Rosemont Copper Project
Updated Feasibility Study

Volume I
NI 43-101 Technical Report

Prepared for:

AUGUSTA RESOURCE CORPORATION

ARCHITECTURE

ENGINEERING

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NI 43-101 Technical Report
For The Rosemont Copper Project
Updated Feasibility Study
Pima County, Arizona, USA

Date: January 14, 2009
(Amended March 17, 2009)

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1.3 SUMMARY

1.3.1 Property

The Rosemont Property is primarily a copper mining project with appreciable amounts of molybdenum and silver by-products. Rosemont is being developed by Augusta Resource Corporation (Augusta). The Property consists of 132 patented lode claims comprising about 1969 acres (797 hectares) and a contiguous package of 949 unpatented lode mining claims comprising more than 12,000 acres (4,860 hectares) which surround the core of patented claims. There are also 10 blocks of fee land associated with the property, consisting of a number of individual parcels that enclose an additional 911 acres (369 hectares). The area covered by patented claims, unpatented claims and fee land totals approximately 15,000 acres (6,070 hectares), and is situated within the historic Helvetia Mining District on the northwestern flank of the Santa Rita Mountain Range and the Rosemont Mining District on the northeastern flank of the Santa Rita Mountain Range.

Mining activity in the Helvetia and Rosemont Mining Districts dates to the mid 1800s, and by the 1880s production from mines on both sides of the Santa Rita Mountains supported the construction and operation of the Columbia Smelter at Helvetia, on the western side, and the Rosemont Smelter in the Rosemont Mining District on the eastern side. Production ceased in 1951 after production of about 227,300 tons of ore containing an estimated 17.3 million pounds of copper, 1.1 million pounds of zinc and 180,760 ounces of silver.

The copper mineralization of the Rosemont deposit is primarily sulfide with a cap of oxide copper close to the surface. The sulfide and oxide ore will be mined through conventional open pit mining techniques. Sulfide ore will be processed by crushing, grinding, and flotation to produce a copper concentrate product and a molybdenum concentrate product for market. The run of mine (ROM) oxide ore will be leached and the resulting leach solution processed through a solvent extraction and electrowinning facility to produce a copper cathode product for market.

1.3.2 Location

The Rosemont copper-molybdenum-silver deposit is located in Pima County, Arizona, USA on the northeastern flank of the Santa Rita Mountains approximately 30 miles southeast of the city of Tucson Arizona. The property occupies flat to mountainous topography at a surface elevation ranging from 4,000 feet to 6,290 feet and at geographical coordinates of approximately 31° 50’ N and 110° 45’ W.

1.3.3 Ownership

The Rosemont deposit is the principal known area of mineralization on the Rosemont property, a group of patented mining claims, unpatented mining claims and fee land that in aggregate total approximately 15,000 acres (6,100 hectares). Augusta first became interested in the Rosemont deposit in 2005 and after completing a two phase drilling program in 2005 and 2006, Augusta completed the purchase of a 100 percent interest in the property in March 2006. The purchase is subject to a 3% Net Smelter Return (NSR).

Augusta maintains offices in Denver, Colorado, USA, and Vancouver, British Columbia, Canada. The company is traded on the American Stock Exchange and the Toronto Stock Exchange under the symbol AZC and on the Frankfurt Stock Exchange under the symbol A5R.

1.3.4 Geology and Mineralization

The Rosemont deposit is a typical representative of the porphyry copper class of deposits. Similar to many other southwestern USA deposits in this class, Rosemont consists of broad-scale skarn mineralization developed in Paleozoic-aged carbonate sedimentary rocks adjacent to their contact with quartz-latite or quartz-monzonite porphyry intrusive rocks. The deposit has been extensively drilled using diamond core holes. Broadly disseminated sulfide mineralization occurs in the Paleozoic units. Near surface weathering has resulted in the oxidation of the sulfides in the overlying Mesozoic units.

1.3.5 Exploration and Sampling

Augusta has recently completed a 20-hole, 17,522 foot diamond drilling program, along with the sampling of 10 previously drilled geotechnical holes. Previously in 2006, Augusta completed a 40-hole, 68,727 foot diamond drilling program on the deposit, consisting of resource, geotechnical, and metallurgical holes. In 2005, Augusta carried out a 15-hole, 27,402 foot diamond drilling program. The results of all of these drilling programs have been integrated with approximately 210,000 feet of previous drilling, conducted by other companies prior to Augusta’s involvement, to estimate the mineral resources presented in this report. This work was incorporated into an updated mineral resource statement provided in a WLRC Technical Report dated December 4, 2008.

1.3.6 Mineral Resource and Mineral Reserve Estimates

A block grade model of the Rosemont deposit was constructed using MEDSystem® software using a geologic model developed in Gemcom® by Augusta personnel and contract geologists. Statistical studies were conducted to identify outliers to the distribution of assays and to estimate the ranges of influence for block grade estimation. Block grade estimations were conducted by rock type using 50-foot composited data and ordinary kriging interpolation methods. Blocks were also classified into measured, indicated and inferred resources in a manner that conforms to Canadian National Instrument 43-101 standards. The mineral resource estimation work was performed by or under the direction of Mr. William Rose, P.E., WLR Consulting Inc.'s (WLRC's) Principal Mining Engineer and an independent Qualified Person under the standards set forth by Canadian NI 43-101.

Updated measured and indicated mineral resource estimates for the Rosemont deposit are summarized in Tables 1-1 and 1-2, respectively. The combined measured and indicated mineral resource estimates are presented in Table 1-3. Inferred mineral resource estimates are shown in Table 1-4. US units are used in these estimations, where tons refer to short tons (2000 lbs). The mineral resource estimates contained herein are effective as of October 22, 2008.

Table 1-1 Rosemont Deposit Measured Mineral Resources

Material / Cutoff (% Cu)	Ktons	% Cu	% Mo	Ag Oz/ton	% CuEqv*	lbs Cu (millions)	lbs Mo (millions)	oz Ag (millions)	lbs CuEqv* (millions)
Oxides:									
0.10	21,600	0.20	-	-	0.20	85	-	-	85
0.15	14,600	0.23	-	-	0.23	68	-	-	68
0.20	7,500	0.30	-	-	0.30	45	-	-	45
Mixed:									
0.15	4,900	0.65	0.007	0.08	0.78	64	0.7	0.4	76
0.20	4,800	0.66	0.007	0.08	0.79	64	0.7	0.4	76
0.25	4,700	0.67	0.007	0.08	0.80	63	0.7	0.4	75
0.30	4,500	0.69	0.007	0.08	0.82	62	0.6	0.4	73
Sulfides:									
0.15	132,300	0.50	0.016	0.14	0.78	1,330	42.3	18.4	2,060
0.20	119,100	0.54	0.016	0.15	0.82	1,280	38.1	17.6	1,950
0.25	106,900	0.58	0.017	0.16	0.87	1,230	36.4	16.6	1,870
0.30	96,100	0.61	0.017	0.16	0.91	1,170	32.7	15.6	1,750

* Equivalency based on prices of $1.25/lb Cu, $18.00/lb Mo and $8.50/oz Ag, with no applied recovery factors.

Table 1-2 Rosemont Deposit Indicated Mineral Resources

Material / Cutoff (% Cu)	Ktons	% Cu	% Mo	Ag Oz/ton	% CuEqv*	lbs Cu (millions)	lbs Mo (millions)	oz Ag (millions)	lbs CuEqv* (millions)
Oxides:									
0.10	81,700	0.20	-	-	0.20	332	-	-	332
0.15	51,400	0.25	-	-	0.25	260	-	-	260
0.20	27,400	0.33	-	-	0.33	180	-	-	180
Mixed:									
0.15	34,300	0.49	0.005	0.05	0.58	334	3.4	1.5	394
0.20	33,500	0.50	0.005	0.05	0.58	332	3.3	1.5	391
0.25	32,200	0.51	0.005	0.05	0.59	326	3.2	1.5	383
0.30	29,400	0.53	0.005	0.05	0.62	311	2.9	1.4	363
Sulfides:									
0.15	464,500	0.44	0.014	0.11	0.68	4,120	130.1	52.0	6,340
0.20	404,700	0.48	0.015	0.12	0.74	3,910	121.4	49.0	5,990
0.25	351,200	0.52	0.016	0.13	0.80	3,680	112.4	45.7	5,610
0.30	305,200	0.56	0.016	0.14	0.84	3,430	97.7	42.1	5,120

* Equivalency based on prices of $1.25/lb Cu, $18.00/lb Mo and $8.50/oz Ag, with no applied recovery factors.

Table 1-3 Rosemont Deposit Combined Measured and Indicated Mineral Resources

Material / Cutoff (% Cu)	Ktons	% Cu	% Mo	Ag Oz/ton	% CuEqv*	lbs Cu (millions)	lbs Mo (millions)	oz Ag (millions)	lbs CuEqv* (millions)
Oxides:									
0.10	103,400	0.20	-	-	0.20	417	-	-	417
0.15	66,000	0.25	-	-	0.25	328	-	-	328
0.20	35,000	0.32	-	-	0.32	224	-	-	224
Mixed:									
0.15	39,100	0.51	0.005	0.05	0.60	398	4.1	1.9	471
0.20	38,300	0.52	0.005	0.05	0.61	396	4.0	1.9	467
0.25	36,900	0.53	0.005	0.05	0.62	389	3.9	1.9	458
0.30	33,900	0.55	0.005	0.05	0.64	373	3.5	1.8	436
Sulfides:									
0.15	596,800	0.46	0.014	0.12	0.70	5,440	172.4	70.4	8,410
0.20	523,800	0.50	0.015	0.13	0.76	5,190	159.5	66.6	7,940
0.25	458,100	0.54	0.016	0.14	0.82	4,910	148.8	62.3	7,480
0.30	401,300	0.57	0.016	0.14	0.86	4,600	130.4	57.7	6,870

* Equivalency based on prices of $1.25/lb Cu, $18.00/lb Mo and $8.50/oz Ag, with no applied recovery factors.

Table 1-4 Rosemont Deposit Inferred Mineral Resources (Excludes Measured & Indicated)

Material / Cutoff (% Cu)	Ktons	% Cu	% Mo	Ag Oz/ton	% CuEqv*	lbs Cu (millions)	lbs Mo (millions)	oz Ag (millions)	lbs CuEqv* (millions)
Oxides:									
0.10	30,400	0.24	-	-	0.24	147	-	-	147
0.15	17,800	0.33	-	-	0.33	117	-	-	117
0.20	12,700	0.39	-	-	0.39	100	-	-	100
Mixed:									
0.15	21,100	0.35	0.004	0.02	0.41	148	1.7	0.3	175
0.20	19,100	0.37	0.004	0.01	0.43	141	1.5	0.3	164
0.25	14,500	0.42	0.004	0.02	0.48	121	1.2	0.2	139
0.30	12,200	0.45	0.003	0.02	0.49	109	0.7	0.2	121
Sulfides:									
0.15	208,800	0.38	0.007	0.06	0.50	1,600	29.2	12.1	2,110
0.20	160,600	0.45	0.008	0.07	0.59	1,440	25.7	10.9	1,880
0.25	133,800	0.49	0.008	0.08	0.63	1,320	21.4	10.0	1,700
0.30	105,000	0.56	0.008	0.09	0.70	1,170	16.8	8.9	1,470

* Equivalency based on prices of $1.25/lb Cu, $18.00/lb Mo and $8.50/oz Ag, with no applied recovery factors.

Augusta's 2008 drilling campaign at the Rosemont deposit has increased both the quantity and confidence level of the estimated mineral resources, which presently totals about 562 million tons of measured and indicated sulfide mineral resources grading 0.50% Cu, 0.015% Mo, and 0.12 ounces per ton Ag, at a 0.20% Cu cutoff. An additional 180 million tons of inferred sulfide mineral resources are estimated at a grade of 0.44% Cu using the same cutoff. Augusta's recent drilling program was successful in converting significant tonnages of inferred material into measured and indicated classifications. *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*

In addition, geologic and metallurgical studies conducted by Augusta have shown the potential for considering the oxide copper mineralization that overlies the sulfide deposit. Estimated measured and indicated oxide mineral resources total nearly 103 million tons grading 0.20% Cu, at a 0.10% Cu cutoff. An additional 30 million tons of inferred oxide mineral resource are estimated at a grade of 0.24% Cu, using the same cutoff.

The classification of currently inferred sulfide and oxide mineral resources can potentially be improved with further drilling. Additional mineral resources may be found in extensions to the north and east of the Rosemont deposit. Mineralization also is known to occur in the Broadtop Butte, Copper World and Peach-Elgin deposits on the

Rosemont property, which could potentially add to the total mineral resource base of the Rosemont area.

The Rosemont deposit's proximity to the topographic surface makes it amenable to open pit mining methods. Lerchs-Grossman analyses of economic pit limits were conducted using a variety of metal prices and operating costs. A base case mining pit shell generated at metal prices of $1.75/lb Cu, $15.00/lb Mo and $10.00/oz Ag and anticipated operating costs was used to design an ultimate pit for mineral reserve estimation and subsequent mine planning. The mineral reserve estimation work was performed by or under the direction of Mr. Robert Fong, P. Eng., Moose Mountain Technical Services (MMTS) Principal Mining Engineer and an independent Qualified Person under the standards set forth by Canadian NI 43-101.

Rosemont mineral reserves have been estimated from only measured and indicated mineral resources; all inferred resources have been treated as waste. Net Smelter Returns (NSRs) were computed as a means of aggregating the net recoverable value of the three primary metals in sulfide rock types; only copper was used in calculating oxide NSRs. No recovery of molybdenum and silver is projected from oxide ore leaching and only quartz monzonite porphyry (QMP), andesite and arkose rock types were considered as potential oxide leach ore (no NSRs were computed for other oxide rock types). An internal NSR cutoff of $3.56/ton was used for sulfide mill ore and $2.19/ton was used for oxide leach ore. Table 1-5 summarizes the estimated mineral reserves for the Rosemont deposit as of the date of this report.

Table 1-5 Rosemont Mineral Reserves

Classification	Sulfides >= 3.56 $/ton NSR Cutoff					Oxides >= 2.19 $/ton NSR		
	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %
Proven	141,999	14.19	0.48	0.015	0.13	16,250	3.91	0.18
Probable	404,339	13.12	0.45	0.015	0.11	53,724	3.77	0.17
Total	546,338	13.40	0.45	0.015	0.12	69,974	3.80	0.17

At prices of $1.75/lb Cu, $15.00/lb Mo and $10.00/oz Ag, combined proven and probable sulfide mineral reserves within the designed Rosemont ultimate pit total nearly 546 million tons grading 0.45% Cu, 0.015% Mo and 0.12 oz/ton Ag. Proven and probable oxide mineral reserves total about 70 million tons grading 0.17% Cu. The pit contains a total of about 1.85 billion tons of material, of which 616 million tons are mineral reserves and 1.23 billion tons are waste rock, resulting in a stripping ratio of 2.0:1 (tons waste per ton of ore). Contained metal in the sulfide (proven and probable) mineral reserves is estimated at 4.93 billion pounds of copper, 161 million pounds of molybdenum and 65 million ounces of silver. Contained metal in proven and probable oxide mineral reserves is estimated at 241 million pounds of copper. *All of the mineral reserve estimates reported above are contained in the mineral resource estimates presented in Tables 1-1 through 1-3.*

The Rosemont ultimate pit contains approximately 54 million tons of inferred sulfide mineral resources and nearly 8 million tons of inferred oxide mineral resources that are above respective sulfide and oxide NSR cutoffs of $3.56/ton and $2.19/ton. These resources are included in the waste estimates presented in the previous paragraph. *Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Inferred mineral resources have a great amount of uncertainty as to their existence and as to whether they can be mined legally or economically. It cannot be assumed that all or any part of inferred mineral resources will ever be upgraded.*

1.3.7 Mining

Six internal mining phase designs were also developed, bringing the total number of phases to seven. A production scheduling analysis was conducted to determine preproduction and long-term waste rock stripping rates. This scheduling was based on a milling rate of 75,000 tons per day (tpd), operating 365 days per year, for a total sulfide ore feed of 27.375 million tpy. Oxide ore will be delivered to the leach pad as it is encountered during the course of mining. Mine and plant operations will be scheduled for continuous coverage, using two 12-hour shifts per day, seven days per week. Ramp-up schedules were developed for preproduction stripping and sulfide ore milling during the first year of plant operations.

Mining sequence plans were developed on a quarterly basis through the end of Year 2 and on an annual basis through Year 7. Additional plans include mining progress through the end of Year 10, Year 15 and Year 21 (end of mining). A production schedule was then generated from these mining plans, indicating a project operating life of 20.1 years using only proven and probable mineral reserves. Peak mining rates of 318,000 tpd of total material (ore and waste) will be realized in Year 1. Typical mining rates during Years 3-6 will be 224,000 tpd of waste rock and oxide ore, or 299,000 tpd of total material (including 75,000 tpd of sulfide ore). Minimum oxide ore will be recovered after Year 6, and typical mining rates during Years 7 to 10 will be 299,000 tpd of ore and waste. A 15-month preproduction stripping program will be required to open the deposit up for initial ore deliveries to the mill.

Overburden and other waste rock encountered in the course of mining will be placed into a waste rock storage (WRS) area located to the southeast and south of the planned open pit and into the dry stack tailings area, where dewatered mill tailings will be placed behind waste rock containment buttresses. The dry stack tailings area is north of the WRS area and east-northeast of the pit. The oxide ore heap leach pad will be located between the dry stack tailings area and the initial WRS area.

The proposed pit operations will be conducted from 50-foot-high benches using large-scale equipment, including up to: three 12.25-inch-diameter rotary blasthole drills, three 70-cu-yd electric mining shovels, two 36-cu-yd front-end loaders, twenty four 320-ton off-highway haul trucks, five 580- to 850-hp crawler dozers, three 500-hp rubber-tired dozers, three 270- to 500-hp motor graders and three 30,000-gallon off-highway water trucks. Four rotating crews will be used for continuous operator and maintenance coverage. Peak manpower (and equipment) levels will occur in Years 11-15, with 45 supervisory and technical personnel, 150 workers in mine operations and 79 in mine maintenance, totaling 274 people.

1.3.8 Metallurgical Testing

The earliest existing records of metallurgical testing are from the period 1974 - 1975, at which time grinding and flotation tests were performed. In the first half of 2006, Augusta initiated test work to provide a better understanding of the metallurgy of the Rosemont deposit and establish the design criteria for the design of a process facility.

The Rosemont sulfide ore was tested to determine grinding and flotation criteria. The test work indicates a process of crushing and grinding the ore to 80% passing 105 micron size distribution followed by bulk flotation to recover copper and molybdenite minerals. A molybdenite concentration circuit to treat the bulk flotation concentrate will be able to produce a molybdenite concentrate.

The Rosemont oxide ore was tested to determine heap leaching design criteria. The test work indicates that a heap leach process on run of mine ore can recover the copper into a pregnant leach solution (PLS) that can be subsequently processed in a solvent extraction – electrowinning (SX-EW) circuit.

1.3.9 Process Flowsheet

Both sulfide and oxide copper ore will be processed. Sulfide ore will be transported from the mine to the primary crusher by off-highway haulage trucks then conveyed to the concentrator facilities. Oxide ore will be transported from the mine to a run of mine heap leaching facility by the off-highway haulage trucks. Copper concentrate produced at the concentrator facility will be loaded into highway haul trucks and transported to a concentrate smelter and metal refinery. Molybdenum concentrate produced at the concentrator facility will be bagged and loaded onto trucks for shipment to market. Oxide ore will be leached with acidic solution and the leach solution will be processed using solvent extraction electrowinning (SX-EW) technology to produce high purity cathode copper plates (cathodes). The copper cathodes will be loaded onto trucks for shipment to market.

The process selected for recovering the copper and molybdenite minerals can be classified as “conventional”. The sulfide ore will be crushed and ground to a fine size and processed through mineral flotation circuits.

The process selected for the recovery of copper from the oxide ore can be classified as “conventional”. The oxide ore will be heap leached and the copper recovered from the leach solution using solvent extraction – electrowinning technology.

1.3.10 Extraction Rates

Sulfide ore metal recoveries for operating years 1 through 3 are indicated by the test work to be for copper (85%), gold (73%), and silver (77%) in a copper concentrate, and molybdenum (72%) in a molybdenite concentrate.

Sulfide ore metal recoveries for operating years 4 through 7 are indicated by the test work to be for copper (83%), gold (73%), and silver (76%) in a copper concentrate, and molybdenum (65%) in a molybdenite concentrate.

Sulfide ore metal recoveries for subsequent years are indicated by the test work to be for copper (84%), gold (73%), and silver (78%) in a copper concentrate, and molybdenum (56%) in a molybdenite concentrate.

Oxide ore copper recovery is indicated by the test work to be 65%.

1.3.11 Process Reagents

Reagent consumption rates for the full scale plant operation have been estimated from the test results. The reagents that will be used in the sulfide circuit are considered to be "conventional". Consumption rates for collectors is estimated to be about 0.164 lbs/ton of sulfide ore, lime about 1.797 lbs/ton, and modifiers, frothers and other about 0.166 lbs/ton. The molybdenite recovery circuit will consume about 0.2125 lbs/ton of sulfide ore in modifiers, collectors, and frothers.

In the oxide ore leaching circuit, sulfuric acid consumption is estimated to be 30.0 lbs/ton ore. In the SX-EW circuit, extractant consumption is estimated to be 0.0002 lbs/lb cathode copper, diluent at 0.001 lbs/lb, all other electrowinning additives 0.0107 lbs/lb, and solution filtering additives at 0.08 lbs/lb.

1.3.12 Power

The power supply for the Rosemont mine and process facilities will be administered by Tucson Electric Power (TEP) under a shared service agreement with TRICO, a local cooperative. The estimated connected load for the project is 139 MW, and will be supplied by a minimum of a 138 kV line to site. The estimated operating load for the project is approximately 106 MW.

The "Option D" proposed by Rosemont, accesses initial construction power from an existing 46 kV line at the Greaterville substation (4.5 miles new line).

For the higher power load required to operate the mine, new construction of 16 miles of 138 kV line is required. The first 4 miles upgrade the TEP transmission system to a new Rosemont substation at or near Wilmot Junction (Section 25). These 4 miles provide a system upgrade to allow a cross tie between the Vail and South Substations. Either South or Vail could provide source to the new Rosemont Substation. From Rosemont substation, a new 12 mile long radial 138 kV line would be built. This radial line is assumed retained by Rosemont. This "Option D" was developed by KR Saline engineers of Arizona to efficiently utilize planned and scheduled system upgrades as included in long term planning documents on file with the Arizona Corporation Commission.

The Arizona State Line-Siting-Committee has established the process to review new power line routes for Rosemont, and the preferred routing and permit application is scheduled for completion during mid 2009.

1.3.13 Water

The fresh water requirements for the Rosemont facilities are about 5,000 acre-feet per year with a peak demand of 5,000 gallons per minute (gpm) and an average demand of 3,370 gpm. All gallons in this report are United States gallons. Water will come from wells located west of the Santa Rita Mountains and will be pumped to the fresh water and fire water storage tank located at the Rosemont site.

The daily usage for potable water is about 17,000 gallons per day, fresh water makeup is 4.8 million gallons per day, and the recycle process water is 37 million gallons per day. There is also a fire water distribution system throughout the mine site.

Augusta has committed to recharging the Santa Cruz aquifer with available Central Arizona Project (CAP) water.

A summary description of the fresh water system is included in Section 1.25.7 of this Technical Report.

1.3.14 Permits

Permitting for the Rosemont Copper Project involves federal approvals and requires compliance with the National Environmental Policy Act (NEPA). This in turn requires an Environmental Impact Statement (EIS) and compliance with the Endangered Species Act (ESA) and the National Historic Preservation Act (NHPA). A Mine Plan of Operation was submitted to the US Forest Service on July 11, 2007 to initiate the EIS and start the permitting process. Major federal permits required to construct and begin operation of the Rosemont Project includes a Clean Water Act (CWA) Section 404 permit for discharge of fill material to onsite washes. Major state permits include an aquifer protection permit, a 401 Certification, and an Arizona Pollution Discharge Elimination System (AZPDES) general storm water permit. The only major local permit required is a Pima County Clean Air Act (CAA) Title V air quality permit. Other permits which do not affect the timeline for project permitting and subsequent start up include explosives permits, nuclear instrumentation licenses, hazardous waste identification, tracking numbers and spill control plans. A list of permits is provided in Section 1.25.8.

1.3.15 Operating Costs

The mine operating costs were derived from equipment hours and cycle times developed by Moose Mountain from their Mine Plan. Rebuild costs for major equipment were generated from vendor supplied component replacement schedules and URS Washington Mining Division's data base for similar projects and equipment. Mining costs supplied by others were checked by URS who built the estimate and was the QP. The average life of mine operating costs for the mining operation is $0.83 per ton mined. These costs include: clearing of vegetation, removal of topsoil, drilling, blasting, loading, hauling, road and dump maintenance, regrading, mine operations supervision, craft labor and subcontractor costs.

Mill process operating costs in Year 2 average $3.34/ton of mill ore which includes crushing and conveying, grinding and classification, flotation and regrind, concentrate thickening, filtration and dewatering, tailings disposal and mill ancillary services. In addition, these operating costs are broken into the major categories of labor, power, reagents, maintenance, supplies and services.

Operating costs for the SX-EW process in Year 2 average $0.92/ lb. of cathode copper which includes heap leach pad, solvent extraction, tank farm, electrowinning and SX-EW ancillary services. In addition, these operating costs are broken into the major categories of labor, power, reagents, maintenance, supplies and services.

The average operating cost for the supporting facilities and general administrative expenses in Year 2 is $0.27/ton of sulfide ore. The supporting facilities include laboratory, safety and environmental, accounting, human resources, security and the general manager's office.

The overall site direct operating cost estimate by cost center in Year 2 is shown in Table 1-6 below. All costs are estimated in fourth quarter 2008 US dollars at an accuracy of ± 10%.

Table 1-6 Summary of Operating Costs
Based on Year 2 of Operations

	Annual Cost ($000)
Mining	70,141
Mill Operations	91,452
SX-EW Operations	18,398
Support Facilities and G&A	8,974
Total	188,965

1.3.16 Capital Cost Estimate

The total capital cost estimate to design, construct and commission the Rosemont facilities is estimated to be $897.2 million for the combined sulfide and oxide plant. The estimate includes the direct field cost for constructing the project at $712.7 million as well as $184.5 million for the indirect costs associated with the design engineering, procurement and construction, commissioning, spare parts, contingency and Owner's cost. An incremental cost for the oxide plant was estimated to be $64.7 million with $53.6 million for the direct costs and $11.1 million for indirect costs. All costs are expressed in fourth quarter 2008 US Dollars at an accuracy of ± 15% with no allowance provided for escalation, interest, foreign currency, hedging, or financing during construction.

1.3.17 Financial Analysis

The Rosemont Project economics were done using a discounted cash flow model. The study evaluated a sulfide concentrate plant with a heap leach SX-EW plant for the treatment of the oxide copper reserves. Costs are in constant fourth quarter 2008 US dollars with no provisions for escalation. The financial indicators examined for the project included the Net Present Value (NPV), Internal Rate of Return (IRR) and payback period (time in years to recapture the initial capital investment). Annual cash flow projections were estimated over the life of the mine based on capital expenditures, production costs, transportation and treatment charges and sales revenue. The life of the mine is 21 years.

The sales revenue is based on the production of three commodities: copper, molybdenum and silver. Gold is also present in the copper concentrates in the form of a saleable by-product credit. The estimates of capital expenditures and site production costs have been developed specifically for this project.

Metal sales prices used in the evaluation are listed in Table 1-7.

Table 1-7 Base Case and Historical Metals Prices

	60/40 Weighted Average *	3 Year Historical Average
Copper	$ 2.47 / pound	$ 3.14 / pound
Molybdenum	$22.70 / pound	$29.05 / pound
Silver	$12.40 / ounce	$13.32 / ounce
Gold	$784.65 / ounce	$723.48 / ounce

**60/40 weighted average of the 36 month historic price and the 24 month futures price forecast*

In addition to the above metal sales price cases, a case of long term metal prices was also evaluated. Long term metal prices were assumed at $1.85/lb Cu, $15.00/lb Mo, $12.00/oz Ag and $750.00/oz Au.

Table 1-8 Long Term Metals Prices

Copper	$ 1.85/lb
Molybdenum	$ 15.00/lb
Silver	$ 12.00/oz
Gold	$750.00/oz

The after tax financial results for the three metal pricing scenarios are shown in Table 1-9.

Table 1-9 Financial Indicators (After Tax)

	Base Case (60/40 split)	Historical 36 Months	Long Term Metal Prices
NPV 0%	4,850.0	6,999.9	2,715.0
NPV 5%	2,417.6	3,628.9	1,200.3
NPV 10%	1,254.2	2,006.2	488.4
IRR	28.5%	37.5%	17.8%
Payback Years	3.1	2.3	5.0

1.3.18 Author's Conclusions

The after-tax IRR is above the Owner's project criteria of 15%, therefore the project should continue to advance with basic engineering and permitting. In the meantime, the copper price should stabilize somewhat, as it is presently below the $1.85/lb price used in this study, although it is not below either the last three (3) years historical plus two (2) years futures average or the three (3) year historical average. Using the spot prices of end of month December 2008 of $1.36/lb Cu, $11.00/lb Mo, $10.79/oz Ag, $869.75/oz Au yields a after-tax IRR of 7.7%.

The downward trend in capital equipment and commodity cost that started in October 2008 is not reflected herein. It may result in even more favorable economics.

1.3.19 Author's Recommendation

The project should proceed with basic engineering and permitting. While that is ongoing, the copper price trend should become more evident following the financial market turmoil of 2008.

1.4 INTRODUCTION AND TERMS OF REFERENCE

Augusta retained a number of contractors, including M3 Engineering and Technology Corporation (M3), to provide a review of prior work on the project and prepare technical and cost information to support a bankable level Feasibility

Study and this Technical Report following the outline as defined in Canada National Instrument NI 43-101 and in compliance with Form 43-101F1. Dr. Conrad Huss, P.E. of M3 Engineering and Technology Corporation (M3) is the Principal Author and Qualified Person responsible for preparation of this report. Dr. Huss has visited the site on numerous occasions prior to this Updated Feasibility Study and is familiar with the site. In addition, the following M3 employees, under the supervision of Dr. Huss, visited the site during the initial Feasibility Study and/or this Updated Feasibility Study on the dates noted.

- Thomas L. Drielick, P.E., Senior Vice President; August 21, 2007
- Rex Henderson, P.E., Project Manager; August 9, 2006, September 27, 2006 and June 12, 2007
- David Moll, P.M.P., Asst. Project Manager, November 13, 2008
- Randy Hensley, Construction Manager, June 12, 2007
- Enrico Laos, P.E., Electrical Supervisor; August 9, 2006
- Tim Oliver, P.E., Environmental Specialist; September 27, 2006
- Daniel Roth, P.E., Reclamation Lead; May 25, 2007, July 11, 2007, November 13, 2008
- Craig Hunt, Civil Lead, November 13, 2008
- David Caldwell, Civil Designer, May 25, 2007 and July 11, 2007
- Tony Ottinger, Civil Designer, May 25, 2007 and July 11, 2007
- Robert Davidson, Project Engineer, August 9, 2006
- Francisco Espinosa, Civil Designer, August 9, 2006

Other contributing authors and Qualified Persons responsible for preparing both this Updated Feasibility Study and this Technical Report include; Mr. William L. Rose, P.E., of WLR Consulting, Inc.; Mr. Robert Fong, P.E., of Moose Mountain Technical Services; Mr. John Ajie, P.E., of URS Washington Division; and Mr. Thomas L. Drielick, P.E., of M3 Engineering and Technology Corp.

Augusta retained WLR Consulting (WLRC – Lakewood, Colorado) to develop and oversee the resource estimate. Mr. William Rose, Principal Mining Engineer, is the Qualified Person responsible for this effort. Mr. Rose visited the site on August 9, 2005. Mr. Rose directed the mineral resource calculations prepared by Mr. Donald Elkin, Principal Geological Engineer of Mine Reserves, Associates, Inc.

Mr. Robert Fong, P.E., of Moose Mountain Technical Services (MMTS – British Columbia, Canada) was contracted to estimate and oversee the calculations of the open pit reserves and to develop the LOM Mine Plan which includes a Lerchs-Grossman analysis, pit design, mine production schedule, mine access and haul roads and waste rock stockpiles. Mr. Robert Fong visited the project site on November 20, 2008.

Mr. John Ajie, P.E. - VP of Engineering and Operations Support, of URS Washington Division (Denver, Colorado) was contracted to supervise and review the development of the mine estimate.

Mr. David Nicholas of Call & Nicholas Incorporated (CNI – Tucson, Arizona) prepared a slope stability study for the pit walls and prepared run of mine (ROM) fragmentation analysis for sulfide and oxide ore and the waste rock. Mr. William Rose, Principal Mining Engineer, has reviewed and incorporated the CNI work into the mine design sections of this report.

Mr. Michael Clarke, previous Augusta Vice President of Exploration, has been on-site numerous times and was responsible for the geologic interpretations for the resource model. Mr. Mark Stevens, current Vice President of Exploration for Augusta, compiled the drill hole data files and prepared the geology section of this Updated Feasibility Study based on earlier published reports and internal reports (Anzalone – 1995, Wardrop – 2005, Augusta – 2007). Mr. Clarke and Mr. Stevens have visited the site on numerous occasions over the last several years.

Tetra Tech of Golden, Colorado and Tucson, Arizona were responsible for the site geotechnical investigations consisting of a site geotechnical study, a geologic hazards assessment, and a baseline geochemical characterization study. Tetra Tech also provided a site water management plan, waste management plan, and oxide leach facilities. Tetra Tech provided the design and material quantities for the leach pad, pregnant leach solution (PLS) pond, raffinate pond, storm water pond, and compliance point dam and M3 estimated the capital cost based on the material quantities.

Tetra Tech was also responsible for preparation of the reclamation and closure plan with some support from M3. Tetra Tech developed the concurrent reclamation plan and Tetra Tech and Augusta estimated the annual costs for reclamation. Tetra Tech was also responsible for the Aquifer Protection Permit, supported by Errol L. Montgomery of Tucson, Arizona, who will prepare the ground water model to confirm the impact of the project on the ground water.

AMEC of Denver, Colorado and Tucson, Arizona provided the design and material quantities for the dry stack tailings facility and process water temporary storage pond and M3 estimated the capital cost based on the material quantities.

Errol L. Montgomery & Associates (ELM) of Tucson, Arizona was responsible for the ground water hydrology modeling and studies to support Tetra Tech with the Aquifer Protection Permit. ELM was also responsible for the exploration drilling and testing of water wells to locate a system of wells to supply fresh water for the project. ELM provided the production well cost and design up to the well head.

Stantec Consulting, Inc. (Tucson, Arizona) was responsible for the design of the fresh water pipeline from the well fields to the project site, including the necessary pumping stations and a water surge analysis for the system. Stantec provided the design and the quantity take-offs for construction and M3 estimated the installed cost.

WestLand Resources, Incorporated (Tucson, Arizona) was responsible for preparation of the Mine Plan of Operations to the US Forest Service. WestLand also prepared the Environmental section of the initial Feasibility Study and this Updated Feasibility Study.

M3 Engineering and Technology Corporation (M3) of Tucson, Arizona was retained by Augusta to prepare the process and infrastructure design, capital and operating costs for the process and infrastructure, and integrating the work by other consultants into this Updated Feasibility Study including the overall project capital cost estimate, operating cost estimate, implementation schedule for the project, and an economic analysis. M3 also reviewed previous metallurgical test reports and coordinated additional metallurgical testing programs conducted by SGS Lakefield Research Limited (SGS) of Toronto, Ontario, Canada; Mountain States Research & Development Inc. (MSRDI) of Tucson, Arizona; Hazen Research, Inc. (HRI) of Golden, Colorado; and G&T Metallurgical Services of Kamloops, British Columbia, Canada, all under contract with Augusta. SGS Lakefield was contracted to conduct ore grindability characterization tests and establish a preliminary grinding circuit design utilizing Comminution Economic Evaluation Tool (CEET) software. MSRDI was contracted to conduct batch and locked cycle flotation tests to define ore variability, grind / grade / recovery parameters, and reagent screening to define a reagent scheme. MSRDI also conducted dewatering tests for concentrate and tailings and column leach tests for oxide leach recovery. Hazen Research was contracted to conduct Bond rod and ball mill index tests. G&T Metallurgical Services Ltd. was contracted to assess mineral content, mineral liberation, and association and mineral fragmentation characteristics on two ore samples from MSRDI. The SGS Lakefield report, MSRDI report and Hazen Report are referenced in this technical report and formed the basis for establishing the plant design parameters, concentrate grades, metal recoveries, mill sizing and reagent consumptions.

1.4.1 Units and Abbreviations

The units of measure in this report are US Units and all costs are in US dollars, unless otherwise noted. The unit of mass is the short ton (ton, T, or t). A short ton is 2,000 pounds. Other units used include dry ton (DT, dt), miles (mi), feet (ft), inches (in), acres (ac), square feet (ft^2, sq. ft.), square inch (in^2, sq. in.), cubic feet (ft^3, cu. ft.), gallon (g), gallons per minute (gpm), pound (lb, lbs), pound per ton (lb/t), Fahrenheit temperature (° F), year (Y, y), day (D, d), hour (h), minutes (m) and seconds (s). Silver metal production is in troy ounces (oz).

Metal assays are in ounces per ton (opt), parts per million (ppm), and parts per billion (ppb). Acid solutions are in grams per liter (gpl).

Acronyms and abbreviations used in this report are noted below:

AA	Atomic Absorption Spectrometry
AAC	Arizona Administrative Code
ACC	Arizona Corporation Commission
ADEQ	Arizona Department of Environmental Quality
Ag	Silver
Anaconda	Anaconda Mining Company
Anamax	Anamax Mining Company
ANPL	Arizona Native Plant Law
APP	Aquifer Protection Permit
ARS	Arizona Revised Statutes
ASARCO	American Smelting and Refining Company
Au	Gold
Augusta	Augusta Resource Corporation
AZPDES	Arizona Pollutant Discharge Elimination System
BADCT	Best Available Demonstrated Control Technology
Banner	Banner Mining Company
BLM	Bureau of Land Management
BMP	Best Management Practices
CAA	Clean Air Act
CAP	Central Arizona Project
CESQG	Conditionally Exempt Small Quantity Generators
CFR	Code of Federal Regulations
CGP	Construction General Permit
CLS	Conservation Land System
Cu	Copper
CuEqv	Copper Equivalent
CWA	Clean Water Act
EIS	Environmental Impact Statement
EPA	Environmental Protection Agency
ESA	Endangered Species Act
HAP	Hazardous Air Pollutants
HRI	Hazen Research Incorporated
IP	Individual Permit
IRA	Important Riparian Area
kWh	Kilowatt Hour
LOM	Life of Mine
LQHUW	Large Quantity Handlers of Universal Wastes
M3	M3 Engineering and Technology Corporation
Mo	Molybdenum
MSGP	Multi-Sector General Permit
MSRDI	Mountain States Research and Development, Inc.

MW	Megawatts
NAAQS	National Ambient Air Quality Standards
NAVD 88	North American Vertical Datum 1988
NEPA	National Environmental Policy Act
NHPA	National Historic Preservation Act
NOI	Notice of Intent
NPDES	National Pollutant Discharge Elimination System
NSPS	New Source Performance Standards
NSR	Net Smelter Return
NWP	Nation Wide Permit
QA/QC	Quality Assurance and Quality Control
PAH	Pincock, Allen & Holt, Inc.
PCDEQ	Pima County Department of Environmental Quality
RCRA	Resource Conservation and Recovery Act
RQD	Rock Quality Data
SGS	SGS Lakefield Research Limited
Skyline	Skyline Assayers and Laboratories, Inc.
SQG	Small Quantity Generators
SQHUW	Small Quantity Handlers of Universal Wastes
SRM	Standard Reference Material
Stantec	Stantec Consulting, Inc.
SWPPP	Storm Water Pollution Prevention Plan
SWTC	South West Transmission Cooperative
SX-EW	Solvent Extraction - Electrowinning
TCLP	Toxic Characteristic Leaching Procedures
TCP	Traditional Cultural Properties
TCu	Total Copper Concentrations
TEP	Tucson Electric Power
TPD	Tons Per Day
USFWS	US Fish & Wildlife Service
UTM NAD 83	Universal Transverse Mercator – North American Datum 1983
WAPA	Western Area Power Administration
Wardrop	Wardrop Consultants
WECC	Western Electricity Coordinating Council
WGI	Washington Group International
Winters	The Winters Company
WLRC	WLR Consulting, Inc.
XRF	x-ray fluorescence

1.5 RELIANCE ON OTHER EXPERTS

M3 Engineering and Technology Corporation (M3) has relied on the data and information from Tetra Tech regarding the site geotechnical investigations, a geologic hazards assessment, the site baseline geochemical characterization, the site water management plan, the initial dry tailings facility design, the leaching facilities design, the ground water protection plan, the reclamation and closure plan, the waste management plan, and the soil salvage estimates for the operational and storage areas of the site. The Tetra Tech reports are referenced in the Technical Report in Section 1.23 and are attached to this Updated Feasibility Study in Appendix B.

M3 has also relied on data and information from Stantec Consulting, Inc. on the design of the fresh water pumping systems from the well fields to the site as well as Errol L. Montgomery on the ground water study; the construction, development and testing for the initial exploration water well; and the cost estimate for the development of the production wells for the project. This data and information is referenced in Section 1.23 and is attached to this Updated Feasibility Study in Appendix B.

The primary Qualified Persons responsible for preparing this Technical Report relied on the various reports and documents listed in Section 1.23. These reports and documents were prepared by technically qualified and professional persons and were found to be generally well organized, to industry standards, and where applicable, the conclusions reached were judged to be professionally sound. It is assumed that the information and explanations given to the Qualified Persons and those assisting the Qualified Persons by the employees of Augusta and third party consultants, who provided the reports referenced in Section 1.23 during the preparation of this Rosemont Copper Project Updated Feasibility Study and this Technical Report, were essentially complete and correct to the best of each employee's or consultant's knowledge and that no information was intentionally withheld.

1.6 PROPERTY DESCRIPTION AND LOCATION

1.6.1 Location

The Rosemont Property consists of a group of patented mining claims, unpatented mining claims and fee land that cover most of both the Rosemont Mining District and the adjacent Helvetia Mining District. The Rosemont Property is located approximately 30 miles (50 km) southeast of Tucson, Pima County, Arizona (see Figure 1-1). The Rosemont Property geographical coordinates are approximately 31º 50'N and 110º 45'W.

1.6.2 Land Tenure

The present land position is a combination of fee land, patented mine claims, and unpatented mine and mill site claims. Taken together, the land position is sufficient to allow mining of the open pit, processing of ore, storage of tailings, disposal of waste rock, and operation of milling equipment. These lands are accessible under the provisions of the Mining Law of 1872, subject to obtaining approval from the US Forest Service after completion of an Environmental Impact Statement (EIS) process. The EIS process includes interagency consultation on endangered species and cultural resources. The use of the project surface rights will require obtaining a number of federal, state, and local permits and approvals, which is now in progress.

The core of the Rosemont Property consists of 132 patented lode claims that in total encompass an area of 1969 acres (797 hectares) as shown in Figure 1-2. A contiguous package of 949 unpatented lode mining claims with an aggregate area of more than 12,000 acres (4,860 hectares) surrounds the core of patented claims. Associated with the property are 10 blocks of fee land consisting of a number of individual parcels that enclose a total of 911 acres (369 hectares). Most of the unpatented claims were staked on Federal land administered by the United States Forest Service, but a limited number of claims in the northeast portion of the property are on Federal land administered by the Bureau of Land Management. The area covered by the patented claims, unpatented claims and fee lands totals approximately 15,000 acres (6,070 hectares).

Surveyed brass caps on short pipes cemented into the ground mark the patented mining claim corners. Cairns and wooden posts mark the unpatented claim corners, end lines and discovery monuments, most of which have been surveyed. The fee lands are located by legal description recorded at the Pima County Recorders Office.

The patented lode claims and fee land parcels have no expiration date and are subject to annual property taxes amounting to a total of approximately ten thousand U.S. dollars. The unpatented lode claims also have no expiration and are maintained through the payment of annual maintenance fees of US$125.00 per claim, for a total of approximately one hundred twenty thousand U.S. dollars, payable to the Bureau of Land Management. A 3% Net Smelter Return (NSR) royalty applies to the patented claims, the bulk of the unpatented claims, and some of the fee land. On March 31, 2006, Augusta completed the purchase of a 100% interest in the property for a total of US$20.8M and continues to maintain the property in good standing.

Augusta retained the legal firm of Fennemore Craig to handle the legal transfer of the Rosemont Property. Augusta's land information has come from 2006 property purchase legal documents and has been subject to further validation contracted by Augusta, including a mining claim specialist, Daniel Mead of Tucson, Arizona, and registered mining claim surveyors at Darling Environmental & Surveying, Ltd.

of Tucson, Arizona. Darling Environmental & Surveying, Ltd. has conducted an extensive field and office review of the patented and unpatented claims. Fennemore Craig has continued to have legal involvement with the property lands.

1.7 Accessibility, Climate, Local Resources, Infrastructure and Physiography

1.7.1 Accessibility

The Rosemont Property is accessible from both the east and west directions. The primary access road will be from State Route 83 (SR-83) between mile markers 46 and 47 which is east of the plant site. The access road to the west will be a secondary access road that will go over the Santa Rita Mountains and join Santa Rita Road at Helvetia Road.

The intersection of the primary access road with SR-83 will be at a point that provides clear line of sight for up to 2,500 feet in each direction. SR-83 will be modified to provide safe ingress and egress from the access road. Modifications will include a 500 foot long center lane in each direction for accelerating and decelerating. A 220 foot deceleration lane and 500 foot acceleration lane will be constructed on the southbound side of SR-83 for safe access into and out of the plant site.

1.7.2 Climate

The climate is generally dry with precipitation being limited for the most part to a rainy season in the months of July, August, and September. Annual precipitation for the area is approximately 18 inches, the majority of which falls in the rainy season. Temperatures range between 92 deg F in the summer and 36 deg F in the winter. Augusta has maintained an automatic weather station on the project site since the second quarter of 2006.

1.7.3 Local Resources

The Rosemont Project is located 30 miles from the city of Tucson (population 500,000+) which provides sufficient resources for staffing a project of this size. Mining has been a part of the Tucson area for decades and includes three major operating copper mines within 75 miles of the project site: the ASARCO Silver Bell Mine near Marana, the ASARCO Mission Complex near Sahuarita, and the Phelps Dodge Sierrita Mine near Green Valley.

1.7.4 Infrastructure

The site is located 11 miles from Interstate 10. From I-10, State Route 83 can be used to gain access to the plant road. This system of interstates and highway will allow for quick access to the site. There is also a train yard at the Port of Tucson for getting supplies close to the project site. The majority of the labor and supplies for construction and operations can come from the surrounding Pima County area (pop. 1,000,000+ including the 500,000+ in Tucson).

Water will be pumped to the project site from new well fields, which lie about 18 miles northwest of the Rosemont Project in the Santa Cruz basin under a Mineral Extraction Water Permit. To mitigate the effects of pumping, water has been purchased from the Central Arizona Project (CAP) and is being recharged into the upper Santa Cruz basin aquifer at the Pima Mine Road recharge facility and the Lower Santa Cruz and Avra Valley sites near Marana.

The power will come from the scheduled upgrade to the transmission line from the Tucson Electric Power (TEP) Vail substation to Nogales. The project will build a switching station at the intersection of the transmission line where it crosses the northern boundary of the Santa Rita Experimental Range and route a new transmission line to the project site which is approximately 9 miles away.

1.7.5 Physiography

The Rosemont Property occupies flat to mountainous topography in the northeastern and northwestern flanks of the Santa Rita Mountains at a surface elevation ranging from 6,290 to 4,000 feet above sea level. The area is considered part of the Basin and Range physiographic province characterized by high mountain ranges adjacent to alluvial filled basins.

1.8 HISTORY

The early history and production from the Rosemont Property has been described in Anzalone (1995), as well as by Augusta (2007) from which that following summarization is taken.

Sporadic prospecting reportedly began in the middle 1800s in the northwestern portion of the Property and subsequently extended into the eastern part. In 1880, both the Helvetia Mining District (to the west) and the Rosemont Mining District (to the east) were established. Production from mines on both sides of the northern Santa Rita Mountains area supported the construction and operation of the Columbia Smelter at Helvetia on the west side of the Santa Rita Mountains and the Rosemont Smelter in the Rosemont Mining District on the east side of the Santa Rita Mountains. Copper production ceased in 1951 after the production of about 227,300 tons of ore containing 17,290,000 pounds of copper, 1,097,980 pounds of zinc and 180,760 ounces of silver. An unknown, but minor portion of the production came from the Rosemont Deposit.

Since shutdown in 1951, the area stretching from Peach-Elgin (on the northwest, see Figure 1-2) to Rosemont (on the southeast) has seen a progression of exploration campaigns. Churn drilling at Peach-Elgin deposit in 1955 and 1956 by Lewisohn Copper Company began the definition of that deposit. Drilling in 1956 by American Exploration and Mining Company initiated exploration of the Broadtop Butte prospect. Banner Mining Co. had acquired most of the claims in the area by the late 1950s and drilled the discovery hole into the Rosemont deposit.

Anaconda Mining Company acquired the property in 1963 and carried out a major exploration program that identified Rosemont as a major porphyry copper deposit and advanced the Broadtop Butte and Peach-Elgin prospects. In 1973, Anaconda joined with Amax in the Anamax partnership that continued until 1986 when Anamax sold the Rosemont – Peach-Elgin property to a real estate company during the corporate dissolution of Anaconda. By the end of the Anaconda-Anamax programs, exploration drilling totaled in excess of 297,321 feet (90,623 meters), of which approximately 195,000 feet (59,500 meters) define the Rosemont deposit. The results of these programs are described in Wardrop (2005).

In 1964, Anaconda produced a geological resource estimate for the Peach-Elgin deposit that was based on assays from 67 churn and diamond drill holes. After calculation of that resource, Anaconda and Asarco drilled approximately 140 additional diamond drill holes, but did not update the 1964 estimate. The estimated resources are briefly summarized in Section 17 (Adjacent Properties) of this report.

In 1977, Anamax commissioned Pincock, Allen & Holt, Inc. (PAH) to calculate a resource for the Rosemont Deposit. The resulting calculation estimated a geological resource of about 445 million tons at an average grade of 0.54% Cu using a cut off grade of 0.20% Cu. The methodology has been described in Wardrop (2005), which is available on SEDAR. *Augusta Resource Corporation has not done the work necessary to verify the classification of this resource and is not treating the resource figure as a NI 43-101 defined resource verified by a Qualified Person and, therefore, the resource figures should not be relied upon by investors.*

Anamax carried out a resource estimate for the Broadtop Butte deposit in 1979 that was based on approximately 18 widely-spaced diamond drillholes. The resources for this deposit are also summarized in Section 17 (Adjacent Properties) of this report.

ASARCO purchased the property in 1988, renewed exploration of the Peach-Elgin deposit and initiated engineering studies on Rosemont. ASARCO drilling on Rosemont was limited to 12 diamond drillholes. ASARCO sold the entire property to real estate interests in 2004, shortly before the ASARCO takeover by Grupo Mexico S.A. de C.V.

ASARCO generated a resource estimate of the Rosemont Deposit that was incorporated into a 1997 consulting report by The Winters Company that comprised an “order of magnitude” mining study of the deposit. The resulting “mineable resource” totaled nearly 341 million tons at an average grade of 0.64% Cu. The results and methodology have also been described in Wardrop (2005). *Augusta Resource Corporation has not done the work necessary to verify the classification of this resource and is not treating the above resource figure as a NI 43-101 defined resource verified by a Qualified Person and, therefore, the resource figures should not be relied upon by investors.*

Augusta Resource Corporation became interested in the Rosemont Property in 2005 and began a program to confirm the results from previous work. Augusta completed the purchase of the property in March 2006. In 2005, Augusta completed a Phase I drilling program consisting of 15 core holes. Based on the new Augusta and previous Anaconda drilling, WLR Consulting, Inc. in conjunction with Mine Reserve Associates, Inc. prepared a mineral resource estimate that was presented in an April 21, 2006 report entitled *Mineral Resource Estimate Revised Technical Report For The Rosemont Deposit, Pima County, Arizona, USA.*

Based on the encouraging results of that program, Augusta continued with a Phase II drilling program in 2006 that consisted of 40 core holes for resource definition, metallurgical, and geotechnical purposes. Additional drill holes were incorporated into a resource estimate update that was announced in a March 16, 2007 press release, which was documented in an April 26, 2007 report entitled *2007 Mineral Resource Estimate Update for the Rosemont Project, Pima County, Arizona, USA*, by WLR Consulting, Inc.

Augusta initiated a Feasibility Study with M3 Engineering & Technology Corporation of Tucson, Arizona in the middle of 2006, which was completed in August 2007. The findings were presented in an August 2007 report entitled *Rosemont Copper Project Feasibility Study*, which documents the Rosemont Mineral Reserves.

As part of a post-feasibility update, Augusta conducted further drilling in 2008. Twenty core holes were drilled to further define the northwestern part of the deposit. In addition, 10 previously drilled geotechnical holes from Augusta’s

2006 drilling campaign were sampled and analyzed. This additional drilling and sampling data was incorporated into a resource estimate that was announced in an October 23, 2008 press release, documentation for which is provided by this Technical Report.

1.9 GEOLOGICAL SETTING

The regional, local and property geology of the Rosemont deposit is described by Anzalone (1995), Wardrop (2005), and by Augusta (2007), from which that following summarization is taken.

At Rosemont, Precambrian meta-sedimentary and intrusive rocks form the regional basement beneath a Paleozoic sedimentary sequence of limestone, quartzite, and siltstone. Paleozoic limestone units are the predominant host rocks for the copper mineralization. Structurally overlying these older units are Mesozoic clastic units, including conglomerates, sandstones, and siltstones. Some andesitic volcanic beds occur within the Mesozoic sedimentary section.

Regionally, the Mesozoic and early Cenozoic Laramide Orogeny was marked by compressional tectonism accompanied by extensive calc-alkaline magmatism. The regional compressional forces caused folding and thrust, transverse and reverse faulting. Coeval magmatism, recorded in voluminous batholithic and smaller intrusions and their associated volcanic equivalents, was responsible for the generation of the porphyry copper deposits of the region. At Rosemont, mineralizing quartz monzonite and quartz latite intruded into a package of Precambrian intrusive rocks and Paleozoic and Mesozoic sedimentary rocks at the intersection of regional basement structures.

Tertiary extensional tectonism followed the Laramide Orogeny, accompanied by voluminous felsic volcanism. Steeply- to shallowly-dipping normal faults became active during this time, most likely including rotational listric faulting. At Rosemont, it appears that Tertiary faulting has significantly segmented the original deposit, juxtaposing mineralized and unmineralized rocks. The extensional tectonics culminated in the large-scale block faulting that produced the present Basin and Range Provence physiography.

A generalized geologic map of the Rosemont Property is presented in Figure 1-3. Figure 1-4 shows a stratigraphic column of the Rosemont District. Faulting has generally divided the deposit into three generalized structural blocks. The north-trending Backbone Fault separates Precambrian granodiorite and Lower Paleozoic quartzite to the west from younger Paleozoic limestone units to the east. The subhorizontal Flat Fault places Paleozoic limestone (minor) and Mesozoic sedimentary rocks over the top of the older Paleozoic units. In addition, partially consolidated gravel of Tertiary age fills a paleochannel on the south side of the deposit area. To the north and east are significant thicknesses of Tertiary volcaniclastic material.

To the north of the Rosemont deposit, the Broadtop Butte deposit is associated with related fault systems. The Copper World Mine deposit is located to the northwest of Broadtop Butte, situated in a complexly faulted block of Paleozoic rocks. Further to the northwest, the Peach-Elgin deposit occurs in a structural block floored by a low-angle fault and may represent the upper part of the Copper World mineralization.

1.10 DEPOSIT TYPES

The Rosemont Deposit consists of skarn-hosted copper-molybdenum-silver mineralization related to quartz-monzonite porphyry intrusions. Genetically, it is a style of porphyry copper deposit, although intrusive rocks are volumetrically minor within the resource area. The skarns formed as the result of thermal and metasomatic alteration of Paleozoic carbonate and to a lesser extent Mesozoic clastic rocks.

Mineralization is mostly in the form of primary (hypogene) copper-molybdenum-silver sulfides, found in stockwork veinlets and disseminated in the altered host rock. Some oxidized copper mineralization is also present in the upper portion of the deposit. The oxidized mineralization is primarily hosted in Mesozoic rocks, but is also found in Paleozoic rocks where those outcrop or are near-surface on the west side of the Rosemont Deposit. The oxidized mineralization occurs as mixed copper oxide and copper carbonate minerals. Minor amounts of enriched, supergene chalcocite and associated native copper mineralization are found in and beneath the oxidized mineralization.

The Twin Buttes Mine, operated by Anaconda and later by Cyprus, was developed on an analogous deposit located about 20 miles (32 kilometers) to the west of Rosemont. The Twin Buttes mine was in production from 1969 to 1994. In addition, the ASARCO Mission Mine, also located about 20 miles (32 kilometers) to the west of Rosemont, has some common geologic characteristics.

1.11 MINERALIZATION

The Rosemont Deposit contains copper-molybdenum-silver primarily hosted in an east-dipping package of Paleozoic-age sedimentary rocks. Two horizontal plans and a vertical cross section of the geology of the Rosemont Deposit are shown in Figures 1-5, 1-6 and 1-7. Drilling has identified a significant mineral resource 3,500 feet (1,100 meters) in diameter that extends to a depth of at least 2,000 feet (600 meters) below the surface. The steeply east-dipping Backbone Fault offsets the mineralization, with limited mineralization occurring to the west of it. To the south, the mineralization appears to weaken and eventually die out. Mineralization in the Paleozoic rocks continues to the north amid complex faulting and to the east beneath increasingly-thick Mesozoic cover, to the present limits of drilling. The subhorizontal Flat Fault separates the strongly mineralized Paleozoic sequence from overlying, weakly-mineralized Mesozoic and lesser

Paleozoic rocks. Oxide copper and chalcocite mineralization occurs widely in the Mesozoic-age rocks.

The main Paleozoic host rocks include, from oldest to youngest, the Escabrosa Limestone, Horquilla Limestone, Earp Formation, Colina Limestone, and Epitaph Formation. The Horquilla Limestone is the most significant, accounting for almost half of the mineralized sulfide material. Significant mineralization also occurs in the Earp Formation and in the Colina Limestone, while relatively minor mineralization is found in other Paleozoic units.

The Mesozoic host rocks consist predominantly of arkosic siltstones, sandstones, and conglomerate. Within the arkose is a local andesite unit that ranges from a few tens of feet to several hundred feet thick. Near the base of the arkose is the Glance Conglomerate, a limestone-cobble conglomerate.

The mineralization is primarily in garnet-diopside (with minor magnetite) skarn that formed in the Paleozoic rocks as a result the intrusion of quartz latite to quartz monzonite porphyry. Marble was developed in the more pure carbonate rocks, while the more siliceous, silty rocks were converted to hornfels. Bornite-chalcopyrite-molybdenite mineralization occurs as veinlets and disseminations in the garnet-diopside skarn and associated marble and hornfels, accompanied by quartz, amphibole, serpentine and chlorite alteration. Quartz latite to quartz monzonite intrusive rocks host strong quartz-sericite-pyrite mineralization with minor chalcopyrite, molybdenite and bornite. Where the mineralized package of Paleozoic rocks and quartz-latite intrusives outcrops on the western side of the deposit, near surface weathering and oxidation has produced disseminated and fracture-controlled copper oxide minerals.

Weakly-mineralized to unmineralized Paleozoic limestone and Mesozoic siltstone, sandstone, conglomerate and andesite comprise the near-surface portion over most of the deposit area, separated from underlying, better-mineralized Paleozoic rocks by the subhorizontal Flat Fault. The highly-variable, but relatively minor, mineralization above the Flat Fault is typically oxidized and supergene chalcocite is locally present. Oxidized and supergene copper mineralization above the Flat Fault appear to be especially well-developed in the andesitic rocks.

Silver occurs in minor, but economically significant quantities in the primary sulfide mineralization in the Paleozoic sequence. The silver is associated with the copper mineralization. The gold content of the deposit is generally very low, but contributes to a by-product credit.

1.12 EXPLORATION

Prospecting began in the Rosemont and Helvetia Mining Districts sometime in the middle 1800s and by the 1880s copper production is recorded, which continued sporadically until 1951. By the late 1950s exploration drilling had resulted in the discovery of the Rosemont Deposit. A succession of major mining companies subsequently conducted exploratory drilling of the Rosemont Deposit and other deposits of the region.

Additional information regarding exploration and evaluations performed on the Rosemont Deposit is presented in Sections 1.8 (History) and 1.13 (Drilling).

1.13 DRILLING

Extensive drilling has been conducted at the Rosemont Deposit by several successive property owners. The most recent drilling was by Augusta, with prior drilling campaigns completed by Banner Mining Company, The Anaconda Company, Anamax and ASARCO. Augusta's drilling was focused on infill drilling the pre-existing drill hole pattern, thereby expanding and increasing the confidence in the database for the current NI 43-101 compliant resource estimate. Table 1-10 summarizes the drill holes used in the current resource estimate.

Table 1-10 Rosemont Deposit Drilling Summary

Company	Time Period	Drill Holes		
		Number	Feet	Meters
Banner	1950s-1963	3	4,226	1,288
Anaconda	1963-1973	113	136,728	41,675
Anamax	1973-1986	52	54,350	16,566
ASARCO	1988-2004	11	14,695	4,479
Augusta	2005-2008	75	113,876	34,709
Total		254	323,875	98,717

The drill holes utilized in the database were all drilled using diamond drilling (coring) methods. In some cases the tops of the older holes were drilled using a rock bit to set the collar; in other cases the upper parts of older holes were drilled with rotary drilling, switching switched to core drilling before intercepting mineralization. A map showing the location of the drill holes used in the resource calculation is provided in Figure 1-8, along with a general outline of the Rosemont deposit limits. Exploration holes drilled using rotary or older "churn" drill holes were excluded from the resource database.

In all of the drilling campaigns, efforts were consistently made to obtain representative samples by drilling larger N (1.9 inch diameter) and H (2.5 inch diameter) size core. Core recoveries were generally good (typically in the range of 86-93%), lending confidence that quality samples were obtained. All of the Rosemont drilling was been conducted on east-west lines that are approximately 200 feet apart. Currently, the average spacing of drill holes along these lines average about 250 feet.

Most of the Anaconda, Anamax and ASARCO drill core was still available on site or was obtained by Augusta and brought back to the Rosemont property, where it was rigorously relogged by Augusta personnel to be geologically consistent with the current Augusta drill hole logging. Along with relogging, this core was also resampled for additional geochemical analyses as described in the sampling section (Section 1.14).

1.13.1 Banner Mining Company Drilling

The first significant core drilling campaign on the Rosemont Property was by the Banner Mining Company, beginning in about 1961. Banner completed mostly shallow diamond drill holes, many of which were subsequently deepened by Anaconda. Three drill holes included in the resource database were shallow holes started by the Banner Mining Company that were significantly deepened during subsequent Anaconda drilling programs. These holes have a combined length of 4,226 feet.

1.13.2 The Anaconda Company Drilling

Anaconda took over Banner's Rosemont holdings around 1963 and conducted exploration at the Rosemont Deposit and in adjacent mineralized areas. Between the years of 1963 and 1973 they completed 113 diamond drill holes at Rosemont for a total of 136,728 feet. These holes were primarily drilled vertically. Down-hole surveys were conducted during drilling or immediately following drill hole completion for selected holes. Drill hole collars were surveyed by company surveyors. Anaconda drilled approximately 85 percent of the larger N-sized core (1.9 inch diameter) and 15 percent of the smaller B-sized core (1.4 inch diameter). Overall core recovery was more than 85 percent.

Exploration subsequently transferred to the Anamax Mining Company (an Anaconda-AMAX joint venture) around 1973, which continued the extensive diamond drilling and analytical work until 1986. Anamax completed 52 core holes for a total of 54,350 feet. These holes were almost exclusively drilled as angle holes inclined -45° to -55° to the west, approximately perpendicular to the east-dipping, Paleozoic, metasedimentary host rocks. Down-hole surveys were conducted during drilling or immediately following drill hole completion for the majority of the holes. Drill hole collars were surveyed by company surveyors. Anamax drilled approximately 80 percent N-sized core (1.9 inch diameter) and 20 percent B-sized core (1.4 inch diameter), with an overall core recovery of more than 88 percent.

During drilling, the core was placed in standard cardboard core boxes by the drillers, with wooden blocks marking the beginning and ending footages of core runs. Core boxes were labeled with the drill hole number, footage interval and other information by the drillers.

1.13.3 ASARCO Mining Company Drilling

ASARCO acquired the Rosemont Property in 1988 and conducted exploration until 2004, completing 11 vertical drill holes for a total of 14,695 feet in the deposit area (a 12th hole was drilled to the east of the deposit). Data was available from eight of the ASARCO core holes and was incorporated into Augusta's resource database. Down-hole survey data, if taken, was not available for the ASARCO holes. Drill hole collars were surveyed by company surveyors. The size of core collected by ASARCO was predominantly N-sized (1.9 inch diameter). Core recovery information was not available but Augusta relogging indicated it to generally be good, similar to that of other drilling campaigns.

ASARCO sold the Rosemont property in 2004 to real estate interests.

1.13.4 Augusta Drilling

Augusta has conducted diamond drilling in three campaigns, the first starting in the second half of 2005 and continuing into early 2006 (Phase I), the second starting in mid 2006 and continuing into early 2007 (Phase II), and the third starting in December 2007 and continuing to July 2008 (2008 Drilling). In total, Augusta has completed 75 core holes for a total of 113,876 (98,717 meters). Of these, 57 drill holes were planned as resource holes to infill where previous drilling had left gaps in the classification of measured or indicated mineral resources, with 3 being exploration holes outside of the potential pit area. The remaining 15 Augusta core holes were drilled in support of geotechnical (13) or metallurgical (2) studies. The relevant geotechnical hole intercepts were sampled and analyzed as part of the 2008 work.

Augusta drill holes were rock-bitted through overburden, then drilled with larger HQ-sized core as deeply as possible and finished with NQ-sized core (1.9 inch diameter) when a reduction in core size was required by ground conditions. Most of the holes were oriented vertically, although a few of the holes were inclined in order to intercept target blocks from reasonably accessible drill locations. Layne-Christensen and/or Boart Longyear were the drilling contractors. All drill holes were surveyed down-hole with a Reflex EZ-Shot survey instrument that measured inclination/dip and azimuth direction, with readings generally taken every 100 feet down the hole during 2008 and every 500 feet down the hole during 2005 and 2006. Phase I drill hole collar locations were surveyed by Putt Surveying of Tucson, Arizona, while Phase II and 2008 drilling locations were measured by Darling Environmental & Surveying. Augusta drill core was approximately 63 percent N-sized (1.9 inch diameter) and about 36 percent being larger H-sized (2.5 inch diameter), with less than 1 percent being smaller B-sized (1.4 inch diameter). Augusta's overall core recovery was approximately 95 percent.

During drilling, the core was placed in standard cardboard core boxes by the drillers, with wooden blocks that marked the footages of core runs. Core boxes were labeled with the drill hole number, footage range and other information by the drillers.

1.14 SAMPLING METHOD AND APPROACH

The Rosemont resource database is based on core samples recovered from diamond drill holes. The drill core from mineralized intervals was generally sampled continuously down the hole, at a nominal five-foot sample length. In taking a sample, the core is generally halved (split) along the long axis, taking care to evenly distribute veinlets and other small-scale mineralized features, where present, into both halves of the core.

1.14.1 Banner, Anaconda and Anamax Sampling

The Banner, Anaconda and Anamax sampling is discussed as a group because the sampling took place as part of a more-or-less continuous program. The analytical data in the resource database for the three Banner drill holes came from the Anaconda laboratory, as most of the length of these holes came from subsequent Anaconda drilling that significantly deepened these holes. The exploration transition from Anaconda to Anamax (Anaconda-Amax Joint Venture) drilling did not appear to immediately utilize a different laboratory or techniques.

In analyzing the Banner, Anaconda and Anamax drill core, the geochemical suite was determined by whether an interval retained its primary sulfide mineralization or had been oxidized. Core with primary sulfide mineralization above trace levels was comprehensively analyzed for total copper and molybdenum. For some intervals, lead and zinc metal concentrations were analyzed where indicated by mineralogy, but that was not common. Relatively late in the program, particularly in the Anamax drill core, silver analysis was routinely included in the sulfide zone, especially for well-mineralized intervals. Oxide zone drill core with visible copper oxide mineralization (chrysocolla, cuprite, copper wad, etc.) was analyzed for acid-soluble copper in addition to total copper, and molybdenum was excluded or only intermittently analyzed in the oxide zone core.

The core was sampled at geologic intervals, based on changes in mineralization and alteration, that generally ranged from one to six feet in length and averaged about five feet in length. In poorly mineralized intervals, analytical samples were collected only intermittently, typically with one five-foot sample collected every 20 to 30 feet, to characterize the rock as having low to no grade values.

The core was first logged to record the core run intervals and percent recovery, along with lithology, structure, alteration and mineralization. After sampling intervals were assigned, the core was split with a mechanical splitter along its long axis, and one-half of the core was retained in the original core box. Sample preparation during the Banner, Anaconda and Anamax programs was conducted by employees of those companies. Other details of the sampling process are not well known, but since this work was carried out by major copper companies for their internal use, it is believed that they used the standard industry practices for that time.

1.14.2 ASARCO Sampling

The ASARCO drill core was routinely analyzed for total copper, acid-soluble copper and molybdenum. Oxide zone core does not appear to have been analyzed differently than the sulfide-bearing core. The core was sampled with preference towards a 10-ft sample length, but, as for the Banner, Anaconda and Anamax core, the geologists appear to have had considerable latitude in choosing longer or shorter intervals. In some poorly-mineralized intervals, it appears that only one analysis was run for intervals exceeding 100 feet in length, although that is rare. The ASARCO drill core was apparently logged and sampled in much the same style as is described above for the Banner, Anaconda and Anamax core.

1.14.3 Augusta Sampling

Augusta Core

Sampling of new Augusta drill holes took place at the Rosemont Ranch sampling facility for the 2005 Phase I and the 2006 Phase II drilling programs. The 2008 drill hole sampling took place at the Hidden Valley Ranch sampling facility. Core drilled for the resource database by Augusta Resource (2005-2008) was analyzed using a geochemical suite that varied depending on whether or not the core retained its primary sulfide mineralization or had been oxidized, similar to the approach described above for Anaconda. In the oxidized zone, the core was routinely analyzed for total and acid-soluble copper. Sulfide zone core was analyzed for total copper, molybdenum and silver. Some of the earlier core was also analyzed for gold, although that was discontinued late in 2006 when the gold content had been adequately characterized and the cost of additional gold analyses was no longer considered warranted.

Augusta core was sampled at even five-foot intervals, except where massive copper or molybdenum veining, structures or lithologic breaks warranted special investigation through the selection of shorter intervals. Sample intervals would return to footages evenly divisible by five as soon as possible thereafter.

Geotechnically oriented Rock Quality Data (RQD) logging was performed on all core drilled by Augusta to systematically quantify core recovery, rock quality, fracture frequency, core hardness, joint condition, large-scale joint expression and down-hole water conditions. Then experienced exploration geologists familiar with the project lithologies logged the rock type, alteration, mineralization, and structure evident in the core. After logging, the geologist assigned and marked the sample intervals and cut-lines directly on the core and on the core box interior with a black marker. Each sample was given a unique, sequenced sample number with the footage noted in a sample tag booklet and on a paper copy. The drill core boxes were then photographed with a digital camera.

The core was split by cutting it in half with a diamond rock saw. All cuts were carefully planned and marked on the core by the logging geologist to evenly divide mineralization between the two halves of the core. All core cutting was done with water using no additives and the sawed drill core was placed directly back in the core box to dry before sampling. When dried, the left-hand half of the split core was placed in bags labeled according to the sequenced paper sample tags, with a sample tag also placed inside the bag. The plastic bags were then sealed with adhesive tape, leaving the sample number visible.

Banner, Anaconda, Anamax and ASARCO Core Resampling

Augusta also sampled available core drilled by Anaconda, Anamax, and ASARCO to fill-in missing analytical information and to validate the older analyses. Resampling of older pre-Augusta drill holes took place at the Hidden Valley Ranch sampling facility in 2006. Oxide zone intervals were resampled and analyzed for both total and acid-soluble copper in cases where total copper was estimated to be >0.1% Cu, but which had not yet been analyzed. All sulfide zone drill core from within the deposit area that had not been analyzed for both total copper and molybdenum were sampled and analyzed to provide complete, continuous copper and molybdenum data.

In addition to infilling the missing copper and molybdenum analyses in the sulfide zone, all of the available Banner, Anaconda, Anamax and ASARCO core that, on average, contained greater than 0.2% Cu over a 50-foot continuous length was resampled and analyzed for silver and sometimes gold, both of which were usually absent from the previous analytical work. Gold analyses were discontinued late in 2006 after the gold mineralization was sufficiently characterized, as described above for the Augusta-drilled core.

Whenever possible, the sample intervals for additional analyses conformed to the original sample intervals as determined from the historic core logs and analytical results. Augusta required all samples to be seven feet or shorter. Where previously only intermittent samples had been collected (i.e., a five-ft sample every 20-30 feet), original intervals were divided into multiple new sample intervals of approximately five feet in length, preserving the starting and ending footages of the original sample intervals. Another circumstance that required deviation from the original sample intervals was when core was missing – either lost or previously taken for metallurgical work. In such cases, Augusta sample intervals were aligned to reflect the missing core intervals.

Augusta geologists identified intervals requiring additional (infill) analyses by referring to the previously logged mineral and analyzed geochemical content of the core. Whenever possible the sample intervals for additional analyses conformed to the original sample intervals as determined from the historic core logs and analytical results. New Augusta assays were assigned unique, sequenced sample numbers from sample tag books. Intervals and corresponding sample numbers were recorded in an Excel-based computer file. For the purposes of silver (and for a time gold) grade determinations, the new sample intervals were combined into length-weighted 50-ft composite samples before analysis, reducing the total number of samples. This compositing was performed on pulp samples at the analytical laboratory using relative weight contributions for each component sample calculated by Augusta geologists.

After sample intervals and sequential sample numbers were assigned for the core to be re-analyzed, the core boxes were carefully photographed using a digital camera. Photos were inspected and archived before samples were collected. The assigned intervals were then measured and collected by sampling technicians, taking the entire remaining core with the exception of some small, representative archive samples. The individual samples were placed in plastic sample bags marked with the new sample number. The paper sample tags from the sample book in which drill hole identification and sample interval had previously been recorded were placed in the bag with the core.

1.15 SAMPLE PREPARATION, ANALYSIS AND SECURITY

1.15.1 Sample Handling and Security

Sample handling during the Banner, Anaconda, Anamax, and ASARCO programs was conducted by employees of those companies, for which some of the protocol records are limited. Augusta notes that these were major mining companies conducting work for their internal use. It is assumed that professional care was taken in the handling of samples by these company employees and no evidence to the contrary has been found.

For the new Augusta drilling program, the drilling contractors kept the core in a secure area next to the drill rig before delivering it to the Rosemont Ranch (2005, 2006) or Hidden Valley (2008) sampling facility, approximately three miles from the drilling area. Resampling of old pre-Augusta core occurred at the Hidden Valley Ranch sampling facility during 2006.

At the Rosemont Ranch facility in 2005 and 2006 and subsequently at Hidden Valley in 2008, samples were logged, marked, cut and placed in sample bags by geologists and helpers contracted by Augusta. At the Hidden Valley facility in 2006, samples were marked to conform to the original sample intervals and placed in sample bags by geologists and helpers contracted by Augusta. At both locations, the samples were kept in a locked storage unit on site until they could be transported to the analytical laboratory in Tucson. The logging and sampling areas were kept under closed-circuit video surveillance to provide a record of the personnel that had accessed the logging and sampling areas. Additional security was afforded by ranch personnel that oversaw the premises at night. No core handling or core security issues were experienced during the drilling or sampling program.

Locked sample boxes were picked up by Skyline employees, who officially took custody of the samples at the two sampling facilities, which were set up on the Rosemont Property. After completion of the laboratory work, the pulp samples and coarse rejects were returned to site for long-term storage and possible future use.

1.15.2 Banner, Anaconda and Anamax Sample Preparation and Analysis

The Banner, Anaconda and Anamax sampling is discussed as a group because the sampling took place as part of a more-or-less continuous program. The analytical data in the resource database for the three Banner drill holes came from the Anaconda laboratory, as most of the length of these holes came from subsequent Anaconda drilling that significantly deepened these holes. The exploration transition from Anaconda to Anamax (Anaconda-Amax Joint Venture) drilling did not appear to utilize a different laboratory or analytical techniques.

Geochemical analyses for the Banner, Anaconda and Anamax core were conducted in-house at Anaconda and Anamax laboratories. The following information was obtained from Mr. Dale Wood, Anaconda Chief Chemist in meetings and telephone conversations on November 28, 2005 and January 21, 2006. Copper and molybdenum were determined by wet chemical analyses and by x-ray fluorescence (XRF) methods, using analytical procedures that were industry standard for the 1960s and 1970s. Crushing and grinding reduced all pulp samples to minus 100 mesh size, with constant screen size testing. Pulp samples for the wet chemical method were brought into solution by hot acid digestion on a shaker table with hydrochloric acid, nitric acid and perchlorate acid added to the boiling solution followed by a few drops of hydrofluoric acid. Analyses for molybdenum were by the colorimetric iodine titration method. Copper analyses were done by the colorimetric phenolthylanaline titration method. The XRF analytical technique consisted of either a quick screening method by compressing a pulp sample on mylar film and placing it under the x-ray beam or, alternatively, adding cellulite to the pulp sample, pressing it into a ring and then placing under the x-ray beam. Samples with XRF-determined grades above 0.2% Cu and 0.02% Mo were selected for wet chemical analyses.

1.15.3 ASARCO Sample Preparation and Analysis

The ASARCO geochemical analyses that Augusta obtained from ASARCO were conducted by Skyline Analytical Laboratory, Tucson, Arizona. Skyline was a large, certified, commercial laboratory that utilized industry-standard analytical techniques; therefore these data obtained for the ASARCO core are considered reliable. No detailed descriptions of Skyline's sample preparation and analytical methods during those years are available at this time.

1.15.4 Augusta Sample Preparation and Analysis

Geochemical analyses for Augusta-drilled core and for the Augusta resampling of the Anaconda, Anamax, and ASARCO core were primarily performed by Skyline Assayers and Laboratories (Skyline) in Tucson, Arizona. During 2005, Skyline was formally known as Actlabs-Skyline and had been owned by ACTLABS (Ancaster, ON, Canada) since 1997. Skyline became independent of ACTLABS in January, 2006. Skyline is accredited in international quality standards through ISO/IEC 17025, with CAN-P-1579 for specific registered tests through the Standards Council of Canada. Augusta had both primary and secondary (duplicates) analyses done at Skyline in 2006 and 2007. ALS Chemex (Vancouver, BC, Canada) was used by the project for duplicate checks sample analyses in 2005. ALS Chemex has accreditation through ISO 9001:2000 in North America.

At Skyline, the entire sample was crushed using a TM Terminator to produce a greater than 80% pass 10-mesh product. Samples were blended and divided using a two-stage riffle splitter, from which a 300-400 gram split was pulverized to a 90% passing 150-mesh product using a TM Max 2 Pulverizer. Wash gravel and sand were used by Skyline to clean the crushers after each batch of samples were processed. Pulverizers were cleaned after each batch of samples and/or after each sample if the material adhered to inside walls of the grinding vessel. Coarse reject and pulp material was saved and returned to Augusta.

For the determination of total copper and molybdenum, Skyline digests 0.2000 to 0.2300 grams of the sample with 10.0 milliliters (ml) of hydrochloric acid, 3.0 ml nitric acid and 1.0 ml perchloric acid at 250° C, in a 200-ml phosphoric acid flask. When the only remaining acid present is perchloric acid and the volume of the liquid in the flask is less than 1 ml, the solution is allowed to cool. About 25 ml demineralized water and 10.0 ml hydrochloric acid is then added and the solution is gently boiled for 10-20 minutes. The flask is again cooled to room temperature and the contents are diluted with demineralized water and shaken well to mix. Copper and molybdenum content are determined by atomic absorption, with reference to standards made up in 5% hydrochloric acid.

Acid soluble copper is determined by leaching one gram of pulverized sample in 10% sulfuric acid solution for one hour. The copper content of the resulting solution is determined by atomic absorption.

For the determination of silver, Skyline digests 0.25 grams of sample with 0.5 ml nitric acid and 1.5 ml hydrochloric acid in a disposable, 18-mm x 150-mm borosilicate glass test tube. After agitation and the cessation of any effervescence due to carbonates, the test tubes are placed in a test tube rack in a hot water bath that is maintained between 90 °C and 95 °C, where digestion continues for 90 minutes. After cooling to room temperature the contents are diluted to 10 ml with demineralized water and again agitated to mix well. The solutions are then read by atomic absorption for silver.

1.15.5 Quality Assurance and Quality Control Protocol

General

The Quality Assurance and Quality Control (QA/QC) protocols in place during the Anaconda, Anamax and ASARCO exploration programs are not documented in records available to Augusta, although all the available evidence shows that they took great care in sample handling and storage, and that the laboratories analyzing the geochemical samples used industry standard practices (see Section 14).

Augusta adopted a systematic QA/QC protocol to support its analytical laboratory results. QA/QC oversight was provided initially by Kenneth A. Lovstrom, Geochemist, and was subsequently continued by Shea Clark Smith, Geochemist, who assumed guidance for QA/QC after January 10, 2006 to the present. The QA/QC procedures used by Augusta consisted of the routine use of standards, blanks, as well as repeat analysis of pulps.

1. Standards were submitted with a frequency of one per 20 samples for the 2005, 2006, and 2008 drilling campaigns.

2. Blank samples were submitted with a frequency of one per 40 samples for the 2005, 2006, and 2008 drilling campaigns.

3. Marble preparation blanks were submitted for the 2005 and 2006 drilling campaigns, as needed following select high-grade sample intervals, as a check of the subsequent cleanliness of the preparation equipment. This was implemented in the middle of the Augusta Phase II drilling program. This material was presumed to be, but was not, certifiably blank. These were not included for the 2008 drilling campaign samples.

4. Duplicate pulp reanalysis was conducted for the 2006 and 2008 drilling campaigns, with 600 and 121 duplicate pulps, respectively, resubmitted to Skyline. Each batch of 16 pulps

was accompanied by a copper standard, a molybdenum standard, a silver standard and a blank. In addition, each batch included the laboratory's own internal standards. This check served to evaluate the repeatability of the sample values.

In addition to Augusta's QA/QC work, Skyline had their own internal control procedures that included standards and repeat analyses. Augusta's primary laboratory data reports contained internal laboratory quality control data. For each laboratory job, an original, certified report(s) was sent to Augusta and has been filed with each drill hole.

Quality control results for 2005 drilling campaign, including drill holes AR-2000 through AR-2014, were discussed in a previous mineral resource report (WLRC, 2006). Quality control results for the 2006 drilling campaign, including drill holes AR-2015 through AR-2043, as well as Augusta fill-in sampling of older Anaconda core, were also discussed in a previous mineral resource report (WLRC, 2007). The following update focuses on the quality control results for the new 2008 Augusta drill holes.

Standard Reference Materials (External Laboratory Standards)

The suite of standard reference materials (SRMs) for the 2008 drilling included five SRMs that were used in the previous 2006 analytical work, incorporating a range of copper, molybdenum and silver concentrations that approximate the range of metal concentrations encountered in Rosemont drilling. These included: R4-A, R4-B, R4-C, R4-E, and R4-G. The R4-suite was prepared at MEG Labs (Carson City, NV) from naturally mineralized rock that had been collected at the Rosemont Project area. Round robin assays were compiled from a minimum of 25 samples of each SRM that had been sent to five or more laboratories. MEG Labs has certified the R4-suite of standards. Statistical analysis by MEG, based on round robin analysis of the standard material, has provided a mean grade, as well as +/- 2 standard deviations (95% confidence interval) acceptable limits.

There is a good match between the SRMs used and the average economic metal concentration in the drill samples. As such, the SRM grades are appropriate for the grade of the material being sampled for copper, molybdenum, and silver. A total of 196 standard samples were run within the analytical sequence for copper (not all of these had molybdenum or silver analyses).

Copper analysis performed well, with only 2 standards out of 196 being outside of the mean +/- 2 standard deviation limit (95% confidence interval). When these samples were later rerun by the laboratory, they returned with values within acceptable limits. When rerunning standards, the routine practice was to also run the samples that occurred before and after the standard in the analytical sequence. These before/after samples returned with values similar to the initial values, indicating that the actual sample analyses were repeatable and dependable, and that the difference in the standard grades was due to statistically normal variability.

Molybdenum analysis performed reasonably, with 18 standards out of 179 being outside of the mean +/- 2 standard deviation limit (95% confidence interval). When these samples were later rerun by the laboratory, 16 returned with values within acceptable limits and 2 that were not. The before/after samples returned with values similar to the initial values, even for the 2 rerun standards that were still out of limit. This indicates that the analyses for the actual samples were repeatable and dependable, and that statistically normal variability in the standard is attributed to the difference.

Silver analysis performed well, with 3 standards out of 110 being outside of the mean +/- 2 standard deviation limit. When these samples were later rerun by the laboratory, they returned with similar values. The before/after samples also returned with values similar to the initial values. The high degree of repeatability for both the reanalysis of standards and samples indicates these data to be reliable.

As was the case for the previous sample analysis work, the performance of the standard reference materials in the analytical stream was acceptable for the three economic metals under consideration.

Internal Laboratory Standards

The laboratory personnel internally included their own standard samples in the analytical sequence. These standards showed good repeatability of the analyses at or close to the certified values for copper, molybdenum and silver.

Blanks

Materials that contain metal concentrations at or below the analytical limits of detection (blanks) were also submitted with the 2008 drill cuttings and SRMs to monitor the limit of detection concentrations of

Cu, Mo and Ag at the assay lab. This was similar to blank material as was used in the previous drilling campaign.

A total of 93 blank samples were run within the analytical sequence for copper (not all of these had molybdenum or silver analyses). These consisted of quartz sand, identified as MEG S108002X blanks. Statistical analysis by MEG, based on round robin analysis demonstrated the absence of metallic elements.

Copper analysis performed well, with only 2 blanks out of 93 being slightly above the analytical threshold limit (0.01%). When these blanks were later rerun by the laboratory, they returned with acceptable values below analytical threshold limits. When rerunning blanks, the routine practice was to also run the samples that occurred immediately before and after the standard in the analytical sequence. These before/after samples returned with values similar to the initial values indicating that the drill hole samples themselves were repeatable and dependable.

Molybdenum analysis performed reasonably, with only 8 blanks out of 84 being slightly above the analytical threshold limit (0.001%). When these blanks were later rerun by the laboratory, 4 returned with acceptable values below analytical threshold limits. Three samples returned with values still above the analytical threshold limits (1 remaining blank was inadvertently not rerun). The before/after samples returned with values similar to the initial values indicating that the drill hole samples themselves were repeatable and dependable.

Silver analysis performed modestly, with 26 blanks out of 52 being slightly above the analytical threshold limit (0.1 g/t or 0.00292 opt). When these blanks were later rerun by the laboratory, 8 returned with acceptable values below analytical threshold limits. Fifteen samples returned with values still above the analytical threshold limits (3 remaining blanks were inadvertently not rerun or had insufficient pulp for rerunning). The before/after samples returned with values generally similar to the initial ones indicating general sample repeatability. Silver values for blanks, particularly in the 0.1 g/t (0.00292 opt) to 0.5 g/t (0.0146 opt) range, show some inconsistent results, part of which was checked further by a program of repeat analyses as discussed below.

Check Assays (Second Pulp Analysis)

Duplicate pulps were generated for 121 samples and a repeat analysis performed as a check of the original analytical values. Duplicate pulps were collected over time and were all run at the end of the sampling program. These pulps were run in batches of 16, accompanied by copper, molybdenum and silver standards, as well as a blank. In addition, each batch included the laboratories own internal standards.

Copper check analyses compares well with the original values. Of the 121 samples, 112 duplicate check analyses were within +/- 10 percent of the original values or 93 percent of the data. Most of the checks were less than 5 percent different and for those with differences did not indicate a bias. One data point reflects an apparent mixed up duplicate pulp sample.

Molybdenum check analyses compare reasonably with the original values. Molybdenum is present at relatively small levels, with the differences not indicating a bias. Again, the same data point mentioned above is present, reflecting an apparent sample mix up.

Silver check analyses compare reasonably with the original values. Silver is also present at relatively small levels, showing some variability, especially in the 0.1 g/t (0.00292 opt) to 0.5 g/t (0.0146 opt) range. It is important to note that the silver variability did not show any sort of preferential bias.

The external standards run in the duplicate pulp sample analytical sequence all returned values within acceptable limits. The external blanks came back with no values for copper or molybdenum. A few of the blanks returned silver values just slightly above analytical detection limits, just as was observed with the blanks that were contained within the routine sampling program.

Summary

The analytical quality assessment/quality control program demonstrated that the copper, molybdenum and silver grade values returned by Skyline were reliable for resource estimation work. The quality control results found for the 2008 drill samples were similar to those found in the previous drilling and analytical campaigns. Because of the relatively low levels of silver being measured, some variability was observed; however, there was no obvious grade bias. It is noted that copper accounts for approximately 80% of the mineral valuation, while molybdenum accounts for 15% and silver accounts for about 5%.

1.16 DATA VERIFICATION

Augusta took a number of steps to verify the results of earlier exploration results by other companies. Augusta's own work was conducted with appropriate sampling handling and QA/QC measures to ensure that resulting data were reasonable. Quality control measures for sample assaying are described in detail in Section 15.

A number of checks were made to appraise the validity of the data entry in the database after the completion the 2005, 2006 and 2008 drilling programs. A visual inspection was conducted comparing a random sampling of the values shown on the original assay certificates to those listed in the database files to check for data entry errors. For the 2005 drilling, fifty-two individual drill holes were inspected, representing approximately 14% of the total database up to that time. The sampling included some data from each of the drilling campaigns conducted by Anaconda, Anamax and Augusta. As no assay value errors were found, the data entry error rate for the group sampled was zero. Computer editing techniques were also employed as an additional check to search for out-of-range values, duplicate entries and depth from-to inconsistencies. One collar location elevation bust was found and corrected. No other errors were encountered.

Augusta's 2006 drilling campaign added 25 new resource drill holes to the database. A similar program to check assay certificates against entered values in the database was conducted. Seven of the new drill holes, representing approximately 28% of the total, were checked. One transposition error in a Cu value was found and one error involving an assay standard value replacing a Cu value was noted. Also, two from-to footage errors were also found. No other problems were found, and the errors were corrected in the database. The error rate for this sampled group was 0.20%.

During 2008 another 20 holes were drilled and 10 previously unsampled geotechnical holes from 2006 were sampled. Assay certificates for portions of 5 drill holes were checked against the drill hole database, representing 232 sample intervals, or approximately 6% of the new drill hole sample intervals, with no errors found.

WLRC is satisfied that the drill hole database is representative of the deposit. WLRC has not conducted any of its own sampling, as this was not deemed necessary.

1.17 ADJACENT PROPERTIES

The Peach-Elgin, Broadtop Butte and Copper World Mine deposits occur within 1.5-2.5 miles to the north and northwest of the Rosemont Deposit. These deposits consist of similar types of mineralization along related structural trends and are within the property package acquired by Augusta. The following summarizes the historical resource estimates for two of these deposits for informational purposes only. None of the resources estimates presented below are included in the Rosemont mineral resource estimates presented in Section 19.

In 1964, Anaconda produced a geological resource estimate for the Peach-Elgin deposit that identified 13,700,000 tons of sulfide material averaging 0.78% Cu and 9,700,000 tons of oxide material averaging 0.72% Cu. The estimate was based on assays from 67 churn and diamond drillholes. After calculation of that resource, Anaconda and Asarco drilled approximately 140 additional diamond drillholes, but did not update the 1964 estimate. The methodology of the 1964 estimate did not conform to modern NI 43-101 requirements but, as it was made by a reputable major copper company, it is taken as a fairly reliable estimate to be viewed in an historical context. *Augusta Resource Corporation has not done the work necessary to verify the classification of this resource and is not treating the above resource figures as a NI 43-101 defined resource verified by a Qualified Person and, therefore, the resource figures should not be relied upon by investors.*

Anamax carried out a resource estimate for the Broadtop Butte deposit in 1979. That estimate, based on approximately 18 widely-spaced (200-500 feet, or 60-150 meters) diamond drillholes, was 8,800,000 tons at an average grade of 0.77% Cu and 0.037% Mo. The estimate was made by a reputable major copper company and on that basis it is taken as a fairly reliable estimate to be viewed in an historical context. *Augusta Resource Corporation has not done the work necessary to verify the classification of this resource and is not treating the above resource figure as a NI 43-101 defined resource verified by a Qualified Person and, therefore, the resource figures should not be relied upon by investors.*

There are no historical or modern resource estimates for the Copper World Mine area.

1.18 MINERAL PROCESSING AND METALLURGICAL TESTING

1.18.1 Metallurgical Testing

The earliest existing records of metallurgical testing are from the period 1974 - 1975, at which time grinding and flotation tests were performed. In the first half of 2006, Augusta initiated test work to provide a better understanding of the metallurgy of the Rosemont deposit and establish the design criteria for the design of a process facility.

The copper sulfide ore contains two main types of copper mineralization: chalcopyrite and bornite/chalcocite. There are three major and several minor lithological units within which the two types of sulfide mineralization occur:

- Horquilla
- Earp
- Colina

- Other including Epitaph and Escabrosa

Two samples of ground Horquilla sulfide ore were examined by detailed mineralogical modal analysis. The result of this analysis indicates that there is a large difference in copper mineralogy within the Horquilla rock type. Silver appears to be associated mainly with the copper sulfide minerals as is minor gold. Molybdenite, MoS_2, is the only molybdenum mineral identified.

The copper oxide mineralization is principally chrysocolla, tenorite, malachite, and azurite. Oxide resources are distributed in three major rock units as follows:

- Arkose
- Porphyry – Quartz Manzanite (QMP) or Quartz Laterite (QLP)
- Andesite

For the most part, core samples from exploration drilling were used for metallurgical testing. Split core samples were used for most of the comminution and some leach tests, while coarse rejects were used for flotation testing. Whole core was used for some tests including the JK Drop-weight and impact crushing tests. Bulk surface samples were also taken for some of the column leach tests.

A fragmentation study was performed to predict the size distribution of ROM ore. The fragmentation study indicates that the ROM ore fed to the primary crusher will have a "Best Estimate" 80% passing size (P_{80}) of about 30 inches, a size distribution readily handled by a size (60" x 110") crusher.

The comminution test program consisted of:

- JK Drop-weight and Abrasion Test
- MinnovEX SAG Power Index Test (SPI)
- MacPherson Autogenous Grindability Test
- Bond Low-energy Impact (Crushing) Test
- Bond Rod Mill Work Index Test
- Bond Ball Mill Work Index Test
- Bond Abrasion Test
- Specific Gravity Determination

Grinding mill sizing parameters were provided to mill manufacturers for use in their mill sizing methods. The mill sizing parameters are shown in Table 1-11.

Table 1-11 Grinding Mill Sizing Parameters

Parameter	Value
CW_i	4.90
RW_i	12.40
BW_i	11.40
Tonnage	3,400 tph
SAG Mill Feed Size	150,000 µ
Transfer Size	3,000 µ
Ball Mill Product Size	105 µ

Flotation test work was performed during the years 1974-1975 and 2006-2008. The tests included bench-scale rougher-scavenger and cleaner tests, rougher variability tests, and rougher cleaner optimization tests. Based on the test results the flotation conditions were indicated to be as follows:

- Primary grinding to P80=105µ
- Rougher flotation pH= 9.7 to 10.8
- AP-238 and AX-343 collectors
- Regrind to P80= 74µ
- One stage of cleaner flotation

The rougher flotation variability tests examined the effect of grind size, ore grade, ore mineralogy, and ore depth on metal recovery. The result of the variability tests indicated that there is not a strong correlation between head grade, copper mineralogy (as determined by logging), and mining level and copper recovery in the samples tested. Previous early-stage testing determined that the degree of sample oxidation was the most significant factor in the metallurgical response.

The result of the variability tests indicated that the grind size has an effect on both copper recovery and rougher concentrate grade. The mineralogical modal analyses indicate that the chalcopyrite liberates at a coarser size, between 150 and 75µ, than do the bornite and chalcocite. The moly begins to liberate from the gangue between 150 and 75µ, but remains locked to a significant degree with gangue to about 22µ.

In the variability tests, only about 10 percent of the samples gave molybdenum recovery of 75% or higher, indicating that the variability test conditions were probably not optimum for moly recovery. Normally a molybdenum recovery of about 80 percent can be expected with a typical southern Arizona copper rougher concentrate. The result of sorting the variability test result for molybdenum recovery and ore elevation indicates no correlation between these variables.

Copper-moly and moly cleaner flotation tests indicate that the Rosemont sulfide ores should respond well to widely used and proven techniques. Reagent screening tests were performed that indicated recovery from the rock type composites could be improved by reagent selection.

The rougher cleaner optimization tests continue in progress. The tests are being conducted on two composite samples comprising first, the weighted ore type mix for the first three years of mine production, and second the weighted ore type mix for mine years four through six. Tests completed to date indicate that by using a newly developed flotation collector reagent the copper, molybdenum, and silver recovery and rougher concentrate grade will be improved from previous test work results.

Column leach tests were performed at a -1 inch particle size distribution on three composite samples. The samples used in the test work were Arkose, Quartz Latite Porphyry, and Andesite rock units. The copper minerals in the samples were: chrysocolla, tenorite, malachite, azurite, chalcocite, covellite, and minor chalcopyrite and bornite. The results of these tests indicate copper recovery for Arkose to be 41.2% and net acid consumption to be 20.3 lbs acid per lb of copper, for Quartz Latite Porphyry 60.5% and 2.2 lbs/lb, and for Andesite 53.1% and 10.2 lbs per lb.

Additional column leach tests were performed on Arkose, QMP, and Andesite ore samples. Column tests were run on the Andesite and QMP ore samples at particle sizes of -1, -2, and -4 inch. Column tests were run on the Arkose ore sample at particle sizes of -1 and -2 inch. The column leach test at -4 inch on the Arkose sample was not run since the as-received sample was nearly all -2 inch. Tests were performed at various irrigation rates and two tests were cured before leaching. The results of these tests indicate that at a -4 inch particle size distribution the copper recovery for Arkose can be predicted to be 75% and acid consumption to be 50 lbs acid per ton of ore leached, for QMP 70% and 10 lbs/ton, and for Andesite 70% and 60 lbs per ton.

1.18.2 Mineral Processing

Both sulfide and oxide copper ore will be processed. Sulfide ore will be transported from the mine to the primary crusher by off-highway haulage trucks then conveyed to the concentrator facility. Oxide ore will be transported from the mine to a run of mine heap leaching facility by the off-highway haulage trucks. Copper concentrate produced at the concentrator facility will be loaded into highway haul trucks and transported to a concentrate smelter and metal refinery.

Molybdenum concentrate produced at the concentrator facility will be bagged and loaded onto trucks for shipment to market. Oxide ore will be leached with acidic solution and the leach solution will be processed using solvent extraction electrowinning (SX-EW) technology to produce high purity cathode copper plates (cathodes). The copper cathodes will be loaded onto trucks for shipment to market.

The process selected for recovering the copper and molybdenite minerals can be classified as "conventional". The sulfide ore will be crushed and ground to a fine size and processed through mineral flotation circuits. The following items summarize the process operations required for sulfide ore:

- Size reduction of the sulfide ore by using a primary gyratory crusher to reduce the ore from run of mine (ROM) to minus 6 inches.

- Stockpiling primary crushed ore in a coarse ore storage building and then reclaiming by feeders and conveyor belt.

- Size reduction of the ore in a semi-autogenous (SAG) mill - ball mill grinding circuit prior to processing in a flotation circuit. The SAG mill will operate in closed circuit with a trommel screen and a pebble crushing circuit. The ball mills will operate in closed circuit with hydrocyclones.

- The flotation circuit will consist of copper and molybdenum flotation circuits. The copper and molybdenum minerals will be concentrated into a bulk copper/molybdenite concentrate. The molybdenite mineral will then be separated from the copper minerals in a molybdenite flotation circuit. The bulk (copper-moly) flotation circuit will consist of rougher flotation, concentrate regrind, cleaner flotation, and cleaner scavenger flotation circuits. The molybdenite flotation circuit will consist of copper-moly concentrate thickener, molybdenite rougher flotation, rougher cleaner flotation, concentrate regrind, second cleaner flotation, and third cleaner flotation circuits.

- Final copper concentrate will be thickened, filtered, and loaded in trucks for shipment. Final molybdenite concentrate will be filtered, dried, and packaged into shipping containers for shipment.

- Flotation tailing will be thickened, filtered, transported by a conveyor system, and dry stacked in a tailing impoundment area at the mill site.

The process selected for the recovery of copper from the oxide ore can be classified as "conventional". The oxide ore will be heap leached and the copper recovered from the leach solution using solvent extraction – electrowinning technology.

ROM ore will be trucked from the mine to the leaching area. The ore will be stacked on the leach pad and irrigated with an acidified leach solution (raffinate). The leach solution will percolate through the leach pile and dissolve soluble copper from the ore before being directed along the impermeable leach pad liner system to the solution collection system. The copper bearing solution called pregnant leach solution, or PLS, will be treated in the solvent extraction electrowinning (SX-EW) circuit.

Copper contained in the PLS (aqueous phase) will be extracted from the aqueous phase solution by contact with organic reagents carried in an organic solution (organic phase) in the solvent extraction circuit. Copper transferred to the organic phase will be stripped from the organic solution by contact with an aqueous solution (aqueous phase), acidic electrolyte solution (lean electrolyte) that will have circulated through the electrowinning cells in the electrowinning circuit. This transfer of copper enriches the electrolyte solution to form the rich electrolyte. The rich electrolyte will be returned to the electrowinning cells for copper electrowinning onto stainless steel cathode blanks. Copper loaded on the stainless steel blanks will be harvested from the electrowinning cells on a weekly schedule. Copper will be removed from the stainless steel blanks by processing through a stripping machine. Copper plates produced by this process, LME Grade A, will be weighed and bundled into 2 to 3 ton packages for shipment to market.

The solvent extraction plant will consist of one train of mixer-settler tanks. The train will have two stages of extraction in series and one stage of stripping. The electrowinning circuit tankhouse will contain twenty-four electrowinning cells.

1.19 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

Updated mineral resource estimates for the Rosemont Deposit are presented in this report to support a press release made on October 23, 2008. The new mineral resource estimates include additional sample assays taken from 20 new drill holes and from 10 geotechnical holes that were previously unsampled.

A three-dimensional (3D) block model of the Rosemont Deposit was constructed and mineral resources were estimated using Mintec's MineSight® mining software package. The subsections that follow describe the parameters and methodology for this work.

1.19.1 Model Extents

The mine coordinate system is based on UTM NAD 83 standards. The UTM NAD 83 Zone 12 coordinates in metric values were converted to Imperial units (i.e., feet). Block dimensions of 50 ft by 50 ft by 50 ft were selected as appropriate to adequately model the deposit geology and to also reflect the proposed mining bench height for the project. Table 1-12 summarizes the limits of the 3D block model expressed in mine coordinates. The current model covers the same area as the previous 2007 model.

Table 1-12 Deposit Model Limits

Direction	Minimum	Maximum	Block Size (ft)	No. of Blocks
X (East)	1,710,000	1,722,000	50	240
Y (North)	11,550,000	11,560,000	50	200
Z (Elevation)	2,500	6,500	50	80

1.19.2 Surface Topography

The topographic data for the project area was captured from an aerial survey flown by Cooper Aerial Surveys Company of Tucson, Arizona in the summer of 2006. The vertical datum is based on the NAVD 88 standard. Cooper provided electronic files with elevation data on 10-foot contour intervals covering the project area.

The topographic surface elevations were then loaded into 2D surface and 3D block model files in MEDSystem®. A block model variable stores the percentage of each block below topography.

1.19.3 Drill Hole Database

The Rosemont Deposit drill hole database contains collar locations, down-hole deviation surveys, sample assay results and geological information from several recent drilling programs by Augusta Resources and from a series of exploration drilling campaigns conducted by a number of companies in the past (see Table 1-10). This mineral resource update includes sample assay information from 30 additional drill holes. Ten of the holes were drilled primarily for geotechnical information but have since been sampled and assayed. The remaining 20 new holes tested areas of extended mineralization and provided some in-fill information. In all, 249 resource drill holes now comprise the drillhole database contained within the area used for grade modeling. Some holes drilled for metallurgical information were used for geologic modeling purposes. The drilling programs included a good mix of vertical and inclined holes designed to test both the shallower stratagraphic units and the high angle structures. The majority of the holes were drilled using diamond core, although a small number of holes, less than five percent, were started by open-hole rotary techniques, but the mineralized zones were drilled with core.

Stored in the drill hole database used for grade modeling are 56,499 individual sample values representing approximately 323,800 feet (98,700 meters) of drilling. Each sample interval record contains values for Cu, Mo and Ag. Intervals in the upper parts of the drill holes were also commonly assayed for recoverable oxide Cu.

During the period from 1964 to 1983, Rosemont samples analyzed by the Anaconda and Anamax labs were processed via a first-pass x-ray method to screen out low grade or waste samples. Sample values greater than 0.2% Cu and 0.02% Mo were then re-assayed by wet chemical techniques. Values for both methods are entered in the database; however, the question previously arose as to the suitability of the lower XRF assay values for grade estimation in the model. A statistical study was conducted to determine the correlation coefficient between XRF and wet chemical values for both Cu and Mo. The study shows excellent agreement with correlation coefficients of 0.944 for Cu and 0.874 for Mo (see Figures 1-9 and 1-10 in Section 1.26). These results indicate that the lower grade XRF values would be valid for use in grade estimation in the model.

A number of checks were made to appraise the validity of the data entry in the database. A visual inspection was conducted comparing a random sampling of the values shown on the original assay certificates to those listed in the database files to check for data entry errors. Fifty two individual drill holes were inspected in March 2006, representing approximately 14% of the total database up to that time. The sampling included some data from each of the drilling campaigns conducted by Anaconda, Anamax and Augusta. As no assay value errors were found, the data entry error rate for the group sampled was zero. Computer editing techniques were also employed as an additional check to search for out-of-range values, duplicate entries and depth from-to inconsistencies. One collar location elevation bust was found and corrected. No other errors were encountered. Augusta's 2006 drilling campaign added 25 new resource drill holes to the database. A similar program to check assay certificates against entered values in the database was conducted. Seven of the new drill holes, representing approximately 28% of the total, were checked. One transposition error in a Cu value was found and one error involving an assay standard value replacing a Cu value was noted. Also, two from-to footage errors were also found. No other problems were found, and the errors were corrected in the database. The error rate for this sampled group was 0.20%. In September 2008 a validity check of a random sampling of assay intervals from five of the new thirty drill holes added in 2008 was conducted. Approximately 6% of the assay intervals were checked with no errors encountered.

A major Ag sampling and assaying program was undertaken in 2006. Limited Ag assaying had been done in the past and the intent of this program was to provide sufficient new sample values to allow Ag grade estimation in the model. Approximately 20% of the drill hole database was checked for Ag data entry problems. No errors were found in the final database compilation.

A statistical study was re-done based on the updated database which included the 30 additional drill holes. Included in the study were frequency distribution histograms for each rock type and lognormal cumulative probability graphs for Cu, Mo and Ag for the deposit as a whole (see Figures 1-11, 1-12 and 1-13 in Section 1.26). As one might expect, the addition of samples from only 30 holes had minimal effect on the overall statistics for the deposit. High grade outliers are common in skarn-type deposits and the Rosemont Deposit is no exception. Inspection of the cumulative probability graph for all Cu assays shows an inflection point in the curve at approximately 10% Cu. The high grade outlier portion of the population above the 10% Cu threshold accounts for approximately 0.20% of the total population, but, if left unadjusted, would bias the model grade estimation upward. For that reason, the Cu assays were capped at 10.0% Cu. A similar situation existed with Ag and a cap was applied at 3.0 ounces per ton. The cumulative probability plot of Mo grades exhibits a better behaved population, with no high grade outlier segment; consequently, no Mo grade capping adjustments were made.

1.19.4 Geologic Model

The geologic model of the deposit was re-visited and some minor revisions to fault positions and lithologic contacts were made based on information from the recent drilling campaign. Three new lithologic units have been added to the model (Martin West, Epitaph North and Tertiary Gravel). In all, 19 individual lithology types were delineated (see Table 1-13 in Section 1.19.10). Material not defined in the model was assigned a code of 20.

Previous geologic models identified only oxide and sulfide mineralization. The newly revised model includes interpreted boundaries for oxide, mixed (i.e., transitional) and sulfide zones. Most of the mixed zone was previously considered to be sulfides.

The 3-D model was checked visually on computer screens and by plotting and reviewing model level plans. Problem areas from the block tagging algorithm were noted and adjustments/corrections were made.

1.19.5 Mineralization Controls

In this deposit, all of the rock types are mineralized to some degree. Some lithologies are significantly better hosts due to favorable protolith composition and/or close relationship to feeder structures.

1.19.6 Compositing of Drill Hole Data and Statistics

The drill hole sample assay intervals were weight averaged to 50-foot composites on even level intervals to approximate a potential mining bench height. Geological rock type unit codes were added to the composites by back-assignment from model blocks. All further statistical analyses and model grade estimation were based on these composite data. Frequency distribution histograms and cumulative probability plots were again generated for the individual rock types using the Cu, Mo and Ag composite grades. Coefficients of variation for of all rock types were 1.21 for Cu, 0.97 for Mo and 1.24 for Ag. These values are very much in line with what one would expect in this type of deposit.

1.19.7 Variography

Variograms were re-calculated to determine if the additional 30 holes had caused a change in the continuity directions and ranges of mineralization. Again, each rock type was reviewed separately, but definitive variograms could not be developed for many of the rock types because not enough composite data points were available. This was especially true for Mo composite variograms. Ultimately, all the individual rock type composites were grouped to provide variograms (see Figures 1-14, 1-15 and 1-16) from which parameters could be selected for the block grade estimation equations. A spherical model was fit to each of the experimental variograms and the following parameters were selected:

	Cu	Mo	Ag
Nugget =	0.00326	0.00009	0.00592
Sill =	0.15194	0.00035	0.01998
Range =	264 ft (80 m)	251 ft (76 m)	254 ft (77 m)

The model parameters obtained in this current study were identical to those obtained from the previous 2007 drill hole database (excluding the new 30 holes) and no revisions to the mineralization continuity directions or ranges were required. The general orientations of the primary direction variograms for Cu and Ag were at azimuths of approximately 110-130° and a dips of -40° to -45°. This is consistent with the measured dip angles of the sedimentary rock formations. The secondary direction follows the general strike of the beds at azimuths of 10-30° with a northerly plunge of 0° to -20°. No clear preferential directions could be determined for Mo, so an omni-directional variogram was selected.

1.19.8 Block Grade Interpolations

Ordinary kriging was selected as the interpolation method to estimate model block grades because of the low coefficients of variation exhibited by the Cu, Mo and Ag composite grade populations. The search ellipse alignment and ranges used in the interpolation process were oriented to reflect the mineralized trends and continuity ranges detected in the variogram analysis. The primary direction for Cu is 110° azimuth, -45° dip, with a range of 264 feet (80 m), and the secondary direction is 10° azimuth, 20° plunge, with a range of 227 feet (69 m). Mo used a circular, omni-directional search radius of 251 feet (76 m). The primary direction for Ag is 130° azimuth, -40° dip, with a range of 254 feet (77 m). The secondary direction for Ag interpolation is 30° azimuth, 0° plunge, with a range of 232 feet (71 m). The Z search direction was held to 110 feet (34 m) in all cases.

A maximum of nine and a minimum of two composites, with only three composites allowed from any one drill hole, were used in the calculation of any one block grade. The majority of the rock units were interpolated independently so as to maintain the integrity of the individual formations. However, because of similar grade populations and lithologies, the Horquilla Limestone and Earp Formations were grouped and interpolated together. Oxidation boundaries (i.e., oxide, mixed and sulfide zones) were also respected by independently interpolating block grades in separate passes for each zone.

For purposes of projecting grades for inferred blocks, a second-pass grade interpolation was made with a 350-ft search distance. This was applied only to blocks that did not receive a grade assignment using the above search parameters.

1.19.9 Resource Classification

Resources were classified into measured, indicated and inferred categories following Canadian NI 43-101 compliant standards. The category assignments are based on composite to block distances and the number of composites used in the kriging calculations. A block was designated as measured if it was within 75 feet (22.9 m), roughly 30 percent of the variogram range, and was estimated by at least three drill holes. A block was considered to be indicated if it was within the variogram range of 260 feet (79.2 m) and was estimated by at least two drill holes, or was within 75 feet and less than three drill holes were used for estimation. A block was designated as inferred if it was greater than 260 feet from any drill hole or did not meet the minimum number of drill holes required for the indicated classification.

1.19.10 Material Densities

Table 1-13 lists the bulk tonnage factors that were assigned in the block model according to rock type. A default tonnage factor of 12.00 ft^3/ton was used where no lithology codes exist.

Table 1-13 Rock Types and Bulk Tonnage Factors

Rock/Formation Description	Rock Code	Tonnage Factor ($feet^3$/ton)
Overburden, unconsolidated	1	13.72
Epitaph Formation	2	12.11
Colina Limestone	3	11.69
Earp Formation	4	11.73
Horquilla Limestone	5	11.18
Escabrosa Limestone	6	11.56
Martin Formation	7	11.98
Quartz Monzonite Porphyry	8	12.31
Mesozoic Andesite	9	11.53
Willow Canyon Arkose	10	12.08
Glance Conglomerate/Ls	11	11.68
Scherrer Formation	12	12.00
Abrigo Formation	13	11.35
Concha Limestone	14	12.11
Bolsa Quartzite	15	11.91
Precambrian Granite	16	11.91
Epitaph North	17	12.11
Martin West	18	11.98
Undefined	19	12.00
Undefined	20	12.00
Tertiary Gravel	21	13.72

1.19.11 Mineral Resource Estimate

The mineral resource estimation work was performed by or under the direction of Mr. William Rose, P.E., WLRC's Principal Mining Engineer and an independent Qualified Person under the standards set forth by Canadian National Instrument 43-101 (Mr. Rose's qualifications are described in Section 24). The mineral resource estimates were based on the above described deposit model and bulk tonnage factors, and were constrained by a floating cone pit shell based on a copper price of $3.50/lb, a molybdenum price of $35.00/lb and a silver price of $14.00/oz.

Measured and indicated mineral resource estimates for the Rosemont Deposit are summarized in Tables 1-14 and 1-15, respectively. The combined measured and indicated mineral resource estimates are presented in Table 1-16. Inferred mineral resource estimates are shown in Table 1-17. The mineral resource estimates contained herein are effective as of October 22, 2008. Imperial units are used in these estimates, where tons refer to short tons (2000 lbs). Cu refers to copper, Mo refers to molybdenum and Ag refers to silver. For comparison with previous mineral resource estimates (WLRC, April 21, 2006 and WLRC, April 26, 2007), copper equivalent (CuEqv) values are based on prices of $1.25/lb Cu, $18.00/lb Mo and $8.50/oz Ag, with no applied recovery factors.

Table 1-14 Rosemont Deposit – Measured Mineral Resources

Material / Cutoff (% Cu)	Ktons	% Cu	% Mo	Ag Oz/ton	% CuEqv*	lbs Cu (millions)	lbs Mo (millions)	oz Ag (millions)	lbs CuEqv* (millions)
Oxides:									
0.10	21,600	0.20	-	-	0.20	85	-	-	85
0.15	14,600	0.23	-	-	0.23	68	-	-	68
0.20	7,500	0.30	-	-	0.30	45	-	-	45
Mixed:									
0.15	4,900	0.65	0.007	0.08	0.78	64	0.7	0.4	76
0.20	4,800	0.66	0.007	0.08	0.79	64	0.7	0.4	76
0.25	4,700	0.67	0.007	0.08	0.80	63	0.7	0.4	75
0.30	4,500	0.69	0.007	0.08	0.82	62	0.6	0.4	73
Sulfides:									
0.15	132,300	0.50	0.016	0.14	0.78	1,330	42.3	18.4	2,060
0.20	119,100	0.54	0.016	0.15	0.82	1,280	38.1	17.6	1,950
0.25	106,900	0.58	0.017	0.16	0.87	1,230	36.4	16.6	1,870
0.30	96,100	0.61	0.017	0.16	0.91	1,170	32.7	15.6	1,750

* Equivalency based on prices of $1.25/lb Cu, $18.00/lb Mo and $8.50/oz Ag, with no applied recovery factors.

Table 1-15 Rosemont Deposit – Indicated Mineral Resources

Material / Cutoff (% Cu)	Ktons	% Cu	% Mo	Ag Oz/ton	% CuEqv*	lbs Cu (millions)	lbs Mo (millions)	oz Ag (millions)	lbs CuEqv* (millions)
Oxides:									
0.10	81,700	0.20	-	-	0.20	332	-	-	332
0.15	51,400	0.25	-	-	0.25	260	-	-	260
0.20	27,400	0.33	-	-	0.33	180	-	-	180
Mixed:									
0.15	34,300	0.49	0.005	0.05	0.58	334	3.4	1.5	394
0.20	33,500	0.50	0.005	0.05	0.58	332	3.3	1.5	391
0.25	32,200	0.51	0.005	0.05	0.59	326	3.2	1.5	383
0.30	29,400	0.53	0.005	0.05	0.62	311	2.9	1.4	363
Sulfides:									
0.15	464,500	0.44	0.014	0.11	0.68	4,120	130.1	52.0	6,340
0.20	404,700	0.48	0.015	0.12	0.74	3,910	121.4	49.0	5,990
0.25	351,200	0.52	0.016	0.13	0.80	3,680	112.4	45.7	5,610
0.30	305,200	0.56	0.016	0.14	0.84	3,430	97.7	42.1	5,120

* Equivalency based on prices of $1.25/lb Cu, $18.00/lb Mo and $8.50/oz Ag, with no applied recovery factors.

Table 1-16 Rosemont Deposit – Combined Measured and Indicated Mineral Resources

Material / Cutoff (% Cu)	Ktons	% Cu	% Mo	Ag Oz/ton	% CuEqv*	lbs Cu (millions)	lbs Mo (millions)	oz Ag (millions)	lbs CuEqv* (millions)
Oxides:									
0.10	103,400	0.20	-	-	0.20	417	-	-	417
0.15	66,000	0.25	-	-	0.25	328	-	-	328
0.20	35,000	0.32	-	-	0.32	224	-	-	224
Mixed:									
0.15	39,100	0.51	0.005	0.05	0.60	398	4.1	1.9	471
0.20	38,300	0.52	0.005	0.05	0.61	396	4.0	1.9	467
0.25	36,900	0.53	0.005	0.05	0.62	389	3.9	1.9	458
0.30	33,900	0.55	0.005	0.05	0.64	373	3.5	1.8	436
Sulfides:									
0.15	596,800	0.46	0.014	0.12	0.70	5,440	172.4	70.4	8,410
0.20	523,800	0.50	0.015	0.13	0.76	5,190	159.5	66.6	7,940
0.25	458,100	0.54	0.016	0.14	0.82	4,910	148.8	62.3	7,480
0.30	401,300	0.57	0.016	0.14	0.86	4,600	130.4	57.7	6,870

* Equivalency based on prices of $1.25/lb Cu, $18.00/lb Mo and $8.50/oz Ag, with no applied recovery factors.

Table 1-17 Rosemont Deposit – Inferred Mineral Resources

Material / Cutoff (% Cu)	Ktons	% Cu	% Mo	Ag Oz/ton	% CuEqv*	lbs Cu (millions)	lbs Mo (millions)	oz Ag (millions)	lbs CuEqv* (millions)
Oxides:									
0.10	30,400	0.24	-	-	0.24	147	-	-	147
0.15	17,800	0.33	-	-	0.33	117	-	-	117
0.20	12,700	0.39	-	-	0.39	100	-	-	100
Mixed:									
0.15	21,100	0.35	0.004	0.02	0.41	148	1.7	0.3	175
0.20	19,100	0.37	0.004	0.01	0.43	141	1.5	0.3	164
0.25	14,500	0.42	0.004	0.02	0.48	121	1.2	0.2	139
0.30	12,200	0.45	0.003	0.02	0.49	109	0.7	0.2	121
Sulfides:									
0.15	208,800	0.38	0.007	0.06	0.50	1,600	29.2	12.1	2,110
0.20	160,600	0.45	0.008	0.07	0.59	1,440	25.7	10.9	1,880
0.25	133,800	0.49	0.008	0.08	0.63	1,320	21.4	10.0	1,700
0.30	105,000	0.56	0.008	0.09	0.70	1,170	16.8	8.9	1,470

* Equivalency based on prices of $1.25/lb Cu, $18.00/lb Mo and $8.50/oz Ag, with no applied recovery factors.

Oxide, mixed and sulfide mineral resources have been segregated in the above estimates as the average grades between these material types are significantly different. Moreover, if the project is developed, oxide, mixed and sulfide ore will likely be treated by different processing methods, with different costs, recoveries and cutoff grades. *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*

It should be noted that there is environmental and political opposition to the development of the Rosemont open pit copper mining project. The right to mine and extract the mineral resources will be subject to obtaining permits and approvals from federal and state agencies. There are well documented procedures in place related to obtaining these environmental and permitting approvals, which are subject to background data gathering, technical application preparation, agency review, public review and specified administrative procedures.

In August of 2007, ASARCO filed a lawsuit against Augusta Resource Corporation and others alleging that an unfair sale of the Rosemont property had taken place in 2004. The lawsuit is presently in the discovery stage and no estimate is available as to when a judgment may be rendered. Augusta believes the case is without merit and will prevail in this legal action.

Rosemont Deposit mineral resources are on mostly patented and some unpatented lands owned by Augusta Resource Corporation. Notwithstanding the existence of a 3% NSR mineral royalty, the ASARCO litigation and the existence of environmental and political groups opposing the development of the project as noted above, the estimates of mineral resources are not affected by any other known legal, title, taxation, socio-economic, marketing, political or other relevant issues.

The estimates of mineral resources will not be materially affected by mining, metallurgical, infrastructure or other relevant technical factors. The metallurgical characteristics of the Rosemont mineral resource are substantially similar to other deposits successfully mined and processed in the area. The greater Tucson area has seen the development of numerous large-scale open pit copper mines, and has an experienced labor force and well developed infrastructure to support a new mining project.

1.19.12 Additional Mineral Resource Potential

Previous work by Anaconda, Anamax, and ASARCO found significant areas of mineralization to the north and northeast of the Rosemont Deposit on the Rosemont Property. These deposit areas at Broadtop Butte, Copper World and Peach-Elgin are characterized by similar styles of mineralization and occur along related structural zones to that of the Rosemont Deposit. Historic drilling intercepted significant copper grades in often widely spaced holes, constituting encouraging targets for further exploration.

1.19.13 Pit Limit Analyses

The information outlined in Sections 1.19.13 through 1.19.15 of this report pertains to the estimation of mineral reserves for the open pit development of the Rosemont deposit. Lerchs-Grossman (LG) analyses were conducted using the deposit model described in Sections 1.19.1 through 1.19.10. Only measured and indicated mineral resources were considered to have potential economic value in the generation of economic pit limits and in the definition of mineral reserves; all inferred mineral resources were treated as waste.

An economic subroutine was developed to compute a Net Smelter Return (NSR) value for each block in the deposit model. This computer algorithm incorporates block grades, expected smelting/refining contracts (i.e., payables and deductions), metallurgical recoveries and projected market prices for each metal (Cu, Mo and Ag) to yield a net revenue value expressed in terms of US dollars per ton. The subroutine also applies to mining, ore processing and general/ administration costs to calculate a net dollar value per block, which includes adjustments for surface topography.

Metal recoveries were derived from preliminary results of metallurgical test work conducted by Mountain States Research and Development, Inc. (MSRDI) and SGS (Lakefield and MinnovEX divisions) under the direction of M3 Engineering and Technology Corporation (M3). Table 1-18 presents the metallurgical recoveries used in the pit optimization evaluations and subsequent mineral reserve estimation. Only the three primary metals – copper, molybdenum and silver – were modeled and used in the revenue calculations. No recovery of molybdenum and silver from oxide ore is projected.

Table 1-18 Metallurgical Recoveries Used in Pit Optimization Evaluations

Metal	Oxide Ore	Sulfide Ore
Copper	60 %	90 %
Molybdenum	-	63 %
Silver	-	80 %

Table 1-19 summarizes the economic parameters used in the base case pit optimization evaluations of the Rosemont deposit. Consistent with current market conditions, no price participation charges were included in the concentrate processing costs.

Table 1-19 Base Case Economic Parameters for Pit Optimization

Metal Prices:	
Copper (Cu)	$ 1.75 / lb Cu
Molybdenum (Mo)	$ 15.00 / lb Mo
Silver (Ag)	$ 10.00 / troy oz
Operating Costs (excl oxide leaching):	
Base ore mining	$ 0.775 / ton
Base waste mining	$ 0.862 / ton
Incremental haulage (below pit rim at 5050 ft elevation)	$ 0.028 / ton / bench
Sulfide ore milling & flotation	$ 3.30 / ton ore
General/administration	$ 0.26 / ton ore
Oxide Copper Ore Processing:	
Cu oxide freight & refining	$ 0.00 / lb Cu
Acid consumption	28.6 lbs acid / ton ore
Cost of acid	$ 0.05 / lb acid
Other processing/leaching	$ 0.76 / ton ore
Copper Concentrate Processing:	
Cu grade in concentrate	30 %
Cu realization	96.5 %
Cu concentrate transportation	$ 75.00 / dry ton
Cu concentrate treatment	$ 55.00 / dry ton
Cu refining	$ 0.055 / lb Cu
Ag realization	90.0 %
Ag refining	$ 0.40 / troy oz Ag
Molybdenum Concentrate Processing:	
Mo grade in concentrate	50 %
Mo realization	90.0 %
Mo concentrate transportation	$ 0.00 / dry ton
Mo treatment & refining	$ 0.00 / lb Mo
NSR royalty	3 %

The base and incremental mining costs, when applied to the material contained within the base case Lerchs-Grossman pit shell, yield an average mining cost of nearly $1.07 per ton of material. Mining costs near the pit bottom – below 3750 level, will exceed $1.50 per ton in 2008 US dollars.

Bulk tonnage factors were read from the block model and combined with volume adjustments for surface topography effects, if any, to determine block tonnages. For each optimized pit case, net profit values were calculated for each model block by subtracting on-site operating costs (mining, ore processing and G&A) from the NSR value, then multiplying the result by the block tonnage.

Overall slope angles used for the Lerchs-Grossman analyses were derived from geotechnical recommendations made by Call & Nicholas, Inc. (CNI) for pit slope designs. The overall slopes were adjusted to accommodate CNI's recommended slope angles and the anticipated placement of internal haulage ramps along the pit walls in certain design sectors. The resulting overall slope angles are summarized in Table 1-20.

Table 1-20 Overall Slope Angles Used in Lerchs-Grossman Analyses

Design Sector	Slope Angle
1	42°
2	43°
3	46°
4	39°
5	46°
6	43°
7	41°
8	48°
9	48°
10	28°
11	35°
12	33°
13	37°
14	35°

All Lerchs-Grossman analyses were restricted to prevent the pit shells from crossing the topographic ridge immediately west of the deposit. This was done to minimize potential visual impacts to the ridge line.

The base case Lerchs-Grossman pit shell is defined by the recoveries and economic parameters listed in Tables 1-21 and 1-22, respectively. The metal prices of $1.75/lb Cu, $15.00/lb Mo and $10.00/oz Ag are below a three-year trailing average. This pit shell contains about 535 million tons of measured and indicated sulfide mineral resources above an internal NSR cutoff of $3.56/ton and nearly 72 million tons of measured and indicated oxide mineral resources above a $2.19/ton NSR cutoff. The resulting stripping ratio is about 2.0:1 (tons waste per ton of ore).

Additional Lerchs-Grossman runs were made to evaluate sensitivities to metal prices and to operating, concentrate freight and treatment costs. These sensitivities were generally conducted in 5% increments to +20% and –45% of the base case parameters. A re-blocking methodology was applied to the 3D block model for these sensitivity cases to speed up the computing process. Re-blocking is simply combining several adjacent blocks of the model into one larger block. In this part of the study, two blocks in each of the x, y, and z directions were combined. The Lerchs-Grossman results from these re-blocked cases will be slightly different from the cases with no re-blocking. Tables 1-21 and 1-22 present the results of the pit optimization analysis and sensitivities to metal price and costs. Generally, the pit resource is more sensitive to metal prices than costs. Whether due to higher metal prices or lower costs, the expansion of the pit is limited due to physical constraints of the west ridge and the facilities to the east.

The estimates presented in Tables 1-21 and 1-22 should not be confused with mineral reserves, which are based on open pit designs that incorporate access, operating, geotechnical and other criteria in addition to economic constraints. The Lerchs-Grossman results should not be relied upon, but do provide an indication of potential mineral reserves that must be validated by proper designs. *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*

1.19.14 Pit Designs

The ultimate Rosemont pit was designed for large-scale mining equipment (specifically, 70-cu-yd electric shovels and 320-ton haulage trucks) and was derived from the base case Lerchs-Grossman pit shell described in the previous section. The design process included smoothing pit walls, eliminating or rounding significant noses and notches that may affect slope stability, and providing access to working faces by developing internal ramps.

The slope angles recommended by CNI and used for the design of the Rosemont ultimate pit and internal mining phases are presented in Table 1-23. Figure 1-17 illustrates the locations of the slope design sectors referenced in this table.

Table 1-21 Lerchs Grossman Results* - Metal Price Sensitivities

	Prices			Internal Cutoffs		Sulfide Mineral Resources** Above Internal NSR Cutoffs					Oxide Mineral Resources** Above Internal NSR Cutoffs			Waste	Total	Strip
Sensitivity	Cu $/lb	Mo $/lb	Ag $/oz	Sulfide	Oxide	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %	Ktons	Ktons	Ratio
+20%	2.10	18.00	12.00	3.56	2.19	601,381	16.24	0.44	0.014	0.11	77,796	4.43	0.17	1,313,906	1,993,083	1.93
+15%	2.01	17.25	11.50	3.56	2.19	596,591	15.58	0.45	0.014	0.12	77,713	4.24	0.17	1,317,356	1,991,660	1.95
+10%	1.93	16.50	11.00	3.56	2.19	587,650	14.96	0.45	0.014	0.12	77,711	4.07	0.17	1,311,612	1,976,973	1.97
+5%	1.84	15.75	10.50	3.56	2.19	575,361	14.33	0.46	0.015	0.12	72,054	4.02	0.17	1,288,901	1,936,316	1.99
Base	**1.75**	**15.00**	**10.00**	**3.56**	**2.19**	**566,632**	**13.68**	**0.46**	**0.015**	**0.12**	**72,011**	**3.83**	**0.17**	**1,286,685**	**1,925,328**	**2.01**
-5%	1.66	14.25	9.50	3.56	2.19	548,512	13.07	0.47	0.015	0.12	65,848	3.77	0.18	1,243,702	1,858,062	2.02
-10%	1.58	13.50	9.00	3.56	2.19	522,739	12.49	0.48	0.015	0.12	65,600	3.59	0.18	1,175,366	1,763,705	2.00
-15%	1.49	12.75	8.50	3.56	2.19	510,835	11.82	0.48	0.015	0.12	59,424	3.52	0.19	1,166,786	1,737,045	2.05
-20%	1.40	12.00	8.00	3.56	2.19	470,554	11.26	0.49	0.015	0.13	53,248	3.44	0.19	1,050,138	1,573,940	2.00
-25%	1.31	11.25	7.50	3.56	2.19	450,771	10.59	0.50	0.016	0.13	47,587	3.34	0.20	1,022,570	1,520,928	2.05
-30%	1.23	10.50	7.00	3.56	2.19	413,260	10.01	0.51	0.016	0.13	42,172	3.26	0.21	950,978	1,406,410	2.09
-35%	1.14	9.75	6.50	3.56	2.19	370,205	9.42	0.53	0.016	0.14	35,974	3.17	0.22	872,122	1,278,301	2.15
-40%	1.05	9.00	6.00	3.56	2.19	334,853	8.76	0.54	0.016	0.14	30,210	3.07	0.23	825,124	1,190,187	2.26
-42.5%	1.01	8.63	5.75	3.56	2.19	317,190	8.45	0.56	0.016	0.14	25,046	3.10	0.24	816,299	1,158,535	2.39
-45%	0.96	8.25	5.50	3.56	2.19	284,464	8.12	0.56	0.016	0.15	21,084	3.09	0.26	755,939	1,061,487	2.47

* Re-blocking applied - 2 blocks in x,y,z directions
** Only measured and indicated mineral resources are reported above; all inferred mineral resources are treated as waste.

Table 1-22 Lerchs Grossman Results* - Cost Sensitivities

	Prices			Internal NSR Cutoffs		Sulfide Mineral Resources** Above Internal NSR Cutoffs					Oxide Mineral Resources** Above Internal NSR Cutoffs			Waste	Total	Strip
Sensitivity	Cu $/lb	Mo $/lb	Ag $/oz	Sulfide	Oxide	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %	Ktons	Ktons	Ratio
+20%	1.75	15.00	10.00	4.27	2.63	511,379	14.26	0.48	0.015	0.12	59,351	4.13	0.19	1,166,231	1,736,961	2.04
+15%	1.75	15.00	10.00	4.09	2.52	517,380	14.16	0.48	0.015	0.12	59,424	4.13	0.19	1,164,052	1,740,856	2.02
+10%	1.75	15.00	10.00	3.92	2.41	537,912	13.99	0.47	0.015	0.12	65,600	3.98	0.18	1,225,555	1,829,067	2.03
+5%	1.75	15.00	10.00	3.74	2.30	551,609	13.83	0.47	0.015	0.12	65,848	3.98	0.18	1,253,962	1,871,419	2.03
Base	**1.75**	**15.00**	**10.00**	**3.56**	**2.19**	**566,632**	**13.68**	**0.46**	**0.015**	**0.12**	**72,011**	**3.83**	**0.17**	**1,286,685**	**1,925,328**	**2.01**
-5%	1.75	15.00	10.00	3.38	2.08	573,141	13.58	0.46	0.015	0.12	72,054	3.83	0.17	1,282,310	1,927,505	1.99
-10%	1.75	15.00	10.00	3.20	1.97	585,954	13.42	0.45	0.014	0.12	77,711	3.70	0.17	1,296,930	1,960,595	1.95
-15%	1.75	15.00	10.00	3.03	1.86	597,127	13.28	0.45	0.014	0.12	77,713	3.70	0.17	1,316,819	1,991,659	1.95
-20%	1.75	15.00	10.00	2.85	1.75	605,224	13.17	0.44	0.014	0.11	83,277	3.57	0.16	1,317,488	2,005,989	1.91

* Re-blocking applied - 2 blocks in x,y,z directions
** Only measured and indicated mineral resources are reported above; all inferred mineral resources are treated as waste.

Table 1-23 Pit Slope Angle Recommendations

Design Sector	Sector Location	Slope Angles		Bench Face Angle	Bench Height (ft)
		Interramp	Overall		
1	Regional Fault – NW	42°	-	58°	50
2	Limestone – North	43°	-	60°	100
3	Limestone – N to NE	46°	-	66°	100
4	Limestone – South	45°	-	65°	100
5	Limestone – West	46°	-	66°	100
6	Limestone – ENE	46°	-	66°	100
7	Limestone – Central	48°	-	68°	100
8	Bolsa – NW	48°	-	68°	100
9	Bolsa - West	48°	-	68°	100
10	Alluvium / Overburden	-	28°	-	50
11	Willow Canyon Frm	-	35°	-	50
12	Willow Canyon Frm	-	33°	-	50
13	Willow Canyon Frm	-	37°	-	50
14	WC Outside Sectors	-	35°	-	50

The slope angles allow for catch bench widths of 50-53 feet in the limestones/skarns and Bolsa Formation where the pit slopes are double-benched (i.e., vertical catch bench intervals of 100 feet). Slopes will be single-benched (i.e., on 50-foot intervals) in alluvium and arkose rock types, providing catch bench widths – toe to crest – of 25 to 48 feet. Interramp slopes and, hence, catch bench widths in alluvium and arkose (Willow Canyon Formation) vary according to the slope height and presence of groundwater.

The remaining parameters used in the designs of the ultimate pit and mining phases are presented in Table 1-24.

Table 1-24 Pit Design Parameters

Bench height	50 ft
Bench face angle	58-68°
Catch bench interval – alluvium & arkose	50 ft
Catch bench interval – all other rock types	100 ft
Road width (including ditch & safety berm)	125 ft
Nominal road gradient	10 %
Minimum pushback width	300 ft

Mining phase, or pushback, widths are typically in excess of 300 feet, although operating widths were occasionally reduced to about 250 feet in limited areas. For the bottom of the ultimate pit, ramps were reduced to a single 70-foot lane (with berm and ditch) and maximum gradients were increased to 12%.

The ultimate pit design (Phase 7) is presented in Figure 1-18. Six internal phases, or pushbacks, were also developed to define the starter pit and general extraction sequence for the proposed mine. The ultimate pit will be about 6,000 feet wide east-west and 6,500 feet wide north-south. The pit bottom will reach the 3,050 foot elevation. The west wall will be about 2,900 feet high, while the east wall height will reach over 2,000 feet in some areas.

1.19.15 Mineral Reserve Estimate

Rosemont mineral reserves have been estimated from only measured and indicated mineral resources; all inferred resources have been treated as waste. Inferred mineral resources have a great amount of uncertainty as to their existence and as to whether they can be mined legally or economically. It cannot be assumed that all or any part of inferred mineral resources will ever be upgraded to a higher category.

The mining phase and ultimate pit (see Figure 1-18) designs were applied to the 3D block model of the deposit to estimate contained tonnages and grades. All mineral reserve estimates are reported in US units.

The base case price and operating cost estimates presented in Section 1.19.13 (see Tables 1-18 and 1-19) were used to define ore grade material in the mineral reserve estimates. Cutoff grades were based on computed NSR values, which were derived from metal prices of \$1.75/lb Cu, \$15.00/lb Mo and \$10.00/oz Ag. All prices and costs are in US dollars.

No recovery of molybdenum and silver is projected from oxide ore leaching and only quartz monzonite porphyry (QMP), andesite and arkose rock types were considered as potential oxide leach ore (no NSRs were computed for oxide Paleozoic formations and other oxide rock types). An internal NSR cutoff of \$3.56/ton was used for sulfide mill ore and \$2.19/ton was used for oxide leach ore.

Bulk material densities, which vary by rock type, were read from values stored in the block model. These assignments are described in more detail in Section 1.19.10 (see Table 1-13). Generally, rock tonnage factors range between 11.18 to 13.72 ft^3/ton and average about 11.85 ft^3/ton for the rock contained within the ultimate pit

The Rosemont deposit is a well-disseminated polymetallic deposit that has large ore zones above the anticipated internal cutoff grade. It was felt that the sample compositing and block grade interpolation process used to construct the deposit block model incorporated sufficient dilution and, hence, no additional internal dilution factors were applied to the resource model.

Mineralized zones represented in the 3D block resource model are made up of relatively large contiguous blocks of ore, with ore being defined as the mineralization above the NSR cutoff value. However, there are areas where blocks of ore are isolated and surrounded by waste. Conversely, there are areas where isolated blocks of waste are surrounded by ore. Large cable shovels will be used and high mining rates are being planned to ensure the lowest possible unit costs for the mine operation. It will be unproductive to selectively mine either the isolated ore or waste blocks. It will also be difficult to determine the precise ore-waste contact represented in the resource model. With the planned bulk mining method, ore dilution to the reserve model is necessary to reflect the run of mine production from the mining operation. Preliminary assessment of the ore distribution represented by the resource model estimates that the mining dilution is 4%, and this factor was applied to the pit sulfide resource to arrive at the run of mine pit reserve. The dilution grades used are 0.199% Cu, 0.007% Mo, and 0.05 oz/ton Ag. No ore losses were applied as the strategy will be to recover ore zones with some dilution rather than treat them as waste.

The mineral reserve estimates presented in this report were prepared by Mr. Robert Fong, P. Eng., Principal Mining Engineer for Moose Mountain Technical Services. Mr. Fong meets the requirements of an independent qualified person under NI 43-101 standards. The mineral reserve estimates are effective as of the date of this report.

Proven mineral reserves for the Rosemont deposit are summarized by mining phase in Table 1-25 and probable mineral reserves are presented in Table 1-26. Table 1-27 lists the combined proven and probable mineral reserve estimates and waste rock for the Rosemont deposit.

Table 1-25 Proven Mineral Reserves by Phase

	Sulfides >= 3.56 $/ton NSR Cutoff					Oxides >= 2.19 NSR Cutoff		
Phase	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %
1	15,119	16.03	0.54	0.017	0.14	7,780	3.81	0.17
2	14,832	13.27	0.44	0.014	0.12	2,315	4.71	0.21
3	17,928	13.20	0.44	0.013	0.12	3,893	3.80	0.17
4	15,258	15.04	0.51	0.014	0.16	418	3.94	0.18
5	18,344	14.49	0.50	0.013	0.13	1,825	3.57	0.16
6	36,291	13.95	0.49	0.015	0.13	19	2.21	0.10
7	24,227	13.93	0.43	0.020	0.13	0	0.00	0.00
Total	141,999	14.19	0.48	0.015	0.13	16,250	3.91	0.18

(NSR values are based on metal prices of $1.75/lb Cu, $15.00/lb Mo and $10.00/oz Ag.)

Table 1-26 Probable Mineral Reserves by Phase

	Sulfides >= 3.56 $/ton NSR Cutoff					Oxides >= 2.19 NSR Cutoff		
Phase	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %
1	34,317	15.90	0.53	0.017	0.14	20,168	3.67	0.17
2	28,481	13.06	0.43	0.014	0.13	9,084	4.82	0.22
3	43,231	11.47	0.39	0.012	0.10	10,825	3.62	0.16
4	39,329	14.09	0.48	0.014	0.13	2,453	3.53	0.16
5	41,564	13.76	0.47	0.013	0.12	10,421	3.33	0.15
6	129,827	12.59	0.45	0.014	0.10	773	2.83	0.13
7	87,590	12.91	0.41	0.017	0.12	0	0.00	0.00
Total	404,339	13.12	0.45	0.015	0.11	53,724	3.77	0.17

(NSR values are based on metal prices of $1.75/lb Cu, $15.00/lb Mo and $10.00/oz Ag.)

Table 1-27 Combined Proven and Probable Mineral Reserves by Phase

	Sulfides >= 3.56 $/ton NSR Cutoff					Oxides >= 2.19 NSR Cutoff			Waste	Total	Strip
Phase	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %	Ktons	Ktons	Ratio
1	49,436	15.94	0.53	0.017	0.14	27,948	3.71	0.17	120,651	198,035	1.56
2	43,313	13.13	0.43	0.014	0.13	11,399	4.80	0.22	85,368	140,080	1.56
3	61,159	11.98	0.40	0.012	0.11	14,718	3.67	0.16	97,205	173,082	1.28
4	54,587	14.36	0.49	0.014	0.14	2,871	3.59	0.16	107,533	164,991	1.87
5	59,908	13.98	0.48	0.013	0.12	12,246	3.36	0.15	110,954	183,108	1.54
6	166,118	12.89	0.46	0.014	0.11	792	2.81	0.13	479,066	645,976	2.87
7	111,817	13.13	0.41	0.018	0.12	0	0.00	0.00	230,688	342,505	2.06
Total	546,338	13.40	0.45	0.015	0.12	69,974	3.80	0.17	1,231,465	1,847,777	2.00

(NSR values are based on metal prices of $1.75/lb Cu, $15.00/lb Mo and $10.00/oz Ag. Inferred mineral resources are included in waste estimates.)

At prices of $1.75/lb Cu, $15.00/lb Mo and $10.00/oz Ag, proven and probable sulfide mineral reserves within the designed Rosemont ultimate pit total nearly 546 million tons grading 0.45% Cu, 0.015% Mo and 0.12 oz Ag/ton. Proven and probable oxide mineral reserves total about 70 million tons grading 0.17% Cu. The pit contains a total of about 1.85 billion tons of material, of which 616 million tons are mineral reserves and 1.23 billion tons are waste rock, resulting in a stripping ratio of 2.0:1 (tons waste per ton of ore). Contained metal in the sulfide (proven and probable) mineral reserves is estimated at 4.93 billion pounds of copper, 161 million pounds of molybdenum and 65 million ounces of silver. Contained metal in proven and probable oxide mineral reserves is estimated at 241 million pounds of copper.

Nearly 26% of the sulfide mineral reserves in the Rosemont ultimate pit are classified as proven and the remainder (74%) is considered probable. Only about 23% of the oxide mineral reserves are classified as proven. The classifications are limited by the relatively wide-spaced exploration drilling in the Rosemont deposit. *All of the mineral reserve estimates reported above are contained in the mineral resource estimates presented in Section 1.19.11.*

The Rosemont ultimate pit contains approximately 54 million tons of inferred sulfide mineral resources and nearly 8 million tons of inferred oxide mineral resources that are above respective sulfide and oxide NSR cutoffs of $3.56/ton and $2.19/ton. These resources are included in the waste estimates presented in Table 1-27. *Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Inferred mineral resources have a great amount of uncertainty as to their existence and as to whether they can be mined legally or economically. It cannot be assumed that all or any part of inferred mineral resources will ever be upgraded.*

All of the mineral reserve estimates presented in this report are dependent on market prices for the contained metals, metallurgical recoveries and ore processing, mining and general/administration cost estimates. Mineral reserve estimates in subsequent evaluations of the Rosemont deposit may vary according to changes in these factors. There are presently no other known mining, metallurgical, infrastructure or other relevant factors that may materially affect the mineral reserve estimates.

It should be noted that there is some local environmental and political opposition to the development of the Rosemont open pit copper mining project. However, the right to mine and extract the mineral reserves is subject to obtaining permits and approvals from federal and state agencies. There are well documented procedures in place related to obtaining these environmental and permitting approvals, which are subject to background data gathering, technical application preparation, agency review, public review, and specified administrative procedures.

Rosemont mineral reserves are on mostly patented and some unpatented lands owned by Augusta Resource Corporation. Notwithstanding the existence of a 3% NSR mineral royalty and the existence of opposition to the development of the project as noted above, the estimates of mineral reserves are not affected by any known legal, title, taxation, socio-economic, marketing, political, or other relevant issues.

1.20 OTHER RELEVANT DATA AND INFORMATION

To the best of the Author's knowledge, all relevant data and information has been addressed elsewhere in this report.

1.21 INTERPRETATION AND CONCLUSIONS

1. Rosemont is a common porphyry deposit with appreciable amounts of molybdenum and silver by-product.

2. The Rosemont deposit has significant reserves and resources with a current projected mine life of 21 years based on a productive rate of 75,000 short tons per day.

3. The property is located near enough to Green Valley, Sahuarita, and Tucson, Arizona to draw upon the considerable trained personnel in the area. Staff for nearby mines has been stable due to proximity to an attractive urban area.

4. Proper diligence has taken place with respect to drilling program, sampling, and metallurgical testing.

5. Upon success in obtaining the necessary permits, Rosemont will become a major employer of skilled employees in Southern Arizona.

6. The mine will locally alter existing terrain. Efforts beyond the norm (such as dry stacked tailings) are being taken to minimize disturbance to surface and sub-surface features.

7. Reclamation will comply with legal requirements and also take into account local resident concern.

8. Metal prices reflective of the past three years, metal prices of the past three years blended with two years futures pricing, and an assumed more conservative long term metal prices result in after tax IRR's of 37.5%, 28.5% and 17.8%. Financial analysis for current (December 31, 2008) spot prices show an after tax IRR of 7.7%. The latter calculation may represent a condition near the bottom of the metals price cycle.

1.22 RECOMMENDATION

The Rosemont Project has metrics indicative of a stable and continuous hard rock mining operation. Based on this assessment, the project should press forward with engineering in anticipation of receiving the necessary permits. This work would largely consist of two main tasks, for an estimated US$25 million, as outlined below:

- Environmental Permitting: Work to receive environmental permits required for construction and operation. Individual permits detailed in Section 25 of this report. Work is currently in process.
- Supporting Engineering: Post-feasibility study engineering to develop plans in support of the permitting work. This may include the procurement of some vendor engineering data. Work is currently in process.

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1.24 DATE AND SIGNATURES

The effective date of this Technical Report is January 14, 2009.

The principal author and Qualified Person for this Technical Report is Dr. Conrad Huss, P.E. of M3 Engineering and Technology Corporation. Other contributing Qualified Persons to this report are Mr. William L. Rose, P.E. of WLR Consulting, Inc.; Mr. Robert Fong, P.E., of Moose Mountain Technical Services; Mr. John Ajie, P.E., of URS Washington Division; and Mr. Thomas L. Drielick, P.E. of M3 Engineering and Technology Corporation. The Certificate of Qualified Person and resume for each are presented in the appendix of this report.

1.25 Additional Requirements for Technical Reports on Development Properties and Production Properties

1.25.1 Mine Operations

The Rosemont deposit is a large tonnage, skarn-hosted, porphyry-intruded, copper-molybdenum deposit located in close proximity to the surface and is amenable to open pit mining methods. The proposed pit operations will be conducted from 50-foot-high benches using large-scale equipment, including: 12.25-inch-diameter rotary blasthole drills, 70-cu-yd electric mining shovels, 36-cu-yd front-end loaders, 320-ton off-highway haul trucks, 580- to 850-hp crawler dozers, 500-hp rubber-tired dozers, 270- to 500-hp motor graders and 30,000-gallon off-highway water trucks.

Mining Plan

Mining sequence plans were developed to depict mining progress at regular intervals and to serve as the basis for a mine production schedule. The sequence plans were developed from the phase designs described in Section 1.19.14 and target a sulfide (mill) ore production rate of 75,000 tpd. Oxide ore will be delivered to the leach pad as it is encountered during stripping operations.

The operating and scheduling criteria used to develop the mining plans are summarized in Table 1-28 below. Pit and mine maintenance operations will be scheduled around the clock.

Table 1-28 Mine Production Scheduling Criteria

Annual Sulfide Ore Production Rate	27,375,000 tons
Daily Sulfide Ore Production Rate	75,000 tons
Operating Hours per Shift	12
Operating Shifts per Day	2
Operating Days per Week	7
Scheduled Operating Days per Year	365
Number of Mine Crews	4

The mill ramp-up schedule used for Year 1 production targets is presented in Table 1-29. Quarterly mill production in Year 2 will average about 6.844 million tons, equivalent to an annual rate of 27.375 million tons.

Table 1-29 Mill Ramp-up Schedule (Year 1)

Month	% of Full Production	Monthly Ktons	Quarterly Ktons
1	40	913	
2	50	1,141	3,423
3	60	1,369	
4	70	1,597	
5	80	1,825	5,475
6	90	2,053	
7	95	2,167	
8	100	2,281	6,729
9	100	2,281	
10	100	2,281	
11	100	2,281	6,843
12	100	2,281	
Total Year 1	82	22,470	22,470

Preproduction stripping will be conducted over a 15-month time period and will ramp up according to the delivery of mining equipment (particularly electric shovels) and the hiring and training of work crews. The long-term and peak mining rates suggest the use of at least three large (70-cu-yd) electric shovels and two large (36-cu-yd) front-end loaders. The preproduction stripping ramp-up was based on the delivery of the first loading unit 15 months prior to mill startup with successive deliveries of operating shovels and/or front end loaders on three-month intervals until the last shovel is placed into production six months before startup.

Mining crews would typically be expanded every one to two months to allow time for hiring and training. Crew efficiencies would start off at reduced levels and increase with experience. Table 1-30 summarizes the mine's preproduction stripping ramp-up schedule.

Table 1-30 Mine Preproduction Stripping Ramp-up Schedule

Preproduction Month	Monthly Ktons	Quarterly Ktons
1	285	
2	550	1,688
3	853	
4	2,233	
5	2,406	7,803
6	3,164	
7	4,860	
8	5,191	15,564
9	5,513	
10	7,673	
11	8,281	24,078
12	8,124	
13	8,509	
14	8,271	25,454
15	8,674	
Total	74,587	74,587

Mining sequence plans were developed on a quarterly basis through the end of Year 2 and on an annual basis through Year 7. Additional plans include mining progress through the end of Year 10, Year 15 and Year 21 (end of mining). Based on these sequence plans, the estimated mine production schedule is presented in Table 1-31. The proven and probable mineral reserves summarized in this schedule were based on an internal NSR cutoff of $3.56/ton for sulfides and $2.19/ton for oxides. All inferred mineral resources were treated as waste.

The totals in Table 1-31 match the mineral reserve estimates presented in Section 1.19.15 (see Table 1-27). Approximately 2 million tons of proven and probable sulfide mineral reserves will be stockpiled during preproduction stripping, with approximately 1 million tons reclaimed during Year 1 – augmenting direct ore deliveries from the pit. Some of the remaining stockpiled sulfide ore will be reclaimed in Year 11 and most is left until Year 21, representing an average ROM stockpile and crushed (in process) inventory.

Overburden and other waste rock encountered in the course of mining will be placed into a waste rock storage (WRS) area located to the southeast and south of the planned open pit and into the dry stack tailings area, where dewatered mill tailings will be placed behind waste rock containment buttresses. The dry stack tailings area is north of the WRS area and east-northeast of the pit. The oxide ore heap leach pad will be located between the dry stack tailings area and the initial WRS area. The WRS, leach pad and dry stack tailings facilities are fully contained within the Barrel drainage basin.

A mine life of 20.1 years is projected by this development plan. Peak mining rates of 318,000 tpd of total material will be realized in Year 1 and 313,000 tpd in Year 2. Typical mining rates during Years 3-6 will be 224,000 tpd of waste rock and oxide ore, or 299,000 tpd of total material (including 75,000 tpd of sulfide ore). Minimum oxide ore will be recovered after Year 6, and typical mining rates during Years 7 to 10 will be 299,000 tpd of ore and waste.

Table 1-31 Mine Production Schedule - Combined Proven & Probable Mineral Reserves

(NSR values are based on metal prices of $1.75/lb Cu, $15.00/lb Mo and $10.00/oz Ag. All inferred mineral resources are treated as waste.)

Time	Mined Sulfides >= 3.56 $/ton NSR Cutoff					Reclaimed Sulfide Ore Stockpile					Total Mill Feed					Oxides >= 2.19 NSR Cutoff			Waste	Total	Strip
Period	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %	Ktons*	Ktons**	Ratio
PP Q1	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	1,688	1,688	0.00
PP Q2	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	7,802	7,802	0.00
PP Q3	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	142	3.21	0.15	15,421	15,562	108.78
PP Q4	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	3,140	3.69	0.17	20,429	23,569	5.60
PP Q5	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	5,365	3.61	0.16	18,577	23,942	2.70
Y1 Q1	2,448	9.32	0.30	0.011	0.11	977	8.78	0.26	0.013	0.11	3,425	9.16	0.29	0.012	0.11	6,406	3.75	0.17	19,091	28,922	2.16
Y1 Q2	5,476	10.49	0.33	0.013	0.12	0	0.00	0.00	0.000	0.00	5,476	10.49	0.33	0.013	0.12	6,301	3.83	0.17	17,699	29,476	1.50
Y1 Q3	6,730	14.51	0.46	0.017	0.14	0	0.00	0.00	0.000	0.00	6,730	14.51	0.46	0.017	0.14	5,438	3.68	0.17	17,562	29,730	1.44
Y1 Q4	6,844	17.11	0.56	0.020	0.15	0	0.00	0.00	0.000	0.00	6,844	17.11	0.56	0.020	0.15	2,529	4.43	0.20	18,470	27,843	1.97
Y2 Q1	6,844	16.49	0.55	0.018	0.14	0	0.00	0.00	0.000	0.00	6,844	16.49	0.55	0.018	0.14	5,554	5.12	0.23	16,445	28,843	1.33
Y2 Q2	6,844	15.82	0.53	0.017	0.13	0	0.00	0.00	0.000	0.00	6,844	15.82	0.53	0.017	0.13	3,532	4.81	0.22	18,466	28,842	1.78
Y2 Q3	6,844	15.62	0.54	0.015	0.12	0	0.00	0.00	0.000	0.00	6,844	15.62	0.54	0.015	0.12	2,644	3.86	0.17	19,354	28,842	2.04
Y2 Q4	6,844	13.93	0.49	0.012	0.11	0	0.00	0.00	0.000	0.00	6,844	13.93	0.49	0.012	0.11	3,021	3.46	0.16	17,978	27,843	1.82
Y3	27,375	11.51	0.39	0.011	0.09	0	0.00	0.00	0.000	0.00	27,375	11.51	0.39	0.011	0.09	9,629	3.59	0.16	72,369	109,373	1.96
Y4	27,375	13.95	0.46	0.014	0.14	0	0.00	0.00	0.000	0.00	27,375	13.95	0.46	0.014	0.14	3,901	3.64	0.16	78,094	109,370	2.50
Y5	27,375	11.43	0.38	0.014	0.08	0	0.00	0.00	0.000	0.00	27,375	11.43	0.38	0.014	0.08	1,821	3.27	0.15	80,177	109,373	2.75
Y6	27,375	13.07	0.44	0.012	0.13	0	0.00	0.00	0.000	0.00	27,375	13.07	0.44	0.012	0.13	9,758	3.30	0.15	71,241	108,374	1.92
Y7	27,375	13.48	0.47	0.011	0.12	0	0.00	0.00	0.000	0.00	27,375	13.48	0.47	0.011	0.12	0	0.00	0.00	81,997	109,372	3.00
Y8 to Y10	82,125	14.45	0.50	0.013	0.13	0	0.00	0.00	0.000	0.00	82,125	14.45	0.50	0.013	0.13	0	0.00	0.00	245,491	327,616	2.99
Y11 to Y15	136,685	13.06	0.47	0.014	0.11	190	8.78	0.26	0.013	0.11	136,875	13.05	0.47	0.014	0.11	0	0.00	0.00	339,995	476,871	2.49
Y16 to Y21	139,761	13.45	0.42	0.018	0.12	851	8.78	0.26	0.013	0.11	140,612	13.42	0.42	0.018	0.12	0	0.00	0.00	53,911	194,523	0.39
Total	544,320	13.42	0.45	0.015	0.12	2,018	8.78	0.26	0.013	0.11	546,338	13.41	0.45	0.015	0.12	69,181	3.81	0.17	1,232,258	1,847,777	2.00

* Includes 793 k-tons of oxide ore scheduled for after Yr 7 that may be destined as waste
** Includes sulfide ore reclaimed from stockpile

Mine Equipment

Equipment requirements for mine operations were derived from the production scheduling criteria listed in Table 1-28 and the mine production schedule presented in Table 1-31. Specific manufacturer's models used in this study are only intended to represent the size and class of equipment selected. The final equipment manufacture selection will be performed at a later stage of the project.

A summary of fleet requirements by time period for major mine equipment is shown in Table 1-32. This represents equipment necessary to perform the following mine tasks:

- Mine site clearing and topsoil salvage and stockpiling.
- Construction of the main haul road, with exception of initial haul roads built in the first two quarters of preproduction by a contractor.
- Production drilling.
- Loading and hauling of sulfide ore to the primary crusher (located on the east side of the pit), oxide ore to the leach pad and waste material to waste rock storage (WRS) areas.
- Maintain mine haulage and access roads.
- Maintain WRS areas and regrading of slopes and final surfaces.

The majority of mining equipment is new, with the exception of: a 3.5-inch track drill, the water trucks (30,000 gal.), equipment transport unit and fuel and lube trucks. Equipment operating time is based on 10.5 hours out of a 12-hour shift and equipment mechanical availabilities (MA) that vary depending on time period.

Blasting operations will be performed by a licensed contractor, who will provide all specialized storage, mixing and product delivery equipment. Powder factors will vary between 0.25-0.35 pounds per ton, depending on material, and will average 0.31 pounds per ton over the life of the project. Ammonium nitrate and fuel oil (ANFO) blasting agents will be loaded directly into dry holes, while wet holes will be pumped and sleeved first.

Table 1-32 Major Mine Equipment Fleet Requirements

	PP-2	PP-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11-15	Yr 16-20	Yr 21
Major Equipment:															
Diesel Blasthole Drill, 12.25-in.	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Electric Blasthole Drill, 12.25-in.		1	1	1	1	1	1	1	1	1	1	1	1	1	1
Drill, Sec/Pioneer, 3.5-in. (Used)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cable Shovel, 70-cu-yd	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Spare Shovel Dipper		1	1	1	1	1	1	1	1	1	1	1	1	1	1
Front-End Loader, 36-cu-yd	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Haulage Truck, 320-ton	9	13	18	18	20	20	20	20	20	20	20	20	24	12	12
Crawler Dozer, D11T-class	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Crawler Dozer, D10T-class	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
RT Dozer, 834H-class	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Front-End Loader, 8-cu-yd	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Excavator, 8-cu-yd	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Motor Grader, 24H-class	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Motor Grader, 16H-class	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Water Truck, 30,000-gal.	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Water Truck, 20,000-gal.	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel and Lube Truck, 50 Ton	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2

Mine Personnel

Mine manpower requirements were estimated on the basis of working two 12-hour shifts per day, seven days per week, 52 weeks per year. A standard, four-crew rotating work schedule will be used for around-the-clock coverage for craft labor and direct-line supervision. Table 1-33 summarizes the mine personnel requirements for the Rosemont Project. Peak manpower levels are reached in Years 11-15, with 45 salaried and technical personnel and 229 craft workers, for a total of 274 people.

Table 1-33 Mine Manpower Summary

	PP-2	PP-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8-10	Yr 11-15	Yr 16 -21
Mine Supervision & Technical Personnel	28	44	45	45	45	45	45	45	45	45	45	36
Mine Operations	50	105	130	130	136	136	136	136	136	136	150	80
Mine Maintenance	14	59	76	77	75	75	75	75	75	75	79	40
Total Mine Personnel	92	208	251	252	256	256	256	256	256	256	274	156

1.25.2 Metallurgical Testing

The earliest existing records of metallurgical testing are from the period 1974 - 1975, at which time grinding and flotation tests were performed. In the first half of 2006, Augusta initiated test work to provide a better understanding of the metallurgy of the Rosemont deposit and establish the design criteria for the design of a process facility.

The test work is documented in the following reports:

- "Rosemont Copper Project, Arizona Preliminary Development of Flotation Flowsheet Using Selected Composite Sample (Primarily Chalcocite Bornite Mineralization) Project 6054" June 26, 2006, Mountain States R&D International, Inc., Vail, Arizona
- "Preliminary Mineralogical Assessment of the Rosemont Deposit", July 13, 2007, G&T Metallurgical Service Ltd, Kamloops, British Columbia, Canada
- "Comminution Testing Hazen Project 10568 Report and Appendix", May 14, 2007, Hazen Research, Inc., Golden, Colorado

- “Ore Grindability Characterization And Preliminary Grinding Circuit Design For The Rosemont Deposit”, January 2007, SGS Lakefield Research Limited, Toronto, Ontario, Canada
- “Proposed Grinding System For The Rosemont Deposit Based On Small-Scale Data”, February 15, 2007, SGS Lakefield Research Limited, Toronto, Ontario, Canada
- “MSRDI Project 6087”, July 27, 2007, Mountain States R&D International, Inc., Vail, Arizona
- “Final Report Preliminary Column Leach Of Oxide Samples From The Rosemont Deposit Project 6052”, July 18, 2006, Mountain States R&D International, Inc., Vail, Arizona
- “Final Report Rosemont Project Metallurgical Testing and Process Engineering “Part I – Hydrometallurgical Aspects of Heap Leaching” Project No. 6087”, June 15, 2007, Mountain States R&D International, Inc., Vail, Arizona
- “Progress Report Current Standing In Regard To Projected Metallurgy Associated With Mine Plan Years 1 thru 3 and Years 4 thru 7, Reference MSRDI Project Number 6118 and 6118-A”, December 23, 2008, Mountain States R&D International, Inc., Vail, Arizona

1.25.3 Processing Flowsheets

Both sulfide and oxide copper ore will be processed. Sulfide ore will be transported from the mine to the primary crusher by off-highway haulage trucks then conveyed to the concentrator facility. Oxide ore will be transported from the mine to a run of mine heap leaching facility by the off-highway haulage trucks. Copper concentrate produced at the concentrator facility will be loaded into highway haul trucks and transported to a concentrate smelter and metal refinery. Molybdenum concentrate produced at the concentrator facility will be bagged and loaded onto trucks for shipment to market. Oxide ore will be leached with acidic solution and the leach solution will be processed using solvent extraction electrowinning (SX-EW) technology to produce high purity cathode copper plates (cathodes). The copper cathodes will be loaded onto trucks for shipment to market.

The process selected for recovering the copper and molybdenite minerals can be classified as “conventional”. The sulfide ore will be crushed and ground to a fine size and processed through mineral flotation circuits.

The process selected for the recovery of copper from the oxide ore can be classified as "conventional". The oxide ore will be heap leached and the copper recovered from the leach solution using solvent extraction – electrowinning technology.

The process is depicted in the summary flow sheet shown in Figure 1-19.

1.25.4 Extraction Rates

An estimate of the metal production has been made based on the test results applied to the estimated tons of sulfide ore to be mined. The sulfide ore metal recovery is shown in Table 1-34.

Table 1-34 Estimated Concentrate Grade & Metal Recovery

	% Metal Recovery			% Mass Recovery
	Cu	Mo	Ag	
Overall Recovery - Year 1 through 3	85.00%	72.00%	77.00%	1.83%
Overall Recovery - Year 4 through 6	83.00%	65.00%	76.00%	1.83%
Overall Recovery - Year 7 through end	84.00%	55.60%	77.90%	1.83%
Cu and Ag to Copper Concentrate Mo to Molybdenite Concentrate				

An estimate of the metal production from oxide ore has been made based on the test results. The test result copper recoveries must be discounted to allow for ROM particle size, lift heights greater than 20 feet, solution channeling effects, and incomplete leaching on side-slopes to predict the result of a full scale heap. Similarly, test results acid consumption should be reduced to allow for ROM particle size, the normal over-estimation of acid consumption in column tests, and the ability of good operators to minimize acid consumption. It is recommended that operations should maintain a low acid concentration in the raffinate and not "cure" any of the rock types. The adjusted copper recovery and acid consumption indicators are shown in Table 1-35.

Table 1-35 Adjusted Leach Parameters

Item	Units	Ore Sample		
		Andesite	Arkose	QMP
Copper Recovery	%	64	69	64
Acid Consumption	lb/ton ore	40	35	7

1.25.5 Process Reagents

Reagent consumption rates for the full scale plant operation have been estimated from the test results. The estimated reagent consumption rates for sulfide ore processing are shown in Table 1-36.

Table 1-36 Estimated Reagent Consumption Rates for Sulfide Ore

Item	Rate lbs/ton ore
Copper Circuit	
Collector, Aero Promoter 8944	0.040
Collector, C7	0.098
Frother, Methyl Isobutyl Carbinol (MIBC)	0.026
Collector, #2 Diesel Fuel	0.026
Sodium Meta Silicate	0.140
Lime (90% CaO)	1.797
Molybdenite Circuit	
Sodium Hydrosulfide	0.184
Sodium Meta Silicate	0.005
#2 Diesel Fuel	0.003
Methyl Isobutyl Carbinol (MIBC)	0.003
Flomin D-910	0.018

Reagent consumption rates for the full scale heap leach and SX-EW operation are shown in Table 1-37.

Table 1-37 Estimated Reagent Consumption Rates For Heap Leach and SX-EW

Item	Rate
	lb/ton ore
Sulfuric Acid (lb/ton ore)	30.0
	lb/lb cathode
Extractant	0.0002
Diluent	0.001
Cobalt Sulfate	0.0004
Guar	0.010
Mist Suppressor (FC 1100)	0.0003
D.E. (filter precoat)	0.040
Clay	0.040

1.25.6 Power

The supply of power for the Rosemont mine and process facilities will fall within Tucson Electric Power (TEP) and TRICO service territories with the majority being in the TEP service area. TEP will be the main electrical utility service provider for the facility. This will result in one utility rate and bill from TEP with a breakdown of revenue between TEP and TRICO transparent to the project.

The estimated connected load for the project is 139 MW, and will be supplied by a minimum of a 138 kV line to site. The estimated operating load for the project is approximately 106 MW.

Four options to connect to the existing TEP and/or Southwest Transmission Cooperative (SWTC) facilities were explored during the Feasibility Study completed in August 2007. A summary discussion of the options is in Section 1.25.6 of this Technical Report and in Section 8 of the August 2007 Feasibility Study.

Subsequent to the August 2007 study, three more detailed options were considered during preparation of a draft System Impact Study (SIS), conducted by TEP with the assistance of Navigant Consulting, during 2008.

Subsequent to the draft SIS, an additional option, "Option D", was developed and selected as the most direct route from available source to destination at a new Rosemont Substation and to the Rosemont Mill site. Rosemont owns fee lands at both Wilmot Junction and at the Rosemont Mill site. The Option D routing is still subject to review and consideration of additional routing alternatives by the Arizona Corporation Commission (ACC).

The "Option D" proposed by Rosemont, accesses initial construction power from the existing 46 kV line at the Greaterville substation (4.5 miles new line), or from an existing 46kV line at Wilmot Junction (Section 25, 12 miles new line). Either of these two 46 kV lines can provide an interim source of power for initiation of construction.

For the higher operating power load required at the mine, new construction of 16 miles of 138 kV line is required. The first 4 miles upgrade the TEP transmission system to a new Rosemont substation at or near Wilmot Junction (Section 25), and are subject to tax gross-ups. The distal 12 miles of 138 kV line are assumed retained by Rosemont and transmit power from the Rosemont Substation at Wilmot Junction to the Rosemont mill.

This Option D was developed by KR Saline, engineers of Arizona, to utilize planned and scheduled system upgrades as included in long term planning documents on file with the Arizona Corporation Commission.

This Option D is described in detail by KR Saline as follows:

Option D in contrast to Options A, B, and C that have been studied. <u>Option D consists of the following as depicted on the relevant drawings</u>:

- **Stage 1–** Rosemont would site, build and own a <u>12 mi.</u> 138 kV and 46 kV double circuit line from Rosemont mine to the SE corner of Sec 25 of T17S-R14E. Rosemont's 46 kV line would be interconnected to the TEP 46 kV line at that location for construction power. Metering would be at the mine substation.
- **Stage 2 –** Rosemont would site, build and <u>initially</u> own <u>4 mi</u>. of 138 kV line on the section line and traversing westerly from the SE corner of Sec 25 of T17S-R14E and parallel to the existing TEP 46 kV line. This new 138 kV line would tap the existing TEP South to Green Valley 138 kV line in proximity of Trico's Sahuarita substation. This is the proposed point of 138 kV interconnect with TEP. If TEP claims there is not sufficient capacity then TEP

should complete its planned 138 kV loop between Green Valley – Canoa Ranch – Duval Clear substations (2009 and 2012).

- **Stage 3** – Once the Nogales line is converted to 138 kV and terminated at Vail (2012), TEP should interconnect that line with Rosemont's 138 kV line at Rosemont's future substation site at SE corner of Sec 25 of T17S-R14E. TEP would purchase the Rosemont line constructed in Stage 2 and construct Rosemont's proposed substation at SE corner of Sec 25 of T17S-R14E. This establishes another 138 kV path between Vail and South and reinforces the Canoa Ranch loop out of South. It also provides redundant supply paths for the radial line to Nogales.

The Option D line route crosses state lands along section lines, and follows existing utility corridors from TEP Substations to the new Rosemont substation at Wilmot Junction. From Wilmot Junction, a new 138 kV line follows the northern and eastern perimeters of the Santa Rita Experimental Range until it reaches private Rosemont Company lands near the old Helvetia townsite. From Helvetia to Rosemont Mill, the line is on private lands owned by Rosemont.

The Arizona State Line-Siting-Committee has established the process to review new power line routes for Rosemont, and the preferred routing and permit application is scheduled for completion during mid 2009.

1.25.7 Water

The amount of fresh water required by the Rosemont facilities is approximated at 5,000 acre-feet per year with a peak delivery of 5,000 gallons per minute (gpm). The well fields and water supply pipeline will be designed for this peak demand.

Water quantities are limited and environmentally sensitive in the region of the Rosemont mine. Groundwater on the eastern side of the Santa Rita Mountains was, therefore, not considered as a potential source of water for the project. Water for the project will be from the basin-fill deposits of the upper Santa Cruz basin, which lies west of the Rosemont Project and the Santa Rita Mountains. A 53-acre parcel along Santa Rita Road near the Santa Rita Experimental Range has been purchased and explored with one test well. The test well indicated that two wells on this property will produce up to 3,000 gpm. Water samples collected from this well was submitted to a State-approved analytical laboratory for analysis and the results indicate that the quality of the groundwater is suitable for anticipated mine uses, including public water supply. Property for other well locations are currently being acquired for the other 2,000 gpm requirement. It is estimated a total of 5 or 6 production wells will meet the water supply needs for the project including back-up capacity.

The delivery of the water to the project site will be through a 20-inch ductile iron pipe line. The route will follow the power transmission line and run along the north and east boundaries of the Santa Rita Experimental Range and then over the Santa Rita Mountains to the fresh water and fire water storage tank on the Rosemont property. Along the way, there will be booster stations at different elevations which will consist of concrete basins, vertical turbine pumps, and a hydro pneumatic tank. The pipeline will be buried between the pumping stations until crossing the Santa Rita Mountains.

From the fresh water and fire water storage tank, the flow will be by gravity to the plant site and surrounding facilities requiring water. The daily requirement for potable water consumption is approximately 17,000 gallons per day. The fresh water used throughout the process plant will be about 4.8 million gallons per day. The process water system will consist of a collection pond sized to hold 3 days of process flows plus capacity for a 100 year, 24 hour storm event. Recycled process water will be pumped to a head tank located near the fresh water tank and then flow by gravity to the process in the grinding area. The recycled usage of process water is about 37 million gallons per day. Fire water will come from the lower part of the fresh water and fire water tank and be able to supply water to all the facilities around the project site.

Augusta has committed to recharging 105% of the total life of mine water requirements with Central Arizona Project (CAP) water. The CAP water will be recharged as close as possible to the location of the Project groundwater wells.

1.25.8 Permits

Permitting for the Rosemont Copper Project involves federal approvals and requires compliance with the National Environmental Policy Act (NEPA). This in turn requires an Environmental Impact Statement (EIS) and compliance with the Endangered Species Act (ESA) and the National Historic Preservation Act (NHPA).

The Environmental Impact Statement began in July 2007 with the submittal of a Mine Plan of Operation to the US Forest Service who is charged with preparing the EIS. The process to complete the EIS, including holding public hearings and public reviews, is scheduled to take 3 years and will culminate with a Final EIS and Record of Decision in July 2010 (Memorandum of Understanding, US Forest

Service and Rosemont Copper Company, signed Sept. 2008). Application for federal, state and local permits are proceeding concurrently; however, some permits may not be issued until a positive Record of Decision has been issued.

The Mine Plan of Operation for the Rosemont Project was issued to the US Forest Service, Coronado National Forest, on July 11, 2007, to begin the process. The following is a summary of the major permits required to construct and begin to operate the Rosemont mine. A list of permits required during operation is included in Table 1-38.

1.25.8.1 National Historic Preservation Act (NHPA)

This permit addresses treatment of cultural resource sites that are eligible for listing on the National Register of Historic Places and sets forth requirements for tribal consultation regarding the potential presence of Traditional Cultural Properties. The process includes a Class III cultural resource survey, developing a cultural resource treatment plan, executing a Memorandum of Agreement between government agencies and tribal groups, implementing the treatment plan, preparing a final report for approval by the lead government agency, conduct tribal consultation with interested tribal groups regarding the presence of TCPs, and preparing a TCP report for approval by the lead government agency. Register eligible sites are known to be present in the Rosemont area; therefore, it is expected that compliance will be required for the project.

1.25.8.2 Endangered Species Act (ESA) and Other Biological Requirements

The ESA requires the US Fish & Wildlife Service (USFWS) identify species that are potentially at risk for extinction, evaluate available scientific information about the species, and (if warranted) list the species as either threatened or endangered. The USFWS is also required to designate “critical habitat” for listed species if prudent and determinable. A Habitat Conservation Plan must then be developed by the project proponent to offset any harmful effects that the project may have on any listed species. A biological assessment / evaluation is being prepared for the Rosemont Project in accordance with the Forest Service and USFWS requirements. This report will address the likelihood of a listed species occurring on the project area and the potential impact of the project on those species.

1.25.8.3 Water Permitting

Aquifer Protection Permit

The Arizona Department of Environmental Quality is responsible for issuing an Aquifer Protection Permit (APP) to facilities that may potentially discharge pollutants which may adversely impact ground water quality. As part of the permit process, applicants must demonstrate that their facilities are designed to be protective of groundwater quality, either through adoption of presumptive "best available demonstrated control technology" (BADCT) or equivalent facility-specific design. The APP program also requires demonstration of the financial capability of the project proponent to design, construct, operate, close, and assure post-closure care of the facility. Further, hydro-geologic characterization of the site, groundwater quality monitoring, a contingency plan (in case of facility failure), closure strategy, and a post-closure monitoring and maintenance plan are required for the APP application.

Development of the Rosemont project will require APP coverage for, at least, the onsite ponds, tailings facilities, and leaching operations. Development of the draft permit application, including implementation of a hydro-geologic characterization study, has been completed for the APP permitting phase of the project and the application will be submitted by February 2009. The law requires that the permitting process be limited to 329 days under the Arizona licensing timeframes rule.

Clean Water Act – Section 402 (AZPDES)

Discharges of process water and storm water to waters of the US are regulated at the federal level by the EPA under the National Pollutant Discharge Elimination System (NPDES), as outlined in Section 402 of the Clean Water Act (CWA). The EPA delegated this program to the Arizona Department of Environmental Quality (ADEQ), which manages these discharges under the Arizona Pollutant Discharge Elimination System (AZPDES). Individual AZPDES permits must be obtained for each point source discharge from an operating mine site. Permits include effluent limitations consisting of both numeric and narrative standards. The numeric limitations restrict quantities, rates, and concentrations of pollutants that may be present in the discharge, and can be either technology or water quality based. Technology based standards require usage of available pollution control technology, while water quality based standards protect ambient water quality by requiring the discharger to achieve the applicable numeric standard as established by ADEQ.

Storm water discharges from mining facilities require a construction general permit (CGP) or a multi-sector general permit (MSGP). The CGP has historically been used for the exploration and construction phase of a mining operation, while the MSGP has been used to cover selected storm water discharges from an active mine facility. The general storm water permit program requires a project proponent to prepare a Storm Water Pollution Prevention Plan (SWPPP), submit a notice of intent (NOI) to discharge storm water, install appropriate best management practices (BMPs), and conduct regular inspections of the site in accordance with the SWPPP. MSGP coverage also requires the establishment of discharge outfalls and regular analytical monitoring of storm water discharges.

The Rosemont project will require coverage under the MSGP program. Depending upon the nature of discharges from the project area, individual AZPDES coverage may be required as well. Rosemont Copper is currently refining a geochemical management strategy such that materials are excavated and placed to minimize the potential to generate acid or alkaline rock drainage. Mineralized materials will be encapsulated, to the extent practicable, in the low-grade waste deposition areas, thus reducing storm water contact and potential metals leaching.

Clean Water Act – Section 404 (Dredge and Fill)

The discharge of solid material to waters of the US is regulated by the US Army Corps of Engineers (Corps) under Section 404 of the CWA. ADEQ plays a peripheral role in this program, providing certification of water quality (401 Certification) in support of the Corps' permitting process. Because implementation and enforcement of the CWA is ultimately the responsibility of the EPA, the EPA can influence the Corps' permitting decisions.

Two primary permitting avenues are also available to project proponents under Section 404: the Individual Permit (IP) and the Nationwide Permit (NWP). NWPs are available for certain specified categories and sizes of disturbance that result in only "minimal impact to the aquatic environment". IPs are required for larger projects or projects whose activities are not covered by the NWP program. The effort to obtain a Section 404 permit varies considerably both in time and cost, depending on the type and extent of the impacts. It should be noted that NEPA analysis, either as an EA or EIS, is required for an IP. Typically, the project must be demonstrably the "least environmental damaging preferred alternative" to obtain approval from the Corps. In addition, mitigation may be required to offset impacts to waters of the US.

The Rosemont project will result in the discharge of fill material to a network of ephemeral streams comprising the Barrel Canyon drainage. Augusta is in the process of completing a delineation of potentially jurisdictional waters within the project area, which will allow for the determination of loss of waters and guide the Section 404 permitting effort. It should be noted that the Corps and US Environmental Protection Agency (EPA) have recently issued joint guidance related to the identification of waters, which has the potential to affect the jurisdictional status of waters within the project area. The regulatory field offices are currently evaluating and interpreting the guidance and will provide clear guidelines which would allow a definitive evaluation of these waters.

1.25.8.4 Air Permitting

Air quality is regulated at the federal level by the EPA under the Clean Air Act (CAA). National Ambient Air Quality Standards (NAAQS) have been established for each of the criteria pollutants of ozone, carbon monoxide, nitrogen dioxide, sulfur dioxide, particulate matter less than 2.5 microns and less than 10 microns aerodynamic diameter, and lead. Authority for air quality permitting has been delegated by the Environmental Protection Agency (EPA) to the Arizona Department of Environmental Quality (ADEQ). ADEQ has subsequently delegated their authority for permitting to the Pima County Department of Environmental Quality (PCDEQ).

Air emissions are regulated under the CAA in the context of the NAAQS. The law and regulations differentiate between mobile and stationary sources, as well as between new and existing facilities. New or modified existing stationary sources must meet performance standards, referred to as New Source Performance Standards (NSPS), established by the EPA for certain categories of sources. The standard of performance for a particular facility is based on the application of the best available system of emission reduction, taking into consideration cost. New major sources are subject to preconstruction review, with different standards and levels of review applied to facilities proposed within attainment areas ("Prevention of Significant Deterioration" requirements) and non-attainment or non-classifiable areas ("New Source Review" requirements).

Emissions of "hazardous air pollutants" (HAPs) are also regulated under the CAA. EPA set standards for HAPs for both specific pollutants and families of pollutants that are not emitted by a sufficient number of sources to justify development of a NAAQS for that pollutant but that can have serious health implications for humans. The CAA requires identification of major sources of HAPs as well as area sources (sources below the volumetric thresholds for major sources). Sources are required to obtain permits for emitting any of the HAPs, again with variance between new and existing source standards.

The permitting components of the CAA for stationary sources are described in Title V of the CAA; thus, air emission operating permits are commonly referred to as Title V permits. These permits comprehensively address all relevant air emissions limitations, monitoring and reporting requirements, HAPs, and NSPS. ADEQ has established three other classes of permits. Class I permits are required for major sources, solid waste incineration units, affected sources (a defined term), and any source in a category designated by the EPA Administrator and adopted by the ADEQ Director. Mining operations qualify as Class I major sources. Class II permits are required for construction or modification of sources that otherwise do not qualify for Class I permits but that emit pollutants above certain thresholds or for sources that are certain types of facilities. Finally, General Permits are preapproved permits available for a specific class of sources, such as common types of facilities like gasoline stations.

Because the anticipated process at Rosemont will incorporate facilities covered under 40 CFR 60.380 Subpart LL, Title V permitting and New Source Performance Standards (NSPS) review may apply.

Metallic Mineral Processing Plants are covered under Subpart LL and are specific to operations from mining through concentrating. Included are all material transfer and storage operations that precede those operations that produce refined metals from metallic mineral concentrates. In addition to Subpart LL, Subpart Kb for petroleum storage will also apply to the Rosemont facility. Petroleum storage is specific to fuel and reagent tank storage, and would not apply to "flow through" process tanks.

In the arid southwest, fugitive emissions are a problem if not properly controlled. In an effort to conserve water and protect watershed areas, alternative forms of dust control are being investigated. A combination of dust suppressants, water, and cover or hooding will be used to manage fugitive emissions from process areas. Capping, seeding, and land management techniques will be used on waste rock piles and storage areas. In addition, captured water from operations and storm water will be used when and where appropriate to control dust to conserve groundwater resources. Management techniques for operations such as speed control, cleanup, and road maintenance will also be used to conserve resources and manage the potential to create fugitive emissions.

1.25.8.5 Arizona Native Plant Law

The Arizona Department of Agriculture administers the Arizona Native Plant Law (ANPL), which provides protections for listed native plants. The ANPL requires, in part, that private landowners notify the State when native plants will be destroyed in order to allow potential salvage of the plants (A.R.S. §3-904). However, the notification is not required for activities that occur "in the normal course of mining", so this requirement would not apply to the Rosemont project.

1.25.8.6 Pima County Conservation Lands System

The Conservation Land System (CLS) was adopted in 2001 by the Pima County Board of Supervisors as part of its Comprehensive Plan. The CLS is comprised of seven conservation land categories defined as Important Riparian Areas (IRAs), Biological Core Management Areas, Scientific Research Areas, Multiple Use Management Areas, Special Species Management Areas, Agriculture In-Holdings within the CLS, and other Riparian Areas mapped and regulated by Pima County. It also identifies six Critical Landscape Connections, which are broadly defined areas of regional significance with constraints to connectivity of the CLS.

The entire Rosemont property is identified by Pima County as part of its CLS, containing approximately 1,202 acres of IRA; 9,202 acres of Biological Core Management Area; and 5,405 acres of Multiple Use Management Area.

The IRAs traverse the Property in association with Barrel, Sycamore, McCleary, Wasp, Scholefield, and Oak Tree Canyons and various unnamed drainages. The Biological Core occurs within northern portions and on the southwest corner of the Property, while the Multiple Use area occupies the remainder of the Property mainly within its southern half. While it is evident that the IRAs were mapped in association with the drainages on the Property, there is no clear biological association with the mapped Biological Core and Multiple Use areas on the Property.

As specified in A.R.S. Section 11-830, the provisions of Pima County code “shall not prevent, restrict or otherwise regulate in any district or zone the use or occupation of land or improvements for railroad, mining, metallurgical, grazing or general agricultural purposes, as defined herein, provided the tract or premises so used is not less than five contiguous commercial acres.” In addition, CLS designations and Conservation Guideline policies as described in the Pima County Comprehensive Plan “apply only to land uses and activities under the jurisdiction of Pima County and Pima County Flood Control District. Application of these designations or guidelines shall not alter, modify, decrease or limit existing and legal land uses, zoning, permitted activities, or management of lands. These policies apply to new rezoning and specific plan requests, time extension requests for rezoning, requests for modifications or waivers of rezoning or specific plan conditions, including substantial changes, requests for Comprehensive Plan amendments, Type II and Type III conditional use permit requests, and requests for waivers of the subdivision plat requirement of a zoning plan." As such, obligations to comply with the mitigation provisions of the CLS do not apply to the Rosemont project.

Table 1-38 List of Agencies and Permits Timeline

Agency	Item	Description	Term	Conditions
Federal Permits				
U.S. Environmental Protection Agency	NPDES General Storm Water Permit	Discharge of storm water	5 years	Delineated in storm water management plan
Department of Transportation	Hazardous Materials Transportation Registration	Shipment of hazardous materials	Annual or 3 year renewal	Labeling, packaging, and shipping
U.S. Environmental Protection Agency	Hazardous Waste – RCRA, RCRA ID Number	Waste activities and disposal of hazardous waste	Life	Manifests, reporting, and inspections
U.S. Army Corps of Engineers	CWA Section 404 Permit	Discharge of fill material to onsite washes	3 years	Variety
Mine Safety and Health Administration	MSHA Number	Miner registration number	Life	Operate following MSHA rules
Forest Service	Plan of Operations	Plan for mining operations in the National Forest		Prepare a plan and manage according to the plan, update as required
Forest Service	Closure Plan	Bonding requirements for operations in the National Forest		Prepare a plan and manage according to the plan, updates as required
Forest Service	NEPA Review	Review of major federal action with CEQ oversight		Follow the Record of Decision
Bureau of Alcohol, Tobacco, and Firearms	Blasting Operator Registration	Registration of all personnel that may handle blasting materials	As needed	Background and fingerprint checks of all persons with access, update as required by Federal Agencies
Federal Communications Commission	Radio Licenses for Industrial/Business Pool Conventional Use	Communications equipment must be licensed	10 years	Follow license requirements
State Permits				
Arizona Department of Environmental Quality	Aquifer Protection Permit	Dumps, tailings, leaching facilities, processing plant for ground water protection	Life	Inspections, monitoring, maintenance, and reporting

Agency	Item	Description	Term	Conditions
Arizona Department of Environmental Quality	CAA Title V Permit	Mobile and stationary emission sources	5 years	Inspections, monitoring, maintenance, and reporting
Arizona Department of Environmental Quality	AZPDES General Storm Water Permit	Discharge of storm water	5 years	Delineated in storm water management plan
Arizona Department of Environmental Quality	Solid Waste Management Inventory Number	Landfill and waste area requirements	Life	Monitoring, maintenance, and operations
Arizona Department of Environmental Quality	Hazardous Waste Management Number	Management of hazardous waste	Life	Monitoring, maintenance, and operations
Arizona Department of Environmental Quality	Waste Tire Cell Registration	Management of off-road tires greater than 3 feet in diameter	Life	Annual reporting, cover requirements
Arizona Department of Water Resources	Groundwater Withdrawal Permits	Groundwater withdrawal rights	20 years	Groundwater withdrawal
Arizona Department of Water Resources	Safety of Dams Permit	Requirements for dam construction	Life	Monitoring, maintenance
Arizona Department of Water Resources	Water Storage Permit	Underground storage of CAP water		Annual reporting, storage, and CAP purchase contracts
Arizona State Mine Inspector	Reclamation Plan	Post-mining land uses and plans for regrading	Life	Annual updates
Local Permits				
Pima County Air Quality Department	Air Quality Permit	Terms for air emissions control	5 years	Fugitive and stack control

1.25.9 Operating Costs

The operating costs are summarized in Table 1-39 below. The operating costs were developed for Year 2 which is a typical year treating 27.4 million tons of mill ore, 8.9 million tons of oxide ore, and total tons mined of 114.4 million. The estimate is in fourth quarter 2008 US dollars at an accuracy of ±10%.

Table 1-39 Operating Cost - Mine Site Cost Summary Year 2 of Operation

Sulfide Ore Tons (Processed)	**27,376,090**		
Oxide Ore Tons (Processed)	**8,926,815**		
Total Ore Tons (Oxide and Sulfide)	**36,302,905**		
Total Tons - Mined	**114,369,807**		
Mine Cost Area	**Total Annual Cost - $**		**$/total ton ore**
Mining Operations		**$/Total Tons Mined**	
Drilling	$ 4,535,280	$ 0.040	$ 0.125
Blasting	9,079,799	0.079	0.250
Loading	8,673,510	0.076	0.239
Hauling	27,665,666	0.242	0.762
Road & Dumps	12,413,716	0.109	0.342
Mining General	7,773,489	0.068	0.214
Subtotal Mining	**$ 70,141,460**	**0.613**	**$ 1.932**
Processing Operations			
Mill Operations		**$/Sulfide Ton**	
Crushing & Conveying	$ 4,181,221	$ 0.153	$ 0.115
Grinding & Classification	48,722,207	1.780	1.342
Flotation and Regrind	23,114,941	0.844	0.637
Concentrate Dewatering, Filtration & Dewatering	1,603,378	0.059	0.044
Tailing Disposal	12,198,792	0.446	0.336
Ancillary Services	1,631,213	0.060	0.045
		$ 3.341	
SX-EW Operations		**$/Oxide Ton**	
Heap Leach Pad	12,187,500	$ 1.365	$ 0.336
Solvent Extraction	1,635,101	0.183	0.045
Tank Farm	759,080	0.085	0.021
Electrowinning	3,235,340	0.362	0.089
Ancillary Services	580,494	0.065	0.016
		$ 2.061	
Subtotal Processing	**$ 109,849,267**		**$ 3.026**
Supporting Facilities		**$/Sulfide Ton**	
Laboratory	984,147	$ 0.036	$ 0.027
General and Administrative	6,289,452	0.230	0.173
Subtotal Supporting Facilities	**$7,273,599**	**$ 0.266**	**$ 0.200**
Total Operating Cost	**$ 187,264,326**		**$ 5.158**

1.25.10 Capital Cost

The estimate for the initial capital cost for the project is summarized in Table 1-40 below. Costs are shown for a stand alone sulfide plant and a combined sulfide and oxide plant. The oxide plant shown is on an incremental cost basis with the sulfide plant carrying the infrastructure costs for the combined plant. The estimate is in fourth quarter 2008 US Dollars at an accuracy of ± 15%. No allowance has been provided for escalation, interest, hedging, or financing during construction.

Table 1-40 Capital Cost Estimate

Area	Description	Oxide Plant	Sulfide Plant	Combined Plants
000	Site General		$8,245,459	$8,245,459
010	Modifications to Highway 83		$211,743	$211,743
050	Mine		$214,550,649	$214,550,649
080	Heap Leach Pad	$25,621,961		$25,621,961
100	Primary Crushing and Storage		$19,282,614	$19,282,614
150	Overland Conveyor		$15,864,799	$15,864,799
200	SAG Feed Conveyor		$15,763,261	$15,763,261
300	Grinding & Classification		$131,755,808	$131,755,808
400	Cu Flotation & Regrind		$35,591,578	$35,591,578
410	Mo Flotation & Regrind		$2,169,157	$2,169,157
500	Cu Concentrate Thickening / Filtration		$15,872,385	$15,872,385
510	Mo Concentrate Thickening / Filtration		$1,222,889	$1,222,889
540	Solvent Extraction	$6,088,340		$6,088,340
560	Tank Farm	$6,549,245		$6,549,245
580	Electrowinning	$13,617,320		$13,617,320
600	Tailing Disposal		$83,750,667	$83,750,667
650	Water Systems		$46,257,350	$46,257,350
700	Main Substation		$8,779,345	$8,779,345
750	Power Transmission Line		$26,946,322	$26,946,322
800	Reagents	$1,740,487	$5,994,560	$7,735,047
900	Ancillary Facilities		$26,859,424	$26,859,424
	Total Direct Cost including Const. Equip. Labor Directs include field payroll burden and overhead, field supervision, field supervision burden and support.	$53,617,353	$659,118,010	$712,735,363
	Field Mobilization	$0	$500,000	$500,000
	Arizona Transaction Privilege Tax	$763,037	$9,380,007	$10,143,044
	EPCM	$4,513,857	$50,998,143	$55,512,000
	Commissioning & Spare Parts	$685,868	$18,366,012	$19,051,880
	Contingency	$5,096,712	$63,777,773	$68,874,485
	Owner's Cost	$0	$30,355,000	$30,355,000
	Total Direct and Indirect Cost	$64,676,826	$832,494,946	$897,171,772

1.25.11 Financial Analysis

1.25.11.1 Introduction

This study evaluated a sulfide concentrate plant with a heap leach SX-EW plant for the treatment of the oxide copper reserves. The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years after production commences to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures, production costs and sales revenue. The sales revenue is based on the production of three commodities: copper, molybdenum and silver. Gold is also present in the copper concentrates in the form of a saleable byproduct credit. The estimates of capital expenditures and site production costs have been developed specifically for this project and are presented in Sections 10 and 11 of this report.

1.25.11.2 Mine Production Statistics

Mine production is reported as sulfide ore, oxide ore and waste material from the mining operation. The annual production figures were obtained from the Mine Plan as reported in Section 5 of this report.

The life of mine ore and waste quantities and ore grade are presented in Table 1-41 below.

Table 1-41 Total Mine Production Statistics

	Tons (000)	Copper	Molybdenum	Silver – oz/ton
Sulfide ore	546,338	0.45%	0.015%	0.12
Oxide ore*	69,181	0.17%		
Waste**	1,232,258			
Total	1,847,777			

** Includes 793 k-tons of oxide ore scheduled for after Yr 7 that may be destined as waste.*

*******All inferred resources above the NSR cutoff grade (62 million tons) within the ultimate pit are included in total Waste.*

The net smelter return (NSR) reflects the value of all payable metal contained in the concentrates produced less charges related to downstream smelting, refining and transportation charges. Detailed annual production statistics for the Combined Base Case (60/40 Pricing) can be found in the financial model (Tables 1-48, 1-55) at the end of the Financial Analysis section.

1.25.11.3 Plant Production Statistics

Oxide ore will be processed using heap leach and SX-EW technology to produce copper cathode. The average recovery of the leach and SX-EW plant is expected to be 65% for the life of the mine. The estimated copper cathode production for the life of the mine is approximately 155.5 million pounds.

The copper cathode production will be recovered in three leaching cycles. When the ore is first placed on the leach pad, the recovery has been estimated at 55% and the next two leaching cycles will recover 5% each of the copper for a combined total recovery of 65%.

In the pre-production time period, approximately 2.018 million tons of sulfide ore will have been stockpiled, 976 thousand tons will be processed the first year, 190 thousand tons in year 11 and the remainder of 852 thousand tons will be processed at the end of the mine life.

Sulfide ore will be processed using crushing, grinding and flotation technology to produce metals in flotation concentrates. Two concentrate products will be produced; copper concentrate and molybdenum concentrate. Precious metals will be recovered in the copper concentrates.

The estimated recovery for the copper is 83% molybdenum recovery is estimated to be 60% and the recovery for silver is 77%.

Average copper production in the combined sulfide and oxide case is 221 million pounds for the first 8 yrs of production. Molybdenum production averages 4.7 million pounds per year and silver averages 2.42 million ounces per year. Gold as a by-product averages 17 thousand ounces per year.

Life of mine saleable production is presented in Table 1-42 below.

Table 1-42 Life of Mine Metal Production

	Concentrate Tons (000)	Copper Tons (000)	Molybdenum Tons (000)	Silver Ozs (000)
Copper Concentrate	6,633	2,039		50,081
Molybdenum Concentrate	93		48	
Copper Cathode		78		

1.25.11.4 Smelter Return Factors

Copper and molybdenum concentrates will be shipped from the site to smelting and refining companies. The smelter and refining treatment charges will be subject to negotiation at the time of final agreement. A marketing report dated December 2008 prepared by specialist, Robert J. Loewen & Associates, forms the basis for the smelting and refining treatment and transportation charges used for this evaluation.

A smelter may impose a penalty either expressed in higher treatment charges or in metal deductions to treat concentrates that contain higher than specified quantities of certain elements. It is expected that the concentrate will not pose any special restrictions on smelting and refining, and that the concentrates will be marketable to smelting and refining companies.

The smelting and refining charges calculated in the financial evaluation include charges for smelting copper and molybdenum concentrates. The off-site charges that will be incurred are presented in Table 1-43 below.

Table 1-43 Smelter Return Factors

Smelter Return Factors	
Copper Concentrate	
Payable copper	96.5%
Copper deduction	Nil
Treatment charge - $/ton	$50.00
Copper refining - $/lb	$0.055
Shipping charge - $/ton	$43.00
Payable gold	90.0%
Gold refining - $/oz	$7.00
Payable silver	90.0%
Silver refining – $/oz	$0.40
Silver deduction	Nil
Molybdenum Concentrate	
Payable molybdenum	100.0%
Molybdenum deduction	NA
Treatment charge - $/lb	$1.50
Shipping charge - $/ton	FCA site
Copper Cathode	
Payable copper	100%
Shipping charge	FCA site

1.25.11.5 Capital Expenditures

Initial Capital

The total capital of new construction (includes direct and indirect costs) for the combined case (sulfide and oxide) is estimated to be $880.6 million, excluding $16.5 million for spare parts moved to working capital.

Any land acquisition or exploration costs or other owner’s study expenditures prior to and including this Updated Feasibility Study have been treated as “sunk” costs and have not been included in the analysis.

Sustaining Capital

A schedule of capital cost expenditures during the production period has been estimated and included in the financial analysis under the category of sustaining capital. The total life of mine sustaining capital is estimated to be $109.8 million. This capital will be expended during a 19 year period, starting in Year 1 and ending in Year 19.

Working Capital

Working capital for accounts receivables will vary by year depending on sales revenue, and a delay of one and a half months before receipt of sales revenue. Note that the inventory portion remains constant, but that the accounts receivable will vary. In addition, working capital for plant consumable inventory is estimated in Year -1 and Year 1. All the working capital is recaptured at the end of the mine life and the final value of the account is $0.

Salvage Value

An allowance of $54.3 million has been included in the cash flow analysis as a return of capital from the salvage and resale of equipment at the end of mine life. It was calculated using initial equipment capital cost and sustaining equipment capital cost for years 1- 15 at 10% and sustaining capital cost for years 16 – 19 at 50%.

1.25.11.6 Revenue

Annual revenue is determined by applying estimated metal prices to the annual payable metal before treatment, refinery and transportation charges for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. Metal sales prices used in the evaluation are shown in Table 1-44 below and is taken from the Commodity Price Analysis dated December 31, 2008, shown in Table 1-46.

Table 1-44 Base Case and Historical Metals Prices

	60/40 Weighted Average *	3 Year Historical Average *
Copper	$ 2.47 / pound	$ 3.14 / pound
Molybdenum	$22.70 / pound	$29.05 / pound
Silver	$12.40 / ounce	$ 13.32 / ounce
Gold	$784.65 / ounce	$ 723.48 / ounce

** See Table 1-46 for definitions*

In addition to the above metal sales prices, cases with long term metal prices were also evaluated. Long term metal prices were $1.85/lb Cu, $15.00/lb Mo, $12.00/oz Ag and $750.00/oz Au.

Table 1-45 Long Term Metals Prices

Copper	$ 1.85/lb
Molybdenum	$ 15.00/lb
Silver	$ 12.00/oz
Gold	$750.00/oz

Table 1-46 Commodity Price Analysis

December 31, 2008

		Prices used for US Sec filings				Prices used for NI 43-101 filings
Commodity	**Spot Price**	**Historical Price**		**Futures Price Forecast**		**Weighted Average**
	EOM Dec 2008	(36-months)	Source of Data	(24-Month) Projected thru Dec 2010		(60-40)
Gold (USD per Tr Oz)	869.75	723.48	London PM Fix - Kitco	876.40	COMEX Futures	784.65
Silver (USD per Tr Oz)	10.79	13.32	London Fix - Kitco	11.01	COMEX Futures	12.40
Copper (USD per lb)	1.36	3.14	LME Monthly Ave	1.46	COMEX Futures & LME Futures	2.47
Lead (USD per lb)	0.44	0.90	LME Monthly Ave	0.46	LME Futures	0.73
Nickel (USD per lb)	5.170	12.48	LME Monthly Ave	4.78	LME Futures	9.40
Zinc (USD per lb)	0.53	1.26	LME Monthly Ave	0.57	LME Futures	0.98
Molybdenum (USD per lb Mo)	11.00	29.05	Infomine Chart Interp.	13.17	see Note 5	22.70

Notes:

1. Precious Metals updated through End-of-Month (EOM) December 2008
   Base metals updated through EOM December 2008
2. Sources: London Bullion Market Association price fixing for gold (PM) and silver daily historical and EOM futures pricing
   London Metals Exchange (LME) for copper, lead, nickel, and zinc monthly average settlement and futures pricing.
   Copper futures pricing is the weighted average of the LME (65%) and COMEX (35%) copper futures prices.
   Molybdenum prices from Infomine.com website.
3. M3 uses weighted average prices for NI-43-101 reporting purposes, 60 % historical prices; 40% futures forecast prices.
4. Spot prices are from London Bullion Market Association & Kitco Metals for precious metals and base metals, respectively.
5. Historical Molybdenum prices are determined by interpolating values on graphs supplied by Infomine. Moly futures are extrapolated using the 3-yr historical Mo price regression slope and projecting it 2 years into the future from the latest month.

1.25.11.7 Cash Copper Unit Cost Net of By-Product Credits

The average Cash Copper Unit Cost Net of By Product Credits over the life of the mine include mine, process plant, general administrative, treatment and refining charges, transportation, property and severance taxes and reclamation expense. These charges are offset by molybdenum, silver and gold credits.

The three different cost comparison cases evaluated are summarized in Table 1-47 below:

Table 1-47 Cash Copper Unit Cost Net of By Product Credits

	Base Case (60/40)	Historical 36 month	Long Term Metal Prices
Mining	1,483,319	1,483,319	1,483,319
Processing - Mill	1,807,929	1,807,929	1,807,929
Processing - SXEW	169,303	169,303	169,303
G & A [1]	192,954	192,954	173,254
Treatment & Shipping Charges	1,026,377	1,026,377	1,026,377
Severance Taxes [2]	100,164	142,273	59,225
Property Taxes [3]	66,500	66,500	66,500
Reclamation Expense [4]	25,298	25,298	25,298
Total Operating Cost	4,871,844	4,913,953	4,811,205
Moly - by-product credit	(2,156,946)	(2,760,321)	(1,425,294)
Silver - by-product credit	(558,904)	(600,372)	(540,875)
Gold - by-product credit	(211,858)	(195,341)	(202,500)
Net Operating cost	1,944,136	1,357,919	2,642,536
Net Unit Cost per lb Cu	0.459	0.321	0.624

1 *G & A*
The G & A cost has a community endowment component which varies by metal prices.

2 *Severance Taxes*
A severance tax is imposed in Arizona in lieu of sales tax on the mining minerals. The net severance base is 50% of the difference between gross value of production and the production cost. The amount of tax is calculated by multiplying the net severance base by 2.5%.

3 *Property Taxes*
A property tax allowance of $3.5 million per year was included in the cash flow, the basis was a study performed by Donald Ross Consulting.

4 *Reclamation & Closure*
An allowance of approximately $19.0 million for the cost of the final reclamation bond has been included in the cash flow projection. Continual early reclamation is done throughout the life of the mine and costs have included for such, e.g. borrow pits.

Also included in the financial analysis are the following items:

Preproduction Mining Cost

A total of $48.1 million will be spent for pre-production mining. 70% of these costs are expensed and the remaining 30% is amortized over a 5 year period.

Fees and Royalties

Royalties are calculated at 3% of the net smelter returns. The royalty is calculated and will be paid at the end of an annual period.

Depreciation

Depreciation percentages were provided by Augusta for an 8 year period using a half year convention for the first and last year of depreciation and capital assets were depreciated using these percentages. The year after end of production was used as a catch up year to fully depreciate any assets that had not been fully depreciated.

Below are the percentages that were applied:

- Year 1 10.71%
- Year 2 19.13%
- Year 3 15.03%
- Year 4 12.25%
- Year 5 12.25%
- Year 6 12.25%
- Year 7 12.25%
- Year 8 6.13%

Depletion

The percentage depletion method was used in the evaluation. It is determined as a percentage of gross income from the property, not to exceed 50% of taxable income before the depletion deduction. The gross income from the property is defined as metal revenues minus downstream costs from the mining property (smelting, refining and transportation). Taxable income is defined as gross income minus operating expenses, overhead expenses, depreciation and state taxes.

The rates for depletion are as follows:

- Copper 15%
- Silver 15%
- Molybdenum 22%

Income Taxes

Taxable income for income tax purposes is defined as metal revenues minus operating expenses, royalty, property and severance taxes, reclamation and closure expense, depreciation and depletion. Income tax rates for state and federal are as follows:

- State rate 7.0%
- Federal rate 35.0%
- Combined effective tax rate 39.6%

The combined effective tax rate was calculated as follows (use decimal format to calculate): state rate (7.0%) + federal rate 35.0 %*(1-state rate 7.0%)

Income taxes were calculated on the taxable income described above using the federal and state rates.

In addition, the Alternative Minimum Tax (AMT) was calculated to determine the taxes that need to be paid for the federal portion. The main differences in calculating the taxable income is that depletion is not a deduction in the AMT calculation and the mine development cost is amortized over a 10 year period. The AMT rate is 20%. AMT carry-forward amounts were calculated and applied against regular federal income taxes payable as appropriate.

These two results were compared and the higher of the two was the income taxes paid for that year.

As the focus of the economic analysis prepared for this Updated Feasibility Study is to calculate the alternative cash-flows for the expensing treatment noted that is used for tax purposes. For accounting purposes expensing of amounts could very well be different and will be determined by generally accepted accounting principles (GAAP).

1.25.11.8 Project Financing

It is assumed for the purposes of this study that the project will be all equity financed. No leverage or debt expense has been applied in the financial analysis.

Net Income after Tax

Net Income after tax amounts for each of the cases evaluated is shown in Table 1-48 below:

Table 1-48 Net Income After Tax

$ Millions	Base Case (60/40)	Historical 36 month	Long Term Metal Prices
Net Income After Tax	$ 4,832.2	$ 6,982.0	$ 2,697.2

Net Present Value, Internal Rate of Return and Sensitivity Analysis

The combined sulfide & oxide base case (60/40 metal pricing) economic analysis (Table 1-49) indicates that the project has an Internal Rate of Return (IRR) of 28.5 with a payback period of 3.1 years.

A sensitivity analysis was conducted for the metals price, capital expenditures, operating costs and metal production. The results are included in Table 1-49. The project IRR is most sensitive to variation in metals price followed by metal production, operating cost, and capital cost.

Table 1-49 Economic Analysis – Combined Base Case (60/40) ($ millions)

	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR %	Payback years
Combined Base Case (60/40 weighted average)	4,850.0	2,417.6	1,254.2	28.5%	3.1
Metals Price +10%	5,681.8	2,886.2	1,545.1	32.1%	2.7
Metals Price -10%	4,014.3	1,944.8	959.5	24.6%	3.5
Capex +10%	4,791.1	2,358.0	1,195.3	26.4%	3.3
Capex -10%	4,908.9	2,477.1	1,313.2	30.9%	2.8
Opex +10%	4,634.0	2,292.6	1,174.9	27.4%	3.2
Opex -10%	5,066.0	2,542.3	1,333.2	29.5%	3.0
Metal Production +10%	5,615.2	2,849.2	1,522.5	31.8%	2.8
Metal Production -10%	4,083.7	1,984.9	984.9	25.0%	3.5

The combined sulfide & oxide historical case (36 month trailing price) economic analysis shown in Table 1-50 indicates that the project has an Internal Rate of Return (IRR) of 37.5% with a payback period of 2.3 years.

Table 1-50 Economic Analysis – Combined Case - Historical 36 Month Prices ($ millions)

	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR %	Payback years
Combined Historical Case (36 month)	6,999.9	3,628.9	2,006.2	37.5%	2.3
Metals Price +10%	8,046.7	4,218.6	2,372.0	41.5%	2.1
Metals Price -10%	5,953.6	3.039.2	1,640.1	33.2%	2.6
Capex +10%	6,940.9	3,569.6	1,947.5	34.8%	2.5
Capex -10%	7,058.8	3,688.3	2,064.9	40.6%	2.1
Opex +10%	6,783.9	3,504.6	1,927.8	36.5%	2.4
Opex -10%	7,215.9	3,753.3	2,084.6	38.4%	2.3
Metal Production +10%	7,978.6	4,180.9	2,349.1	41.3%	2.1
Metal Production -10%	6,023.2	3,078.8	1,664.8	33.5%	2.6

The combined sulfide & oxide long term price case economic analysis shown in Table 1-51 indicates that the project has an Internal Rate of Return (IRR) of 17.8% with a payback period of 5.0 years.

Table 1-51 Economic Analysis – Combined Case - Long Term Prices* ($ millions)

	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR %	Payback years
Combined Long Term Prices	2,715.0	1,200.3	488.4	17.8%	5.0
Metals Price +10%	3,348.4	1,563.3	717.5	21.1%	4.1
Metals Price -10%	2,051.5	818.4	246.8	14.1%	6.2
Capex +10%	2,639.9	1,129.0	420.6	16.3%	5.4
Capex -10%	2,787.3	1,269.9	555.1	19.6%	4.6
Opex +10%	2,472.8	1,056.3	395.0	16.4%	5.5
Opex -10%	2,951.4	1,340.7	579.5	19.2%	4.6
Metal Production +10%	3,291.6	1,531.7	697.9	20.9%	4.2
Metal Production -10%	2,120.4	857.7	271.4	14.5%	6.1

- ** See Table 1-38 for the prices*

Shown below in Table 1-52 is a graph depicting the project Net Present Values (NPV) for various prices of copper and molybdenum. The scale on the left is the net present value in millions of US$ at a discount rate of 5% and the scale on the right depicts the net present value at a 10% discount rate. For example: at a copper price of $2.50/lb, and molybdenum price of $10.00/lb., the NPV (5%) is approximately US$2.0 billion and the NPV (10%) is US$1.0 billion.

Table 1-52 Metal Price Sensitivity

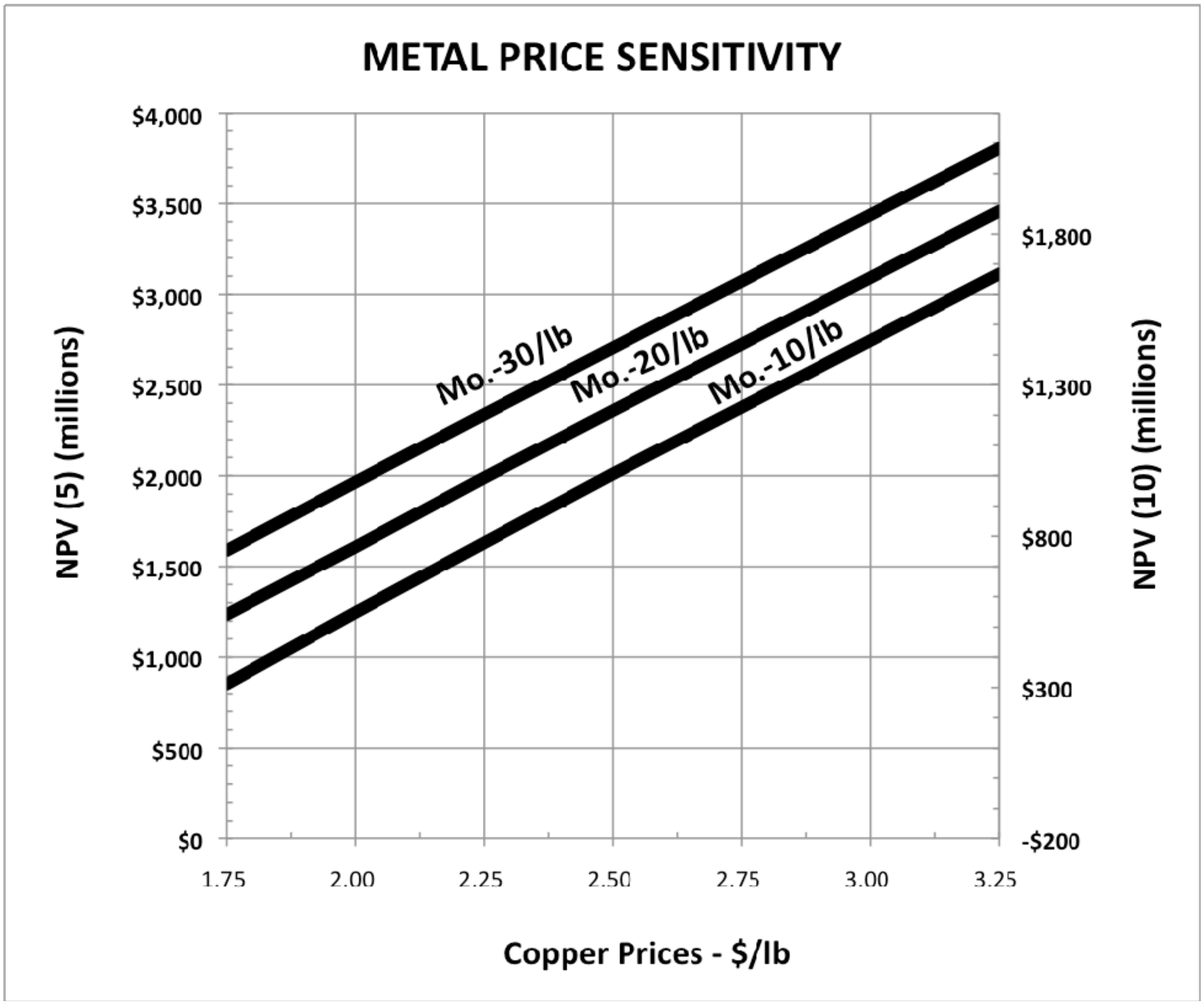


Table 1-53 Combined Base Case (60/40 split)

		Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Mining																													
Oxide Ore	Mined (kt)	69,181	0	0	8,646	20,675	14,751	9,629	3,901	1,821	9,758	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	Copper Grade -%	0.17%	0.00%	0.00%	0.16%	0.17%	0.20%	0.16%	0.16%	0.15%	0.15%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Sulfide Ore	Mined (kt)	546,340																											
	Copper Grade -%	0.45%	0.00%	0.00%	0.26%	0.44%	0.53%	0.39%	0.46%	0.38%	0.44%	0.47%	0.49%	0.49%	0.52%	0.54%	0.40%	0.45%	0.49%	0.48%	0.42%	0.45%	0.46%	0.41%	0.34%	0.40%	0.00%	0.00%	0.00%
	Moly Grade -%	0.015%	0.000%	0.000%	0.013%	0.016%	0.015%	0.011%	0.014%	0.014%	0.012%	0.011%	0.013%	0.012%	0.013%	0.011%	0.011%	0.015%	0.015%	0.016%	0.019%	0.017%	0.018%	0.022%	0.017%	0.012%	0.000%	0.000%	0.000%
	Ag Grade - oz/ton	0.1185	0.0000	0.0000	0.1082	0.1354	0.1216	0.0917	0.1399	0.0830	0.1261	0.1214	0.1279	0.1515	0.1208	0.1379	0.0715	0.0901	0.1104	0.1225	0.1268	0.1238	0.1313	0.1146	0.1211	0.1702	0.0000	0.0000	0.0000
Waste	Mined (kt)	1,232,258	0	1,688	62,230	72,822	72,243	72,369	78,094	80,177	71,241	81,997	81,996	81,995	81,500	77,000	68,000	77,999	64,998	51,998	40,512	4,928	1,434	144	4,369	2,525	0	0	0
Total	Total Material Mined (t)	1,847,779	0	1,688	72,893	114,995	114,370	109,374	109,371	109,374	108,375	109,373	109,372	109,371	108,876	104,186	95,376	105,375	92,374	79,374	67,888	32,304	28,810	27,520	31,745	5,395	0	0	0
Process Plant	Cathode Production (klbs)	155,514	-	-	-	20,015	19,997	19,991	20,010	19,998	20,017	20,062	12,220	2,374	831	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Production (klbs)	4,077,220	-	-	-	165,380	245,640	183,000	210,920	171,020	202,120	212,940	223,200	224,400	237,000	219,600	165,600	197,400	217,800	217,200	193,800	208,200	213,600	188,400	156,600	23,400	-	-	-
	Gold Production (kozs)	300	-	-	-	14	20	15	19	15	18	19	15	15	16	15	11	13	15	14	13	14	14	13	10	2	-	-	-
	Silver Production (kozs)	50,081	-	-	-	2,322	2,564	1,933	2,911	1,727	2,623	2,526	2,732	3,235	2,578	2,912	1,503	1,913	2,348	2,612	2,708	2,644	2,805	2,447	2,586	452	-	-	-
	Molybdenum Production (klbs)	95,016	-	-	-	5,136	5,912	4,336	4,982	4,982	4,270	3,912	3,986	3,678	3,986	3,364	3,358	4,590	4,594	4,904	5,826	5,212	5,520	6,746	5,212	510	-	-	-
Dollars in Thousands																													
Capital Cost																													
Initial Capital - Equity $(000's)		880,643	60,136	272,509	488,864	59,134	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Oxide $(000's)		66,942	669	23,430	42,843	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sulphide $(000's)		591,343	46,125	149,018	337,065	59,134	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine Mobile Equipment $(000's)		222,358	13,341	100,061	108,956	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital		109,760	-	-	-	25,591	-	20,797	150	917	759	1,918	286	3,165	12,264	20,129	1,995	1,110	8,846	959	354	9,815	555	150	-	-	-	-	-
Oxide $(000's)		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sulphide $(000's)		22,581	-	-	-	3,381	-	11,358	-	717	204	1,153	286	-	2,232	1,227	1,440	-	-	204	204	175	-	-	-	-	-	-	-
Mine Mobile Equipment $(000's)		87,179	-	-	-	22,210	-	9,439	150	200	555	765	-	3,165	10,032	18,902	555	1,110	8,846	755	150	9,640	555	150	-	-	-	-	-
Total Capital (Initial + Sustaining) $(000's)		990,403	60,136	272,509	488,864	84,725	-	20,797	150	917	759	1,918	286	3,165	12,264	20,129	1,995	1,110	8,846	959	354	9,815	555	150	-	-	-	-	-
Working Capital																													
Metal WIP and Finished Goods (1 1/2- month) $(000's)		-	-	-	-	73,000	28,000	(24,000)	12,000	(14,000)	9,000	2,000	1,000	(3,000)	3,000	(7,000)	(18,000)	14,000	7,000	1,000	(5,000)	3,000	2,000	(4,000)	(14,000)	(57,000)	(9,000)	-	-
Inventory - Parts, Supplies and Comodities $(000's)		-	-	-	7,226	10,838	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(18,064)	-	-	-
Pre-production mining (amortized)		-	-	1,000	13,421	(200)	(2,884)	(2,884)	(2,884)	(2,884)	(2,684)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Working Capital $(000's)		-	-	1,000	20,646	83,638	25,116	(26,884)	9,116	(16,884)	6,316	2,000	1,000	(3,000)	3,000	(7,000)	(18,000)	14,000	7,000	1,000	(5,000)	3,000	2,000	(4,000)	(14,000)	(75,064)	(9,000)	-	-
Revenue																													
Cathode	Copper	382,628	-	-	-	41,041	49,208	49,200	49,200	49,200	49,241	49,331	33,276	9,889	2,698	344	-	-	-	-	-	-	-	-	-	-	-	-	-
Concentrate	Copper & Moly	12,645,966	-	-	-	546,578	762,435	566,800	661,732	550,628	620,701	637,960	663,579	665,066	695,459	642,886	495,502	605,235	660,201	668,071	633,572	653,963	675,618	638,677	527,500	73,803	-	-	-
		13,028,594	-	-	-	587,619	811,643	616,000	710,932	599,828	669,942	687,291	696,855	674,955	698,157	643,230	495,502	605,235	660,201	668,071	633,572	653,963	675,618	638,677	527,500	73,803	-	-	-
Cash Operating Costs																													
Mine	Operations	1,483,319	-	-	-	61,476	70,141	77,858	79,147	91,626	86,073	86,331	73,712	80,102	85,502	102,883	85,328	93,233	88,189	78,859	64,010	37,679	37,130	46,894	48,005	9,140	-	-	-
Processing	Operations - Mill	1,807,929	-	-	-	76,065	91,452	91,451	91,451	91,452	91,452	91,451	89,717	89,717	89,717	89,717	89,717	89,717	89,717	89,717	89,717	89,717	89,717	89,717	89,717	16,831	-	-	-
Processing	Operations - SXEW	169,303	-	-	-	18,463	18,398	17,789	23,277	24,452	24,676	24,515	15,011	1,875	846	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Other	G & A	192,954	-	-	-	8,974	8,974	9,274	9,274	9,274	9,274	9,274	8,974	8,974	8,974	8,974	8,974	8,974	8,974	8,974	8,974	8,974	8,974	8,974	8,974	8,974	3,000	-	-
Shipping, Refining and Smelting		1,026,377	-	-	-	43,956	62,389	46,388	50,331	41,995	47,415	48,987	55,209	55,187	58,137	53,564	41,373	50,241	54,818	55,242	51,605	53,774	55,459	51,726	42,596	5,988	-	-	-
Total Cash Operating Costs		4,679,882	-	-	-	208,934	251,354	242,761	253,480	258,798	258,889	260,559	242,624	235,855	243,176	255,138	225,392	242,166	241,698	232,793	214,307	190,144	191,280	197,311	189,292	40,933	3,000	-	-
Cash Costs																													
Royalty		361,000	-	-	-	16,000	22,000	17,000	20,000	17,000	19,000	19,000	19,000	19,000	19,000	18,000	14,000	17,000	18,000	18,000	17,000	18,000	19,000	18,000	15,000	2,000	-	-	-
Severance Tax		100,164	-	-	-	3,868	5,338	3,296	4,685	3,169	4,091	4,294	5,374	5,875	6,095	5,196	3,606	4,860	5,587	5,795	5,578	6,150	6,437	5,899	4,576	395	-	-	-
Property Tax		66,500	-	-	-	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	-	-	-	-	-
Total Cash Costs		5,207,546	-	-	-	232,302	282,192	266,557	281,665	282,467	285,480	287,353	270,498	264,229	271,771	281,834	246,497	267,526	268,786	260,088	240,385	217,795	220,217	224,710	208,868	43,328	3,000	-	-
Production Costs																													
Pre-production mining cost		48,068	-	2,332	31,315	200	2,884	2,884	2,884	2,884	2,684	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reclamation bond fee		18,974	-	6,325	6,325	6,325	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reclamation expense		25,298	-	-	-	237	782	426	558	1,064	202	202	202	202	202	1,186	1,186	1,186	1,186	1,186	2,717	2,717	2,717	2,717	2,717	-	851	851	-
Reimbursable expenses/salvage value		(79,354)	-	-	-	(3,989)	(2,091)	-	-	-	-	-	-	-	-	(5,467)	-	-	-	-	-	-	-	-	-	(67,348)	-	(459)	-
Depreciation		990,403	-	-	-	97,058	173,363	138,434	115,008	114,266	113,841	114,068	58,742	3,453	3,695	5,462	6,715	5,732	5,968	6,580	6,093	5,945	4,806	3,100	2,627	5,447	-	-	-
Total Production Cost		6,210,935	-	8,657	37,640	332,132	457,130	408,302	400,115	400,681	402,207	401,624	329,442	267,885	275,669	283,015	254,398	274,444	275,940	267,854	249,196	226,457	227,740	230,527	214,212	(18,572)	3,851	391	-
Income from Operations		6,817,659	-	(8,657)	(37,640)	255,487	354,514	207,698	310,817	199,147	267,735	285,667	367,413	407,070	422,488	360,215	241,104	330,791	384,261	400,217	384,376	427,506	447,878	408,150	313,288	92,375	(3,851)	(391)	-
Taxes with loss carry forward applied		1,985,495	-	-	-	56,647	91,913	51,480	79,648	50,560	62,937	75,147	107,247	124,336	129,371	108,121	68,679	97,948	116,310	122,028	116,932	133,361	140,178	125,597	94,516	32,541	-	-	-
Net Income After Taxes		4,832,165	0	(8,657)	(37,640)	198,840	262,601	156,218	231,169	148,587	204,798	210,521	260,166	282,734	293,117	252,094	172,425	232,843	267,951	278,190	267,444	294,145	307,701	282,553	218,772	59,834	(3,851)	(391)	0
Cash Flow from Property after Tax																													
Net Income After Taxes		4,832,165	-	(8,657)	(37,640)	198,840	262,601	156,218	231,169	148,587	204,798	210,521	260,166	282,734	293,117	252,094	172,425	232,843	267,951	278,190	267,444	294,145	307,701	282,553	218,772	59,834	(3,851)	(391)	-
Add Back - Depreciation		990,403	-	-	-	97,058	173,363	138,434	115,008	114,266	113,841	114,068	58,742	3,453	3,695	5,462	6,715	5,732	5,968	6,580	6,093	5,945	4,806	3,100	2,627	5,447	-	-	-
Operating Cash Flow		5,822,568	-	(8,657)	(37,640)	295,897	435,963	294,653	346,177	262,853	318,639	324,589	318,908	286,187	296,812	257,556	179,140	238,575	273,919	284,769	273,538	300,090	312,506	285,653	221,399	65,281	(3,851)	(391)	-
Working Capital		0	-	(1,000)	(20,646)	(83,638)	(25,116)	26,884	(9,116)	16,884	(6,316)	(2,000)	(1,000)	3,000	(3,000)	7,000	18,000	(14,000)	(7,000)	(1,000)	5,000	(3,000)	(2,000)	4,000	14,000	75,064	9,000	-	-
Capital Expenditures		(990,403)	(60,136)	(272,509)	(488,864)	(84,725)	-	(20,797)	(150)	(917)	(759)	(1,918)	(286)	(3,165)	(12,264)	(20,129)	(1,995)	(1,110)	(8,846)	(959)	(354)	(9,815)	(555)	(150)	-	-	-	-	-
Sunk Cost		17,827	17,827	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cash Flow		4,849,992	(42,309)	(282,166)	(547,150)	127,534	410,847	300,740	336,911	278,821	311,564	320,671	317,622	286,022	281,548	244,427	195,145	223,465	258,073	282,810	278,184	287,275	309,951	289,503	235,399	140,345	5,150	(391)	-
Cumulative Cash Flow			(42,309)	(324,474)	(871,624)	(744,091)	(333,243)	(32,504)	304,408	583,228	894,792	1,215,463	1,533,084	1,819,107	2,100,655	2,345,082	2,540,227	2,763,692	3,021,766	3,304,576	3,582,760	3,870,034	4,179,986	4,469,489	4,704,888	4,845,233	4,850,383	4,849,992	4,849,992
Property Economics Indicators: Before Taxes																													
Cash Flow before Taxes			(42,309)	(282,166)	(547,150)	184,180	502,760	352,220	416,559	329,380	374,501	395,817	424,869	410,358	410,919	352,548	263,823	321,413	374,383	404,838	395,116	420,636	450,129	415,100	329,915	172,886	5,150	(391)	-
NPV at 0% - ($000) after tax		6,835,486																											
NPV at 5% - ($000) after tax		3,467,296																											
NPV at 10% - ($000) after tax		1,868,032																											
IRR		34.8%																											
Property Economics Indicators: After Taxes																													
Cash Flow before Taxes																													
NPV at 0% - ($000) after tax		4,849,992																											
NPV at 5% - ($000) after tax		2,417,584																											
NPV at 10% - ($000) after tax		1,254,249																											
IRR		28.5%																											
Payback - Years from Startup		3.1																											

1.25.12 Mine Life

The Life of Mine of the Rosemont Project is 21 years.

1.26 ILLUSTRATIONS

Figure 1-1
Rosemont Deposit Location Map

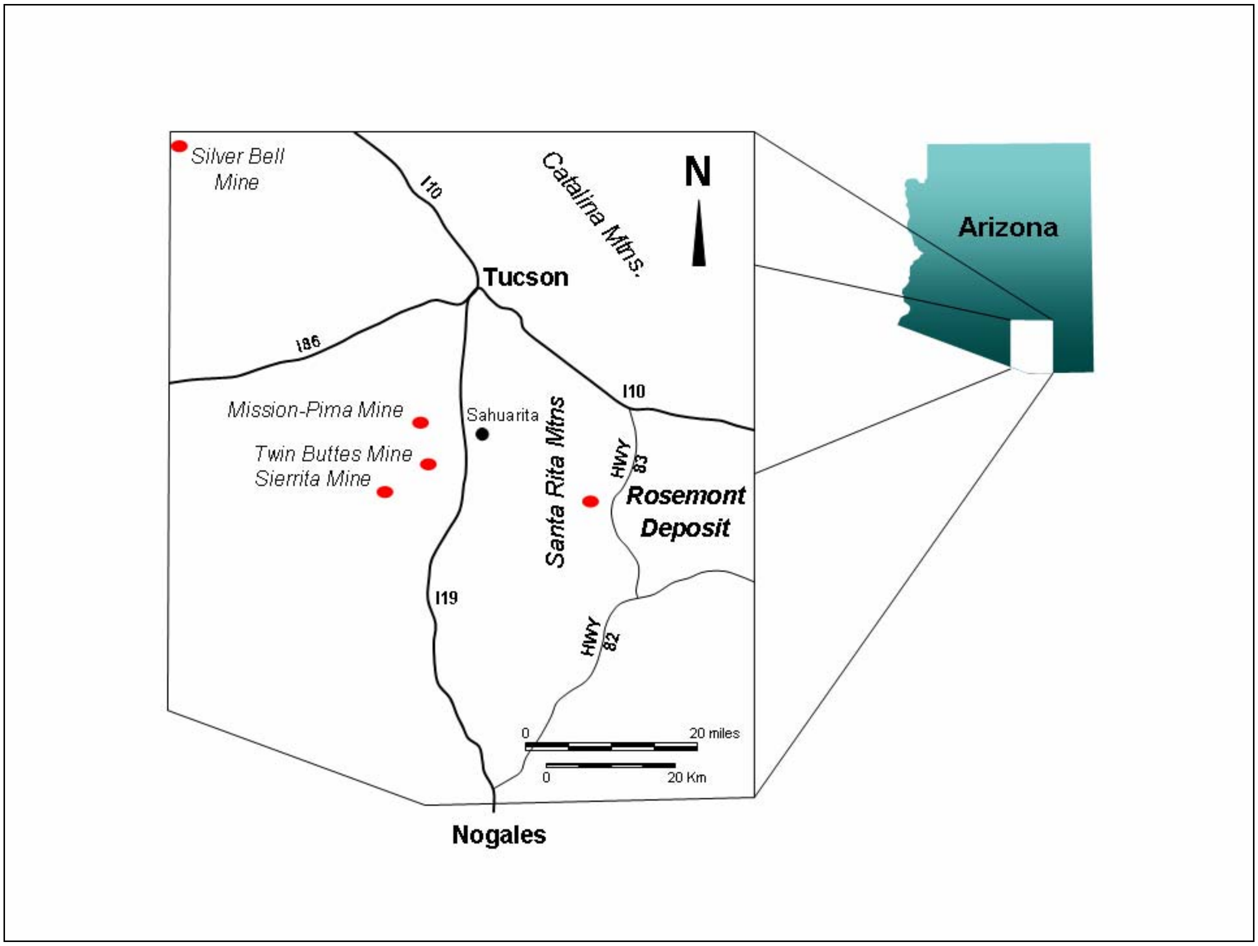


Figure 1-2
Rosemont Property Land Tenure

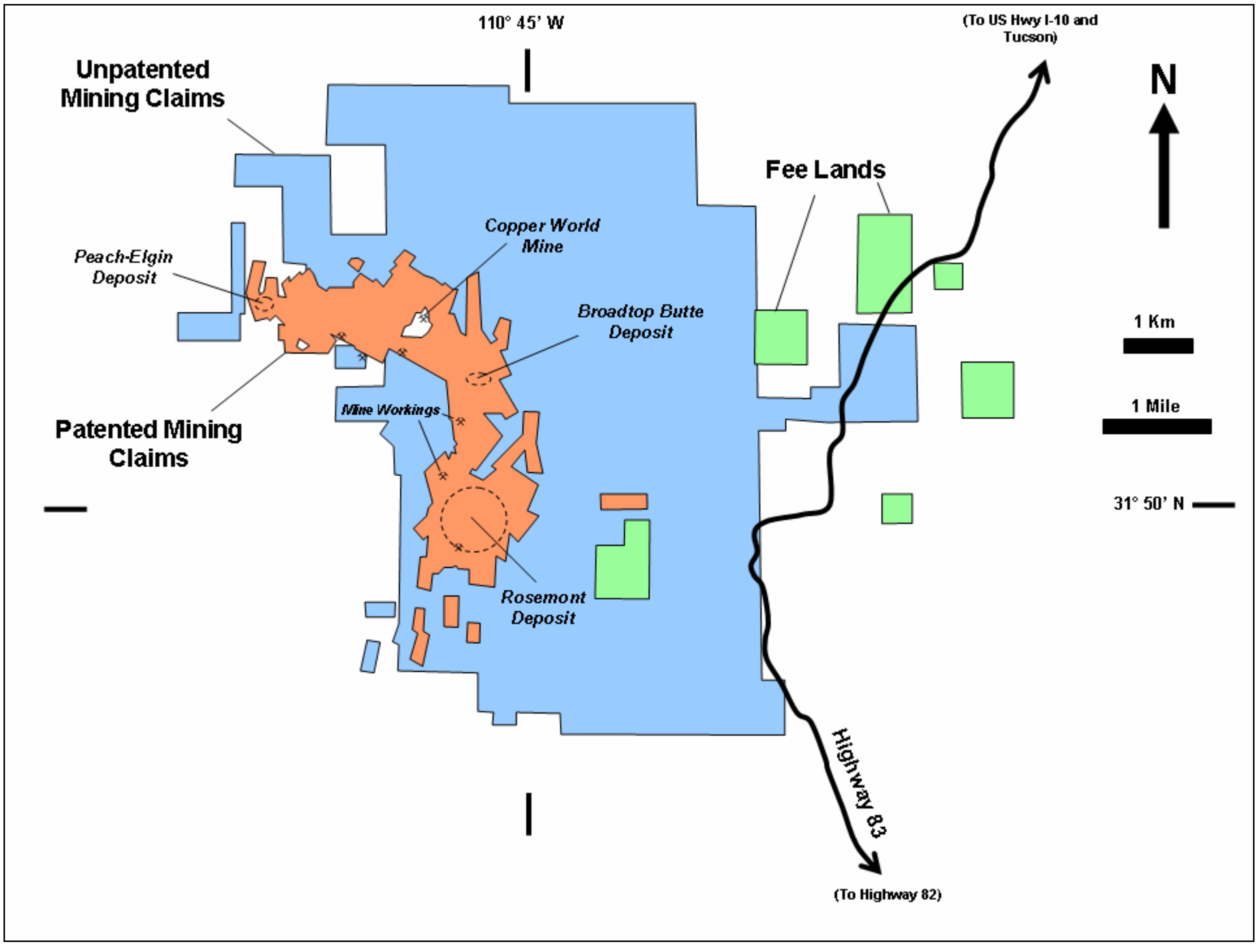
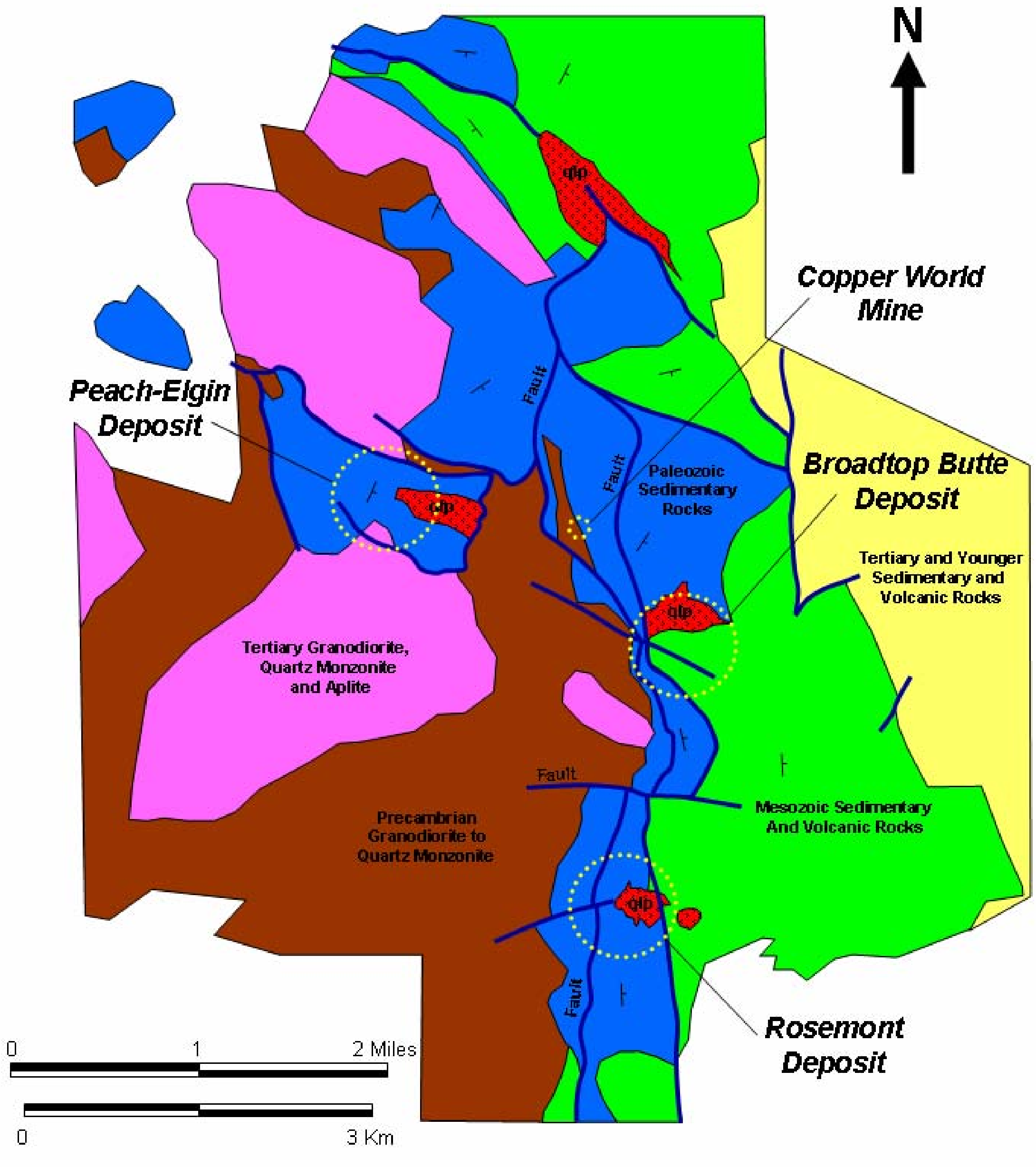


Figure 1-3
Rosemont Property Generalized Geologic Map



Generalized from Hardy, J.J., Jr., 1997.[5]

Figure 1-4
Rosemont District Stratigraphic Column

Era	Period	Formation (lith code)	Thickness (ft)	Section	Lithology	Skarn/Alteration	Mineralization	Relative Copper Content
Cenozoic	Tertiary	Gravel Unit (20)	0-600		Gravel & sand - partially consolidated paleovalley fill.	Not altered.	Not mineralized.	
		Quartz-Feldspar Porphyry (8) Intr.Bx (19)			Quartz monzonite porphyry to quartz latite porphyry intrusive (55.7-56.3 my). Assoc.intr.bx.	Qtz-ksp; qtz-ser-pyr; minor epidote-chlorite	Qtz +/- pyr-cpy-bn-mo veining, commonly oxidized. Mo disseminated and vein controlled.	
Mesozoic	Cretaceous	Willow Canyon Formation (10) Interbedded Andesite (9)	2,200		Interbedded arkosic sandstone, siltstone and conglomerate. Internal andesite flow sequence locally present.	Arkose: wk; ksp, epidote, calcite. Rare qtz veining. Andesite: stronger quartz-chlorite-epidote.	Arkose: wk limonite and secondary Cu mins. Rare qtz-pyr-cpy-bn vns. Andesite: stronger qtz-pyr-cpy-bn veining, commonly oxidized.	
		Glance Conglomerate (11)	0-1500		Limestone conglomerate with clasts of eroded older Paleozoic or Pc lithologies. Locally underlain by "Upper Plate" Concha Ls,+/- Scherrer Qtz,+/- Epitaph Fm. "Flat Fault" at base of sequence.	Weak to locally moderately marblized. Local calc-silicate alteration.	Mineralized locally (rarely).	
Paleozoic	Permian	Rainvalley Fm (18)	0-300		Fossiliferous limestone, dolomite and quartz sandstone	Serpentine-magnetite.	Minor mineralization.	
		Concha Limestone (14)	400-575		Thick-bedded, cherty, fossiliferous limestone	Marblized. Wollastonite and garnet skarns locally.	Secondary Cu and minor sulfides in BT Buttes area	
		Scherrer Fm (12)	720		Quartzite,dolomitic/calcareous, locally x-bedded; dolomite and limestone member. Basal siltstone.	Wk. chlorite in clastics. Garnet skarn after limestone.	Pyr, less common cpy and bn	
		Epitaph Fm (2)	1000		Limestone, dolomite and marl; less quartzite. Distinctive gypsum beds. Basal thick-bedded dolomite. *Gradational lower boundary.*	Strong chlorite-serpentine-magnetite; less garnet-diopside-chlorite skarn; hornfels	Bn-cpy +/- pyr vns and disseminated in skarn	
		Colina Limestone (3)	350		Dark, thick-bedded dolomitic limestone. Fossiliferous. Minor quartzite.	Marblized. Some serpentine-magnetite vns. Some garnet skarn.	Bn-cpy +/- pyr	
		Earp Fm (4)	800		Siltstone, shale, sandstone, chert-pebble conglomerate and limestone. *Grad low bdy.*	Hornfels and garnet-diopside-chlorite skarn.	Qtz-cal-chl and qtz vns with pyr-cpy-bn	
	Pennsylvanian	Horquilla Limestone (5)	1000		Thin- to thick-bedded limestone, siltstone, minor shale. Basal Black Prince Limestone	Skarn, hornfels, and marble. Garnet-pyroxene skarn with lessor chlorite and serpentinite. Local wollastonite.	Qtz and chl-serp-mag vns; bn-cpy-cc +/- pyr in vns and disseminated in skarn. Secondary Cu minerals near faults. Main mineralized formation.	
	Miss.	Escabrosa Limestone (6)	560		Thick-bedded to massive limestone, cherty.	Marble. Serpentine at faults. Garnet-diopside-magnetite at intrusive contact.	Commonly barren. Where altered, strong py-bn-cpy mineralization.	
	Dev.	Martin Formation (7)	400		Thin- to medium bedded dolomite; less limestone, siltstone and sandstone. Faulted contacts.	Marblized; minor chorite and serpentine at faults. Gnt & mixed skarn locally.	Weakly mineralized.	
	Cambrian	Abrigo Formation (13)	740-900		Thin-bedded limestone, siltstone, shale, sandstone.	Marblized. Gnt & mixed skarn locally.	Mneralized locally with py-bn-cpy.	
		Bolsa Quartzite (15)	460		Course-grained, thick-bedded quartzite	Weakly altered, unreactive.	Trace disseminated pyr and cpy	
pC	pC	Continental Granodiorite (16)			Granodiorite porphyry	Weak alteration locally.	Weak mineralization locally	

Notes: Stratigraphic thicknesses taken from H. Drewes, Professional Paper 748 (1972), and may exceed thicknesses found locally at the Rosemont Deposit. Skarn/Alteration and Mineralization from Daffron and others, Augusta Resource internal report, 2007. Overburden/Fill = 1. Unassigned lithologies = 17.

Figure 1-5
Rosemont Deposit Geologic Plan Map
4500 Ft Elevation

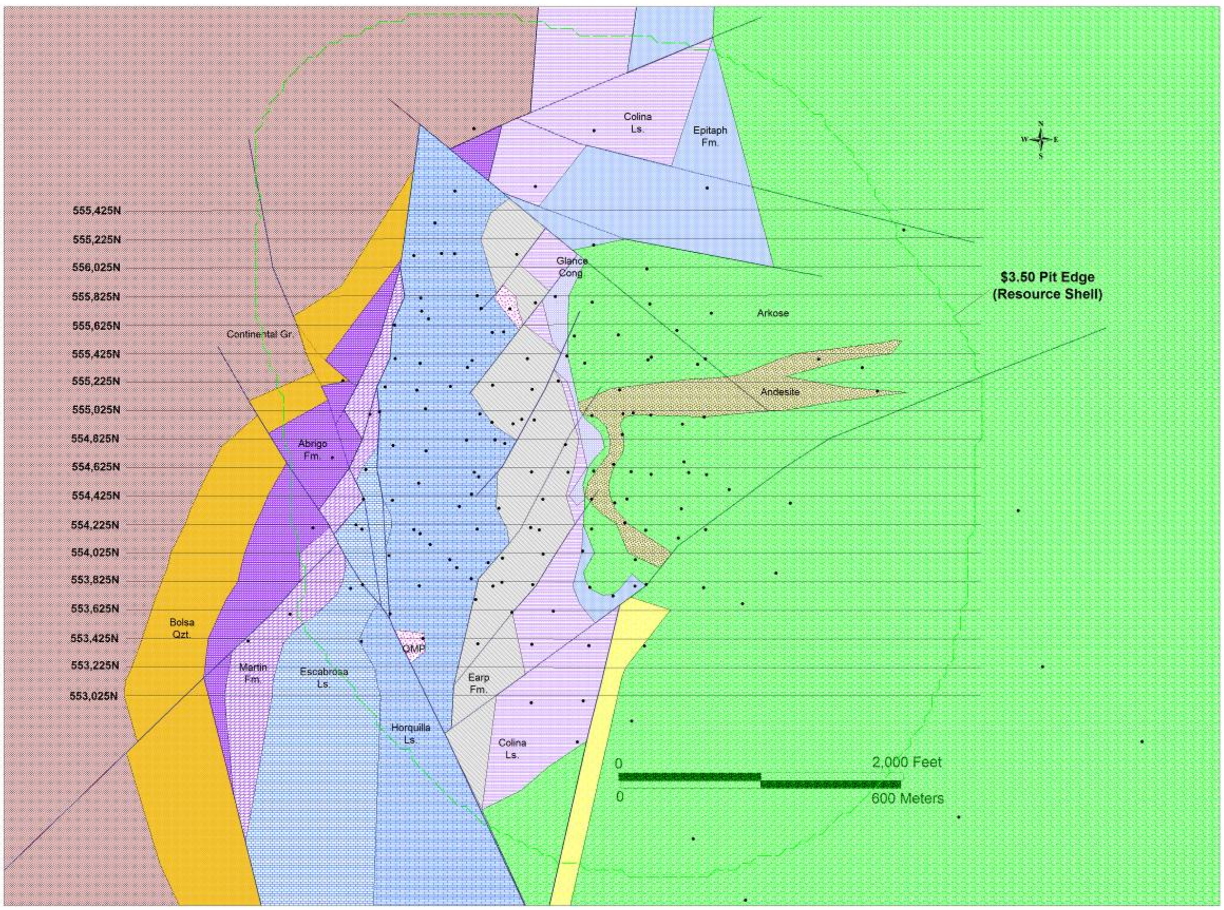


Figure 1-6
Rosemont Deposit Geologic Plan Map
3500 Ft Elevation

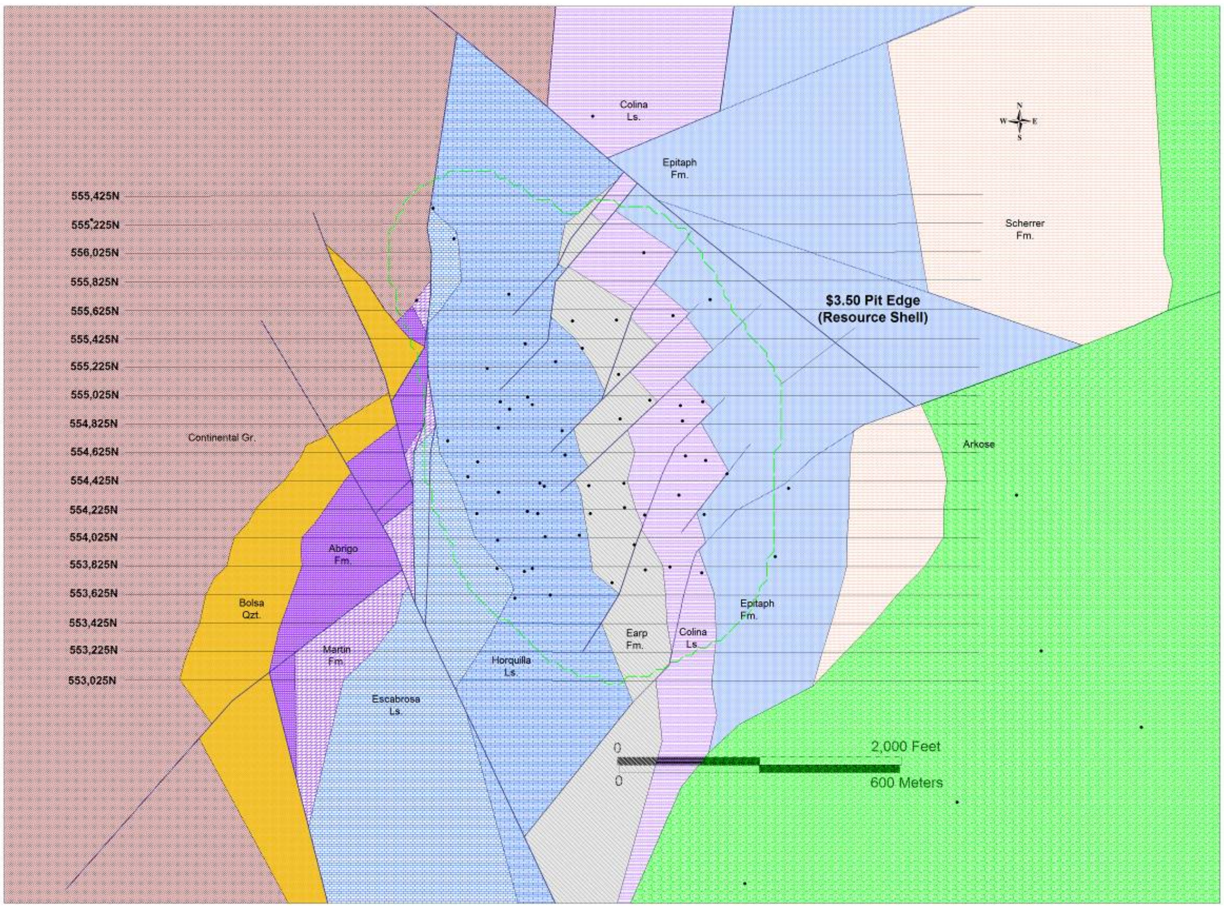


Figure 1-7
Rosemont Deposit Geologic Cross Section
At 11,554,225 N (looking north)

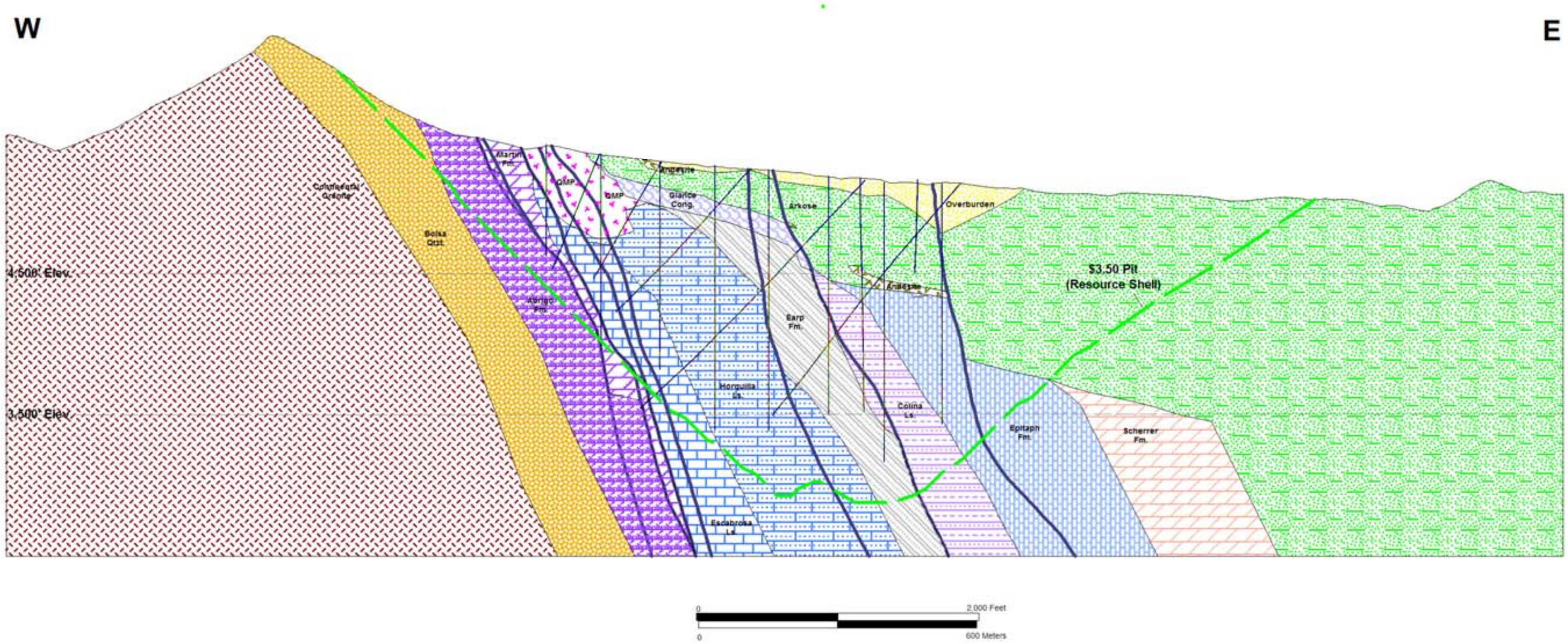


Figure 1-8
Rosemont Drill Hole Collar Locations

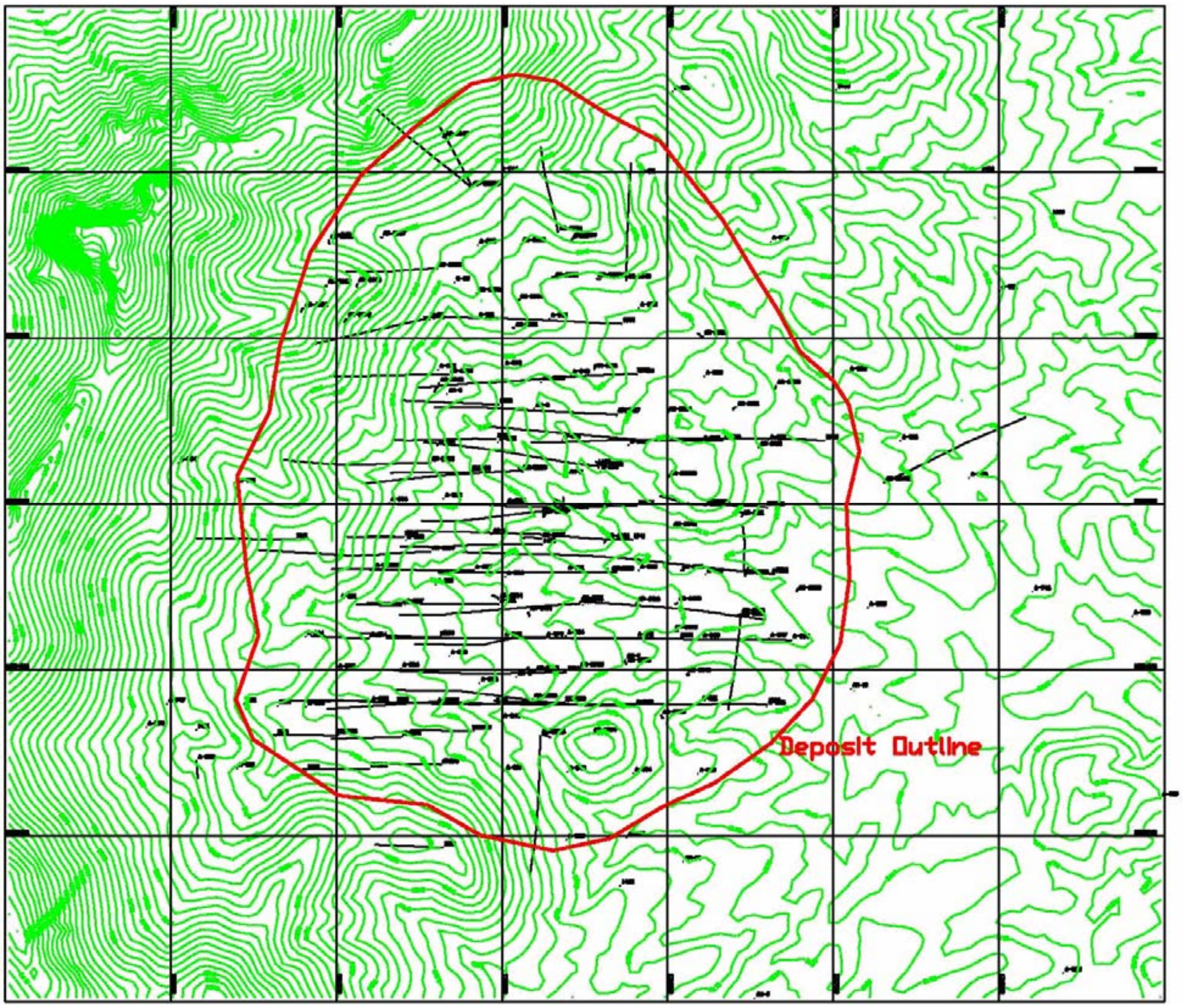
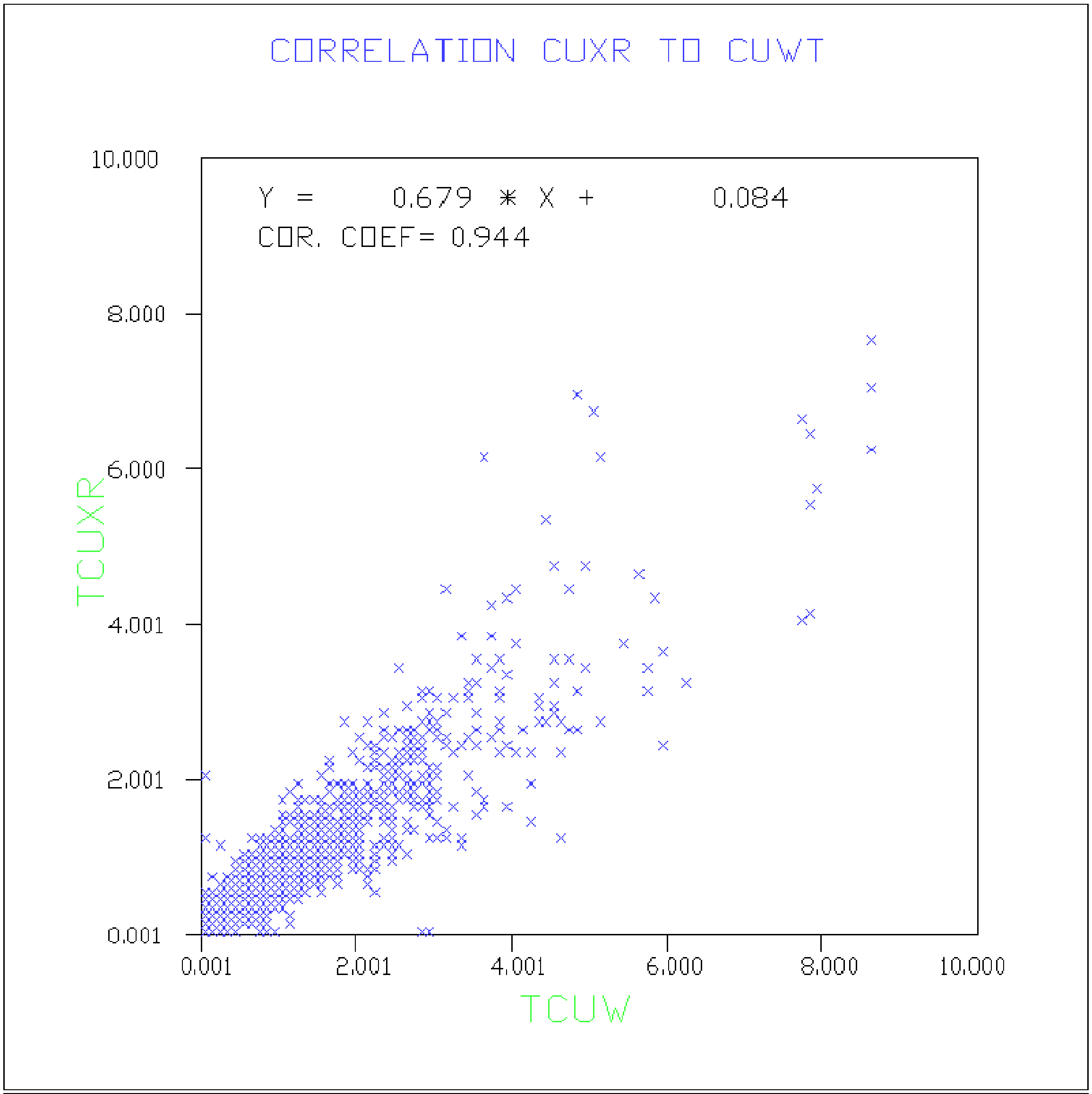
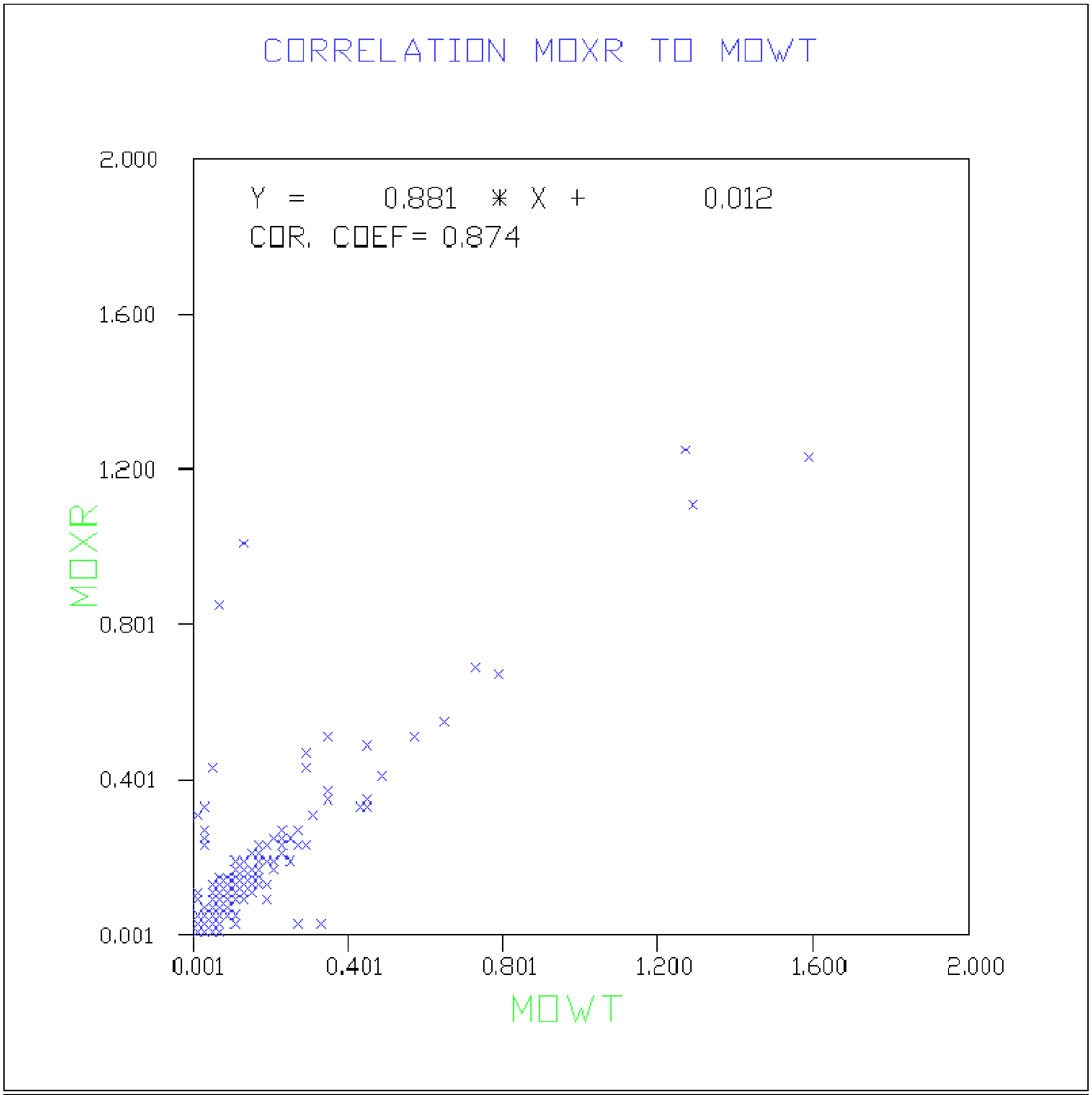


Figure 1-9
XRF-Wet Assay Correlation Plot for Cu



X- MEAN=	0.492	MIN.=	0.000	NO. =	9009
STD DEV=	0.926	MAX.=	24.700	C.V.=	1.882
Y- MEAN=	0.417	MIN.=	0.000	NO. =	9009
STD DEV=	0.665	MAX.=	14.700	C.V.=	1.594

Figure 1-10
XRF-Wet Assay Correlation Plot for Mo



X- MEAN=	0.043	MIN.=	0.000	NO. =	2015
STD DEV=	0.075	MAX.=	1.590	C.V.=	1.745
Y- MEAN=	0.050	MIN.=	0.000	NO. =	2015
STD DEV=	0.076	MAX.=	1.260	C.V.=	1.526

Figure 1-11
Lognormal Cumulative Probability Plot for Cu

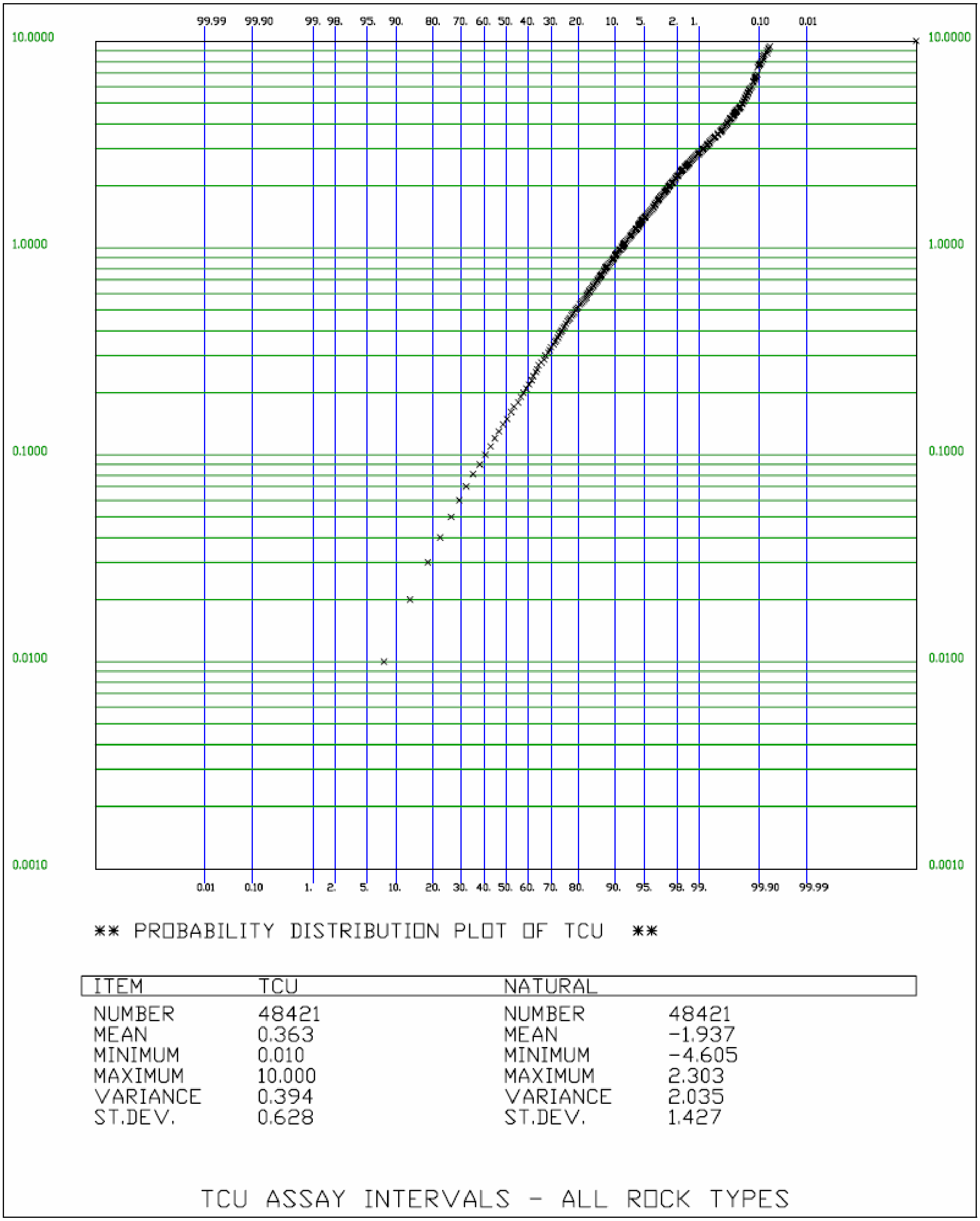
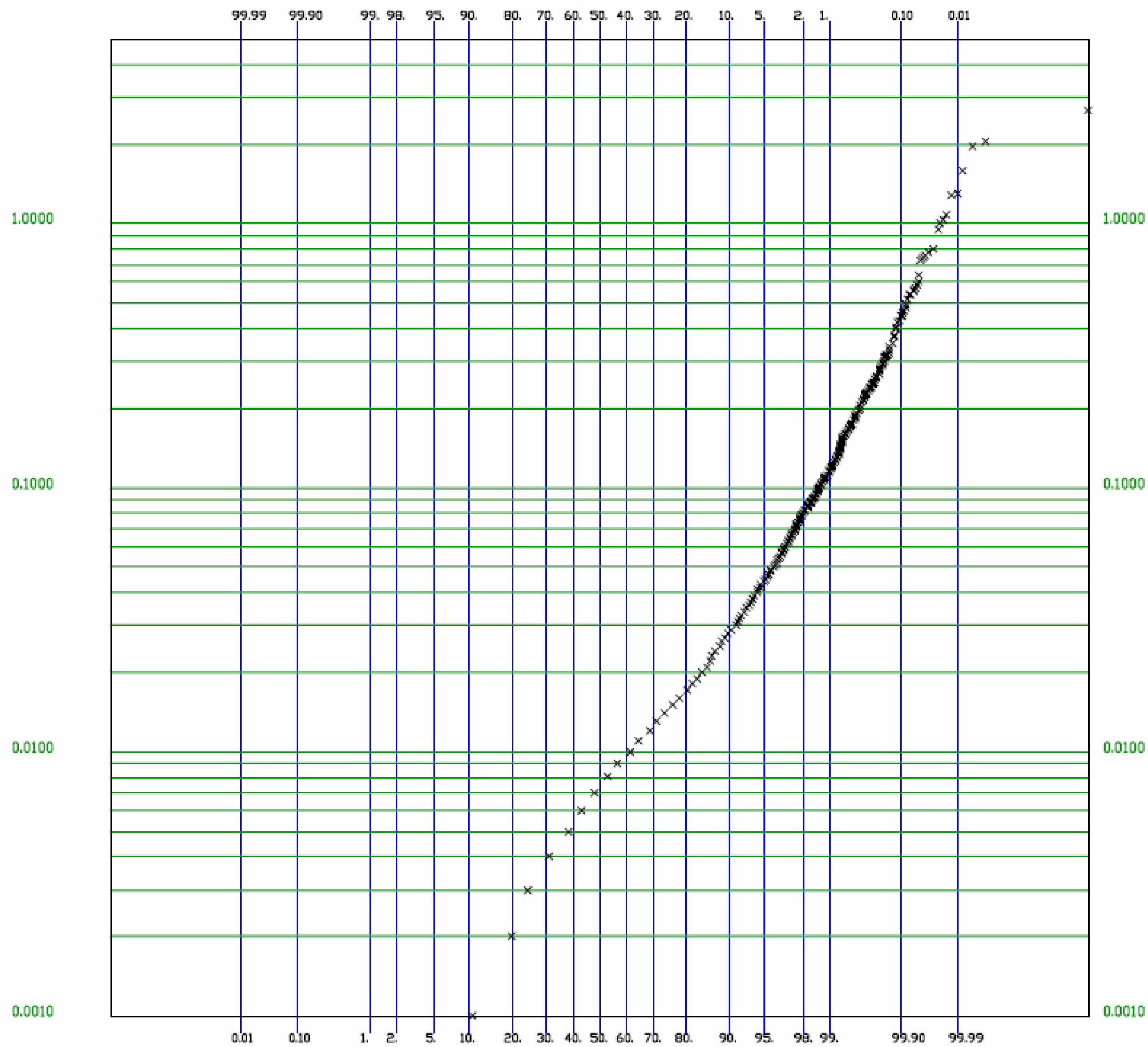


Figure 1-12
Lognormal Cumulative Probability Plot for Mo

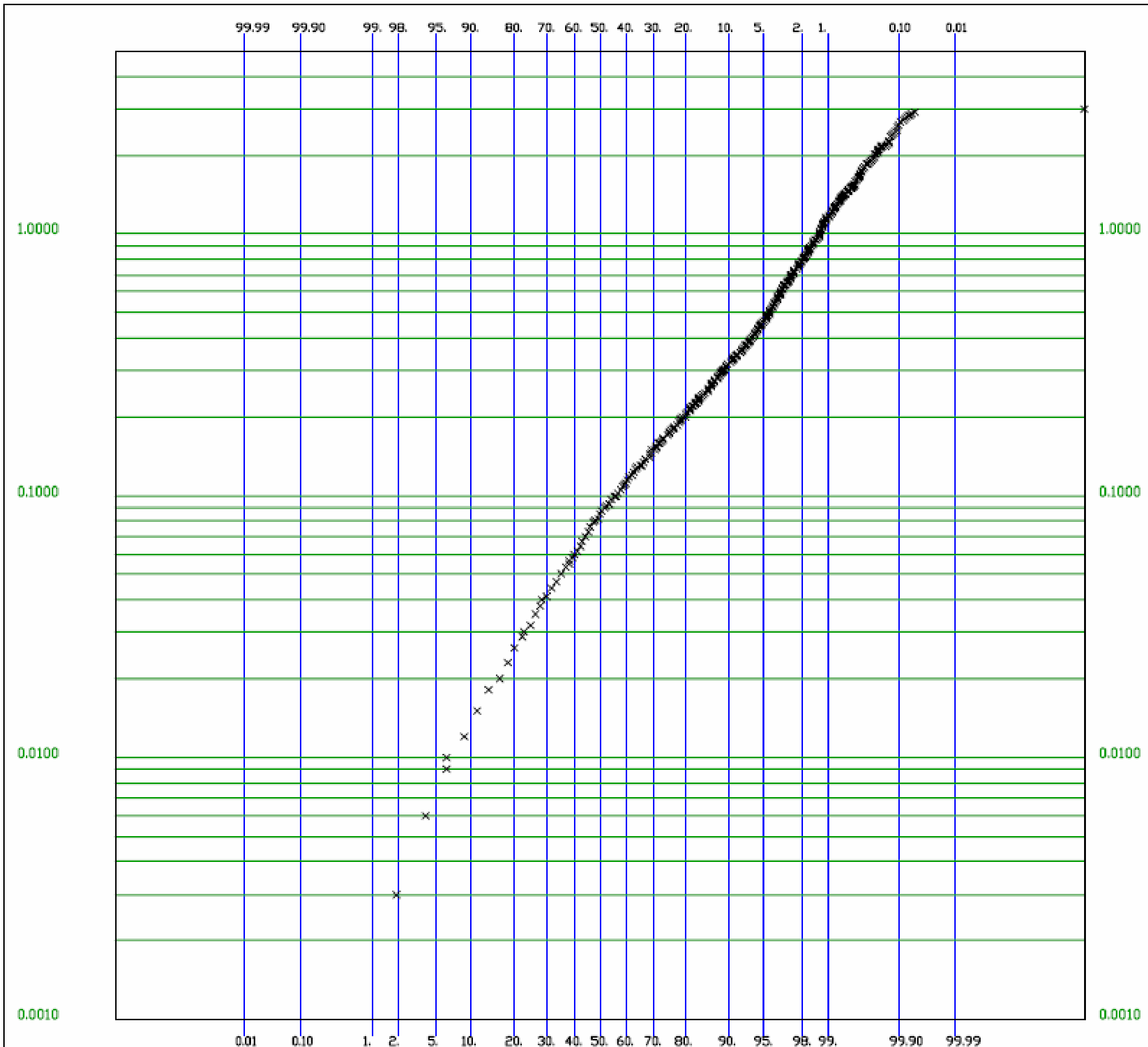


** PROBABILITY DISTRIBUTION PLOT OF MO **

ITEM	MO	NATURAL	
NUMBER	38156	NUMBER	38156
MEAN	0.015	MEAN	-4.922
MINIMUM	0.001	MINIMUM	-6.908
MAXIMUM	2.660	MAXIMUM	0.978
VARIANCE	0.001	VARIANCE	1.272
ST.DEV.	0.038	ST.DEV.	1.128

MO ASSAY INTERVALS - ALL ROCK TYPES

Figure 1-13
Lognormal Cumulative Probability Plot for Ag



** PROBABILITY DISTRIBUTION PLOT OF AG **

ITEM	AG	NATURAL	
NUMBER	30124	NUMBER	30124
MEAN	0.148	MEAN	-2.582
MINIMUM	0.003	MINIMUM	-5.809
MAXIMUM	3.000	MAXIMUM	1.099
VARIANCE	0.050	VARIANCE	1.474
ST.DEV.	0.225	ST.DEV.	1.214

AG ASSAY INTERVALS - ALL ROCK TYPES

Figure 1-14
Variogram of 50-Ft Composited Cu Values

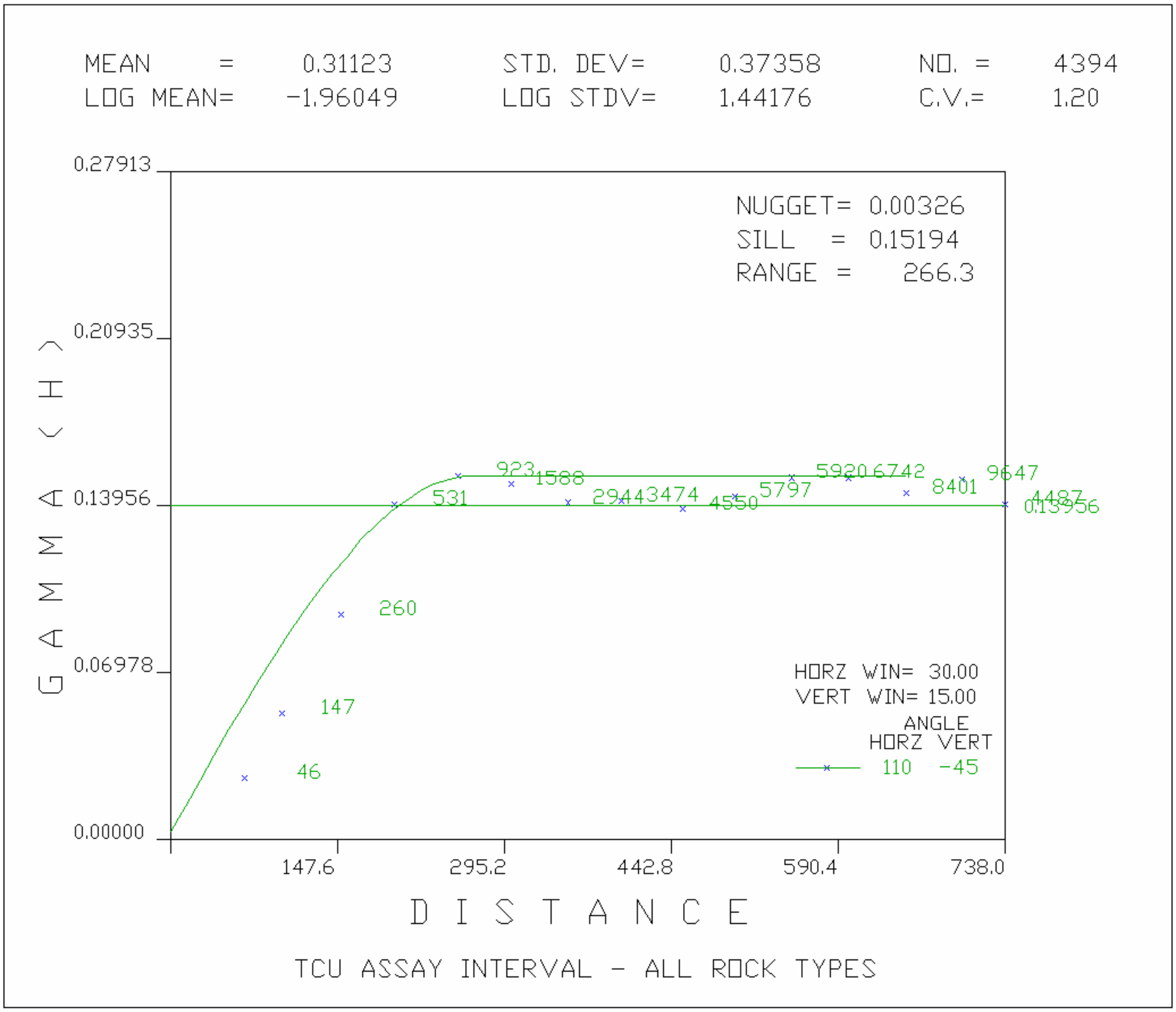


Figure 1-15
Variogram of 50-Ft Composited Mo Values

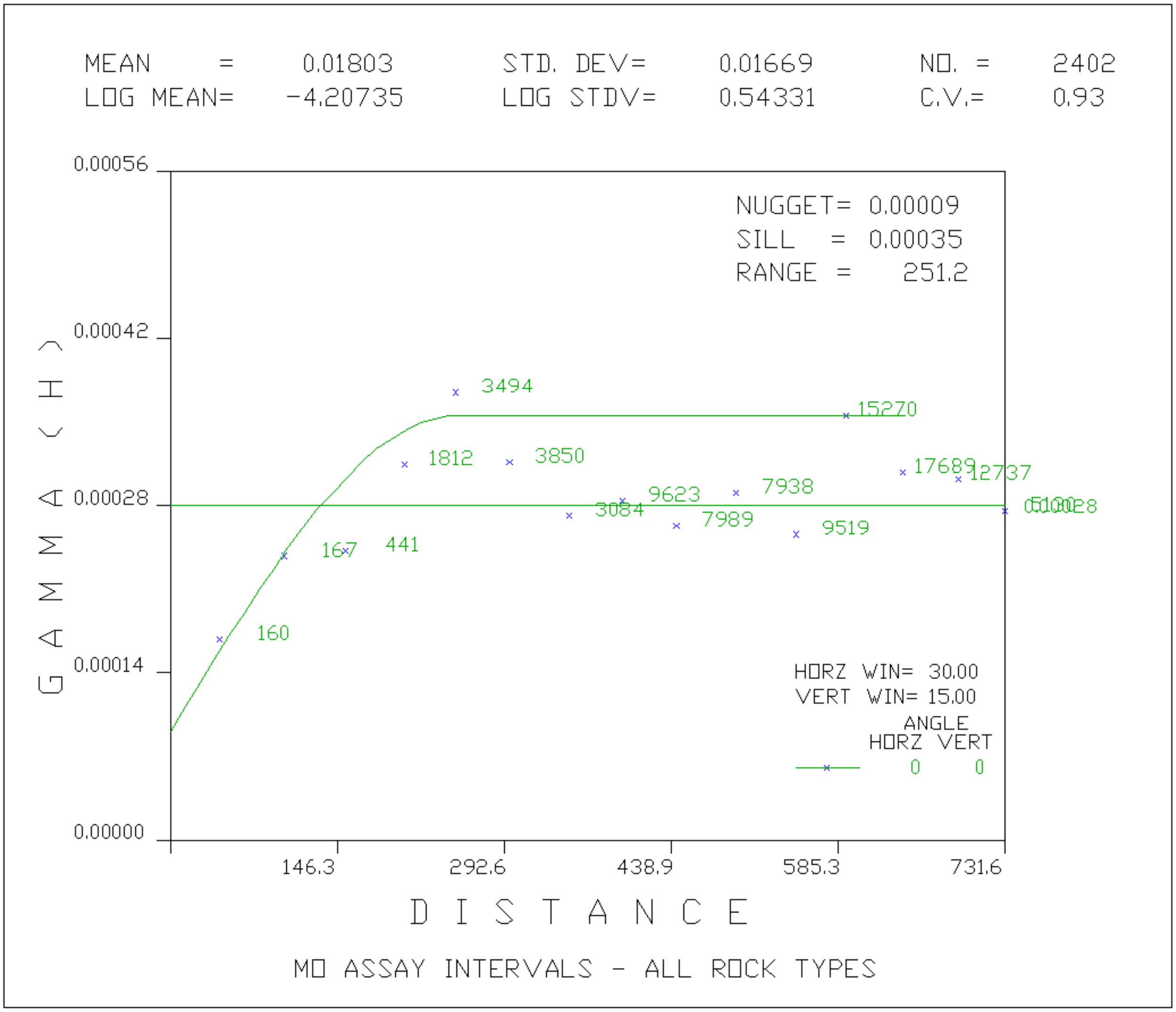


Figure 1-16
Variogram of 50-Ft Composited Ag Values

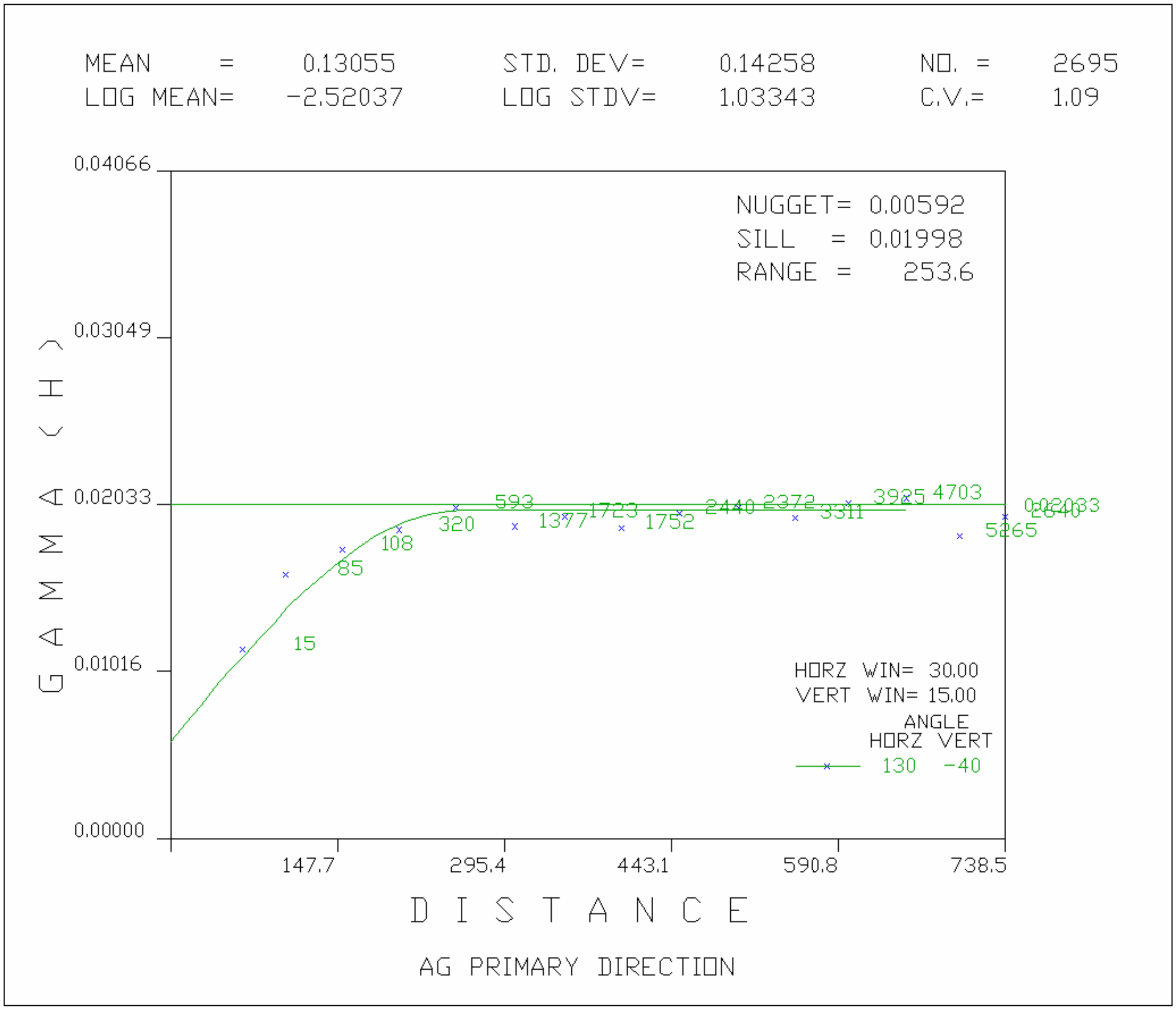


Figure 1-17
Pit Slope Design Sectors and Maximum Slope Angles

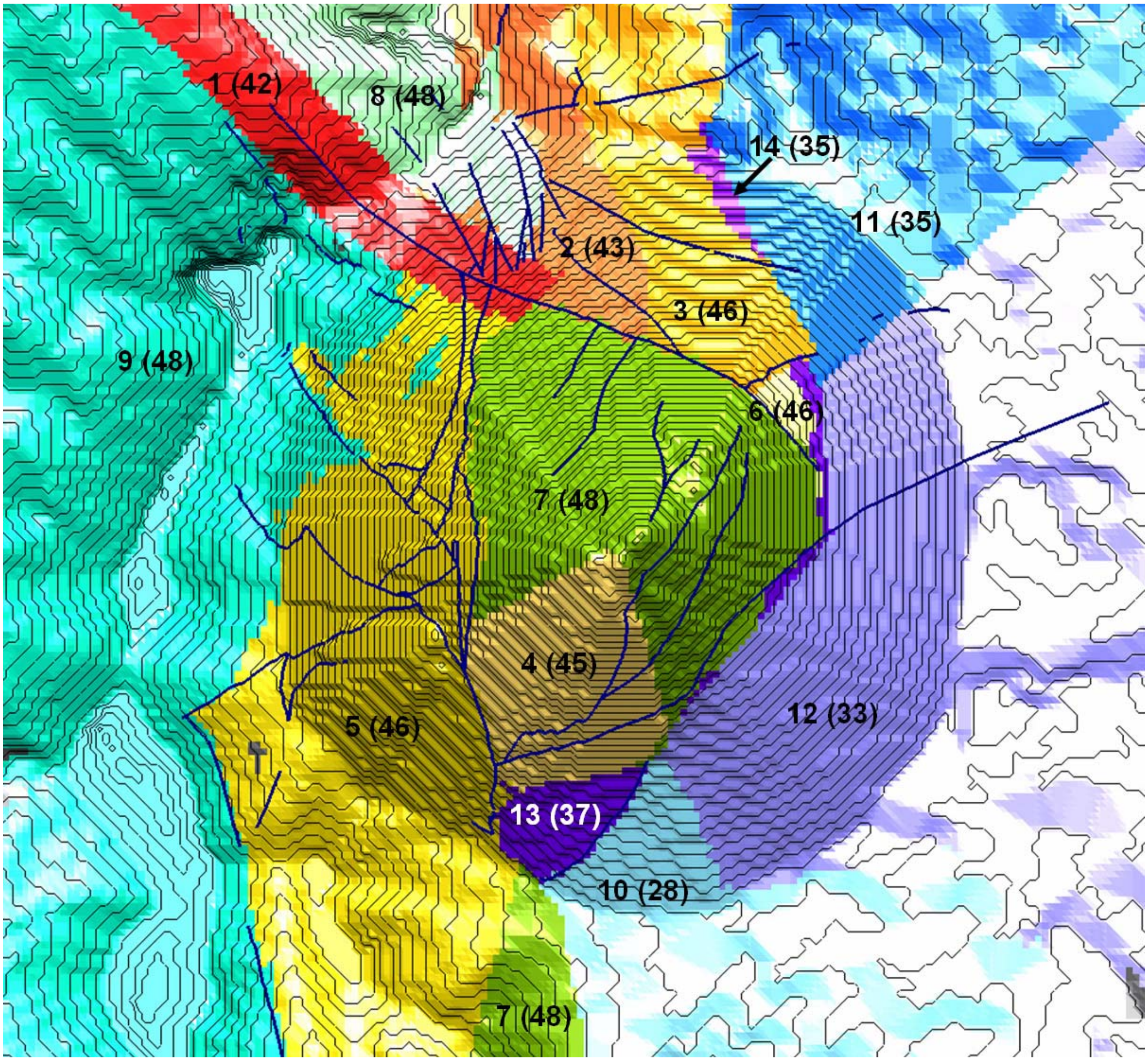


Figure 1-18
Rosemont Ultimate Pit Plan

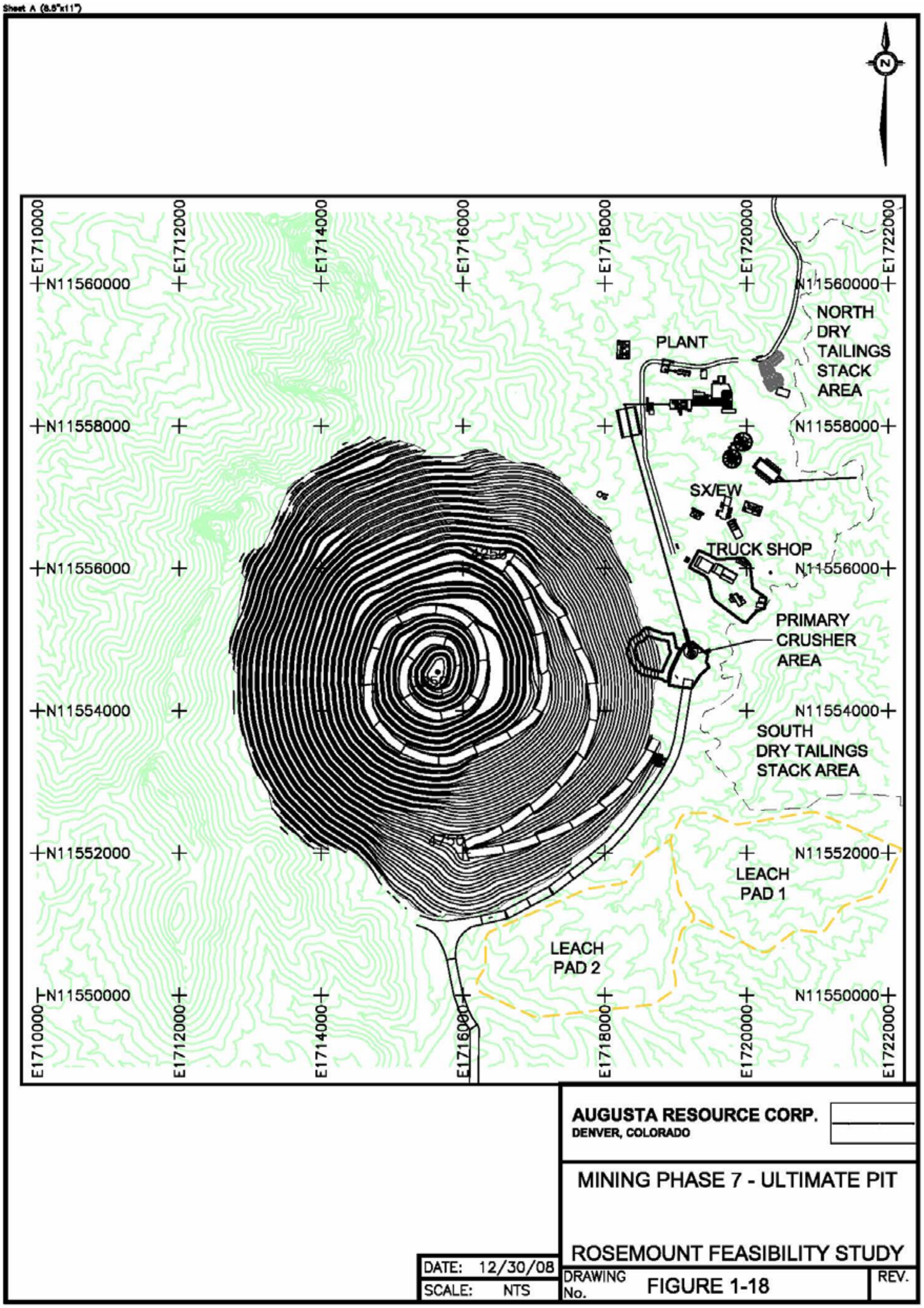
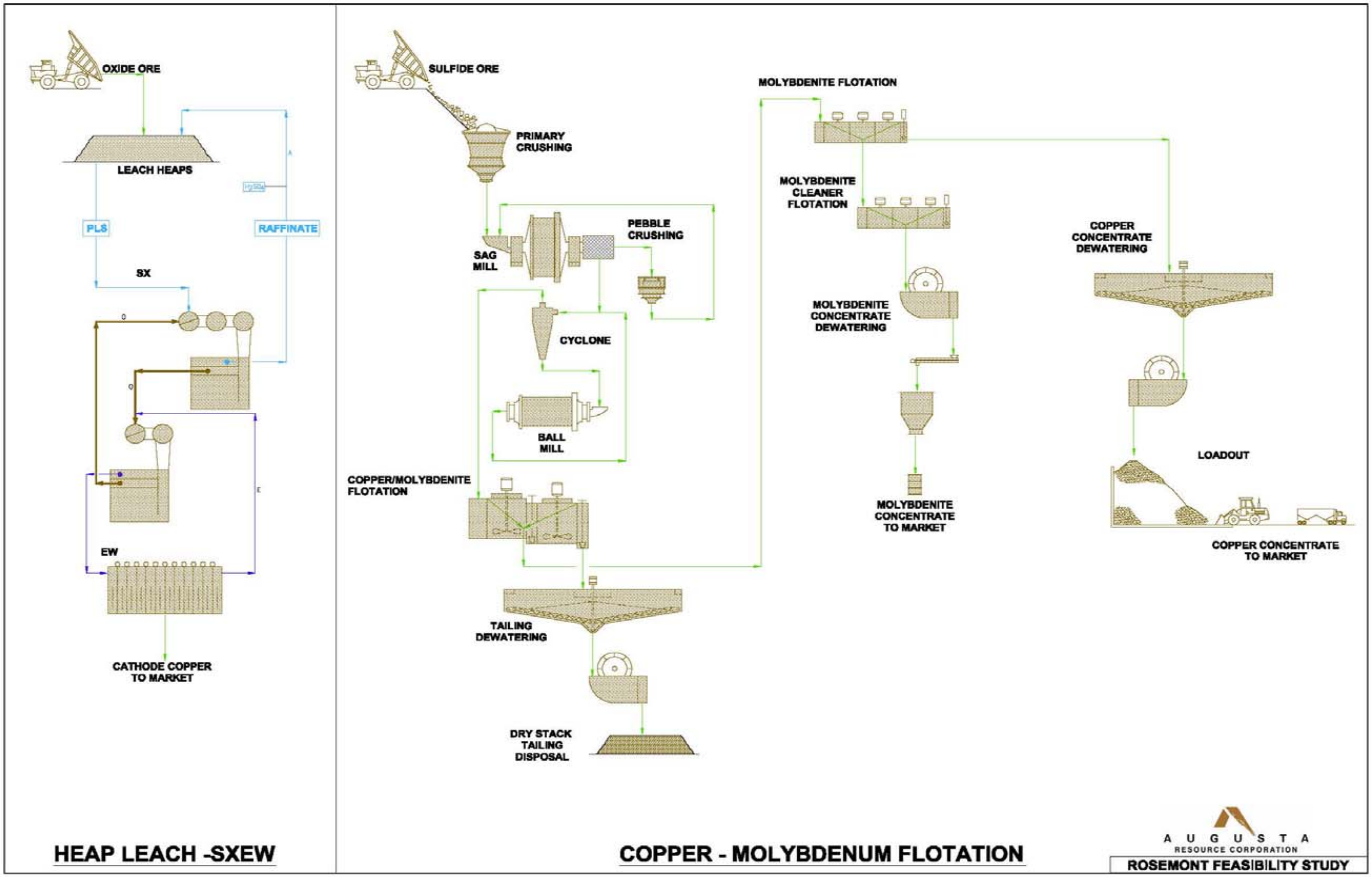


Figure 1-19
Overall Process Flowsheet



APPENDIX A

PROFESSIONAL QUALIFICATIONS

CERTIFICATE of QUALIFIED PERSON

I, Dr. Conrad E. Huss, P.E., Ph.D., do hereby certify that:

1. I am currently employed as Executive Vice President and Chairman of the Board by:

   M3 Engineering & Technology Corporation
   2440 W. Ruthrauff Rd.
   Tucson, Arizona 85705
   U.S.A.

2. I graduated with a degree in Bachelor's of Science in Mathematics and a Bachelor's of Art in English from the University of Illinois in 1963. I graduated with a Master's of Science in Engineering Mechanics from the University of Arizona in 1968. In addition, I earned a Doctor of Philosophy in Engineering Mechanics from the University of Arizona in 1970.

3. I am a professional engineer in good standing in the State of Arizona in the areas of civil and structural engineering. I am also registered as a professional engineer in the states of California, Maine, Minnesota, Missouri, Montana, New Mexico, Oklahoma, Oregon, Texas, Utah and Wyoming.

4. I have worked as an engineer for a total of forty years since my graduation from the University of Illinois. I have taught at the University level part-time for five years and as an assistant professor for one year.

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

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

7. I am responsible for the overall preparation of the "NI 43-101 Technical Report for the Rosemont Copper Project Updated Feasibility Study, Pima County, Arizona, USA" (the "Technical Report"), dated January 14, 2009, prepared for Augusta Resource Corporation; integrating the mineral resource estimate, mineral reserve estimate, and mining sections by WLR Consulting and the property history, geological setting, mineralization, exploration, drilling, sampling and data verification sections by August Resource Corporation.

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

9. I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101.

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

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 14th day of January 2009.

Registered Professional Engineer (Civil)
CERTIFICATE NO. 9648
CONRAD E. HUSS
Date Signed 14 JAN 2009
ARIZONA, U.S.A.
EXPIRES 31 MARCH 2009

____
Signature of Qualified Person

Conrad E. Huss, P.E, Ph.D.
Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, William L. Rose, P.E., do hereby certify that:

1. I am currently employed as Principal Mining Engineer by:

   WLR Consulting, Inc.
   9386 West Iowa Avenue
   Lakewood, Colorado 80232-6441
   U.S.A.

2. I graduated with a Bachelor of Science degree in Mining Engineering from the Colorado School of Mines in 1977.

3. I am a:
   - Registered Professional Engineer in the State of Colorado (No. 19296)
   - Registered Professional Engineer in the State of Arizona (No. 15055)
   - Registered Member of the Society for Mining, Metallurgy and Exploration, Inc. (no. 2762350RM)

4. I have worked as a mining engineer for 31 years since my graduation from college.

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

6. I am responsible for the preparation of the mineral resource estimates contained in the technical report titled *NI 43-101 Technical Report for the Rosemont Copper Project Updated Feasibility Study, Pima County, Arizona, USA* and dated January 14, 2009 (the "Technical Report"), prepared for Augusta Resource Corporation and, in particular, I prepared Subsections 1.19.1 through 1.19.12 and portions of Subsection 1.3.6 (Summary – Mineral Resource and Mineral Reserve Estimates, paragraphs and tables pertaining to the deposit model and mineral resource estimates) of the Technical Report.

7. I have had prior involvement with the property that is the subject of the Technical Report. I have completed prior mineral resource estimates and technical reports during December 2008, April 2007, February-April 2006 and participated in the Preliminary Assessment evaluation that was completed in June 2006. I have visited the subject property on August 9, 2005.

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

9. I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101.

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

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 14th day of January 2009.

Signature of Qualified Person

William L. Rose

Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Robert H. Fong, P. Eng., do hereby certify that:

1. I am a Principal Mining Engineer associated with:

   Moose Mountain Technical Services (MMTS)
   P.O. Box 797
   Elkford, B.C. Canada
   V0B-1H0

2. I am a graduate of McGill University, Montreal, Quebec, and hold a Bachelor of Engineering Degree (B. Eng.) - Mining, 1979.

3. I am a registered professional engineering in good standing with the Association of Professional Engineers, Geologists and Geophysicists of Alberta (No. M59151)

4. I have worked as a mining engineering since graduation from university, and have provided over 12 years of engineering consulting services to projects in Canada, United States, South America, Mexico, Africa and Asia.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, relevant work experience, and affiliation with APEGGA, I fulfill the requirements to be a "Qualified Person" as set out by NI 43-101.

6. I am responsible for the preparation of mineral reserves portion of Section 1.3.6 (Summary – Mineral Resource and Mineral Reserve Estimates), 1.19.13 (Pit Limit Analysis), 1.19.14 (Pit Designs), and 1.19.15 (Mineral Reserve Estimate) of the technical report titled "*NI 43-101 Technical Report for Rosemont Copper Project Updated Feasibility Study, Pima County, Arizona, USA*", dated January 14, 2009 (the Technical Report) relating to the Rosemont property.

7. I have not had prior involvement with the Rosemont property that is the subject of this Technical Report.

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

9. I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101.

10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 14th day of January, 2009.

____
Signature of Qualified Person

Robert H. Fong
____
Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Thomas L. Drielick, P.E., do hereby certify that:

1. I am currently employed as Sr. Vice President by:

   M3 Engineering & Technology Corporation
   2440 West Ruthrauff Rd.
   Tucson, Arizona 85705
   U.S.A.

2. I am a graduate of Michigan Technological University and received a Bachelor of Science degree in Metallurgical Engineering in 1970. I am also a graduate of Southern Illinois University and received an M.B.A. degree in 1973.

3. I am a:
   - Registered Professional Engineer in the State of Arizona (No. 22958)
   - Registered Professional Engineer in the State of Michigan (No. 6201055633)
   - Member in good standing of the Society for Mining, Metallurgy and Exploration, Inc. (No. 850920)

4. I have practiced metallurgical and mineral processing engineering and project management for 37 years. I have worked for mining and exploration companies for 18 years and for M3 Engineering and Technology Corporation for 19 years.

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

6. I am responsible for the preparation of Section 1.3.8 "Metallurgical Testing", Section 1.3.9 "Process Flowsheet", Section1.3.10 "Extraction Rates", Section 1.3.11 "Process Reagents", Section 1.18 "Mineral Processing and Metallurgical Testing", Section 1.25.2 "Metallurgical Testing", Section 1.25.3 "Processing Flowsheets", Section1.25.4 "Extraction Rates", and Section 1.25.5 "Process Reagents" of the technical report titled *Rosemont Copper Project Updated Feasibility Study Volume I NI 43-101 Technical Report Prepared for Augusta Resource Corporation* dated January 2009 (the "Technical Report").

7. I have had prior involvement with the property that is the subject of the Technical Report. I participated in the preparation of the technical report titled *NI 43-101 Technical Report for the Rosemont Copper Project Feasibility Study, Pima County, Arizona, USA* dated August 2007, relating to the Augusta Rosemont property. I participated in the preparation of the technical report titled *2008 Mineral Resource Update for the Rosemont Project, Pima County, Arizona, USA* and dated December 4, 2008, relating to the Augusta Rosemont property. I visited the subject property on August 21, 2007.

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

9. I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101.

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

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 14th day of January 2009.

Signature of Qualified Person

Registered Professional Engineer (Mining)
CERTIFICATE NO
22958
THOMAS L
DRIELICK
Date Signed 1-14-09
ARIZONA U.S.A.
Expires 9/30/2011

Thomas L. Drielick
Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, John Ajie, P.E., do hereby certify that:

1. I am currently employed as Vice President of Engineering and Operations Support by:

   URS Washington Division
   7800 East Union Avenue, Suite 100
   Denver, CO 80237
   USA

2. I graduated with a Masters and Bachelor of Science degrees in Mining Engineering from the University of California at Berkeley, CA and New Mexico Tech University at Socorro, NM in 1981 and 1979 respectively.

3. I am a:
   - Registered Professional Engineer in the State of Texas, 55901
   - Registered Professional Engineer in the State of New Mexico, 14276
   - Registered Professional Engineer in the State of MT, 8420E

4. I have worked as a mining engineer for 28 years since my graduation from college.

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

6. I have had prior involvement with the property that is the subject of the Technical Report. I have visited the subject property on May 18, 2006.

7. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

8. I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101.

9. 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.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes,

including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 14 day of January 2009.

JOHN I. AJIE
NEW MEXICO
14276
REGISTERED PROFESSIONAL ENGINEER

(sealed)

Signature of Qualified Person

John I. Ajie
Print Name of Qualified Person

CONRAD E. HUSS, P.E.
Project Manager

EDUCATION	Ph.D., Engineering Mechanics, University of Arizona M.S., Engineering Mechanics, University of Arizona B.A., English, University of Illinois B.S., Mathematics, University of Illinois
REGISTRATION	Civil and Structural Engineer - Arizona Professional Engineer - California, Idaho, Maine, Minnesota, Montana, New Mexico, Oklahoma, Oregon, Texas, Utah, Washington, Wyoming
EXPERIENCE	Forty-two (42) years of design in industrial, municipal, commercial projects, including material handling, reclamation, water treatment, base metal and precious metal process plants, industrial minerals, smelters, chemical plants, special structures and audits. Career highlights include twenty-five years of design/construct experience, plant startups in South America and Mexico, oceanography/surveying in Alaska and Hawaii, and six years of university teaching.

PROJECT EXPERIENCE

- M3 Engineering & Technology, Project Manager (22 Years)
  - Goldcorp – Minera Peñasquito 130,000 MTPD - Mexico
  - Global Alumina – Republic of Guinea 43-101 Study - Africa
  - Pan American, Alamo Dorado 43-101 Study – Mexico
  - Pan American Manantial Espejo Silver/Gold - Argentina
  - Phelps Dodge Bagdad Primary Crusher Relocation – Arizona
  - Refugio Reopening - Chile
  - Climax Molybdenum Moly Metal Plant - Arizona
  - Phelps Dodge Safford CASC Design - Arizona
  - Phelps Dodge Safford Copper Leach - Arizona
  - CEMEX Victorville Clinker Hall - California
  - Western Silver Peñasquito Scoping, 43-101 Study and Feasibility Studies – Zacatecas, Mexico
  - Piedras Verdes Copper Leach 43-101 Study - Mexico
  - AVESTOR Lithium Vanadium Polymer Battery Plant with Laboratories – Nevada
  - Kennecott Utah Lime Plant Feasibility Study - Utah
  - Phelps Dodge Arizona Closure/Closeout Plans at 7 properties - Arizona
  - Alamos Gold Mulatos Prefeasibility - Mexico
  - Teck Cominco Glamis Gold Feasibility - Mexico
  - Kennecott Rawhide conceptual Closeout Plan for Leach Pile - Nevada
  - Phelps Dodge El Abra Structural and Material Handling Audit - Chile
  - Kennecott Utah Bid Call for Restructure of Maintenance Workforce - Utah
  - Phelps Dodge El Abra SX-EW ER Tank Replacement - Chile
  - Fischer-Watt Copper SX-EW Prefeasibility - Mexico
  - Kerr McGee 1200 MTPY BLVO Plant - Apex, Nevada

- M3 Engineering & Technology, (continued)
  - Billiton/BHP Worsley Alumina Plant Audit - Australia
  - Phelps Dodge Tyrone Closure/Closeout Plans with Water Treatment Plant - New Mexico
  - Mitsubishi Cement Lucerne Valley Plant Upgrades - California
  - Chino Closure/Closeout Plans with Water Treatment Plant - New Mexico
  - Mitsubishi Cement Longbeach Ocean Port - California
  - Cobre Closure/Closeout Plans - New Mexico
  - Phelps Dodge El Abra, Chile, Material Handling and Structural Audit
  - Billiton/ALCOA Alumar Alumina Refinery Plant in Brazil, Material Handling/Structural Audit
  - Peñoles F.I. Madero 8,000 TMPD Greensfield Silver/Lead/Zinc - Mexico
  - Kennecott Greens Creek Flotation Expansion, Silver/Lead/Zinc - Alaska
  - Phelps Dodge Henderson, Colorado, Material Handling and Structural Audit
  - Kennecott Greens Creek Pyrite Circuit for Reclamation - Alaska
  - Phelps Dodge Morenci Coronado Leach -Arizona
  - Cyprus Cerro Verde Crush/Convey - Peru
  - California Portland Cement RIMOD 3 Expansion - Arizona
  - Phelps Dodge Candelaria Material Handling and Structural Audit - Chile
  - Minera Alumbrera Startup and Performance Test for Copper/Gold Plant - Argentina
  - Echo Bay Gold Aquarius Feasibility Study - Canada
  - Arizona Portland Cement Expansion - Arizona
  - Minera Alumbrera, SAG Mill Run In – 3 month field assignment - Argentina
  - Phelps Dodge Ajo, Open Air Copper Mill - Arizona
  - Echo Bay Paredones Gold Amarillos EPCM Basic Engineering - Mexico
  - Cyprus Sierrita Inpit Crush/Convey - Arizona
  - Kennecott Smelter Upgrade following Audit - Utah
  - Battle Mountain Crown Jewel, Gold and Silver Detail Engineering - Washington
  - Kennecott Greens Creek Reopening and Reclamation, Lead/Zinc/Silver - Alaska
  - Cyprus Bagdad Material Handling and Structural Audit - Arizona
  - Hecla Rosebud Precious Metal Detail Engineering and Reclamation - Nevada
  - Phelps Dodge Morenci Ball Mills A-7 and B-32, Copper Concentrator
  - Phelps Dodge Hidalgo Smelter Upgrade Simulation
  - Kerr-McGee West Chicago Physical Separation Reclamation Facility with Water Treatment
  - Lluvia Del Oro Gold Plant Detail Engineering - Mexico
  - Cyprus Miami Smelter Modifications for Ancillaries - Arizona
  - Phelps Dodge Chino Smelter Upgrade including Uptake Shaft - Arizona
  - Cyprus Miami Smelter Casting Furnace Upgrade - Arizona
  - Phelps Dodge Morenci Material Handling and Structural Audit - Arizona
  - Geomaque Gold - Detail Engineering - Sonora, Mexico
  - Phelps Dodge Morenci Smelter Equipment Relocation - Arizona
  - Phelps Dodge Hidalgo Smelter Fugitive Gas Collection System - Arizona

CONRAD E. HUSS, P.E.
Project Manager

- M3 Engineering & Technology, (continued)
  - Magma San Manuel Smelter Anode Press Plant - Arizona
  - Phelps Dodge Ajo Smelter Demolition - Arizona
  - Penmont La Herradura Gold Plant - Mexico
  - Phelps Dodge Hidalgo Smelter Upgrade including Reaction Shaft - New Mexico
  - Cyprus Casa Grande Roaster Upgrades, Copper - Arizona
  - Zinc Corp Roaster Upgrade - Oklahoma
  - Cyprus Bagdad WaterFlush Crusher, Copper - Arizona
  - ASARCO Hayden Smelter Dust System - Arizona
  - Placer Dome Mulatos Gold Plant Basic Engineering -Mexico
  - Cyprus Bagdad 156,000 TPD Feasibility Study - Arizona
  - Cyprus Bagdad Feasibility Study for Inpit Crushing and Mill Expansion - Arizona
  - Hecla La Choya Gold Plant, Sonora, Mexico
  - Chemstar Lime Plants, Western United States
  - Majdanpek, Yugoslavia, Crush/Convey for Copper Mine
  - Phelps Dodge Chino SX-EW Expansion - New Mexico
  - Magma McCabe Gold Plant Expansion - Arizona
  - Phelps Dodge Morenci Flotation Expansion, Copper - Arizona
  - Phelps Dodge Chino Waterflush Crusher, Copper - New Mexico
  - Granite Sand & Gravel Plant - Arizona
  - Kerr-McGee Manganese Dioxide Chemical Plant - Nevada
  - Cyprus Sierrita Acid Plant (Rhenium Recovery) - Arizona
  - Phelps Dodge Chino Conveyor System Rebuild - New Mexico
  - Molycorp Mountain Pass Crush/Convey System for Rare Earths - California
  - Cyprus Esperanza/Twin Buttes Cross Country Conveyor Upgrade - Arizona
  - Mt. Graham Utilities and Tankage - Arizona
  - Old Tucson Utility Inventory and Upgrade - Arizona
  - ASDM Utility Inventory and Upgrade - Arizona
  - Cyprus Twin Buttes Fuel Stations Demolition and Upgrade - Arizona
  - Cyprus Sierrita Fuel Station Demolition and Upgrade - Arizona
  - Cyprus Sierrita ADM Chemical Plant - Arizona
  - ASARCO Mission Mill Feed Upgrade, Copper - Arizona
  - Cyprus Miami Road and Bridge - Arizona
  - ASARCO Mission Dust Collection, 96,000 CFM - Arizona
  - Magma San Manuel No. 4 Head Frame Upgrade - Arizona
  - Cyprus Sierrita Ferro Moly Dust Collection - Arizona
  - St. Cloud Flotation Upgrade, Lead/Zinc/Silver - New Mexico
  - University of Arizona Optical Mirror Laboratory - Arizona
  - Mt. Graham SMT Telescope Facility - Arizona
  - Mt. Graham Observatory Site Programming, Utilities and Maintenance Building - Arizona

- M3 Engineering & Technology, (continued)
  - Phelps Dodge Chino Inpit Crush/Convey Study - New Mexico
  - Philippines Crush/Convey Study
  - Cyprus Bagdad Tankhouse Expansion, Copper
  - Phelps Dodge Morenci Inpit Crush/Convey Checking - Arizona
  - Cyprus Sierrita Inpit Crush/Convey - Arizona
  - AZANG Maintenance Hangar and Hush House - Arizona
  - Cyprus Sierrita Column Cell Expansion I & II, Copper/Moly - Arizona
  - Cyprus Sierrita Moly Roaster Feed Systems I & II - Arizona
  - Magma Pinto Valley #4 Tailing Dam Slurry Pump Station - Arizona
  - Ft. Huachuca General Instruction Building - Arizona
  - Mt. Bell Communication Centers - Arizona
  - Cyprus Sierrita Moly Packaging System Upgrade - Arizona

- RGA Engineering Corporation, Structural Engineer, V. President, Engineering Director (4 Years)

  - Coronado Post Office for USPS - Arizona
  - Amphitheater Elementary School and University of Arizona Science Building - Arizona
  - Tanque Verde and Campbell Avenue Street Lighting - Arizona
  - Reid Park Band Shell and Master Plan - Arizona
  - AZANG Engine Shop, General Purpose Shop, Hush House - Arizona
  - Northern Arizona University Information Center - Arizona
  - Davis-Monthan Combat Support Center - Arizona
  - Ft. Huachuca Communications Facilities - Arizona
  - University of Texas Submillimeter Telescope - Texas
  - Arizona-Sonora Desert Museum Mountain Habitat - Arizona
  - Ina Road Bridge, Tanque Verde Bridge, Clifton Bridge, I-17 AC/DC Bridge, Orange Grove Bridge
  - University of Arizona Submillimeter Telescope - Arizona
  - University Heights Shopping and Parking Complex - Arizona
  - La Paloma Resort Hotel and Office Complex - Arizona
  - Design of warehouses and greenhouses - Worldwide
  - Design of banks, apartments, office buildings - Arizona
  - Design of elephant enclosure - Arizona
  - Design of conveyor head frames and maintenance shops- Arizona
  - Design of schools, libraries, churches - Arizona
  - Project Engineer for conversion of U.S.P.S. power system in Phoenix, Az with 2-350 ton chillers
  - Analysis for parking garage and pedestrian bridge - Arizona
  - Finite element earthquake analysis of ten-story office building - Arizona
  - Design of eight-story reinforced concrete hotel - Mexico
  - Converter Blower Electrification, Project Manager, Inspiration, Arizona

CONRAD E. HUSS, P.E.
Project Manager

- Mountain States Engineers, Vice President and Manager of Engineering (4 Years)
  - 52,000-65,000 TPD Mill Expansion, SPCC - Cuajone, Peru
  - Pennsylvania Fuels Group, Coal Gasification Plant Study
  - Gold Mill Expansion, Newmont, Carlin - Nevada
  - Shuichang Bethlehem International Crush/Convey, 10,000 TPH - China
  - Gold Heap Leach Study, Newmont, Telfer - Australia
  - Fly Ash Disposal System, Tucson Electric - Springerville, Arizona
  - Concentrate Loadout, Cyprus Bagdad - Arizona
  - Tailings System, Cyprus Bagdad - Arizona
  - No. 19 Dump Leach System, Inspiration - Arizona
  - 8000 TPH Crushing/Conveying of Waste, Kennecott Copper Corp. - Arizona
  - 150,000 TPD Crush/Convey, Kennecott - Ray, Arizona
  - 50,000 TPD Crushing/Conveying System, Island Copper - BC, Canada
  - 10,000 TPD Limestone Loadout, Grupo Cementos - Mexico
  - 40,000-54,000 TPD Mill Expansion, Cyprus Bagdad - Arizona
  - Smelter Coal Conversion, Phelps Dodge - Hidalgo, New Mexico
  - Modification of 3000 TPH Wash Plant, Carbon Coal - New Mexico
  - Moly By-Product Plant, Phelps Dodge - Ajo, Arizona
  - 50 TPD Zinc Skimmings Plant, National Zinc - Oklahoma
  - 4000 TPH Portable Crusher, Duval Corp. - Sierrita, Arizona
  - Sulfur Unloading Facility, Duval Corp. - Galveston, Texas

- Mountain States Engineers, Piping Department Head (1 Year)
  - Gulf & Western Sonora Gold - California
  - 1000 TPD Moly By-Product Plant, ASARCO, Mission - Arizona
  - 40,000 TPY Flotation Retrofit, ASARCO, Mission - Arizona
  - Tailings System, Plateau Resources - Utah
  - 2600 TPH Crush/Convey, Climax Molybdenum Co. - Colorado
  - 2000 TPD Moly By-Product Plant, La Caridad - Mexico
  - 6600 TPH Crushing/Conveying - Majdanpek, Yugoslavia
  - Coal Loadout Facility, CF&I, Maxwell Mine - Colorado
  - 12,000 TPD Uranium/Vanadium Plant, Cotter Corp. - Colorado
  - Lined Tailings Pond, Cotter Corp. - Colorado
  - Spent Catalyst Plant, Cotter Corp. - Colorado
  - 750 TPD Uranium Mill, Plateau Resources - Utah
  - Potash Plant Modifications, Duval Corp. - Carlsbad, New Mexico
  - Feasibility Study for Urangesellschaft, Site Determination and Environmental
  - Delamar Silver Plant CCD - Idaho

CONRAD E. HUSS, P.E.
Project Manager

- Mountain States Engineers, Structural Engineer and Department Head (5 Years)
  - Round Mountain, Nevada, Heap Leach Gold including recovery pad
  - Lime Plant, Nafinsa, Job Engineer - Santa Rita, Arizona
  - 250,000 TPD Crushing/Conveying, Job Engineer, Duval Corporation - Sierrita, Arizona
  - Ferro-Moly Plant, Job Engineer, Duval Corp. - Sierrita, Arizona
  - Special Investigations of Towers, Thickener and Frames
- Hughes Aircraft Company, Structural Engineer (½ Year)
  - Finite element analysis of missiles and vibration isolation of missile components
  - Strain gauge layout and destructive testing

- Teaching (6 Years)
  - Adjunct Lecturer, University of Arizona (2 Years)
  - Assistant Professor of Engineering, Northern Arizona University (1 Year)
  - Graduate Fellow in Engineering Mechanics, University of Arizona (4 Years)
  - High School Teacher, West Tampa Junior High School (½ Year)

- LTJG U.S. Coast and Geodetic Survey (3 Years)
  - 3rd in Command, USC & GSS Hydrographer
  - Hydrographical surveys in Hawaii, Alaska and Florida
  - Inspection of damage caused by Alaskan Good Friday Earthquake
  - Photogrammetry and land surveying in Alaska

COURSES

- Cold Regions Engineering Short Course
- Completed MSHA Training

PUBLICATIONS

HUSS, Conrad, and Dan Neff; "Horizontally Stiffened Angular Hoppers Analyzed by Beam Action Versus Finite Element", Bulk Solids Handling, June 1984.

HUSS, Conrad, and Nikita G. Reisler; "A Comparison of Handling Systems for Overburden of Coal Seams", Bulk Solids Handling, March 1984.

HUSS, Conrad, and Dan Neff; "Horizontally Stiffened Membrane Hoppers Analyzed by Virtual Work Versus Finite Element", Bulk Solids Handling, November 1983.

HUSS, Conrad, Nikita G. Reisler, and R. Mead Almond; "Practical and Economic Aspects of In-Pit Crushing Conveyor Systems", SME/ AIME, October 1983.

PUBLICATIONS Continued

HUSS, Conrad, and Dan Neff; "Finite Element Structural Analysis of Movable Crusher Supports", Bulk Solids Handling, March 1983.

HUSS, Conrad; "Cost Considerations for In-Pit Crushing/Conveying Systems", Bulk Solids Handling, December 1982.

ALMOND, R. Mead, and Conrad Huss; "Open-Pit Crushing and Conveying Systems", Engineering and Mining Journal, June 1982.

MUNSELL, Stephen, R. Mead Almond, and Conrad Huss; "The Trend Toward Belt Conveying of Ore and Waste in Arizona Open Pit Mines", SME/AIME, September 1978.

SANAN, Bal, and Conrad Huss; "Foundation Design for Rod and Ball Mills", ACI Conference 1977, Presentation Only.

HUSS, Conrad, and Ralph Richard; "Dynamic Earthquake Analysis of Tucson Federal Office Building", GSA Contract 70-6-02-0058, May 1972.

HUSS, Conrad; "Axisymmetric Shells Under Arbitrary Loading", The University of Arizona, 1970. Doctoral Thesis.

HUSS, Conrad; "Airy's Function by a Modified Trefftz's Procedure" The University of Arizona, 1968. Master's Thesis.

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WLR Consulting, Inc.

Resume of

William L. Rose, P.E.

EXPERIENCE:

2001 – Present	Principal Mining Engineer – WLR Consulting, Inc. Owner of a consulting firm specializing in ore deposit modeling, reserve estimation, and all aspects of open pit mine planning (including feasibility-grade project evaluations, equipment and manpower requirements, and cost estimation). Fluent in Mintec's MEDS® and MineSight® software.
1990 – 2001	Principal Mining Engineer – Mine Reserves Associates, Inc. Conducted numerous prefeasibility- and feasibility-grade evaluations of open pit mining projects throughout the world. Assisted clients with long- and short-range mine planning, including on-site assignments. Developed custom software and procedures for client reserve and royalty reporting requirements. Co-owner of MRA.
1989 – 1990	General Manager – Brewer Gold Company. Managed a 40,000 oz/year heap leach gold mine in the southeast U.S. with 122 employees. Operations included 16,500 tpd open pit mine, 4,500 tpd crusher/ agglomerator system, and a five-stage carbon adsorption circuit. Orchestrated permitting and construction of leach pad facilities and water treatment systems for NPDES-regulated discharges in a high rainfall environment.
1987 – 1989	Mine Superintendent – Brewer Gold Company (a subsidiary of Westmont Mining). Directed 43 employees in mine operations, mine engineering and mobile equipment maintenance. Purchased all mobile equipment, hired work force, initiated pit operations and set up maintenance facilities during project startup of an open pit gold mine near Jefferson, SC. Developed an efficient cost accounting system for operations.
1985 – 1987	Senior Mining Engineer – Westmont Mining Inc. Performed mine planning and permitting work for an open pit talc mine (Montana Talc near Ennis, MT). Conducted project financial evaluations and procured mine equipment fleet during project startup.
1980 – 1985	Senior Mine Planning Engineer – Pincock, Allen & Holt, Inc. Duties included project management; ore reserve estimation; mine planning; equipment selection; manpower, capital and operating cost estimation; and financial analyses. Developed computer applications for mine planning, including long-range production scheduling software.
1978 – 1980	Mining Engineer – U.S. Borax & Chemical Corporation. Performed equipment procurement evaluations, cost estimation, mine planning and feasibility studies for an open pit sodium borate operation (Boron, CA).
1977 – 1978	Assistant Mining Engineer – Atlantic Richfield Company. Duties included mine planning, permitting and quality control studies for an open pit coal mine (Black Thunder Mine near Gillette, WY).

EDUCATION: B.S. Mining Engineering, Colorado School of Mines - 1977.

AFFILIATIONS: Registered Member of the Society for Mining, Metallurgy, and Exploration, Inc. (No. 2762350RM)

REGISTRATION: Professional Engineer in Arizona (No. 15055) and Colorado (No. 19296).

SOFTWARE SKILLS:

MineSight® and MEDSystem® software from Mintec (deposit modeling, reserve estimation and mine planning)
Microsoft Office XP Professional
AutoCAD (through release 2004)
Surfer from Golden Software (surface modeling)
Mathcad 13
FORTRAN

PAST PROJECTS: (partial listing)

Rosemont	-	Augusta Resource Corporation
El Galeno	-	Northern Peru Copper Corp.
Bingham Canyon Mine	-	Kennecott Utah Copper
San Cristóbal	-	Apex Silver Mines
Hycroft and Briggs	-	Canyon Resources Corporation
Cerro Corona	-	Gold Fields
Paredones Amarillos	-	Vista Gold
Permanente Quarry	-	Hanson Permanente Cement
Cerro San Pedro	-	Metallica Resources
El Sauzal	-	Francisco Gold
Amayapampa	-	Vista Gold
Miami Mine	-	Phelps Dodge Miami
Crown Jewel	-	Crown Resources
Chapada Project	-	Echo Bay
Cerro Verde	-	Cyprus
Lihir	-	Kennecott

ROBERT (Bob) FONG, P.Eng.

email: fongrh@telus.net

Phone: 403-860-7113

39 Schiller Cr. NW
Calgary, Alberta, Canada
T3L 1W7

EDUCATION

Bachelor of Engineering - Mining, McGill University (1979), Montreal, Quebec, Canada

REGISTRATION

Association of Professional Engineers and Geoscientists, Alberta (APPEGA)and Geoscientists, Alberta (APPEGA)

EXPERIENCE

A professional mining engineer with over 27 years of experience in operations, management, and consulting. Has undertaken numerous studies on various mining projects world wide. They have included open pit mine designs, mine planning, development of mine costs, production scheduling, project evaluations, reserves estimates, Qualified Person's audits and reviews.

PROJECT EXPERIENCE

Principal Mining Engineering Consultant

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- Study manager on mine planning study for Taseko Resources' Gibraltar mine in central British Columbia;
- Preliminary assessment study on TTM Resources' Chu molybdenum property in central British Columbia;
- Technical study for Hard Creek Nickel's Tournagain nickel property in northern British Columbia;
- Preliminary assessment study on the Sharihada coal property in Mongolia for Canadian Sinosun Energy Corp;
- Technical study on open pit copper project in Ecuador for Corriente Resources;
- Preliminary assessment study on the Sharihada coal property in Mongolia for Canadian Sinosun Energy Corp;
- Qualified Person's review on Shell's Muskeg River Mine and Jackpine Oil Sands project in Fort McMurray, Alberta;
- Qualified Person's review on Petro-Canada's Ft Hills project in Fort McMurray, Alberta;
- Mine development plan for Novagold's Galore Creek Mine Feasibility project located in northern B.C.;
- Design and planning engineer on the EIA mine application for Deer Creek Energy on the Joslyn

North Mine Oil Sands project in Fort McMurray, Alberta;

- Mining Lead in preparation of the EIA mining application for permit approval on Imperial Oil's Kearl Oil Sands project;
- Pre-strip mining study for Kemess North project in northern B.C.;
- Qualified Person's review of mine capital and operating costs on Northern Lights Oil Sands Project for Synenco Energy;
- Preliminary mining studies on early stages of the Kearl Oil Sands project for Imperial Oil;
- Mining audit of an iron ore mine in Brazil;
- Scoping level study for a gold mining project for Novagold;
- Engineering support for Suncor's Millennium and Voyageur oil sands mine expansion projects;
- Preliminary oil sands mining study for PanCanadian Energy on Lease OS9;
- Oil sands screening level mining study for ExxonMobil;
- Audit of mine capital and operating costs for Shell's Muskeg River Mine in Fort McMurray, Alberta;
- Design, planning and costing for Suncor's Millennium Expansion project in Fort McMurray, Alberta.

1995 to 2000 – Consultant Engineer with H.A. Simons Ltd.

Employed as a consultant engineer by H.A. Simons Ltd. (later acquired by AGRA, then AMEC), where he was a member of their mining division – MRDI. His foreign working experience includes projects in Brazil, Peru, Africa, Philipines, Mexico, as well as numerous locations in the United States.

1993 to 1994 – Management and Ownership of Byron Creek Collieries

Was principally involved in the management team buyout of the Byron Creek Collieries coal mine from Esso Resources, where he subsequently assumed the position of Vice President of Engineering and Development. This mine operation was later sold to Fording Coal Ltd in 1994, and was renamed Coal Mountain Operations.

1980 to 1992 – Mine Engineer in Operations

Spent thirteen years in coal mine operations in southeastern British Columbia working in various capacities for Fording Coal, and Esso Resources.

THOMAS L. DRIELICK, P.E.
Principal Metallurgist

EDUCATION M.B.A., Southern Illinois University
B.S., Engineering, Michigan Technological University

REGISTRATION Engineer - Arizona

EXPERIENCE Over thirty-six (36) years of professional management and engineering experience in plant operations, project management, budget control, quality/schedule control, bid evaluations, planning, design development, process flowsheets, and project evaluations. Experience has been international with projects in the U.S.A., Canada, Mexico, Central America, Argentina, Chile, Peru and Australia. Over 100 computerized simulations of flowsheets for ore processing (precious and base metals), chemical plants and water treatment, O&M estimates for reclamation closure/closeout projects.

PROJECT EXPERIENCE

- M3 Engineering & Technology, Project Manager and Metallurgist (18 Years)

  – Pan American Silver, Manatial Espejo, Mexico
  – Pan American Silver, Alamo Dorado, Mexico
  – Alamos Mulatos Gold, Sonora, Mexico
  – Goldcorp Peñasquito Flowsheets, Zacatecas, Mexico
  – Minefinders Dolores Gold Plant, Chihuahua, Mexico
  – AVESTOR Battery Plant, Nevada
  – APC RIMOD 3 Modernization of Cement Plant, Arizona
  – ASARCO Mission Crushing Plant Upgrade, Arizona
  – Kerr-McGee Manganese Dioxide Chemical Plant, Nevada
  – Kerr McGee Physical Separation Facility, Nevada
  – Kerr McGee West Chicago Remediation and Reclamation, Illinois
  – BHP Magma Anode Preparation Plant and Gold Concentrator, Arizona
  – BHP Magma Miami Unit SX-EW Organic Recovery, Arizona
  – BHP Magma San Manuel Selenium Recovery Circuit, Arizona
  – Battle Mountain Gold Crown Jewel Project, Washington
  – Chemical Lime Apex Quicklime Handling, Nevada
  – ASARCO Ray SAG Mill Bypass Study, Arizona
  – ASARCO Ray Secondary Crushing Project, Arizona
  – ASARCO Ray Tankhouse Upgrade Project Chemstar Cosgrave Lime Crushing and Kiln, AZ
  – Chemstar Tenmile Pass Lime Crushing & Kiln, Soda Springs, Idaho
  – Coeur d'Alene Mines Corporation Boleo Project, Mexico
  – Cyprus Bagdad Expansion Studies, Arizona
  – Cyprus Bagdad Mineral Park SX-EW, Arizona

- M3 Engineering & Technology (continued)

  - Cyprus Bagdad 3000 Cu. Ft. Rougher Expansion, Arizona
  - Cyprus Bagdad Tonopah Mills and Cells, Arizona
  - BHP Magma Pinto Valley No. 4 Tailing Reclaim Water System, Arizona
  - BHP Magma Pinto Valley Tailing Pump Station, Arizona
  - BHP Magma San Manuel Smelter Flue Dust Leach and SX Plant, Arizona
  - Majdanpek Flotation Mill Expansion/Modernization, Yugoslavia
  - Cyprus Bagdad "WaterFlush" Crushing Plant, Arizona
  - Cyprus Casa Grande Silver Leaching, Arizona
  - Cyprus Cerro Verde Crush/Convey Upgrade Project, Peru
  - Cyprus Tohono Oxide Ore Process Plant, Arizona
  - Cyprus Tonopah SX-EW Study, Nevada
  - Cyprus Sierrita Copper Larox Pressure Filter Installation, Arizona
  - Cyprus Sierrita Moly Expansion Support, Arizona
  - Cyprus Sierrita Molybdenum Chemicals ADM Plant, Arizona
  - Cyprus Sierrita Roll Crusher Plant Addition, Arizona
  - Cyprus Sierrita Rhenium Plant Expansion, Arizona
  - Echo Bay Gold Aquarius Project including water system, Canada
  - Echo Bay McCoy Gold Crushing/Grinding, Nevada
  - Echo Bay Paredones Amarillos Project including water system, Mexico
  - Francisco Gold El Sauzal Project, Mexico
  - Geomaque San Francisco Gold Project, Mexico
  - Golden Queen Mining Soledad Mountain Project, California
  - Granite Swan Road Sand and Gravel Plant, Arizona
  - Griffin Copper Bale Leach, Arizona
  - Griffin Copper Plant Expansion, Arizona
  - Grupo Mexico Cananea Raffinate Neutralization Project, Mexico
  - Hecla KT Clay Monterrey Expansion Project, Mexico
  - Hecla La Choya Gold Heap Leach, Mexico
  - Hecla Lucky Friday Mill Expansion, Idaho
  - Hecla Noche Buena Gold Plant Study, Mexico
  - Hecla Rosebud Gold Project including reverse osmosis water treatment, Nevada
  - Kennecott Carmen Feasibility Project, Alaska
  - Kennecott Sweetwater Uranium Water Recycle Project, Wyoming
  - Kennecott Greens Creek Recommission Project including three new chemical lime water treatment projects facilities, Alaska
  - Kennecott Greens Creek Pyrite Circuit Study for Reclamation, Alaska
  - Kennecott Greens Creek Mill Enhancements Study, Alaska
  - Kennecott Greens Creek Cleaner Flotation Projects, Alaska

- M3 Engineering & Technology (continued)

  - Kennecott Utah Lime Plant, Utah
  - Liximin/Golden News Luz del Cobre, Mexico
  - Maricunga Refugio Gold Operation Consulting, Chile
  - Minera Alumbrera Prefeasibility Study, Gold/Copper, Argentina
  - Minera Alumbrera Grinding Line No. 3 Project, Argentina
  - Minera Alumbrera Filter Plant, Argentina
  - Minera Alumbrera Crusher/Mill Upgrades, Argentina
  - Minera Alumbrera Mill Expansion Study, Argentina
  - Minera Las Cuevas Fluorspor Calcination and Leaching, Mexico
  - Minera Penmont La Herradura Gold Project, Mexico
  - Molycorp Mt. Pass Rare Earth Minerals Crushing Plant, California
  - Morgain Minera MGM Trona Study, Mexico
  - Newmont Gold Company Gold Mine Dewatering, Nevada
  - Phelps Dodge New Mexico Properties Reclamation Plan, New Mexico
  - Peñoles Fco. I Madero Project, Mexico
  - Peñoles Fresnillo Grinding Expansion Project, Mexico
  - Phelps Dodge Standard Low Cost SX-EW Study, Worldwide
  - Phelps Dodge Arizona Properties Reclamation Plans (7), Arizona
  - Phelps Dodge Ajo Concentrator Project, Arizona
  - Phelps Dodge Chino Waterflush Crusher, New Mexico
  - Phelps Dodge Morenci Coronado Lead Project, Arizona
  - Phelps Dodge Morenci Metcalf 82,000 TPD Expansion Study, Arizona
  - Phelps Dodge Morenci Fine Grind Expansion Study, Arizona
  - Phelps Dodge Morenci Flotation Expansion, Arizona
  - Phelps Dodge Morenci Secondary Crushing Study, Arizona
  - Phelps Dodge New Mexico Reclamation Plans (3) including water treatment, New Mexico
  - Phelps Dodge Tyrone SXEW Raffinate Tank, New Mexico
  - Pinal Creek Group EPCM, Arizona
  - Pinal Creek Group Water Treatment Trade-Off Study, Arizona
  - Pinal Creek Group Water Treatment Pilot Plant, Arizona
  - Placer Dome Cortez In-pit Sizing and Conveying Study, Nevada
  - Placer Dome Mulatos Gold Project, Mexico
  - Questa Water Treatment Study
  - Teck Cominco Morelos Camp Gold Plant Feasibility - Mexico
  - Tucson Electric Power Springerville Lime Plant Study, Arizona
  - Zinc Corporation of America Zinc Sulfide Leach Plant Upgrade, Oklahoma

THOMAS L. DRIELICK, P.E.
Principal Metallurgist

- Newmont Mining Corporation, Project Manager and Project Engineer (8 Years)

  - Idarado Mining Company: Copper tailing pond dust suppression
  - Limestone preparation plant evaluation
  - Magma Copper Company: Copper concentrate drying evaluation
  - Magma Copper Company: Copper slag concentrator project evaluation
  - NHPL Telfer; Australia: Gold plant expansion evaluation
  - NMC, Uchuchaqua: Silver plant expansion evaluation
  - NML, Similkameen: Copper plant expansion evaluation
  - NML, Similkameen: Copper tailing disposal modifications
  - New Celebration Gold Mine, Australia: Gold plant expansion project
  - Newmont Gold Company, Gold Quarry: Gold dump leach crushing plant evaluation
  - Newmont Gold Company, Gold Quarry: Gold dump leach for 3 MM TYP
  - Newmont Gold Company, Maggie Creek: Gold leach solution heating evaluation
  - Newmont Gold Company, North Area: Gold heap leach and carbon in pulp
  - Newmont Gold Company, No. 1 Mill: Gold plant expansion
  - Newmont Gold Company, No. 2 Mill: Gold milling facility for 7,000 TPD
  - Newmont Gold Company, No. 2 Mill: Gold plant expansion evaluation
  - Newmont Gold Company, Rain: Gold project evaluation

- Kennecott Corporation, Process Engineer, Plant Metallurgical Engineer, Operations Foreman (7 Years)

  - Chino Division: Copper, Molybdenite milling facility for 37,000 TPD
  - Chino Division: Copper solvent extraction electrowinning process development
  - Nevada Division: Copper tailing recovery and re-treatment evaluation
  - Ray Division: Copper solvent extraction project evaluation
  - Tintic Division: Lead, Zinc - Responsible for the concentrator metallurgical performance, production budget, quality standards, and development of process treatment methods
  - Utah Division: Copper, Molybdenite concentrator metallurgist responsibilities included commissioning of new process facilities, training plant operators, solving production problems
  - Copper, Molybdenite modernization project development/evaluation
  - Utah Copper Division: Molybdenite plant project development/evaluation
  - Utah Division: Copper, Molybdenite shift supervisor of crushing, grinding & flotation circuits

- U.S. Army, Metallurgical Engineer (3 Years)
  - Frankford Arsenal, Metallurgical engineering research programs in the area of metal fragmentation and liquid metal embrittlement

COURSES
- Completed MSHA Training

John I. Ajie, P.E.

Director Western Operations

Education

M.S., Mining Engineering University of California, Berkeley

B.S., Mining Engineering New Mexico Institute of Mining and Technology

Experience Summary

- Licensed Professional Engineer – Texas, Montana and New Mexico
- Proven Leadership – Diverse management experience in operations, and project engineering, evaluation and estimating.
- Project Management - Demonstrated ability to manage several coal mines with a record of meeting or exceeding project financial goals, facing numerous problems and developing effective solutions.
- Adaptable – Proven ability to adapt to changing business and work conditions.
- Strong People Skills – Proven success in getting the most out of employees and work teams.

Work Experience

2003-present **Director**

Western Operations
Washington Group International, Inc.(formerly Morrison Knudsen)—Denver, Colorado
Oversee the existing mining operations to ensure safety and operational performance. Assist in the startup of new operations. Direct and coordinate the, due diligence, engineering, evaluation and estimating of large international and domestic projects being considered for acquisition or operation.

2001-2002 **General Manager**

Powder River Operations
Washington Group International, Inc.(formerly Morrison Knudsen)—Hardin, Montana
Manage all costs, production, maintenance, reclamation, safety, purchasing/warehousing and administrative personnel activities at the 6 million tpy Sarpy Creek coal mine. The mine uses a 115-cy dragline and employed 100 people. Also, managed the ongoing reclamation and mine support work at several coal mines in Wyoming. The Wyoming operations employed 62 people.

1998-2000 **Design Engineering Manager**
Washington Group International, Inc.(formerly Morrison Knudsen)—Boise, Idaho
Directed and coordinated the engineering, evaluation and estimating of several large international and domestic projects being considered for acquisition or operation. Prepared of all necessary reports and technical documentation. Provided operations support and audits at existing mining operations.

1997-98 **General Manager**
Atascosa Mining Company (a subsidiary of Morrison Knudsen Corporation)—Jourdanton, Texas
Managed all costs, production, reclamation, maintenance, safety, purchasing/warehousing and administrative activities at the 3.2-million tpy lignite strip mine that used two draglines and employed 140 people. Accomplishments include considerably exceeding mine revenue and profit projections as well as exceeding mine production targets.

1996 **Principal Engineer**
Morrison Knudsen Corporation—San Antonio, Texas
Coordinated the engineering, estimating, bid and report preparation on many mining projects. Projects ranged in size from one (1) million to 30 million dollars per year in annual revenue. Performed due diligence at several international coal mining operations.

Navasota Mining Company
(a subsidiary of Morrison Knudsen Corporation)—Carlos, Texas

1995-96 **Mine Manager**
Responsible for all costs, production, reclamation, maintenance, safety, purchasing/warehousing and administrative personnel activities at the 3.5-million tpy lignite strip mine. The mine used two draglines and employed 150 people. Accomplishments include significantly exceeding mine production targets which resulted in manpower reduction and lower mining cost.

1995 **Mining Operation Engineering Manager**
Supervised the Mine Engineering Department at this project. Directed all mine planning, mine sequencing, production forecasting, and cost and productivity analyses. Prepared the annual mine budget and performed all equipment replacement analysis. Was involved with the mine environmental and safety activities.

1991-95 **Staff Mining Operations Engineer**
Planned and coordinated all mine operation activities. This included over five million cubic yards per year pre-strip removal and placement, two 75-cubic yard dragline operations, lignite load and haul, and reclamation activities Prepared short- and long-term mine planning and reviewed engineering designs

prior to construction. Also prepared the annual mine operating cost budget and performed all equipment replacement analysis.

1986-91 **Senior Mining Operations Engineer**

Prepared detailed annual and five-year mine plans. Managed the daily operations and performed any engineering required at the one-million cubic yard per year Ash Landfill. Located and designed mine highwall service roads, lignite haul roads, and dragline move roads, and calculated monthly coal volumes per land tract. Also supervised several construction projects and prepared the annual mine operating cost budget.

1984-86 **Mining Operations Engineer**

Morrison Knudsen Corporation—Hardin, Montana

Monitored and evaluated all mine drills and the draglines at the Sarpy Creek coal mine; performed all short- and long-term mine planning; and assisted in the preparation of the annual mine budget. Was also responsible for the quality of coal mined and shipped; calculated the required blend to ensure that shipped coal met contract specifications; and directed coal sampling. Developed engineering reports.

1981-84 **Mining Engineer**

Morrison Knudsen corporation—San Antonio, Texas

Performed mine sequencing, engineering and estimating, and assisted in the design of 17 ponds and 46 diversions for the 6-million tpy Cummins Creek Mine in Texas. Also sited stockpiles, truck dumps and facilities; designed the stockpiles, haul roads and ramps; and evaluated and selected coal loading and hauling equipment for 3-, 5-, and 9-million tpy lignite mines in Louisiana and Texas. Designed and prepared cost estimates for the surface-water-control structures of the Powell Bend Mine.

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