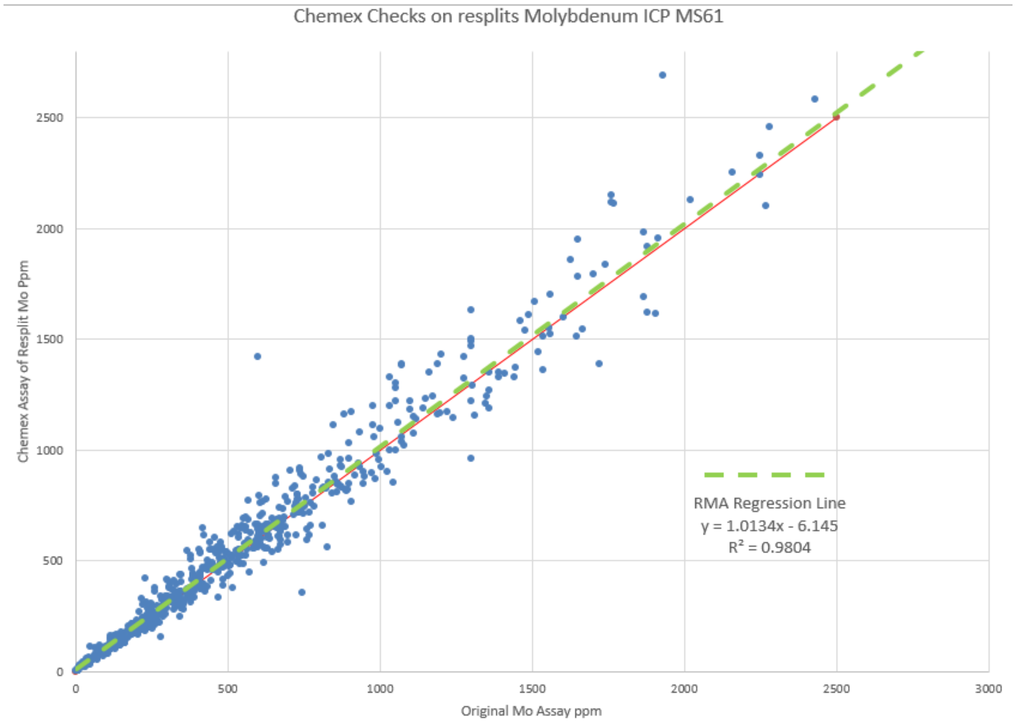
| Item | Claim Name/Number | BLM Serial No. | County Instrument Number | Loc Dt |
|---|---|---|---|---|
| 91 | CUMO #91 | 188234 | 202277 | May-05 |
| 92 | CUMO #92 | 188235 | 202278 | May-05 |
| 93 | CUMO #93 | 188236 | 202279 | May-05 |
| 94 | CUMO #94 | 188237 | 202281 | May-05 |
| 95 | CUMO #95 | 188238 | 202282 | May-05 |
| 96 | CUMO #98 | 188239 | 202366 | May-05 |
| 97 | CUMO #99 | 188240 | 202367 | May-05 |
| 98 | CUMO #100 | 188241 | 202368 | May-05 |
| 99 | CUMO #101 | 188242 | 202369 | May-05 |
| 100 | CUMO #107 FRACTION | 188244 | 202371 | May-05 |
| 101 | CUMO #109 | 188246 | 202373 | May-05 |
| 102 | CUMO #121 | 188258 | 202283 | May-05 |
| 103 | CUMO #122 | 188259 | 202284 | May-05 |
| 104 | CUMO #123 | 188260 | 202285 | May-05 |
| 105 | CUMO #124 | 188283 | 202286 | May-05 |
| 106 | CUMO #125 | 188261 | 202287 | May-05 |
| 107 | CUMO #126 | 188262 | 202288 | May-05 |
| 108 | CUMO #127 | 188263 | 202289 | May-05 |
| 109 | CUMO #128 | 188264 | 202290 | May-05 |
| 110 | CUMO #132 | 188268 | 202294 | May-05 |
| 111 | CUMO #133 | 188269 | 202295 | May-05 |
| 112 | CUMO #134 | 188270 | 202296 | May-05 |
| 113 | CUMO #135 | 188271 | 202297 | May-05 |
| 114 | CUMO #136 | 188272 | 202298 | May-05 |
| 115 | CUMO #137 | 188273 | 202299 | May-05 |
| 116 | CUMO #138 | 188274 | 202300 | May-05 |
| 117 | CUMO #139 | 188275 | 202301 | May-05 |
| 118 | CUMO #140 | 188276 | 202302 | May-05 |
| 119 | CUMO #141 | 188277 | 202303 | May-05 |
| 120 | CUMO #142 | 188278 | 202304 | May-05 |
| 121 | CUMO #143 | 188279 | 202305 | May-05 |
| 122 | CUMO #144 | 188280 | 202306 | May-05 |
| 123 | CUMO #145 | 188281 | 202307 | May-05 |
| 124 | CUMO #146 | 188282 | 202308 | May-05 |
| 125 | CUMO #147 | 188284 | 202309 | May-05 |
| 126 | CUMO #148 | 188285 | 202310 | May-05 |
| 127 | CUMO #149 FRACT. | 188286 | 202311 | May-05 |
| 128 | CUMO #150 | 188257 | 202312 | May-05 |
| 129 | CUMO #151 FRACT. | 188287 | 202313 | May-05 |
| 130 | CUMO #152 | 188288 | 202314 | May-05 |
| 131 | CUMO #153 | 188289 | 202315 | May-05 |
| 132 | CUMO #154 | 188290 | 202316 | May-05 |
| 133 | CUMO #155 | 188291 | 202317 | May-05 |
| 134 | CUMO #156 | 188292 | 202318 | May-05 |
| 135 | CUMO #157 | 188293 | 202319 | May-05 |

## Cumo Claim List 2014 - Page 4

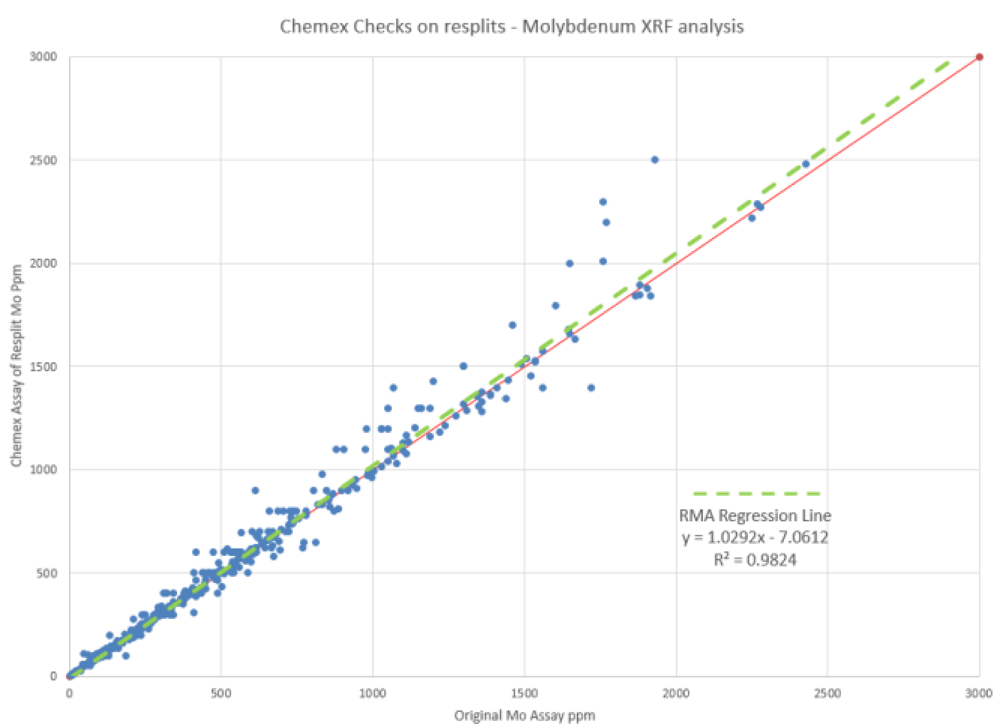
| Item | Claim Name/Number | BLM Serial No. | County Instrument Number | Loc Dt |
|---|---|---|---|---|
| 136 | CUMO #158 | 188294 | 202320 | May-05 |
| 137 | CUMO #159 | 188295 | 202323 | May-05 |
| 138 | CUMO #160 | 188486 | 202321 | May-05 |
| 139 | CUMO #161 | 188491 | 202322 | May-05 |
| 140 | CUMO #176 FRACT. | 188306 | 202324 | May-05 |
| 141 | CUMO #177 FRACT. | 188307 | 202325 | May-05 |
| 142 | CUMO #178 | 188308 | 202326 | May-05 |
| 143 | CUMO #179 | 188309 | 202327 | May-05 |
| 144 | CUMO #180 | 188310 | 202328 | May-05 |
| 145 | CUMO #181 | 188311 | 202329 | May-05 |
| 146 | CUMO #182 FRACT. | 188312 | 202330 | May-05 |
| 147 | CUMO #183 FRACT. | 188313 | 202331 | May-05 |
| 148 | CUMO #184 | 188314 | 202332 | May-05 |
| 149 | CUMO #185 | 188315 | 202333 | May-05 |
| 150 | CUMO #186 | 188316 | 202334 | May-05 |
| 151 | CUMO #187 | 188317 | 202335 | May-05 |
| 152 | CUMO #188 FRACT. | 188318 | 202336 | May-05 |
| 153 | New Cumo 190 Fraction | 203192 | | Oct-10 |
| 154 | New Cumo 191 Fraction | 203193 | | Oct-10 |
| 155 | New Cumo 192 Fraction | 203194 | | Oct-10 |
| 156 | New Cumo 193 Fraction | 203195 | | Oct-10 |
| 157 | Cumo 194 | 203196 | | Oct-10 |
| 158 | Cumo 195 Fraction | 203197 | | Oct-10 |
| 159 | Cumo 196 Fraction | 203198 | | Oct-10 |
| 160 | Cumo 197 Fraction | 203199 | | Oct-10 |
| 161 | Cumo 198 Fraction | 203200 | | Oct-10 |
| 162 | Cumo 199 Fraction | 203201 | | Oct-10 |
| 163 | Cumo 200 Fraction | 203202 | | Oct-10 |
| 164 | Cumo 201 Fraction | 203203 | | Oct-10 |

# 22.0 Appendix 2: Re-Splits of Rejects

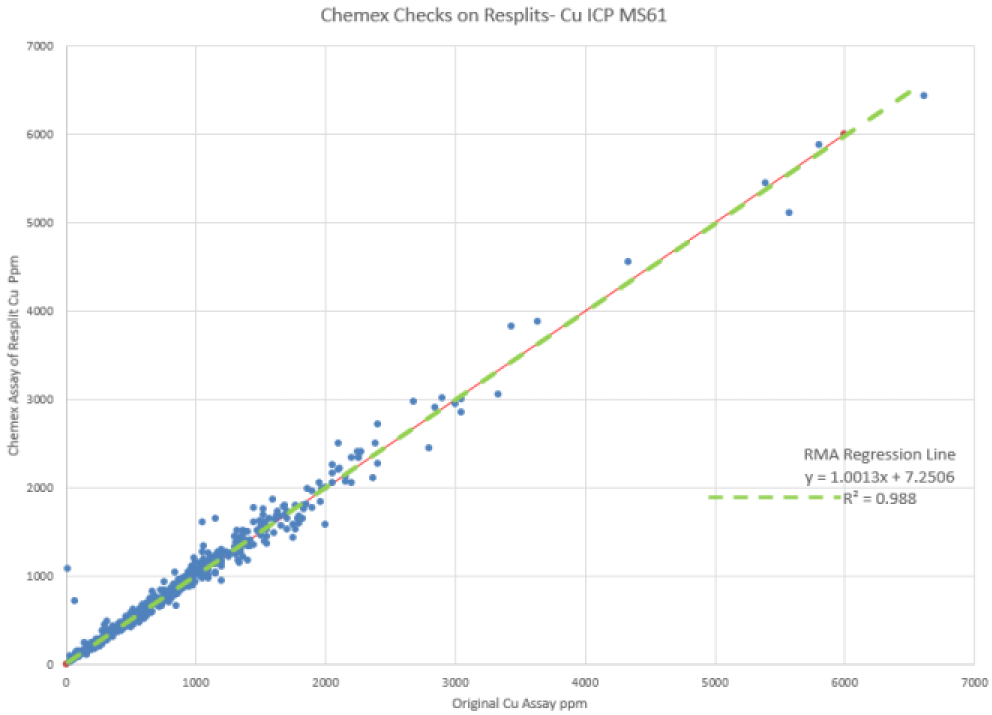
Results for Mo - Chemex - Original vs. ICP Check



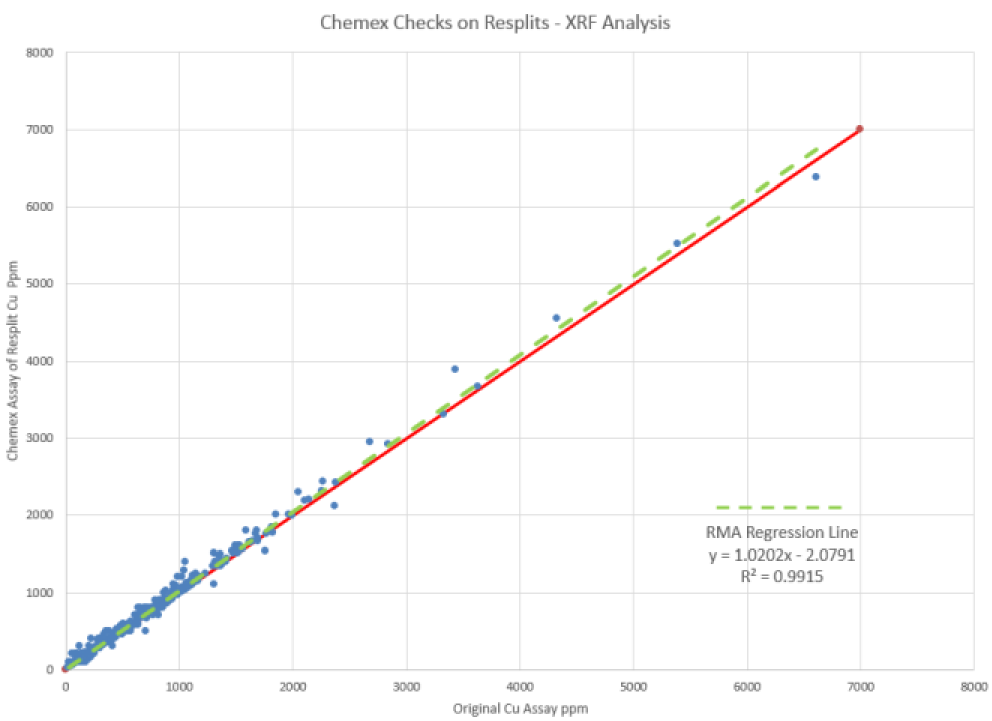

Results for Mo - Chemex - Original vs. XRF Check



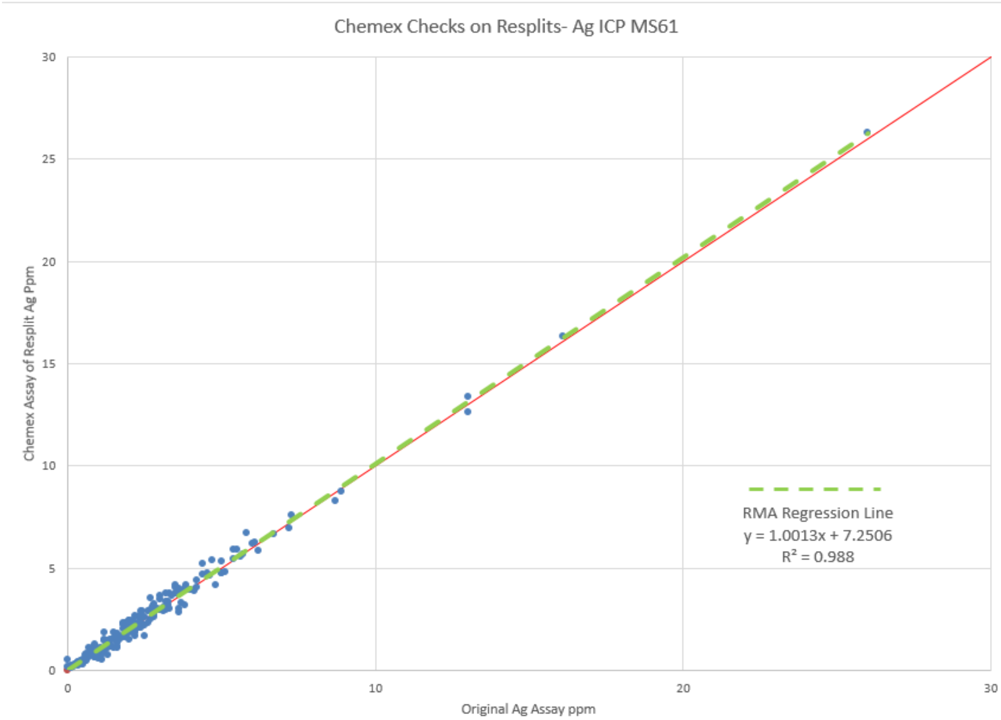
Results for Cu - Chemex - Original vs. ICP Check




Results for Cu - Chemex - Original vs. XRF Check



Results for Ag - Chemex Original vs. ICP Check



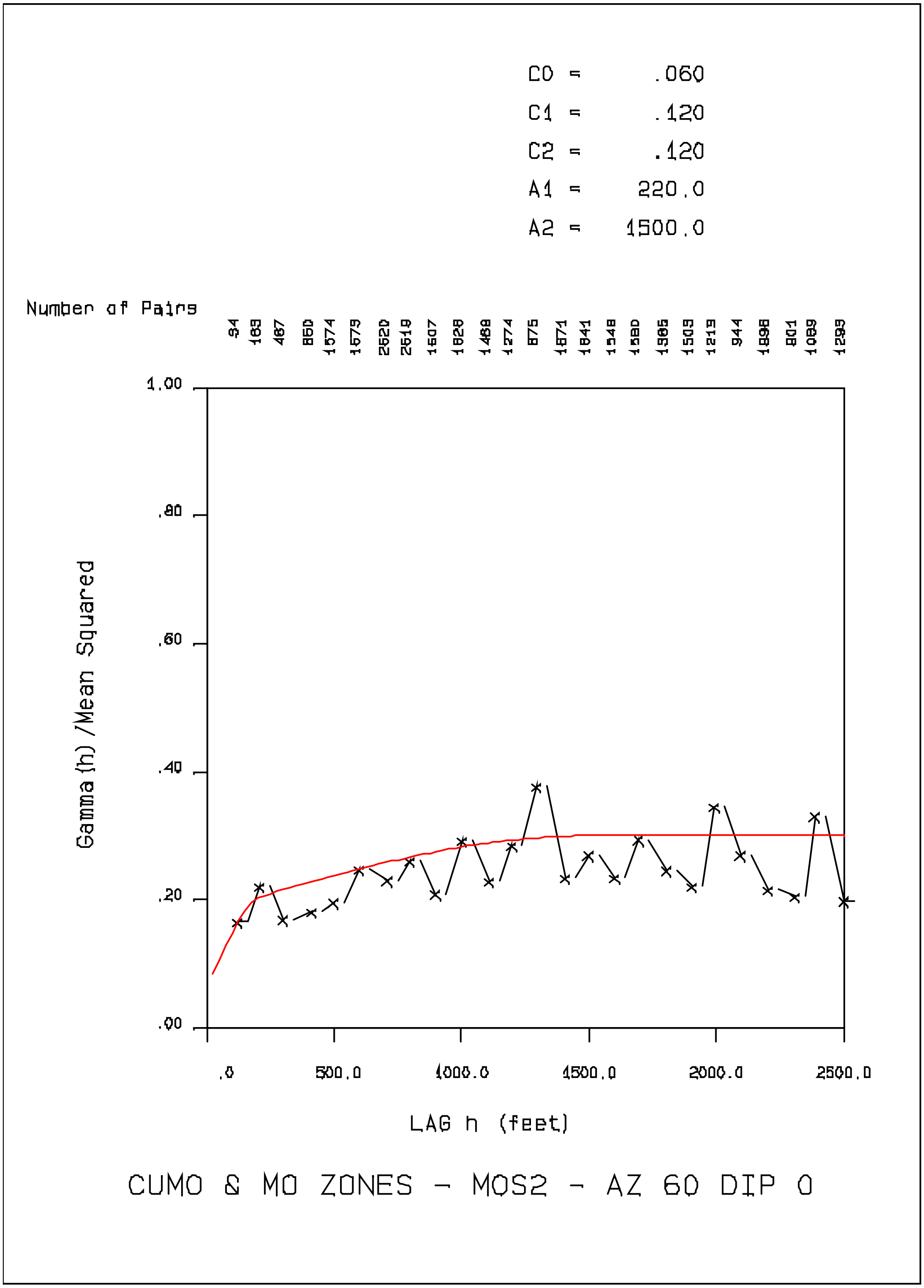
# 23.0 Appendix 3: Drill Holes used in Resource Estimate

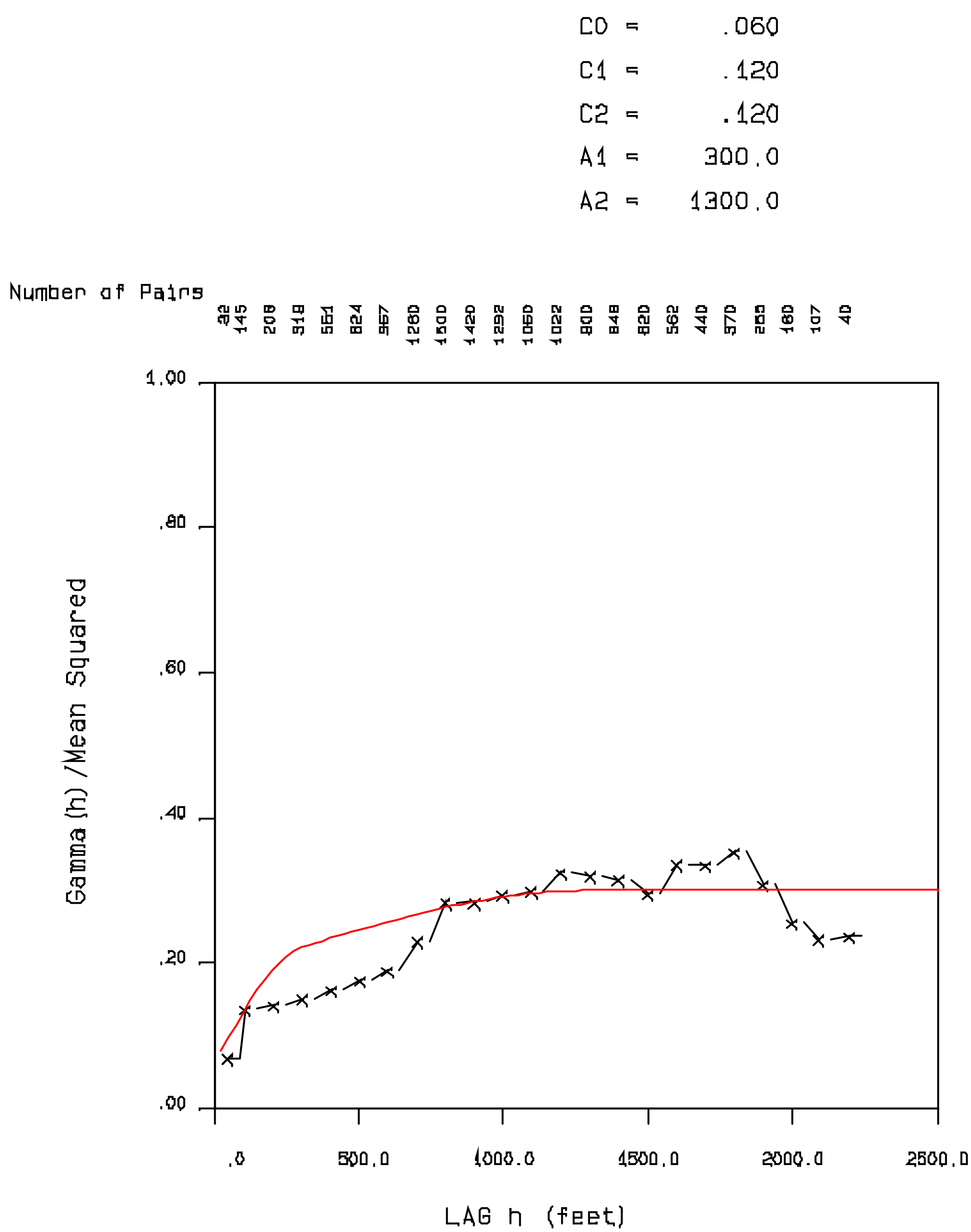
| Hole | Northing | Easting | Elevation | Dip | Azimuth | Length (feet) |
|---|---|---|---|---|---|---|
| 27-06 | 120,016.70 | 220,160.30 | 7105 | -90 | 0 | 1849 completed |
| 28-06 | 119,531.60 | 120,796.40 | 7170 | -90 | 0 | 1711 completed |
| 29-07 | 120,016.70 | 220,160.30 | 6305 | -70 | 140 | 2281.7 completed |
| 30-07 | 119,531.60 | 220,796.40 | 6206 | -90 | 0 | 2416.5 completed |
| 31-07 | 120,016.70 | 220,160.30 | 6305 | -70 | 45 | 2104 completed |
| 32-07 | 119,480.00 | 220,720.30 | 6316 | -70 | 190 | 2044 completed |
| 33-07 | 118,585.30 | 221,268.90 | 6798 | -90 | 0 | 2095 stopped |
| 34-07 | 118,530.50 | 220,343.80 | 6512 | -70 | 95 | 1769 stopped |
| 34-07 | 118,530.50 | 220,343.80 | 6512 | -70 | 95 | 1769 stopped |
| 36-08 | 119,266.80 | 219,322.90 | 6457 | -90 | 0 | 2488 completed |
| 37-08 | 119,755.70 | 221,220.40 | 6341 | -70 | 335 | 2195 completed |
| 38-08 | 118,658.30 | 220,487.40 | 6534 | -70 | 180 | 2441 completed |
| 39-08 | 118,872.70 | 220,777.60 | 6466 | -90 | 0 | 2688 completed |
| 40-08 | 119,539.80 | 220,816.80 | 6321 | -70 | 225 | 2252 completed |
| 41-08 | 119,545.70 | 219,005.80 | 6247 | -90 | 0 | 3018 completed |
| 42-08 | 118,711.90 | 219,886.60 | 6544 | -70 | 270 | 2707 stopped (winter) |
| 43-08 | 120,515.60 | 220,178.60 | 6198 | -80 | 40 | 1308 stopped by fault |
| 44-08 | 118,068.10 | 221,448.90 | 6733 | -65 | 75 | 3047 completed |
| 45-08 | 119,802.30 | 218,821.40 | 6183 | -80 | 330 | 1796 stopped (winter) |
| 46-09 | 220,811.30 | 118,913.90 | 6575.1 | -75 | 110 | 959 stopped |
| 47-09 | 219,421.70 | 120,686.70 | 5832.6 | -90 | 0 | 2530 completed |
| 48-09 | 120,741.30 | 219,432.50 | 5827 | -70 | 305 | 2576 completed |
| 49-09 | 118,881.60 | 221,719.80 | 6668 | -90 | 0 | 2847 completed |
| 50-09 | 121,752.90 | 219,929.40 | 5885 | -75 | 270 | 1826 completed |
| 51-09 | 121,752.90 | 219,929.40 | 5885 | -90 | 0 | 1583.5 completed |
| 52-09 | 118,585.30 | 221,268.90 | 6798 | -75 | 20 | 2772 completed |
| 53-09 | 119,802.30 | 218,821.40 | 6183 | -75 | 15 | 2461 completed |
| 54-09 | 119,802.30 | 218,821.40 | 6183 | -75 | 15 | 2471 completed |
| 55-10 | 117,559.60 | 218,422.40 | 6724.2 | -65 | 0 | 2479 completed |
| 56-10 | 117,559.90 | 218,421.80 | 6724.2 | -65 | 305 | 1294 completed |
| 57-10 | 117,559.30 | 218,422.20 | 6724.2 | -90 | 0 | 534 stopped (winter) |
| 58-11 | 219,970.30 | 119,095.60 | 6451.3 | -90 | 0 | 1885 completed |
| 59-11 | 221,745.90 | 117,559.90 | 6645.3 | -75 | 0 | 1910 completed |
| 60-12 | 218421.86 | 117559.92 | 6724.2 | -50 | 180 | 1455 completed |
| 61-12 | 219911 | 118748.9 | 6549.23 | -75 | 335 | 1318 Stopped |

| | | | | | | |
|---|---|---|---|---|---|---|
| 62-12 | 218040.5 | 116866.1 | 6628.7 | -50 | 135 | 1484 completed |
| 63-12 | 218041.5 | 116866.8 | 6628.7 | -60 | 330 | 807 completed |
| 64-12 | 220811.3 | 118913.9 | 6575.1 | -75 | 25 | 2139 completed |
| 65-12 | 221117.5 | 118148.8 | 6785.7 | -80 | 315 | 1908 completed |
| 66-12 | 221687.8 | 118674 | 6689.7 | -90 | 0 | 2241 completed |
| 67-12 | 220811.3 | 118913.9 | 6575.1 | -70 | 340 | 1978 completed |
| 68-12 | 221745.9 | 119095.6 | 6645.3 | -70 | 310 | 2133.5 completed |

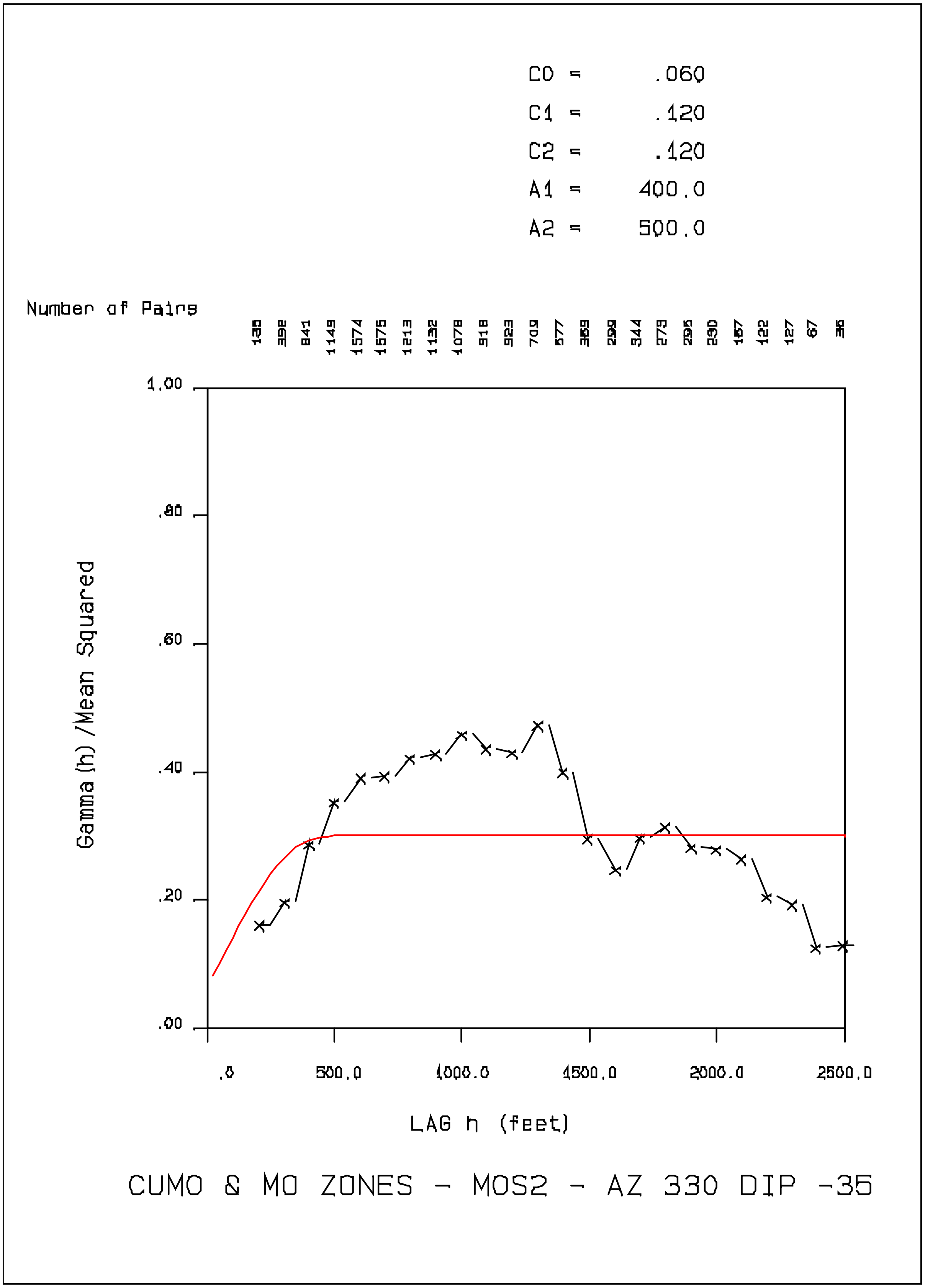
# 24.0 Appendix 4: Semivariograms

4.1 - $MoS_2$ in CuMo and Mo Zones



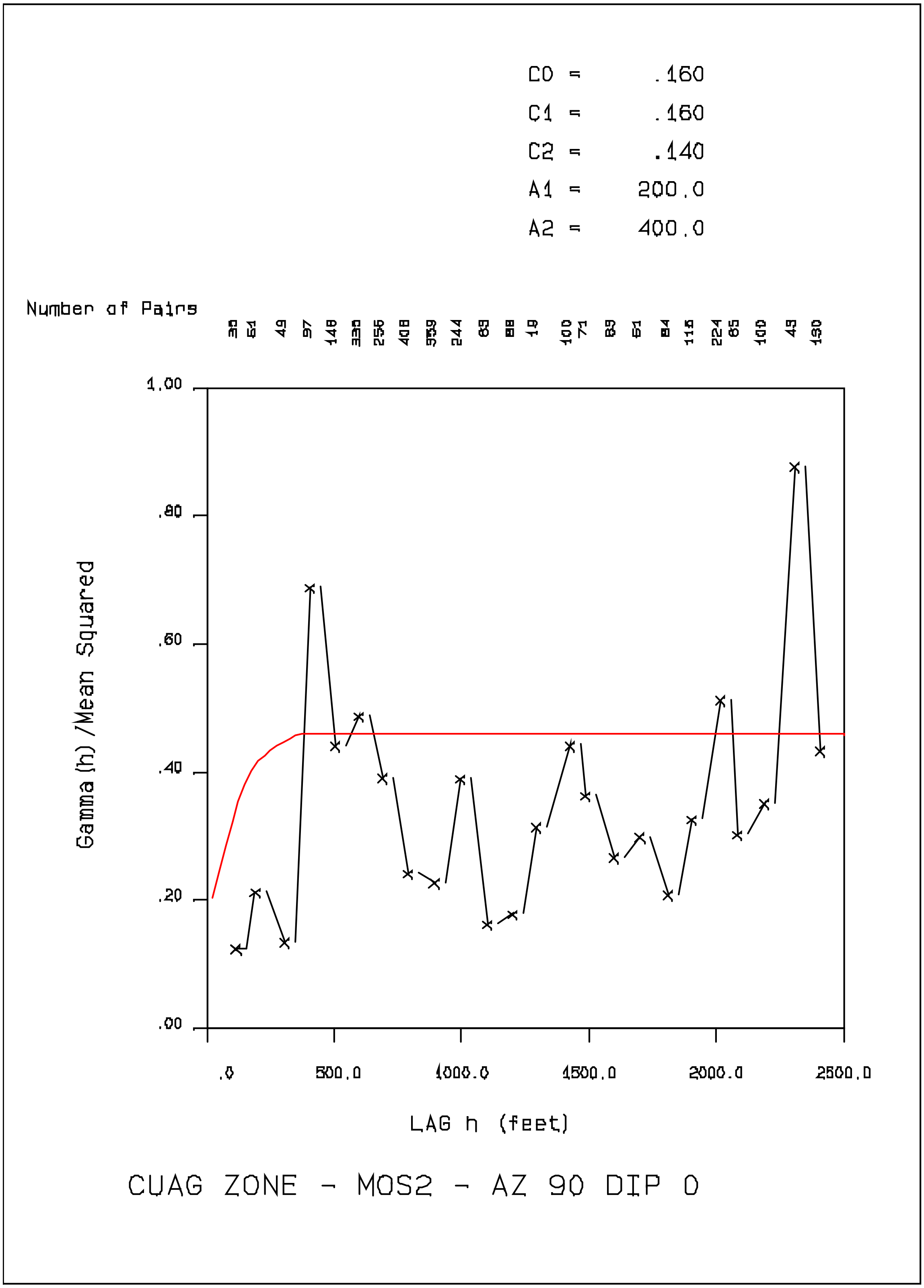


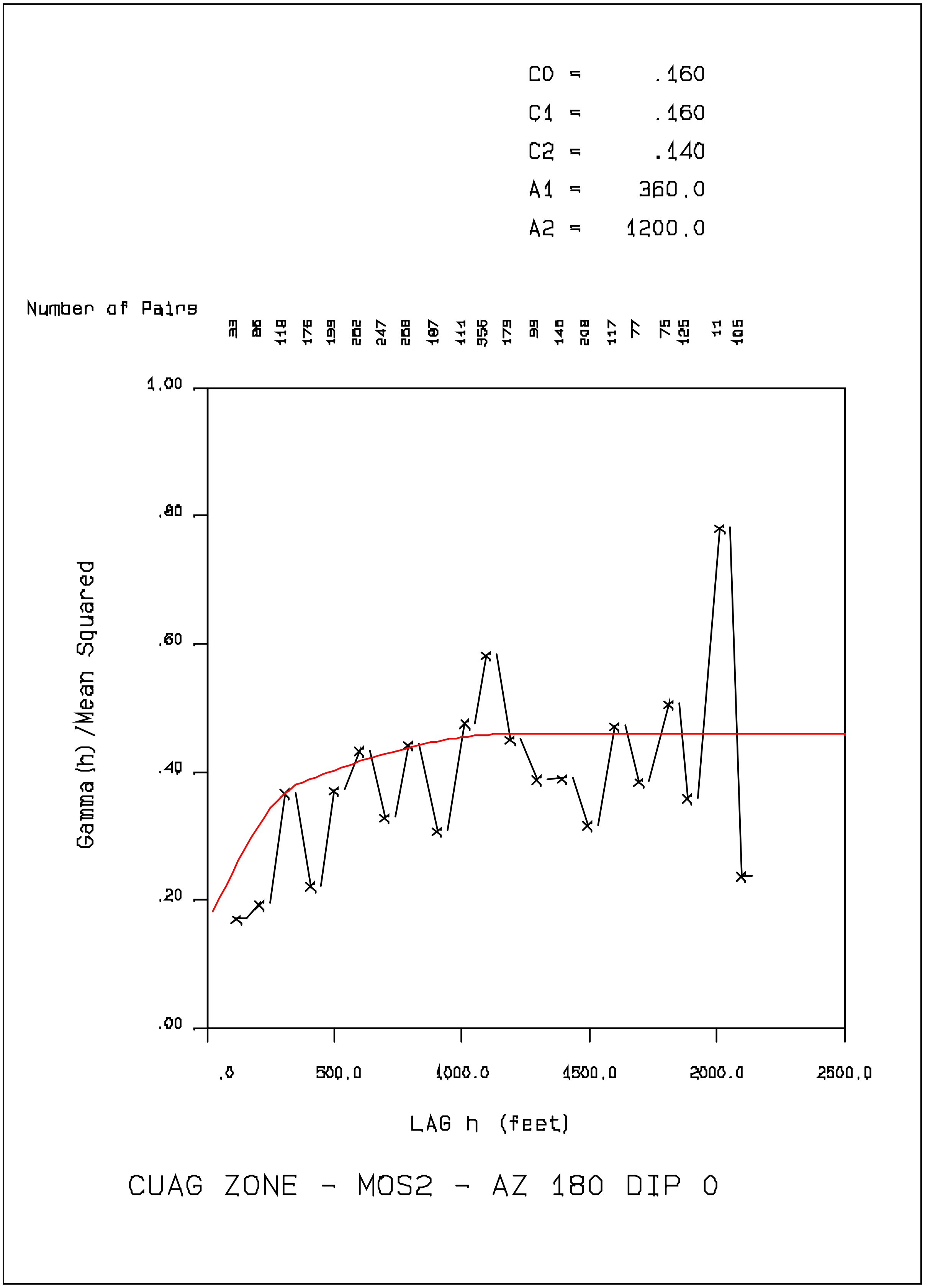

CUMO & MO ZONES - AZ 150 DIP -55

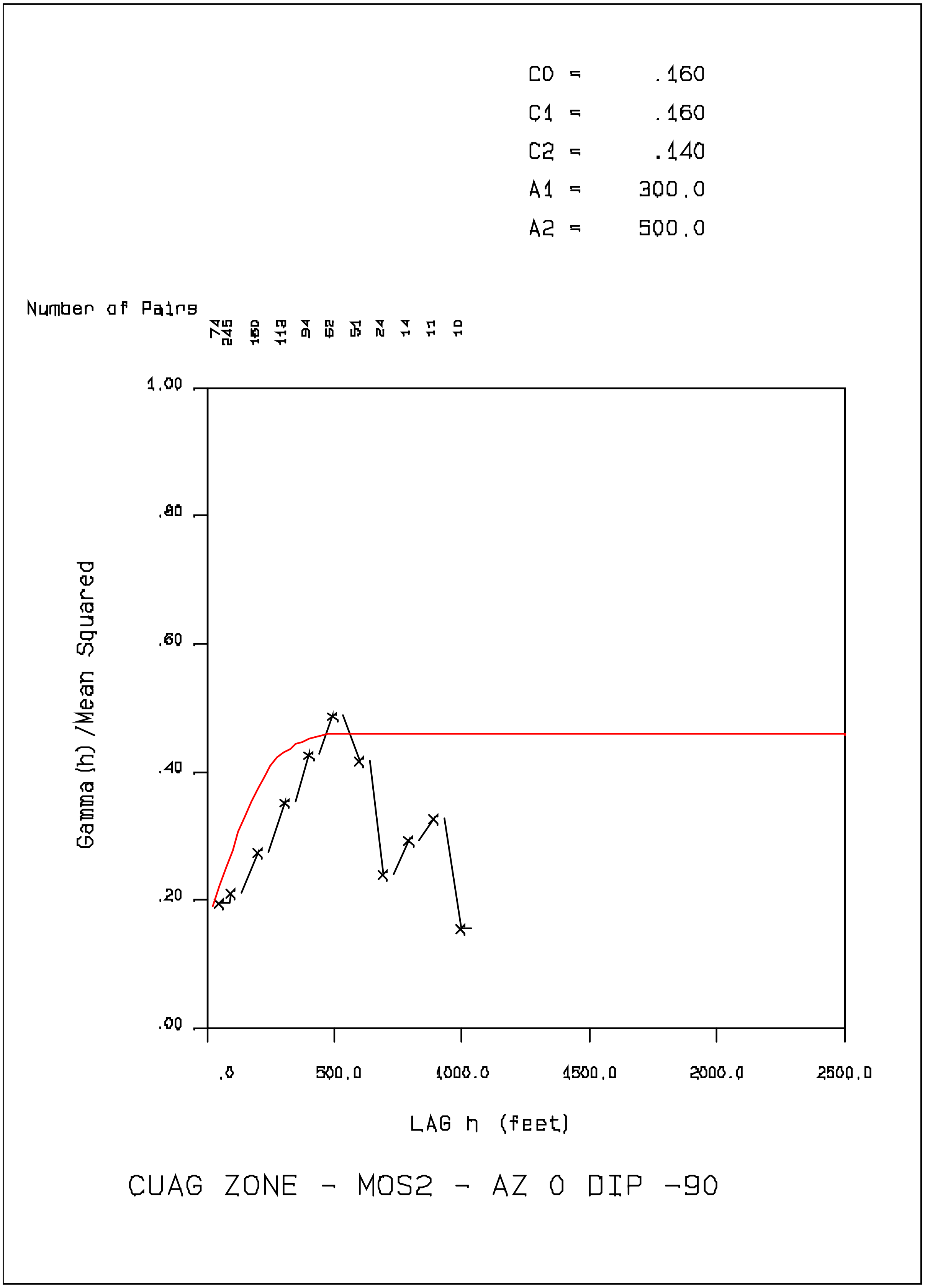


C0 = .060
C1 = .120
C2 = .120
A1 = 400.0
A2 = 500.0
Number of Pairs
135 392 841 1149 1574 1575 1213 1132 1078 918 923 709 577 369 299 344 273 295 290 157 122 127 67 36
1.00
.80
.60
.40
.20
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUMO & MO ZONES - MOS2 - AZ 330 DIP -35

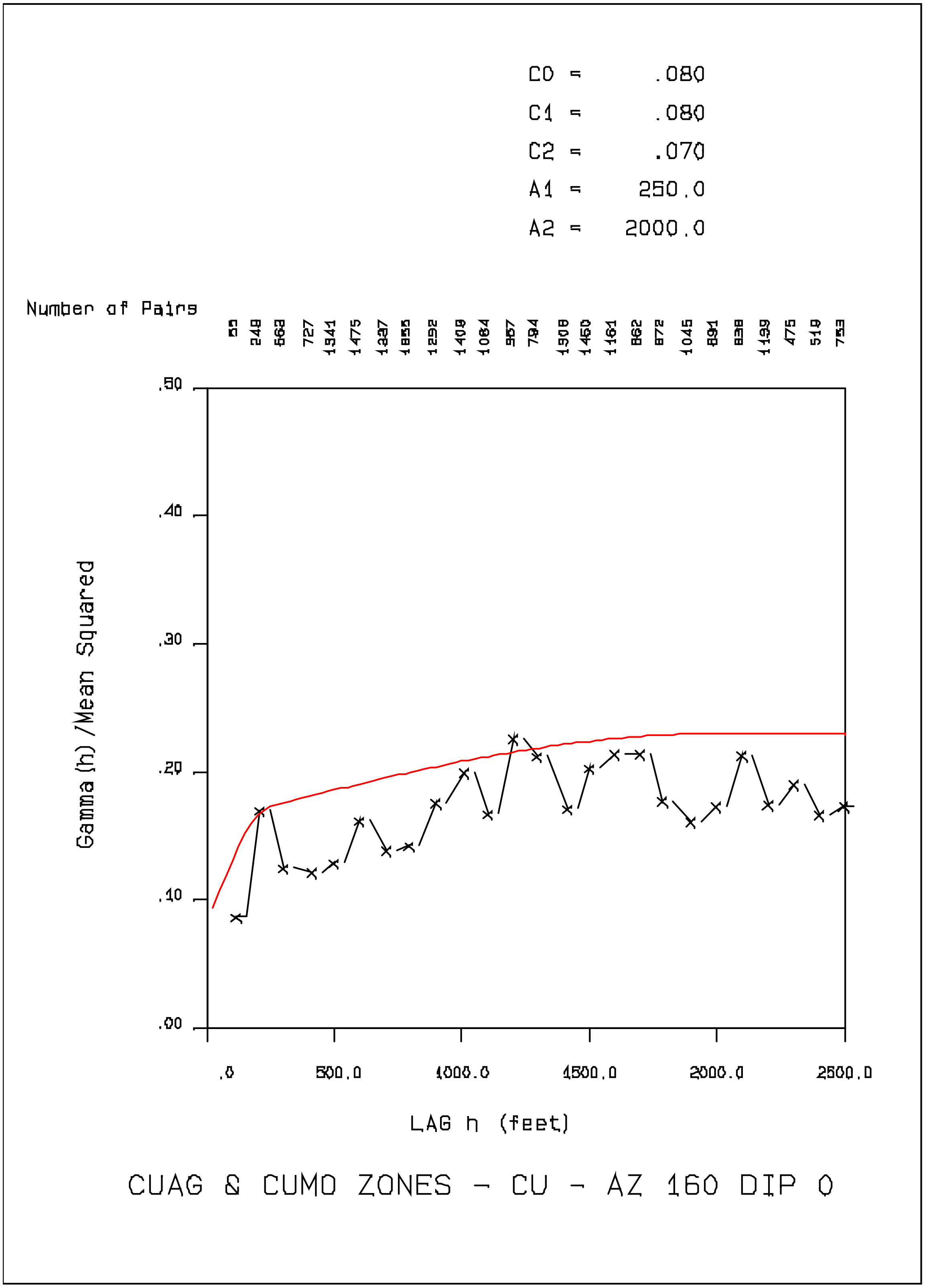
4.2 - $MoS_2$ in CuAg Zone

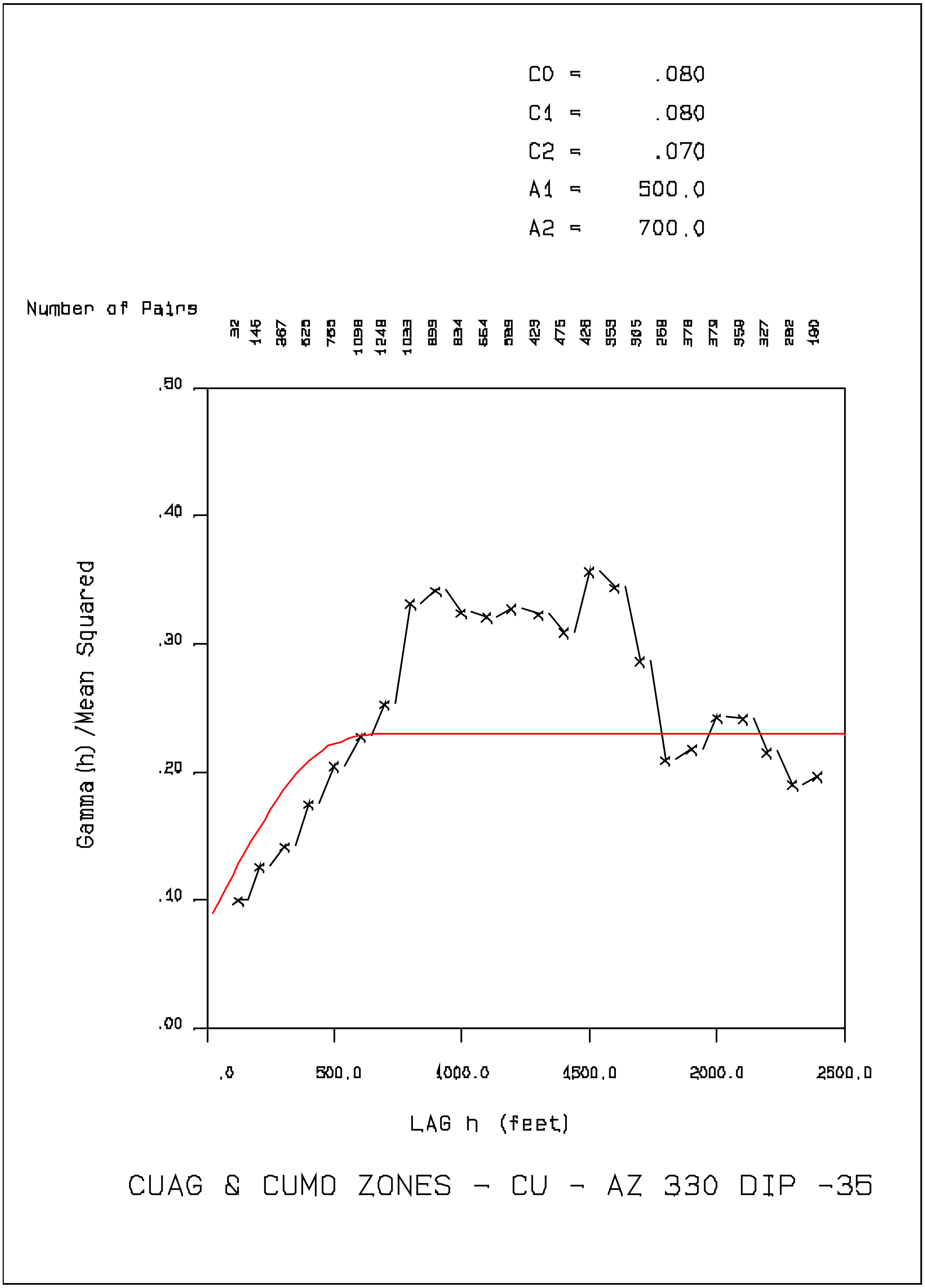


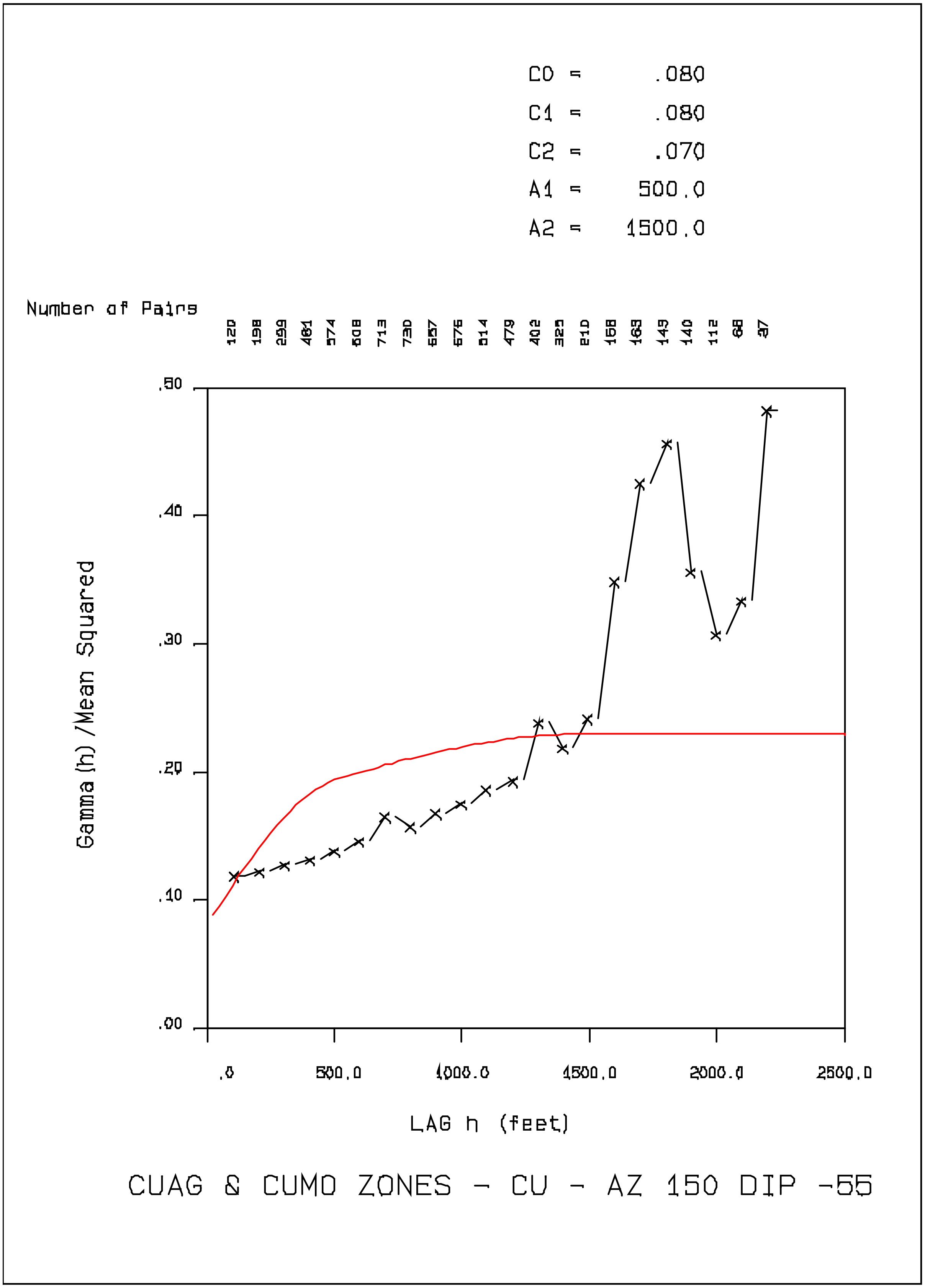

C0 = .160
C1 = .160
C2 = .140
A1 = 360.0
A2 = 1200.0
Number of Pairs
Gamma (h) /Mean Squared
LAG h (feet)
CUAG ZONE - MOS2 - AZ 180 DIP 0


C0 = .160
C1 = .160
C2 = .140
A1 = 300.0
A2 = 500.0
Number of Pairs
74
245
150
113
94
62
51
24
14
11
10
1.00
.80
.60
.40
.20
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG ZONE - MOS2 - AZ 0 DIP -90

4.3 - Cu in CuAg and CuMo Zones

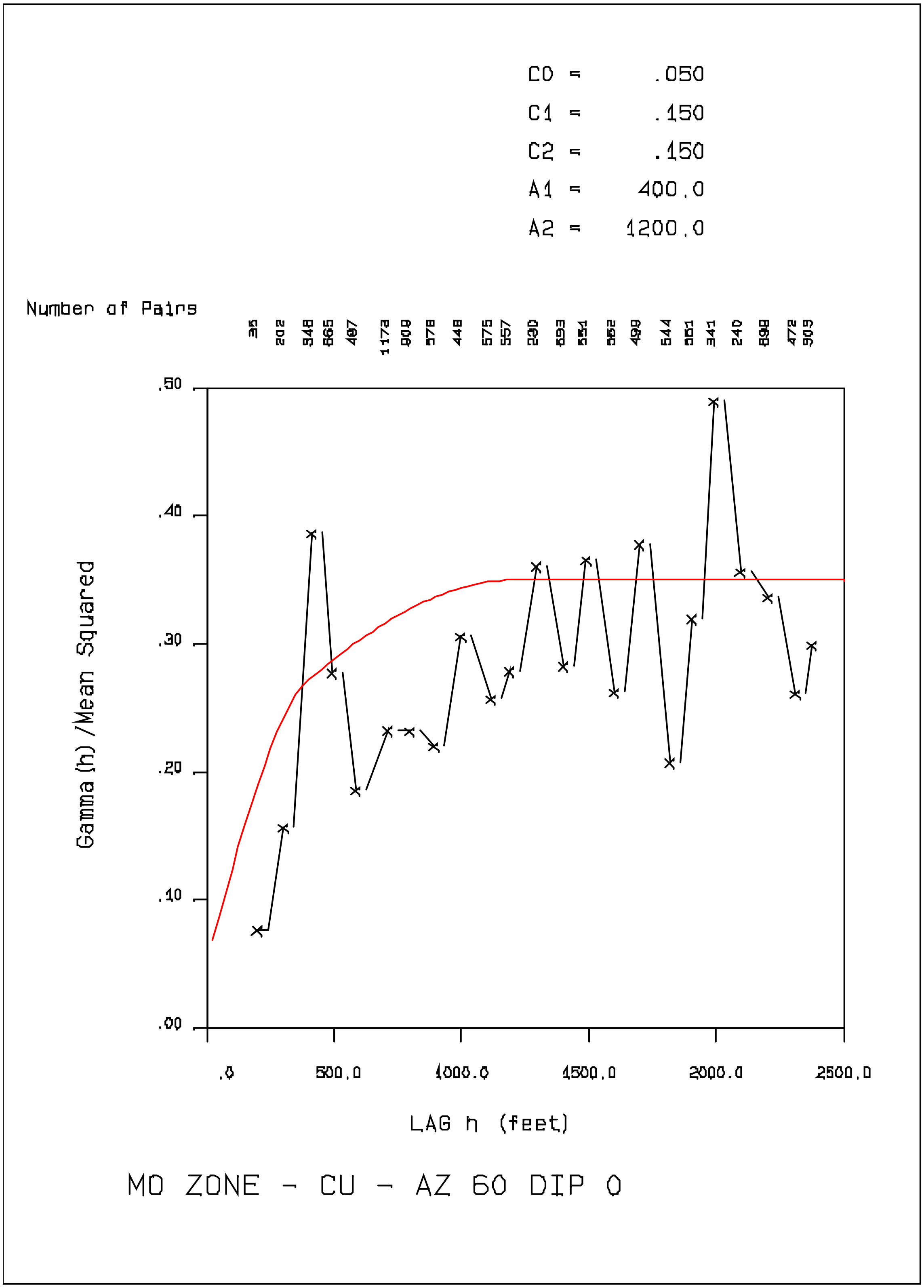


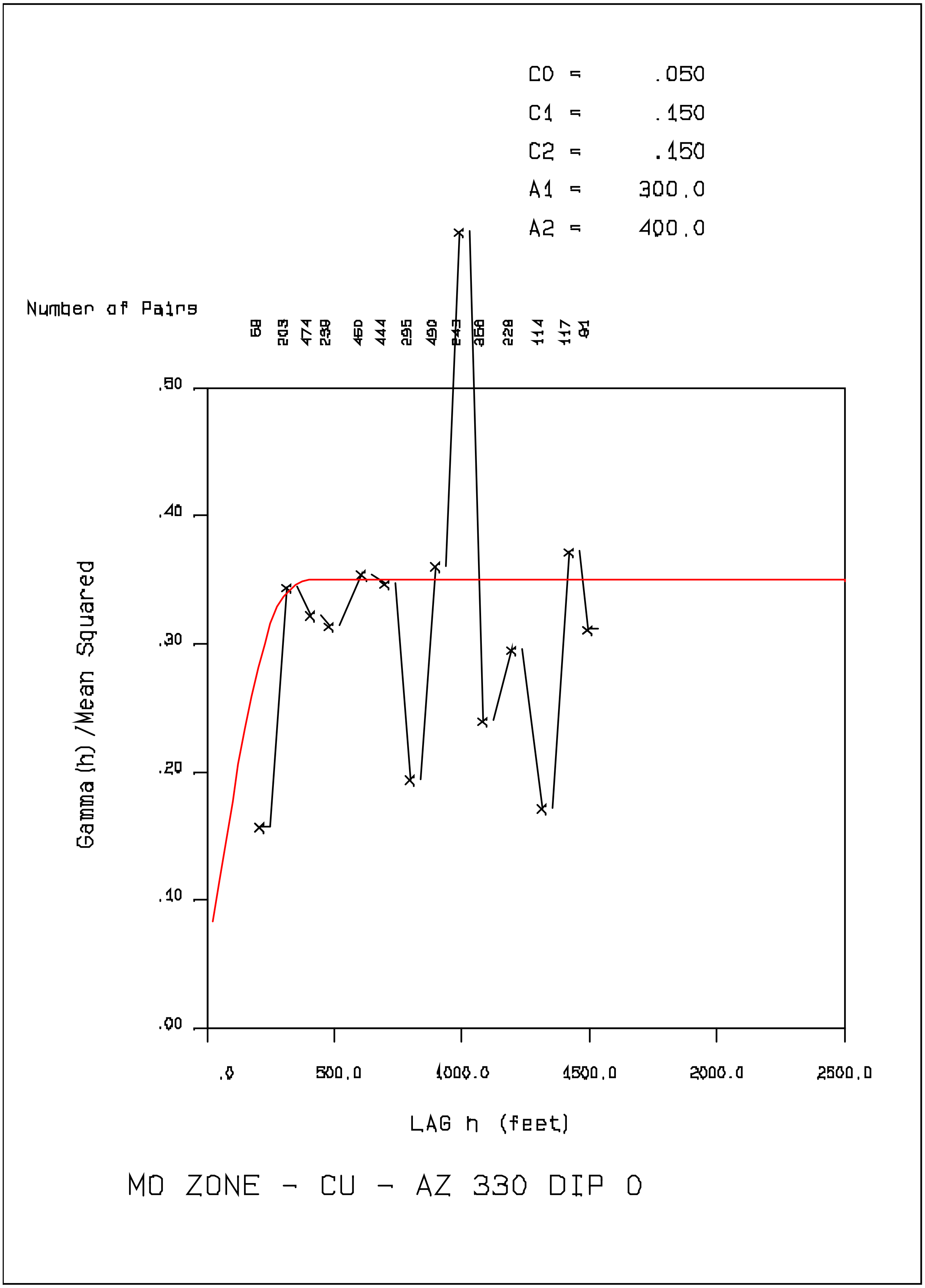

C0 = .080
C1 = .080
C2 = .070
A1 = 500.0
A2 = 700.0
Number of Pairs
Gamma (h) /Mean Squared
.50
.40
.30
.20
.10
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG & CUMO ZONES - CU - AZ 330 DIP -35

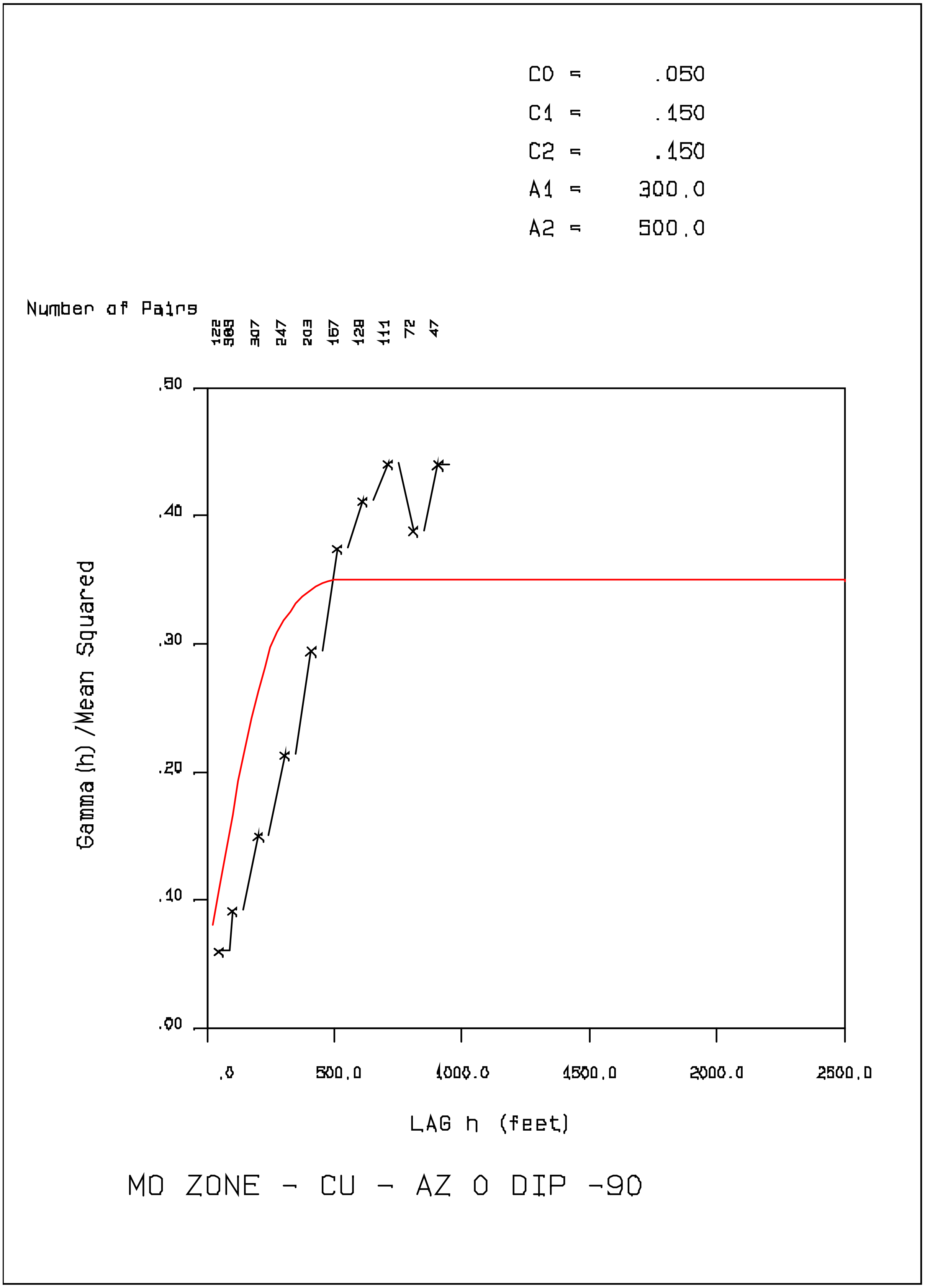


C0 = .080
C1 = .080
C2 = .070
A1 = 500.0
A2 = 1500.0
Number of Pairs
120 198 299 401 574 608 713 730 657 676 514 479 402 325 210 168 163 143 140 112 68 37
.50
.40
.30
.20
.10
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG & CUMO ZONES - CU - AZ 150 DIP -55

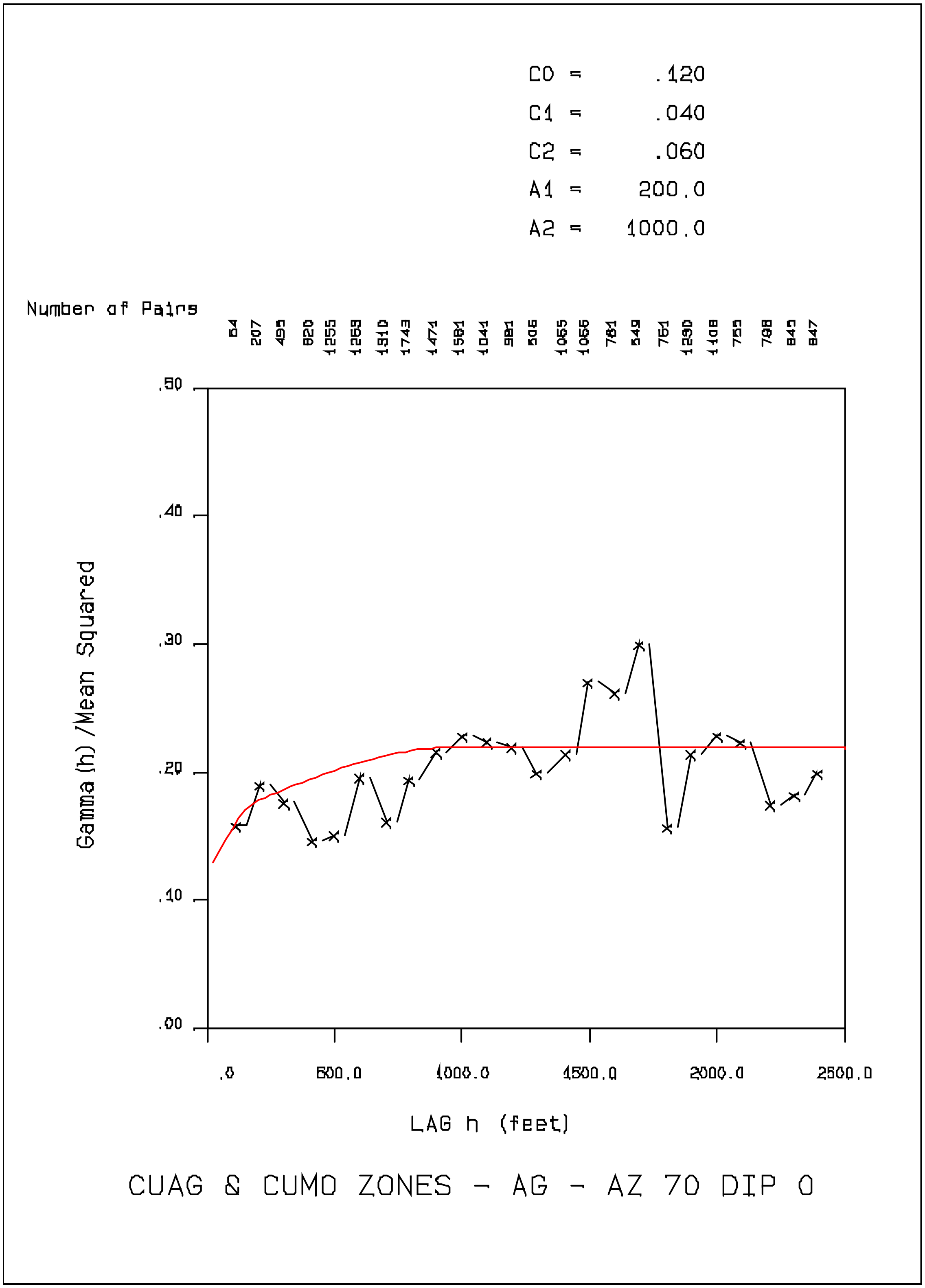
4.4 - Cu in Mo Zone

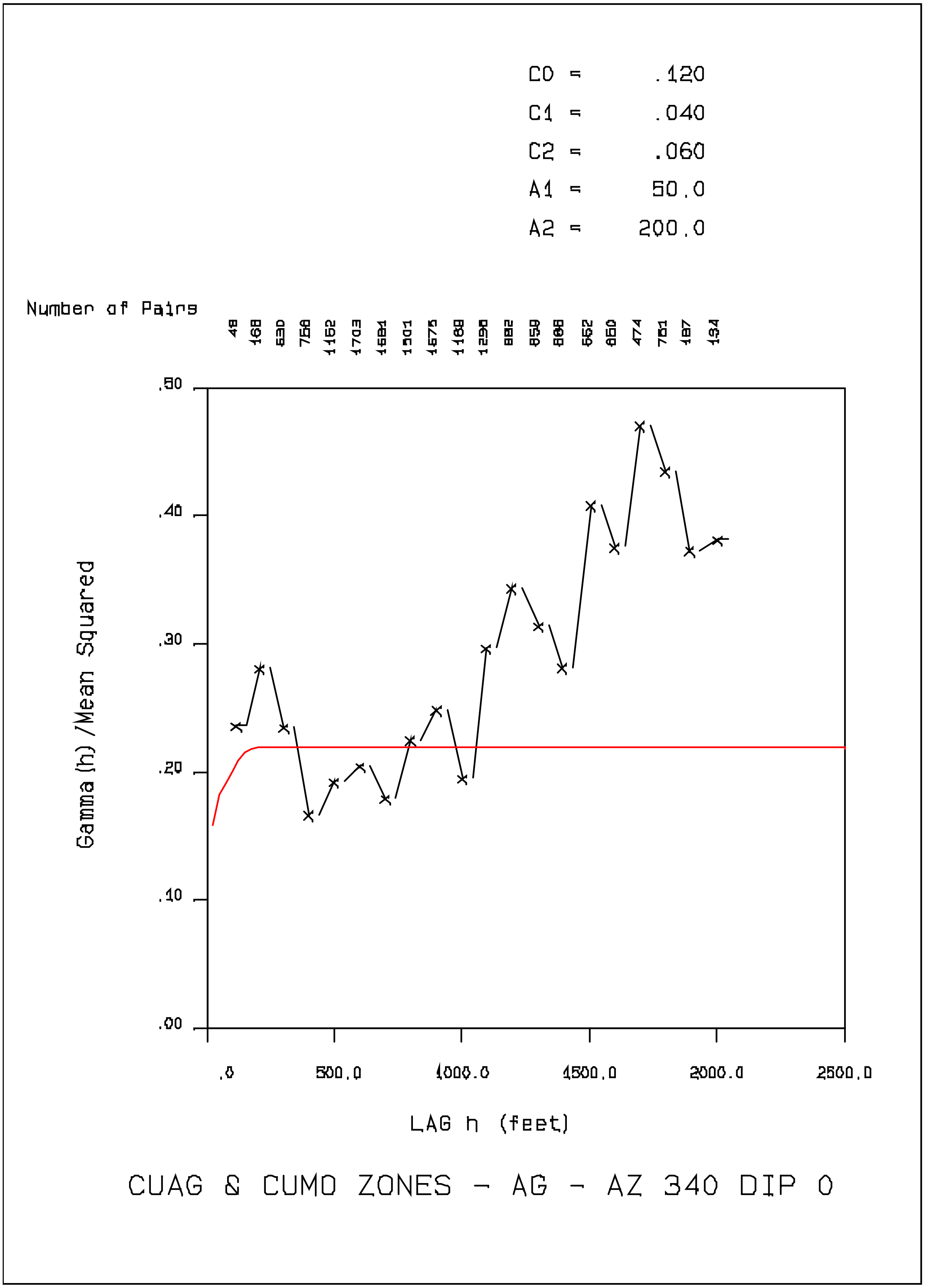


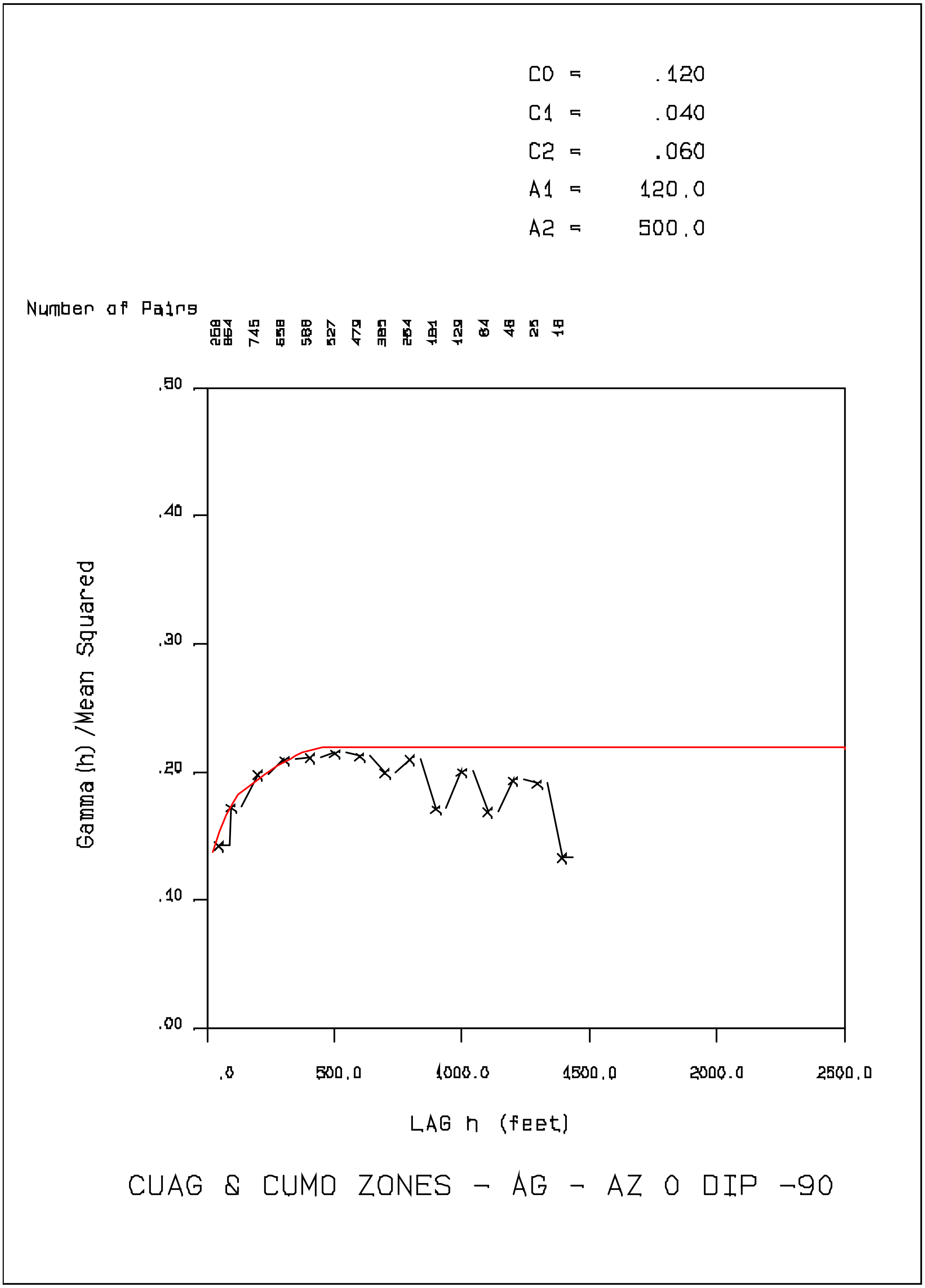

C0 = .050
C1 = .150
C2 = .150
A1 = 300.0
A2 = 400.0
Number of Pairs
58 203 474 238 460 444 295 490 243 356 228 114 117 91
Gamma (h) /Mean Squared
.50
.40
.30
.20
.10
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
MO ZONE - CU - AZ 330 DIP 0


C0 = .050
C1 = .150
C2 = .150
A1 = 300.0
A2 = 500.0
Number of Pairs
122 363 307 247 203 167 128 111 72 47
.50
.40
.30
.20
.10
.00
Gamma (h) /Mean Squared
.0 500.0 1000.0 1500.0 2000.0 2500.0
LAG h (feet)
MO ZONE - CU - AZ 0 DIP -90

4.5 - Ag in CuAg and CuMo Zones

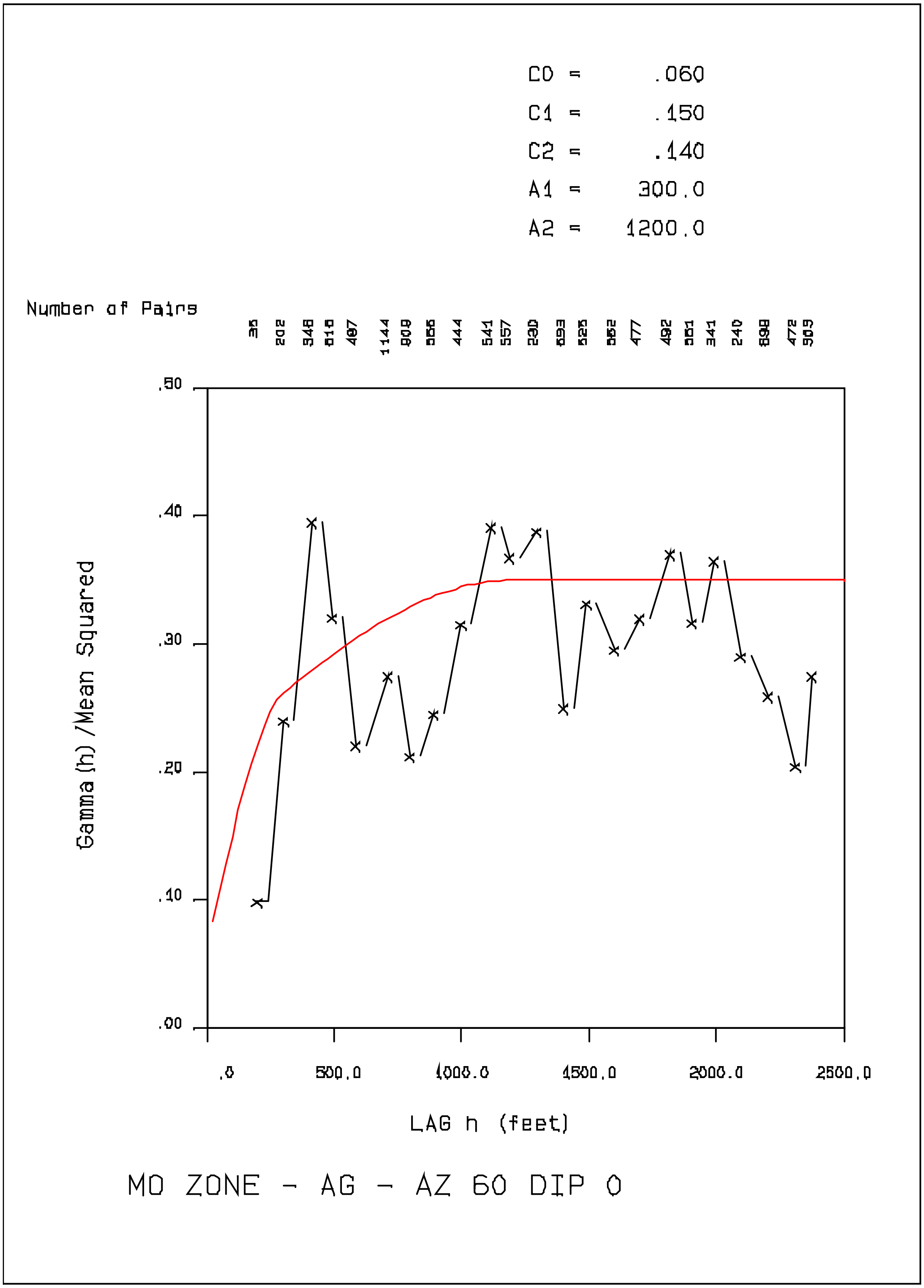


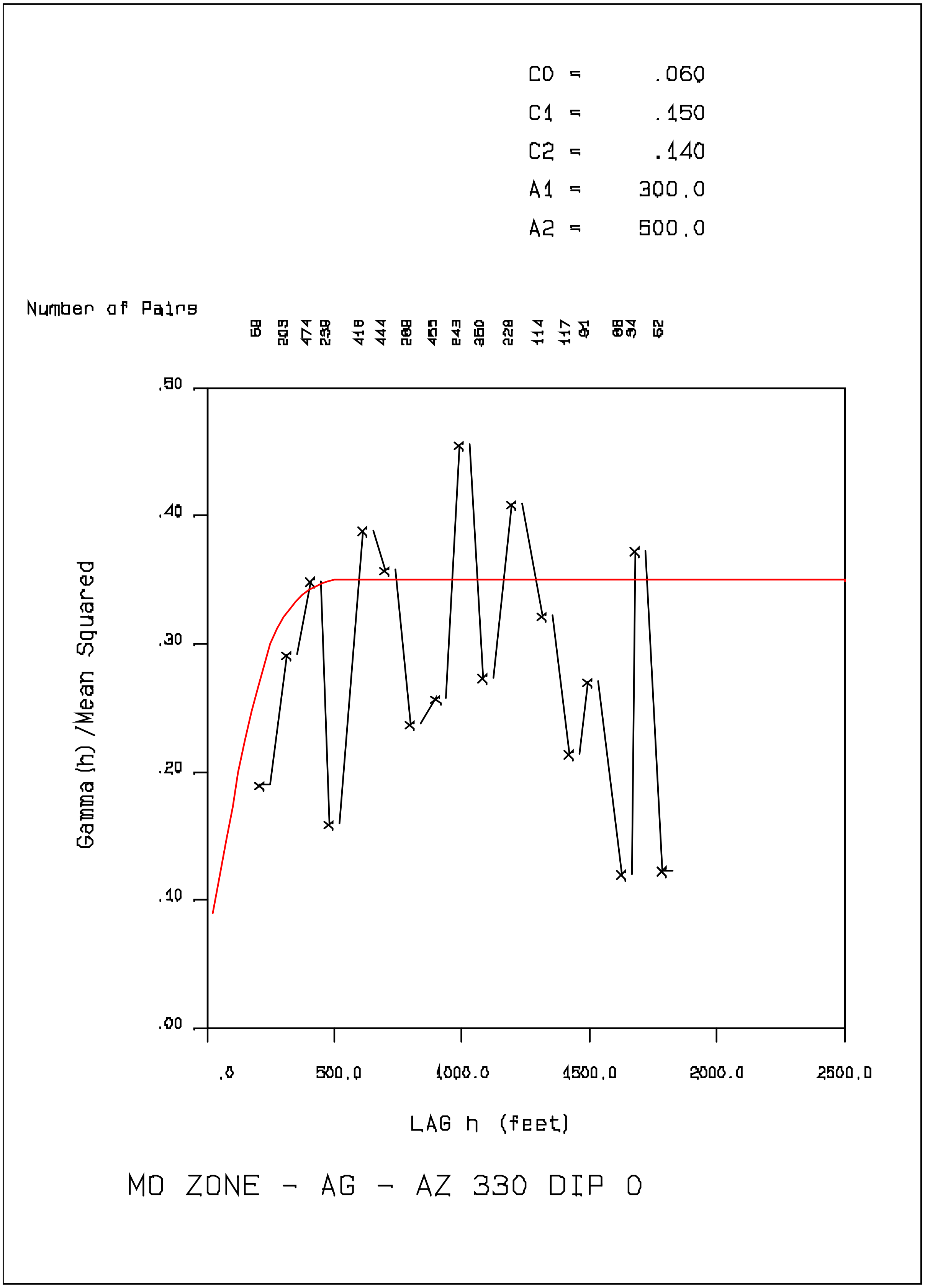

C0 = .120
C1 = .040
C2 = .060
A1 = 50.0
A2 = 200.0
Number of Pairs
48 168 630 756 1162 1703 1684 1501 1675 1168 1296 882 850 806 562 660 474 761 187 134
.50
.40
.30
.20
.10
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG & CUMO ZONES - AG - AZ 340 DIP 0



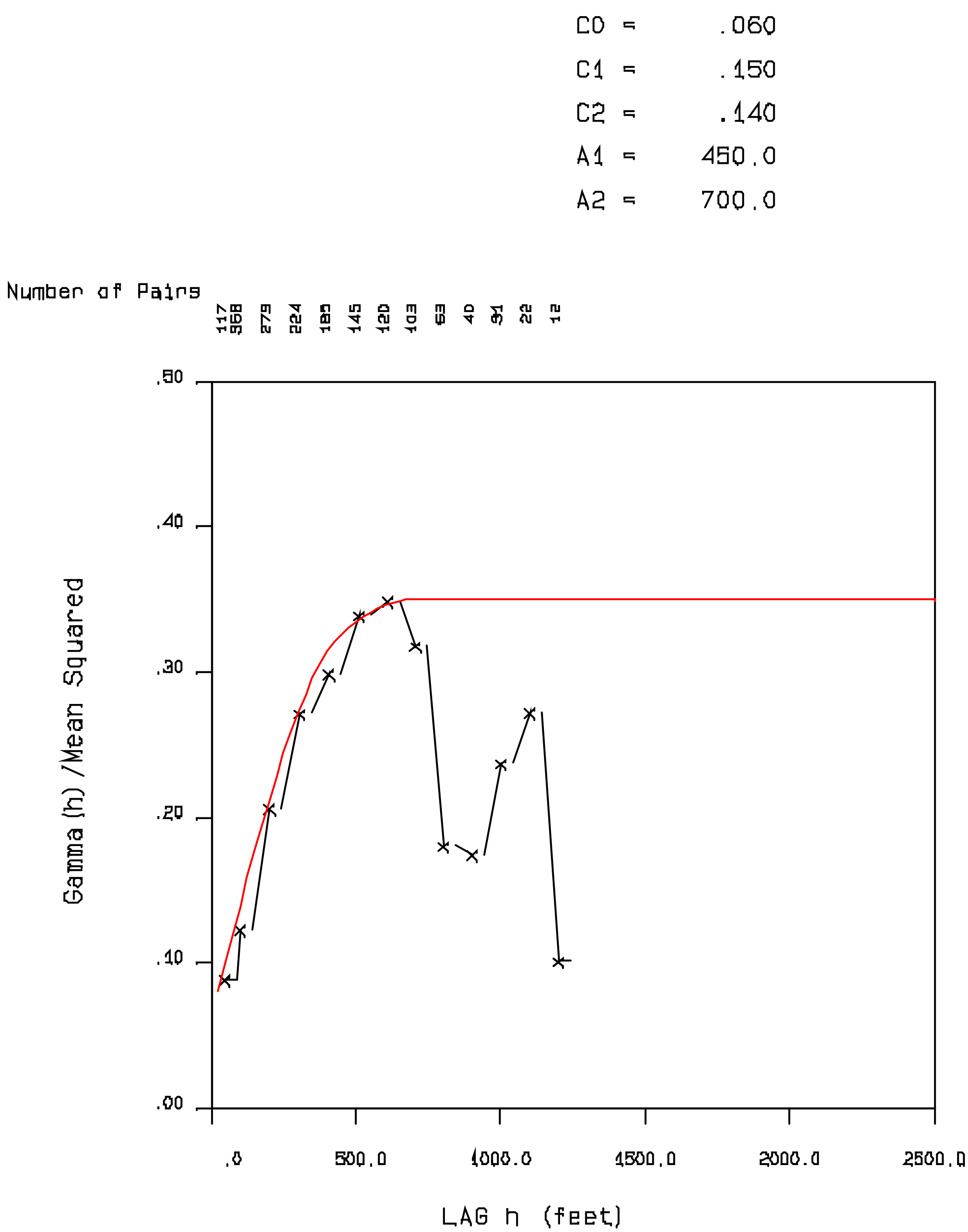
C0 = .120
C1 = .040
C2 = .060
A1 = 120.0
A2 = 500.0
Number of Pairs
258 864 745 658 588 527 479 385 264 181 129 84 48 25 18
Gamma (h) /Mean Squared
.50
.40
.30
.20
.10
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG & CUMO ZONES - AG - AZ 0 DIP -90

4.6 - Ag in Mo Zone



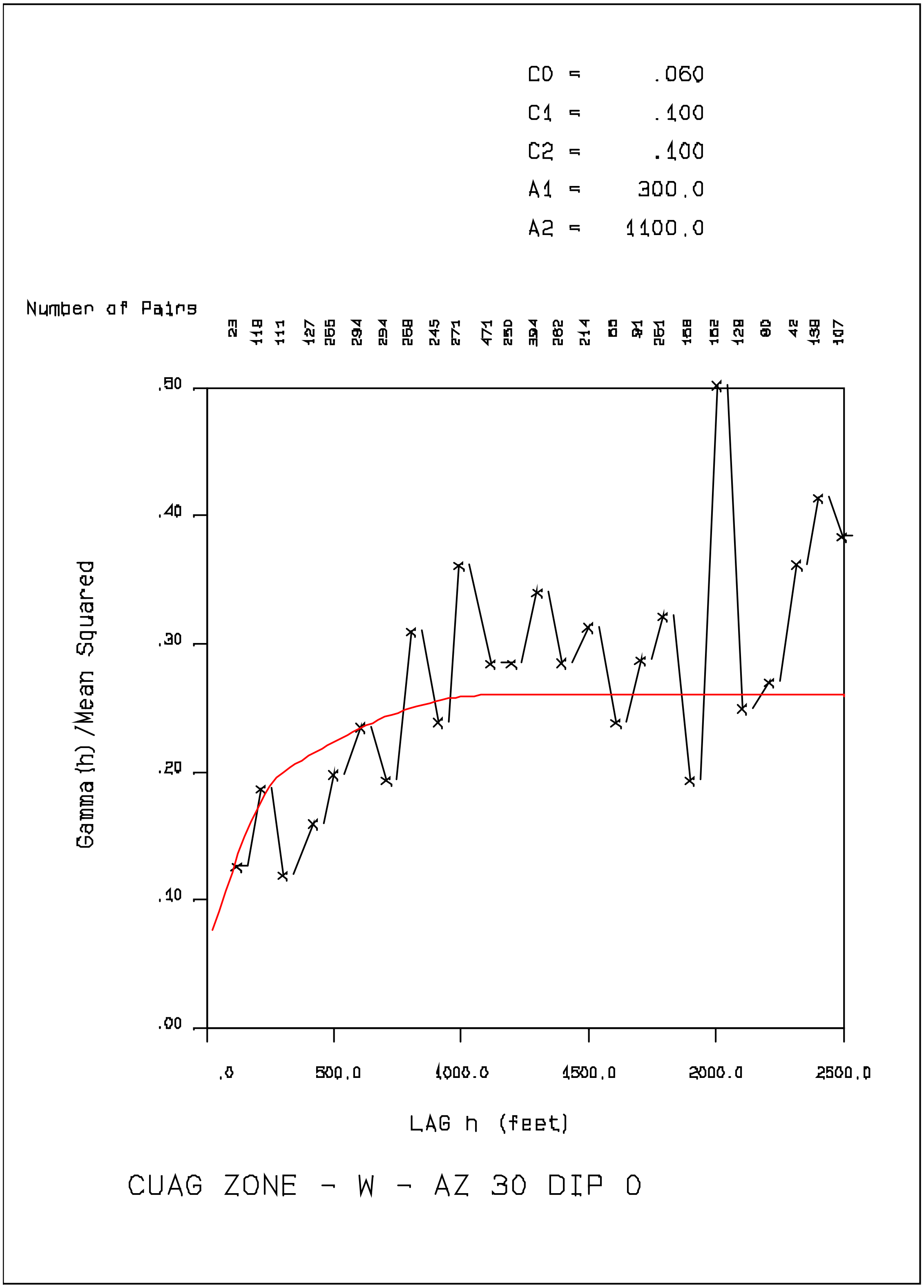


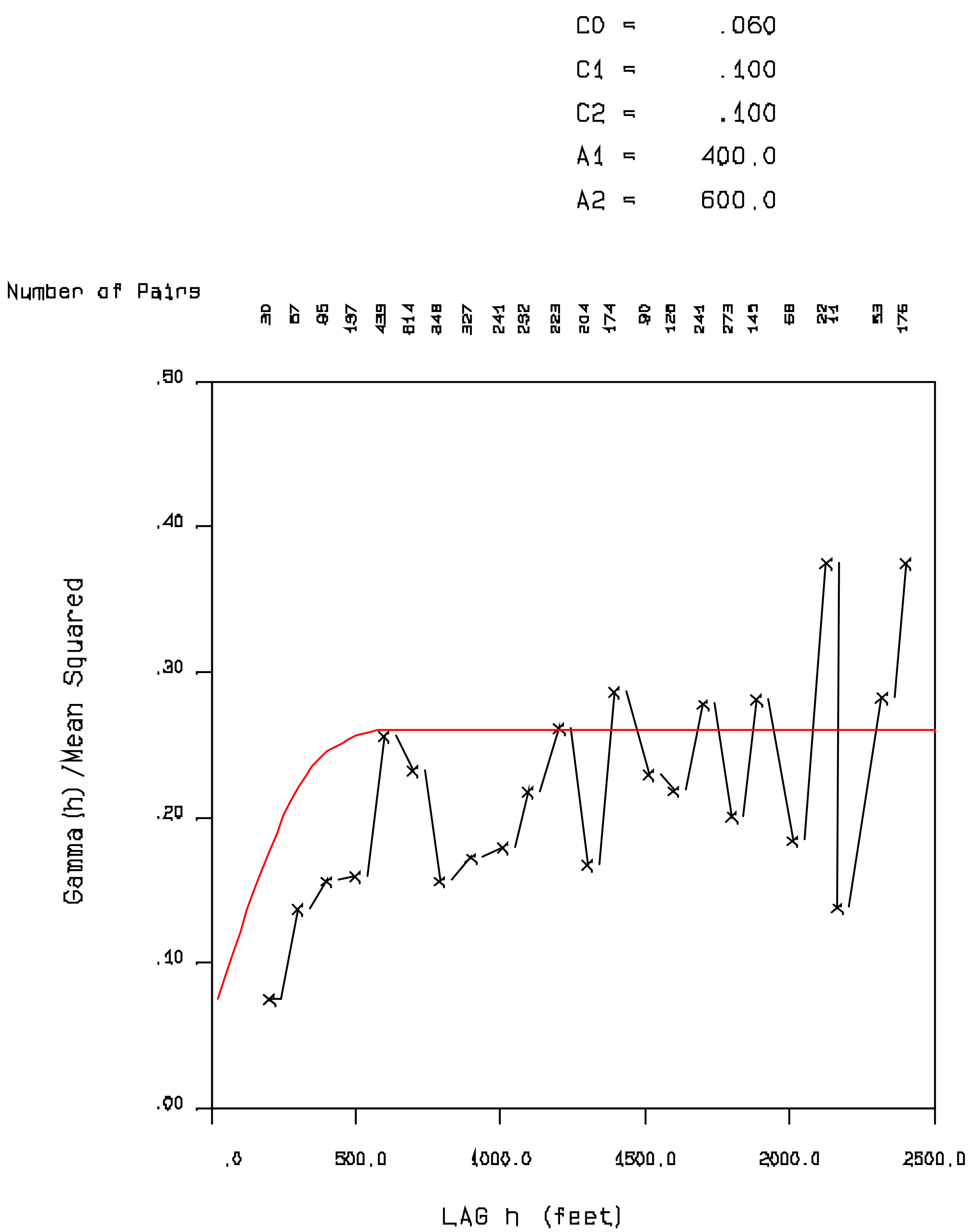
C0 = .060
C1 = .150
C2 = .140
A1 = 300.0
A2 = 500.0
Number of Pairs
58 203 474 238 418 444 288 455 243 360 228 114 117 91 88 34 52
Gamma (h) /Mean Squared
.50
.40
.30
.20
.10
.00
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
MO ZONE - AG - AZ 330 DIP 0



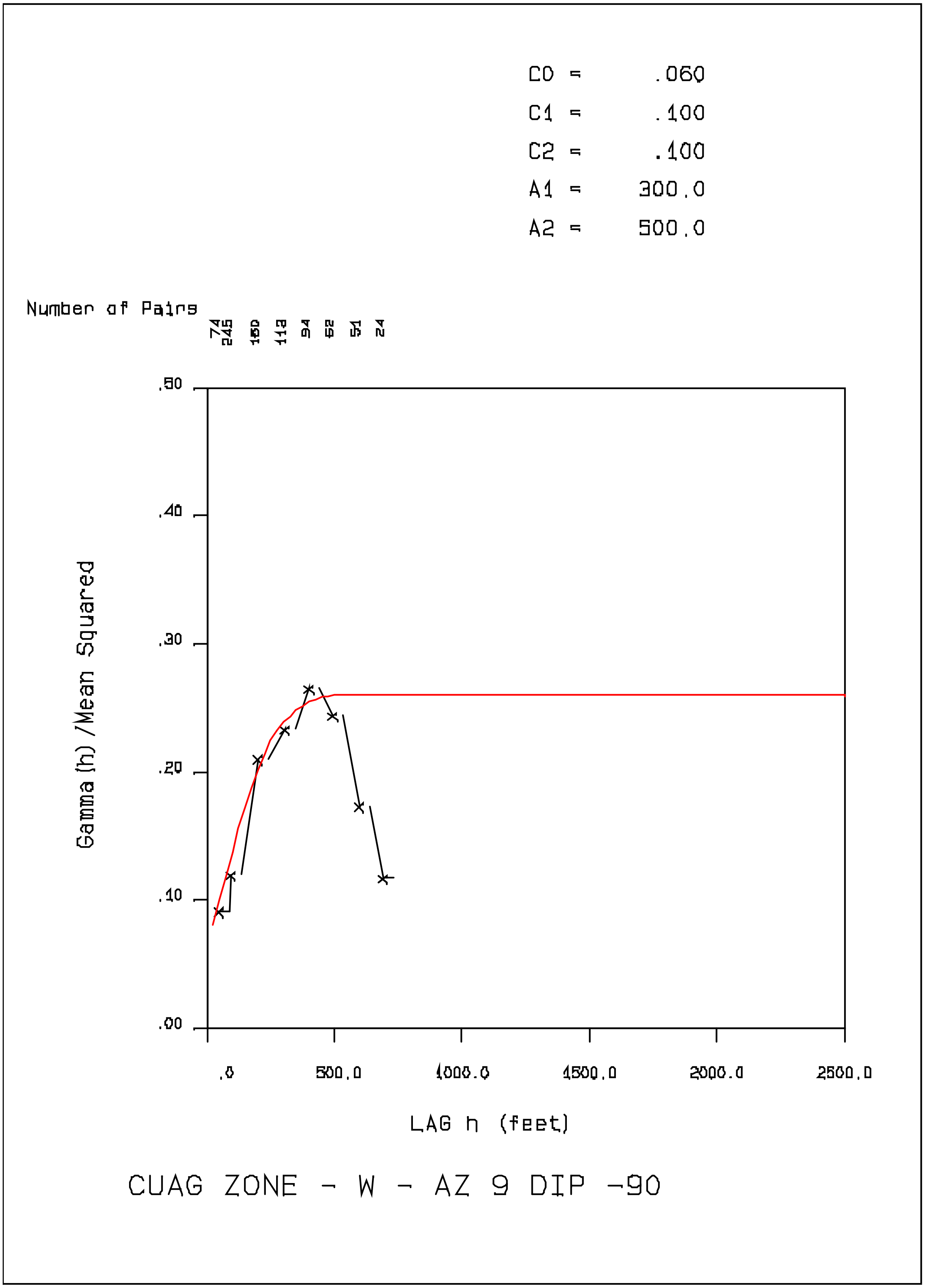

MO ZONE - AG - AZ 0 DIP -90

4.7 - W in CuAg Zone

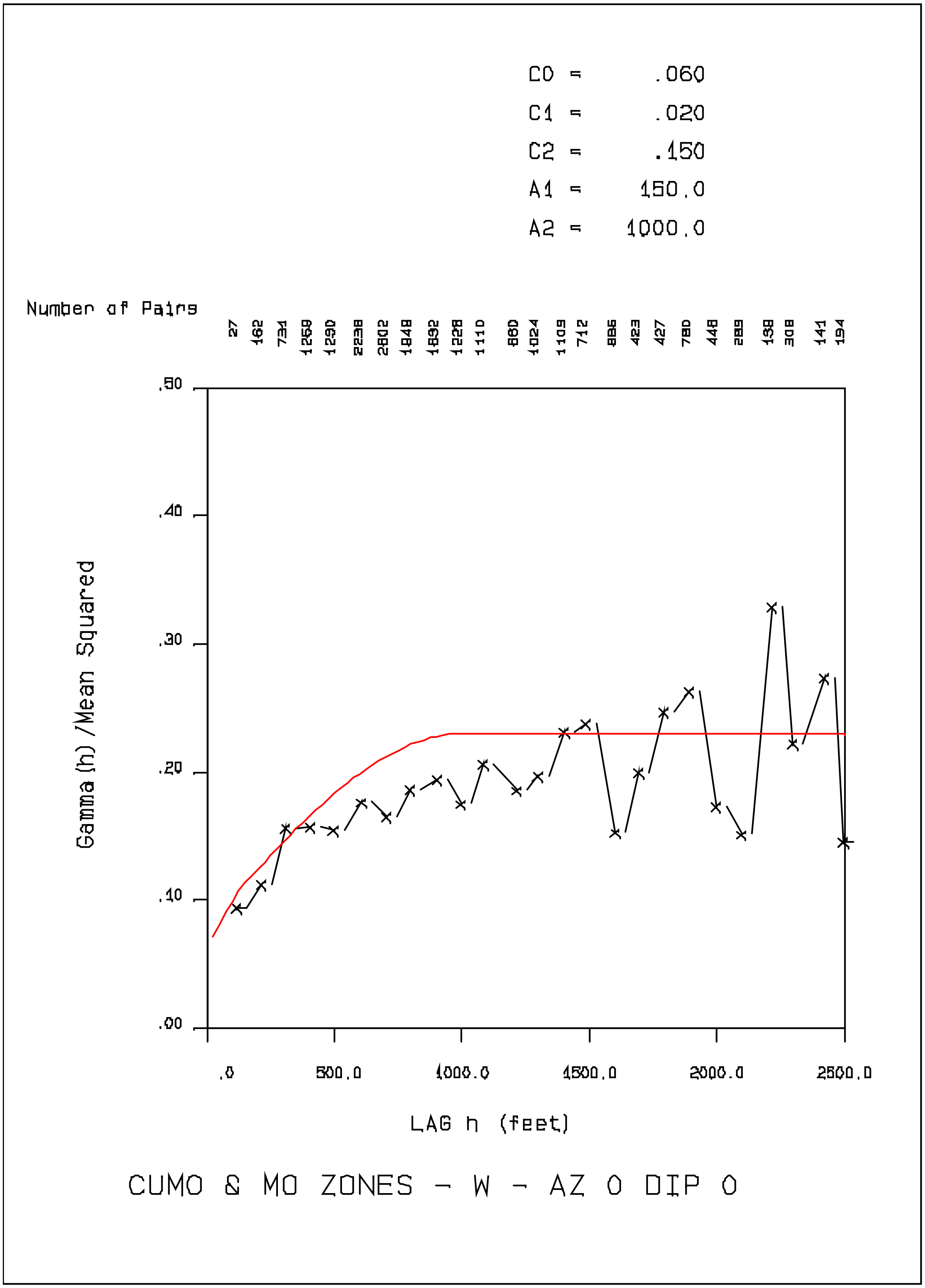


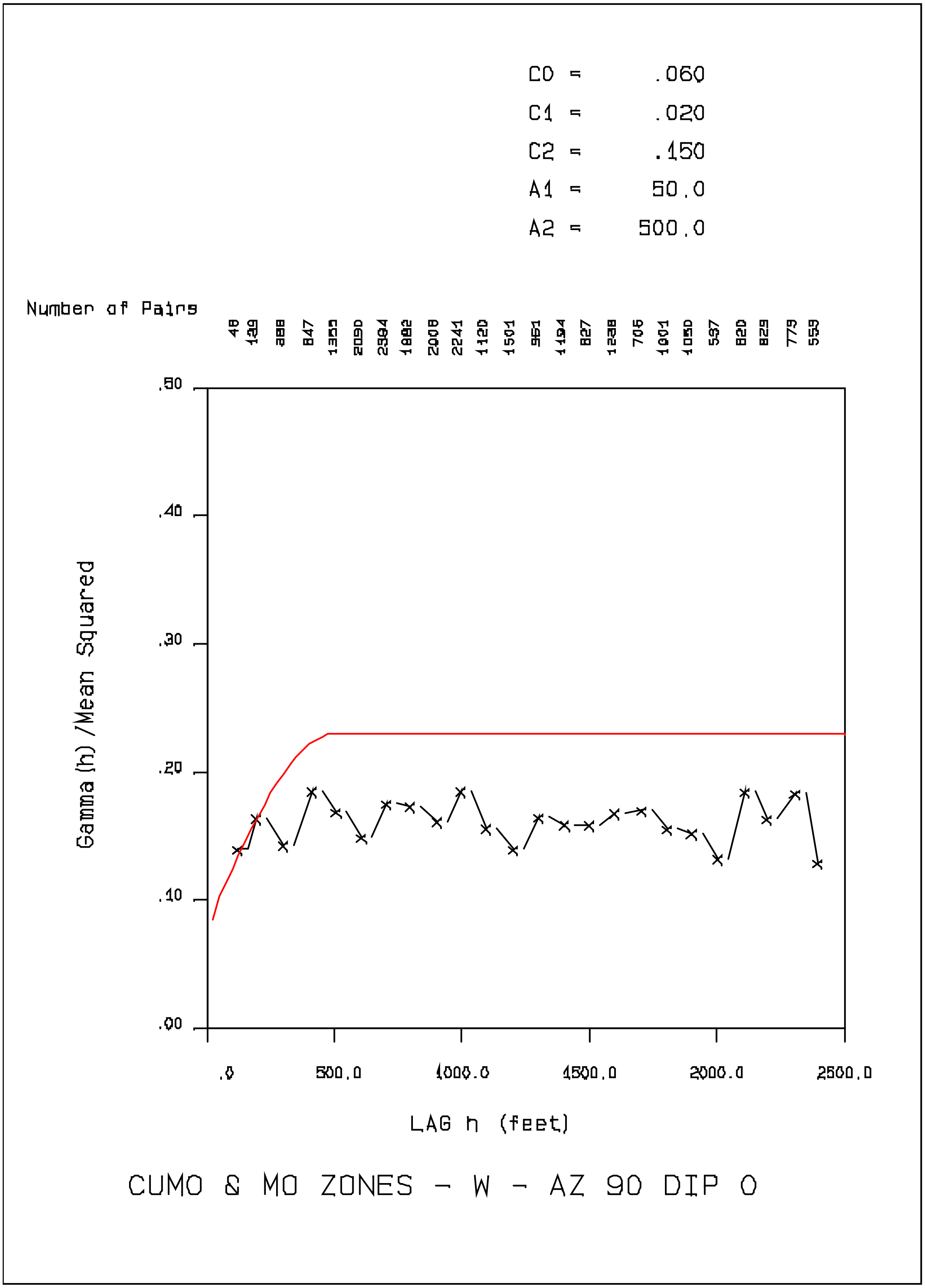



CUAG ZONE - W - AZ 300 DIP 0

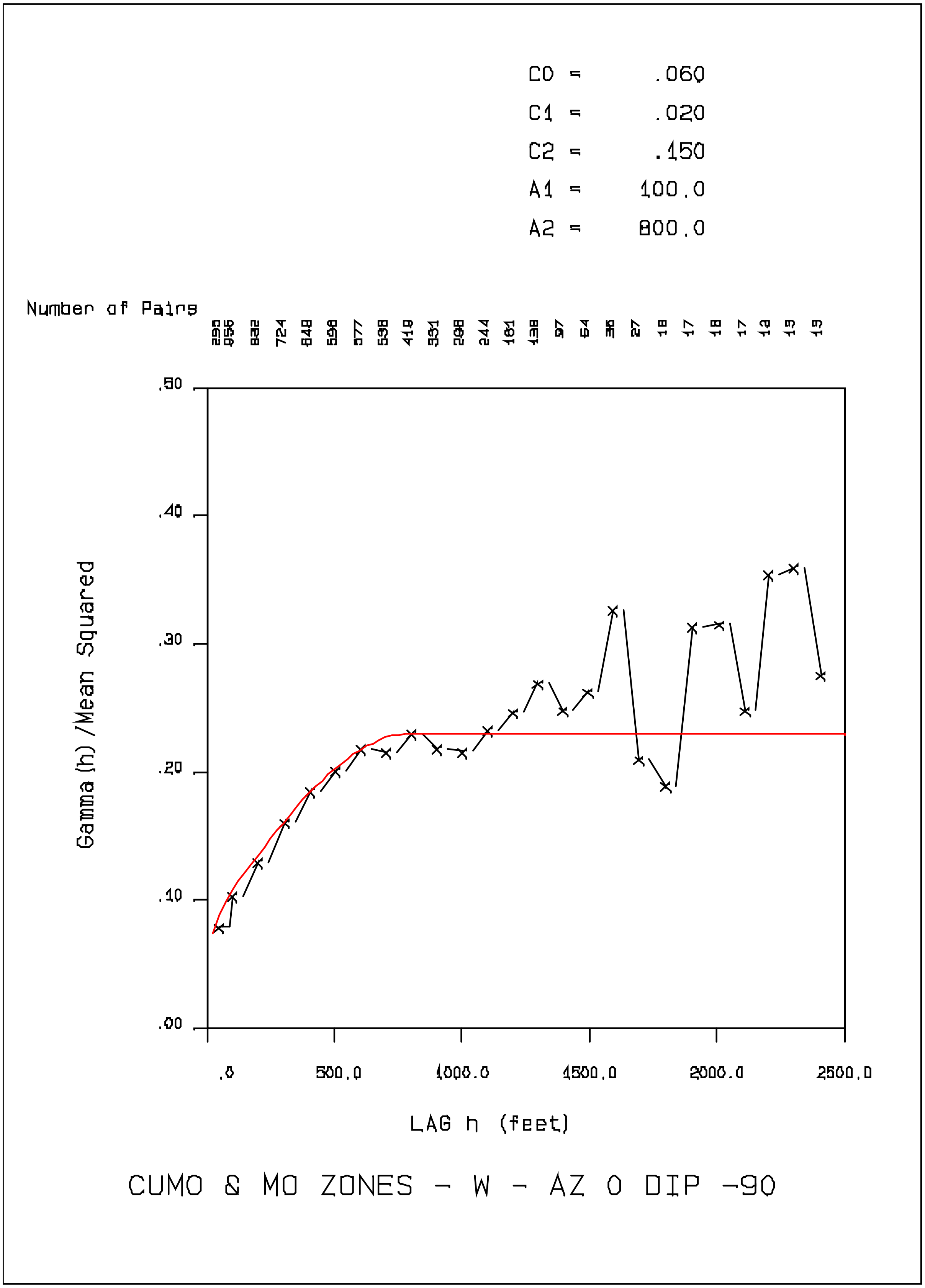

C0 = .060
C1 = .100
C2 = .100
A1 = 300.0
A2 = 500.0
Number of Pairs
74
245
150
113
94
62
51
24
.50
.40
.30
.20
.10
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUAG ZONE - W - AZ 9 DIP -90

4.8 - W in CuMo and Mo Zones





C0 = .060
C1 = .020
C2 = .150
A1 = 50.0
A2 = 500.0
Number of Pairs
48 139 388 847 1355 2090 2394 1882 2008 2241 1120 1501 951 1194 827 1238 705 1001 1050 537 820 829 773 553
.50
.40
.30
.20
.10
.00
Gamma (h) /Mean Squared
.0
500.0
1000.0
1500.0
2000.0
2500.0
LAG h (feet)
CUMO & MO ZONES - W - AZ 90 DIP 0


C0 = .060
C1 = .020
C2 = .150
A1 = 100.0
A2 = 800.0
Number of Pairs
Gamma (h) /Mean Squared
LAG h (feet)
CUMO & MO ZONES - W - AZ 0 DIP -90