Washington Group International

Integrated Engineering, Construction, and Management Solutions

Preliminary Assessment and Economic Evaluation for the Rosemont Deposit Pima County, Arizona, USA

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Prepared for:

Augusta Resource Corporation
4500 Cherry Creek South Drive, Suite 1040
Denver, Colorado 80246

Prepared by:

Washington Group International
7800 E Union Avenue
Denver, Colorado 80237
Tel: (303) –843-2000

Principal Author and Qualified Person: John Ajie, P.E.

Other Contributors:

William L. Rose – WLR Consulting, Inc.

3. SUMMARY

Washington Group International (Washington Group) was retained by Augusta Resource Corporation (Augusta) to prepare a preliminary assessment and economic evaluation (PAEE) of the Rosemont deposit. The property is located in Pima County, Arizona, USA approximately 30 miles southeast of the city of Tucson. The deposit consists of copper-molybdenum skarn deposit and will be mined using a conventional truck/shovel open pit operation. The ore will be processed using conventional crushing/grinding/flotation circuit to produce a copper concentrate product from the sulfide ore. This is considered the base case condition (i.e. oxide recovery is not considered).

To develop this study, Washington Group was dependent on the opinions and information from other experts outside of its corporate structure consisting of:

- Mr. William Rose - WLR Consulting, Inc.: for information on the geology, deposit model and mineral resource estimates.
- Mr. James Sturgess – Augusta Resource Corporation: for information on property descriptions, land ownership, status of patented and unpatented claims and fee lands, historic reports and data, status of permitting and environmental compliance issues.
- Mr. Dave Logue – Stantec, Inc.: for information on the water supply system.
- Mr. Dave Larsen – Navigant Consultants: for information on the power supply system.

This study was based on the deposit model and mineral resource estimates presented in a previous technical report[1] prepared by WLR Consulting, Inc. (*Mineral Resources Estimate, Revised Technical Report for the Rosemont Deposit, Pima County, Arizona, USA* dated April 21, 2006).

Similar to other southwestern USA deposits in this class, the Rosemont deposit consists of large-scale skarn mineralization developed in Palaeozoic-aged carbonate sedimentary rocks around their contact with quartz-latite or quartz-monzonite porphyry intrusive rocks. The estimate of measured and indicated mineral resources above a 0.20% Cu cutoff totals about 442 million tons grading 0.51% Cu and 0.015% Mo, which has not changed from WLR Consulting's (WLRC's) April 21, 2006 technical report. The estimate of inferred mineral resources above a 0.20% Cu cutoff also remains unchanged at 145 million tons grading 0.45% Cu and 0.015% Mo.

WLRC conducted floating cone evaluations on the potential ore at US$1.05/lb Cu and US$7.50/lb Mo to determine the economic pit limits and pushback sequencing. Six conceptual mining phases were generated for the subsequent development of a scoping-level mine production schedule for a sulfide ore milling rate of 75,000 tons per day (tpd). Including inferred mineral resources, the mine production schedule totals about 432 million tons of potential sulfide ore grading 0.47% Cu and 0.015% Mo, 59 million tons of potential oxide ore grading 0.20% cu, and about 759 million tons of waste rock. This potential ore contains just over four billion pounds of copper and 130 million pounds of molybdenum. The base case stripping ratio is projected at 1.9:1 (tons waste plus tons oxide ore per ton of sulfide ore). Internal NSR cutoffs used in the production schedule are US$4.00/ton for sulfide ore.

The above estimates are for an economic assessment that is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the *economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that this preliminary assessment will be realized.*

Peak material handling rates will occur in Years 1-4, averaging about 245,000 tpd of total material before falling off to nearly 225,000 tpd for most of the remaining years. The first six phases of mining are currently projected at 16 years of operation. Major mining equipment will include 12.25-in.-diameter rotary blasthole drills, 50- to 60-cu-yd electric rope shovels and 250- to 355-ton off-highway haulage trucks. A fleet of appropriately-sized auxiliary equipment will also be required to support the proposed mining operations.

Based on recent metallurgical testing and experience at the nearby Mission Mine, the flow sheet for processing 75,000 tpd of sulfide ore includes: a 60" x 110" gyratory primary crusher, two 36.5 ft x 19 ft EGL Semi-Autogenous Grinding (SAG) mills, two 22-ft-diameter by 36.5 ft ball mills in parallel, eleven 4500-cu-ft rougher flotation cells, and a cleaner circuit consisting of two 10.8-ft-diameter by 38-ft-tall column cells and eight 1000-cu-ft scavenger cells. Rougher and scavenger flotation tailings will be combined in a thickener for water recovery and further dewatered by belt filters before being conveyed to the waste rock storage area (dry tailings disposal). An alternative of traditional wet tailings storage in earth impoundments is also being considered.

It is estimated that the process will recover approximately 89% of the Cu at a concentrate grade of 33% and 63% of the Mo at a concentrate grade of 56%.

Based on the input from third party consultants, both the water and power sources are available to support the project.

Washington Group estimates that mine manpower requirements will average 43 people in supervision and technical services and 184 employees in operations, maintenance and warehousing. Average manpower required for the concentrator facility is 86 people, including nine salaried and 76 hourly employees. Project administration personnel requirements are estimated at 35 people. Total direct employment for the project will average 348 people.

The base case overall total capital cost for the project was estimated by Washington Group and other engineering/consulting companies at US$725 million, which consists of:

Description	**Costs, $US M**
Direct Costs	
Water Supply System	21
Power Supply System	18
Concentrator and Process Facility	359
Mining Equipment Capital and shop facilities	156
Subtotal Directs	**554**
Contingency @ 15%	**83**
Indirect Costs	
Working Capital, Spare Parts and First Fills	45
Reclamation Guarantee and Bonding	16
Subtotal Indirects	**61**
Sustaining Capital	**28**
TOTAL CAPITAL COSTS	**725**

Consistent with a scoping-level study, the accuracy of the cost estimates (capital and operating) should be considered ±40%.

Washington Group's estimates of direct operating costs not including depreciation, reclamation, taxes or royalties for the Rosemont project are summarized below:

Mining	US$ 0.74 / ton of material
Processing	US$ 2.51 / ton of ore
General/administration	US$ 0.23 / ton of ore

The base case financial model developed for this preliminary assessment, using US$1.20/lb Cu, US$10.00/lb Mo and US$7.50/oz Ag prices, indicates an after-tax cumulative profit of almost US$1.5 billion over the project's currently projected life of 16 years. At a discount rate of 8% per annum, the project's net present value is estimated at US$442 million. An internal rate of return (IRR) is estimated at 17% (from Table 25.15). The project's net cash costs after by-product credits are estimated to average US$0.42/lb Cu.

Operating cost sensitivity studies at ±10% yield IRRs of 16 to 19% respectively, and capital sensitivities at ±10% exhibit similar IRR ranges. Both cost sensitivities were conducted at the base case metals prices presented above.

Washington Group recommends to proceed with the next phase of the study, either a prefeasibility or final feasibility to further refine engineering and cost projections and advance the project to the next stage of decision making and potential development.

4. INTRODUCTION AND TERMS OF REFERENCE

Washington Group was retained by Augusta to prepare an independent preliminary assessment of the economic potential of the Rosemont deposit located near Tucson, Arizona. The estimates of mineral resources have not changed since the publication of the previous technical report[1] (*Mineral Resources Estimate, Revised Technical Report for the Rosemont Deposit, Pima County, Arizona, USA* dated April 21, 2006 and prepared by WLR Consulting, Inc.). New work represented in this preliminary assessment includes: estimating a mine production schedule (including inferred mineral resources); performing additional metallurgical testing; developing an initial process flowsheet; determining mine and processing equipment and manpower requirements to fulfill the production schedule; estimating capital and operating costs for the project; and performing project economic analyses, including cash flow estimates.

The preliminary assessment presented herein is a scoping-level study that is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that this preliminary assessment will be realized.

Mr. William Rose of WLR Consulting, Inc. conducted floating cone evaluations of potential economic pit limits, developed conceptual mining phases, estimated potential ore and waste quantities, generated a mine production schedule for a sulfide ore milling rate of 75,000 tons per day, and estimated haulage profiles for Washington Group's subsequent estimate of equipment and manpower requirements. Mr. Rose meets the standards for an independent, qualified person as set forth in Canadian NI 43-101.

This PAEE has been prepared under the direction of Mr. John Ajie of Washington Group who visited the site on May 18, 2006. Mr. Ajie evaluated the mine equipment requirements, developed equipment productivities, mine equipment capital and mine operating costs based on the production schedule provided by WLR Consulting. Mr. Ajie meets the standards for an independent, qualified person as set forth in Canadian NI 43-101. Mr. Donald Podobnik, an experienced metallurgist working for Washington Group prepared the process flowsheet, estimated process capital and operating costs. Mr. Paul Carey of Washington Group performed the economic evaluations. Mr. James Sturgess, Augusta Resource Vice President of Projects and Environment addressed the environmental and permitting issues. Vector Engineering provided a preliminary design and cost for tailing management. Navigant Consultants provided a preliminary evaluation and cost estimate for the power supply system. Stantec provided a preliminary evaluation and cost for the water supply system. Skyline Lab of Tucson Arizona and Mountain States provided metallurgical analyses.

Augusta has completed all option payments, totaling US$20.8 million, to acquire a 100% interest in the Rosemont property. A 3% NSR royalty is still in effect. Section 6 has been modified from the previous technical report[1] to reflect this change and to expand the discussion of environmental aspects and considerations in the project's development.

Sections 7 through 11 have been omitted as there have been no changes from the previous technical report[1]. Sections 12 through 17 and subsections 19.1 through 19.11 also have not changed, but are reproduced in this preliminary assessment for the sake of completeness and in compliance with NI 43-101 requirements.

The bulk of the new information presented in this report is presented in Sections 18, 19.12, 20, 25, 26 and 27.

Unless otherwise specified, all units of measurement in this report are Imperial and all costs and/or prices are expressed in United States dollars. Tons refer to short tons (2000 pounds). Company abbreviations include:

Anaconda	Anaconda Mining Company
Anamax	Anamax Mining Company
ASARCO	American Smelting and Refining Company
Augusta (Client)	August Resource Corporation (the Client)
Banner	Banner Mining Company
EIS	Environmental Impact Statement
MSRD	Mountain States Research and Development
NED	National Elevation Dataset
NRRI	Natural Resource Research Institute
PAH	Pincock, Allen & Holt, Inc.
ROD	Record of Decision
RSD	Relative Standard Deviation
SD	Standard Deviation
Skyline	Skyline Labs Inc.
SRMs	Standard Reference Materials
Stantec	Stantec Inc.
Washington Group	Washington Group International
Winters	The Winters Company
WLRC	WLR Consulting, Inc.

Other commonly used acronyms and abbreviations include:

AAS	Atomic Absorption Spectrometry
ADEQ	Arizona Department of Environmental Quality
ADWR	Arizona Department of Water Resources
ADOT	Arizona Department of Transportation
Ag	silver
AGS	Arizona Geological Survey
Au	gold
Cu	copper
CuEqv	Copper Equivalent
FC	Floating Cone
ft	foot or feet
ft^3	cubic feet
g	grams
g/t	grams per ton (metric)
ICP	Inductively Coupled Plasma
IRR	Internal rate of return
kg	kilogram
km	kilometer or kilometers
ktons	tons x 1000
kWh	kilowatt hour
lb	pound
lbs	pounds
m	meter or meters
Mo	molybdenum
NED	National Elevation Dataset
NSR	Net Smelter Return
OES	Optical Emission Spectrometry
oz	troy ounce
oz/ton	troy ounces per ton
PAEE	Preliminary Assessment Economic Evaluation
ppm	parts per million
ton	short ton (2000 lbs)
ton	metric ton (1000 kg or 2204.6 pounds)
USGS	U. S. Geological Survey
XRF	x-ray fluorescence

5. RELIANCE ON OTHER EXPERTS

Washington Group has relied on the opinions of Augusta personnel, Mr. James Sturgess, and documents from Augusta's due diligence evaluation (in relation to its April 2005 agreement to acquire the Rosemont Properties) regarding property descriptions and land ownership, the status of patented and unpatented claims and fee lands, historic reports and data and the status of permitting and environmental compliance issues.

Washington Group has also relied on the expertise of Mr. William Rose of WLR Consulting, Inc. for information on the geology, deposit model and mineral resource estimates.

Washington Group has also relied on the expertise of Mr. Donald M. Podobnik, a consulting metallurgist working for Washington Group International. Mr. Podobnik performed a review and summarization of prior metallurgical test work (Section 18), assisted in: the development of the process flowsheet (Section 20), selecting major process equipment, estimating process manpower requirements and estimating process capital and operating costs (Section 25). Mr.

Paul Carey an experienced mining professional working for Washington Group performed the economic analysis.

Washington Group has also relied on the expertise of Mr. Dave Logue of Stantec in determining the water supply system and of Mr. Dave Larsen of Navigant Consultants in determining the power supply system.

While Messrs. Podobnik, Carey, Logue and Larsen, have extensive experience in their respective fields, none meet all of the requirements to be considered a qualified person as defined in Canadian NI 43-101.

6. PROPERTY DESCRIPTION AND 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 in the American Southwest, approximately 30 miles (50 km) southeast of Tucson, Pima County, Arizona. In geographical terms, the Rosemont Property location coordinates are approximately 31º 50'N and 110º 45'W.

The core of the Rosemont Property consists of 132 patented lode claims that in total encompass an area of 1968.5 acres (797.2 hectares). A contiguous package of 850 unpatented lode mining claims with an aggregate area of approximately 12,000 acres (4,860 hectares) surrounds the core of patented claims. Associated with the property are 14 parcels of fee land grouped into six individual areas that enclose a total of 910.96 acres (368.7 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 14,880 acres 6,026 hectares).

The patented lode claims and fee land parcels have no expiration date and are subject to annual taxes amounting to several thousand US dollars in total. The unpatented lode claims also have no expiration and are maintained through the payment of annual fees of $150.00 (US) per claim. A 3% NSR royalty applies to the patented claims, the bulk of the unpatented claims, and some of the fee land. Augusta announced on June 1, 2005, that it had entered into an option agreement to acquire the entire property for a purchase price of $20.8M (US). On April 3, 2006, Augusta announced that it had fully exercised its option to purchase the property.

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.

As an advanced exploration development property, Rosemont is up to date and compliant with all its environmental obligations and as such there are no material environmental liabilities.

The Rosemont Ranch Lands were surveyed in 2003 for environmental liabilities as part of a land transaction. At that time, the environmental liabilities were characterized as minimal, and were determined to not be material to the land transaction. Specific issues reviewed included mine adits, shafts, exploration holes, and mine wastes from prior production. Areas of potential liability included a modest amount of annual monitoring and maintenance to repair or replace fencing and drainage around mine openings and residual waste products.

6.1 Environmental Aspects and Considerations in the Preliminary Assessment and Economic Evaluation of the Rosemont Project

6.1.1 Introduction

Environmental protection, mitigation, and management have been integrated as fundamental goals of the Rosemont Mine Project. These goals are seen by Augusta Resource Corporation as obligations in all operating plans being evaluated for Rosemont. Early determination of key environmental aspects was accomplished through review of all published and much of the unpublished materials related to the Rosemont Ranch. Interviews with regulatory agency representatives, screening for special resource values, and specific inquiry to community leaders, neighbors, and elected officials have been utilized to develop this outline of a progressive environmental program for the development of the Rosemont mineral resources.

6.1.2 Community Involvement

The local community will be kept informed throughout the planning process. During the course of the feasibility evaluation, public notice, and agency review, the public will be involved as the environmental aspects become more fully developed. This will insure that all environmental aspects are reviewed and publicly discussed. The detailed plan will be formally proposed with enforceable obligations and conditions during the course of project feasibility evaluation, public notice, and agency review. As specific federal, state, and local issues and requirements are negotiated permit by permit, the generalized program outlined below will develop into a detailed Environmental Management System (EMS) for Sustainable Mine Development (SMD) at Rosemont. This EMS/SMD will become a formal part of the detailed Plan of Operations submitted for impact evaluation through an Environmental Impact Statement as part of the review and approval process for the project.

6.1.3 Water Resource Plans for the Rosemont Mining Project

The Rosemont water supply is based on a sustainable and renewable source of makeup water. To insure reliability, the project will use water extracted from groundwater wells, with redundant systems to insure trouble free operation. To ensure sustainability, the project will utilize renewable surface water supplies to replace the project's consumptive use of well water, so that there is no net depletion to the groundwater basin.

Water Conservation. Employing state-of-the-art techniques in eliminating waste and evaporation, and by recycling all possible water within the project, Rosemont's consumptive use water demand projections are less than half of the water amounts used by nearby copper mines. This is primarily achieved by the elimination of tailings ponds and the use of dry tailings storage co-managed with waste rock.

Sustainable Water Supply. Rosemont will construct this project with the least environmental impact possible. To this end, Rosemont is actively acquiring replacement water supplies for all of the groundwater to be consumed. Most of this replacement water will come from the Colorado River, to be delivered through the Central Arizona Project to underground water storage facilities near the project's wells. Storage of water before mining production begins will insure that the water used in the project will not impact other current or future groundwater users within the basin.

Protection of Water Quality. Groundwater quality will be protected by isolation and containment of process waters at Rosemont. The use of primary and secondary containment structures, double liners in process impoundments, elevated tankage, overflow protection, and spill and leak detection systems is now commonplace for newly constructed industrial facilities. Common stormwater runoff will be managed in retention basins with capacity to insure water quality goals are maintained. Process water will be managed for zero discharge. Monitoring programs approved by agencies, monitoring stations, monitoring wells, and data recording devices will insure that designs use best available demonstrated control technology, plans are followed, and goals are met for all aspects of water quality conservation at Rosemont.

6.1.4 Concurrent Reclamation Plans for the Rosemont Mine

To insure protection of the production, aesthetics, and open space character of the Rosemont project site, advance planning has identified the goals for a post mining land use at the area: Dedicated open space for livestock ranching, recreation, and wildlife.

Committing to concurrent reclamation. Planning for mine closure and reclamation has been integrated into the design of the Rosemont Project. Mining plans include topsoil stripping and salvage for reuse in reclamation. Slope grading and contouring of waste rock are included in the mine plans. Other specific steps to expedite concurrent reclamation include design of a perimeter berm along the 5400 elevation waste rock storage pile to be constructed in the first three years of operations. This perimeter berm will be constructed of mine run rock placed in setback terraces to allow for grading to 3:1 interbench slopes, and approximating a 4:1 overall slope including the bench setbacks. The toe of the perimeter berm will be well inside the Barrel wash drainage, and allow for access and runoff controls to be confined to the one drainage area. These plans will be formalized in the Mine Reclamation Plan for the Rosemont Plan of Operations, which will be presented for approval to state and federal agencies prior to mine construction. Financial guarantees for full cost of mine land reclamation will be in place and dedicated solely to that purpose, prior to ground breaking.

Long-term open space goals. The use of progressive proven techniques for mine land reclamation using rangeland restoration and carefully controlled cattle grazing on reclaimed side slopes will bring fast recovery to this important buffer area. A similar visual barrier berm will also be constructed at key locations below the plant site, along the perimeters of the middle and lower Barrel waste rock areas. The goal of the concurrent reclamation program is to provide visual demonstration of the potential for mining to be conducted in a sustainable manner not inconsistent with ranching operations and other open space resource activities.

6.1.5 Dry Tailings Storage, Co-managed with Waste Rock

Conventional disposal of tailings, the material remaining after concentration of the useful minerals, involves pumping sand and water from the mill grinding lines in a slurry form to large settling ponds for separation of solids and liquids. The size, seepage potential, evaporation losses, and dusting potential make these conventional tailings ponds problematic during operation as well as post closure. Rosemont will co-manage waste rock and tailings to optimize benefits for both types of mine waste materials.

Reclamation is facilitated. The closure of a sloped, graded, contoured waste rock pile is considerably simpler than a large embankment of even sized sand. When considering the size of the Rosemont project, the integration of tailings and waste rock into a single co-managed structure is significant.

Recycle potential for water. At Rosemont, Augusta is committed to use water miser criteria for tailing management, filter drying the tailings prior to disposal. Using conventional belt filter technology, Rosemont will reduce the water content from 50-60% to less than 15%, a point at which it is damp sand. The recovered water is recycled in the process of grinding rock and separating the useful minerals from the host rock. The damp solids, at 10-15% moisture, can be managed in the same structure as the run of mine waste rock, with a few special considerations. The damp sands will be placed within rock-lined basin to protect against erosion from wind and rain. The sands will be placed and managed as needed to provide for stability within the rock storage facility and protection against dust entrainment. These considerations will be presented in the detailed management plans as part of the Rosemont Plan of Operations.

Water quality is protected. Both the waste rock and tailings are derived predominantly from sedimentary and high-carbonate rock types such as limestone, dolomite, skarn, and siltstones. These types of waste rock are desirable from an environmental perspective: high pH, low sulfur, low acidity potential, good settling characteristics, and tendency toward structural stability. Co-management of waste rock and tailings will provide for the most compact storage, resulting in smaller surface area impacts and minimizing the potential for surface water runoff or groundwater seepage issues.

6.1.6 Dark Skies Initiative

The open space and low population density surrounding the Rosemont site makes for low levels of fugitive light and dark nights. Designs for Rosemont will follow the intent of the dark skies programs to protect these values and ambient conditions, even though the mining industry is excluded from those requirements.

Dark skies design criteria. Special design sodium lights will be used or other technologies as may be developed, to replace the more traditional, less desirable mercury vapor or metal halide lighting for outside areas. Shielding, placement, total lumens used, and types of fixtures will be carefully considered as a design and purchasing criteria for Rosemont.

6.1.7 Social Issues and Socioeconomics

The issues of socioeconomics are recognized as both important and complex. Jobs are beneficial. Traffic is problematic. Steps to balance these issues have been taken in the early stages of mine design and development of the process flow sheet.

Area employment. The operating mine and mill will employ some 348 full time direct employees. If a concentrate leach circuit is included the number of jobs could be increased to around 400 full time jobs. Indirect employment opportunities for suppliers, contractors, and service personnel will exceed the number of direct employees. Taken together, the number of direct and indirect jobs created by Rosemont will represent job opportunities for a thousand families or more.

Traffic reduction. The use of two 12 hour shifts per day equates to less traffic and conservation of fuel resources. In addition, producing cathode copper on site has potential to reduce the number of concentrate shipments.

Public safety. Augusta has selected an access route to the mine site that avoids the narrow winding areas of State Highway 83 at mileposts 44-46. Augusta will work with the ADOT to design the access road connection with the highway.

6.1.8 Facility Siting and Viewshed Protection

Preliminary siting evaluations have included viewshed analyses to minimize the areas of visual impact.

Mill site located in a back valley. The mill site was selected to utilize natural ridgelines and topography to shield the site from view, insofar as possible. The siting and sizing of waste rock elevations and contours have taken into consideration the elevations of the crusher, mill, mine pit, and conveyor, so as to shield these mine facilities early in the mine life.

The use of berms, terracing setbacks, vegetation, siting, and color selection of facilities will be utilized to minimize the visual impact of the Rosemont mine.

6.1.9 Environmental Management System

The EMS for the Rosemont Project follows the goals of pollution prevention, avoidance of impacts wherever possible, mitigation of unavoidable impacts, and enhancement of natural resources towards sustainability.

The compact nature of the proposed facility layout within the natural basin surrounding the Rosemont Mineral Resources allows for the minimum impact achievable.

6.2 Federal Mine Plan Approvals

Provisions of the General Mining Law of 1872, the Federal Land Policy and Management Act of 1976, the Mining and Mineral Policy Act of 1970; and the National Materials and Minerals Policy, Research and Development Act of 1980 authorize mining on public lands under an approved Mine Plan of Operations provided that all other federal, state and local environmental permits and authorizations are received.

6.3 Environmental Impact Statement

The land required for mining the open pit is privately held. Adjacent public land will be required for milling, utility corridors, access roads, waste rock and tailing disposal, and other incidental operations. Acquiring the right to use and occupy several thousand acres of this public land will require completion of an Environmental Impact Statement.

Completion of an EIS for Rosemont operations will include public scoping, community involvement, technical analysis, field data collection and reporting, endangered species consultation as needed, public notice and comment periods, and publication of the Draft EIS and the Final EIS.

6.4 Threatened and Endangered Species Review

The U. S. Fish and Wildlife Service and Arizona Game and Fish Department maintain lists of Special Status Species; threatened, endangered, proposed endangered, candidate, and conservation agreement species. Mine plans will be subject to review for avoidance or mitigation of impacts to protected species.

6.5 Aquifer Protection Permit

ADEQ requires that potentially discharging facilities are subject to environmental review under the Aquifer Protection Permit Program. This process must demonstrate that discharging facilities will not cause an exceedance of aquifer water quality standards. In addition to this technical demonstration, groundwater quality monitoring will be required during operations and through mine closure. A detailed closure plan is required to show how water quality will be protected after mine operations are completed.

6.6 Air Permits

Any Augusta mining operations must obtain an air quality control permit from the Arizona Department of Environmental Quality. The permit will contain provisions for emission control equipment or practices, recordkeeping and reporting procedures and monitoring.

6.7 Water Quality and Stormwater Permits

Stormwater discharge permits and Stormwater Pollution Prevention Plans will be required for any open pit mining operation.

6.8 Army Corps 404 Permit

Three major washes and several tributaries will require crossings of jurisdictional waters to access the site. Other washes and ravines will be affected by mine pit and waste areas. The total sum of the jurisdictional area for Section 404 will require that Rosemont obtain an individual permit from the US Army Corps of Engineers.

6.9 Arizona Department of Transportation (ADOT) Access Road Approvals

To improve the road access from the existing state highway, Augusta must follow the ADOT permit process for review and approval of construction along an existing scenic highway.

6.10 Mine Reclamation and Closure Plans and Financial Assurance

Mine closure plans are required as part of the Federal Mine Plan of Operations as well as by the Arizona State Mine Inspector and the Arizona State Department of Environmental Quality. The State Mine Inspector Reclamation Plan requires a detailed plan showing what post-mining land uses will be possible on the mined out lands, and must include a program for achieving those post-mining land uses. Federal and state reclamation closure plans require a financial assurance instrument to demonstrate financial ability to complete the reclamation program as described in the closure plans. There are interagency agreements to allow for each agency to recognize the financial assurance held by other agencies, so that duplicate bonding is not required.

6.11 Cultural Resources

All lands required for mine construction and operation will require clearance for cultural resources. The process includes field survey for locating cultural resource sites, and testing of sites determined to be significant.

6.12 Local Permits and Approvals

Several local agencies will be involved in the approval process through Pima County Development Services. These agencies regulate floodplain encroachments, drainage improvements in washes, grading land clearing for roadways and erosion control and impacts to water quality in streams:

6.13 Impact of Permitting Process on Rosemont Project

The time to complete the permit application and review process can be affected by potential public controversy, difficult or unresolved technical issues, legal challenges, changes in operating plans, or unforeseen environmental impacts that are not readily mitigated.

At the end of the environmental review process, the responsible official from each agency must sign a permit, issue a Record of Decision (ROD) or provide some other form of documentation approving, denying, or modifying the permit application. Typically, the approving permit document authorizes a specific project component, and may include a list of conditions and permit requirements that mitigate or minimize the environmental issues determined to be significant in the review analysis. These conditions can affect project schedule, economics, and feasibility.

7. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The information in this section has not changed from the previously submitted report titled: "Mineral Resource Estimate Revised Technical Report for the Rosemont Deposit, Pima County, Arizona, USA" dated April 21, 2006. See Reference 1 for this information.

8. HISTORY

The information in this section has not changed from the previously submitted report titled: "Mineral Resource Estimate Revised Technical Report for the Rosemont Deposit, Pima County, Arizona, USA" dated April 21, 2006. See Reference 1 for this information.

9. GEOLOGICAL SETTING

The information in this section has not changed from the previously submitted report titled: "Mineral Resource Estimate Revised Technical Report for the Rosemont Deposit, Pima County, Arizona, USA" dated April 21, 2006. See Reference 1 for this information.

10. DEPOSIT TYPES

The information in this section has not changed from the previously submitted report titled: "Mineral Resource Estimate Revised Technical Report for the Rosemont Deposit, Pima County, Arizona, USA" dated April 21, 2006. See Reference 1 for this information.

11. MINERALIZATION

The information in this section has not changed from the previously submitted report titled: "Mineral Resource Estimate Revised Technical Report for the Rosemont Deposit, Pima County, Arizona, USA" dated April 21, 2006. See Reference 1 for this information.

12. EXPLORATION AND DEVELOPMENT

Exploration work by previous owners has been described in Section 8 of Reference 1 above.

Exploration work by Augusta includes the 2005 Rosemont Deposit infill diamond drilling program, which is described in Section 13 of Reference 1 (Drilling). Results of the 2006 Rosemont drilling program are not included in this assessment.

13. DRILLING

The Augusta diamond drilling program was designed to augment and be combined with the previous drilling information to create the underlying database for a NI 43-101 compliant resource estimate.

The previous Anaconda and Anamax drilling on the Rosemont Deposit amounted to approximately 184,000 feet (56,100 meters) of mostly NQ (2-inch diameter drill hole) -size diamond drilling in about 150 holes. Most of the holes were vertical, although many were inclined -45° to -55° in order to provide samples cored perpendicular to the hosting lithology bedding, which dips 50-60° to the east. Drillhole orientation is considered of minor importance owing to the large-scale morphology of the deposit that would be most amenable to large-scale, open-pit mining methods. Down-hole surveys were conducted during drilling or immediately following drillhole completions. Drillhole collars were surveyed by company surveyors.

Augusta carried out its diamond-drilling program in the last half of 2005, drilling a total of 27,402 feet (8,352 meters) in 15 holes. Layne-Christensen and Boart Longyear were the drilling contractors. Holes were drilled HQ (2.9 inch diameter drill hole) as far as possible, then finished NQ. Most of the holes were oriented vertically, although three of the holes were oriented in other directions and inclinations in order to accommodate drill targets to the available surface access. All drillholes were surveyed down hole with a Reflex EZ-Shot drillhole survey that measured inclination/dip and azimuth direction every 500 feet down hole. The drillhole collar locations were surveyed by Putt Surveying of Tucson, Arizona.

A total of 161 drillholes have been drilled to date within the Rosemont model area, which are summarized by company in Table 13.1.

Table 13.1 Rosemont Deposit Drilling Summary

Company	Time Period	*Drill Holes*		
		No.	Feet	*Meters*
Banner	1950s	4	4,484	*1,367*
Anaconda	1963-1973	100	130,347	*39,728*
Anamax	1973-1983	42	49,203	*14,996*
Augusta	2005	15	27,402	*8,352*
Total		***161***	***211,436***	***64,443***

Almost all of the holes are diamond drillholes. Most were cored from the surface or after about 10 ft of rock-bit drilling to set the surface casing. However, an as-yet undetermined number of the core holes were pre-collared by hammer or rotary drilling until they approached projected depths of mineralization. Hammer or rotary cuttings were generally sampled on intervals ranging from 10 to 20 ft, although a few holes were sampled on 50-ft intervals. No details have been recorded regarding the methods of sample collection for rotary or hammer cuttings.

14. SAMPLING METHOD AND APPROACH

The present study did not involve any sampling other than the drillhole sampling described below.

Both Anaconda-Anamax and Augusta drillhole core recoveries averaged over 90% in calcareous sedimentary rocks and andesites. Near-surface recoveries in silty and shaley Mesozoic rocks were typically about 65%, but improved to over 90% below depths of about 50-75 feet (15-23 meters). The samples provided by the drilling are considered adequate for purposes of resource estimation.

Sulfide mineralization hosted by carbonate sedimentary rocks occurs in rock alteration that ranges from marble to skarn as described previously. Marble tends to host disseminated sulfide mineralization while sulfide mineral distributions in skarn range from disseminated to massive.

Individual lenses of massive sulfide typically measure a few inches or cm in width, but may rarely range up to about 15 feet (5 meters) in true width. The style of sulfide mineralization in hornfels ranges from veinlet to disseminated. These styles of mineralization occur intermittently over true widths ranging from about 100 feet (30 meters) at the edges of the deposit up to approximately 1,300 feet (400 meters) near the deposit center.

Individual core samples were taken over nominal 5-foot (1.5-meter) lengths during both the Banner-Anaconda-Anamax and Augusta programs. Shorter intercepts of massive or near-massive sulfide mineralization measuring more than a nominal foot (0.3 meter) or so in length were sampled over widths matching the width of such mineralization. Occasionally, long stretches of very uniform mineralization was sampled over widths ranging up to 10 feet (about 3 meters).

Drillhole sample assay values were composited as one of the steps in creating a block model of the deposit as described within Section 19 of this report. This process effectively addresses issues related to the comparison of drilled and true widths.

Additionally, a sample of split core from each rock formation unit was selected from every Anaconda-Anamax hole, as well every Augusta drillhole for specific-gravity measurements. The samples were typically about 6 inches (15 cm) in length. The specific-gravity determinations were carried out by Skyline Labs using standard weight-in-water, weight-out-of-water methods on a total of 392 samples.

Other sample handling, preparation and security procedures are described in the following section.

15. SAMPLE PREPARATION, ANALYSES AND SECURITY

15.1 Sample Handling and Security

Sample preparation from the Banner and Anaconda-Anamax programs was conducted by employees of those companies. During the Augusta program, samples were marked, cut and transported to the laboratory by contractors hired by Augusta. No Augusta employee, director or associate was involved in these tasks.

Banner and Anaconda-Anamax selected drill-core samples for assay and split them using standard mechanical core splitters. Other details are not known, but since this work was carried out by large copper companies for their internal use, it is believed that the work was carried out to industry standards for that time and is reliable.

The historical Rosemont methods of copper and molybdenum wet chemical analyses and copper and molybdenum x-ray fluorescence (XRF) used by Anaconda and Anamax were all standard procedures in the 1960s and 1970s, according to Mr. Dale Wood, Anaconda Chief Chemist (based on meetings and telephone conversations on November 28, 2005 and January 21, 2006). Details of the procedures used are as follows: Crushing and grinding reduced all pulp samples to minus 100 mesh size, with constant screen size testing. Pulp samples for the wet chemical method was brought into solution by hot acid digestion on a shaker table with hydrochloric acid, nitric acid, and perchlorate acid were added to the boiling solution, followed by a few drops of hydrofluoric acid. Analysis for molybdenum was by the colorimetric iodine titration method and 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.

During the Augusta program, the contract drillers placed the core into cardboard core boxes, with the footage marked on wood blocks inside the boxes as well as on the outside of the core boxes. The drilling companies kept the core in a secure area next to the drill rig before delivering it to the Rosemont Ranch core logging area, approximately three miles from the drilling area, on a daily basis. No core handling problems or core security problems during the drilling or sampling program were observed by the drillers, geologist or samplers.

All drillholes were logged by experienced Arizona geologists who measured percent core recovery, rock quality data, rock type, alteration, mineralization and structure. After logging, the geologist marked the sample intervals and cut-lines directly on the core, splitting evenly mineralization or structures for diamond core sawing and sampling. Sample intervals were generally 5 feet (1.5 meters) in length, except where massive copper or molybdenum veining, structures or lithologic breaks exist.

Intervals for core sawing were marked on the core boxes with a black marker and given a sequenced sample number with the footage noted in a booklet and on a paper copy. Each sample interval was labeled with a unique sample number. The drill core boxes were then photographed with a digital camera before delivery to the diamond saw core cutting area. 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 tag, half of which was placed into the bag. The plastic bags were then sealed with adhesive tape, leaving the sample number visible. The samples were delivered directly to the Skyline Labs, Inc. (Skyline) office in Tucson, by contract geologists, either Michael R. Pawlowski or Thornwell Rogers. All samples were delivered to a secured, locked area in the Skyline Laboratory building in Tucson along with a sample inventory paper sheet.

All core boxes, sample coarse, rejects and sample pulps are currently stored at the project site. The core boxes and sample coarse rejects are stored on pallets that are wrapped in plastic and the sample pulps are stored in sealed 55 gallon drums.

The principal assay laboratory for this project is Skyline, formerly known as Actlabs-Skyline and owned by ACTLABS (Ancaster, Ontario, Canada) since 1997. 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. Primary and secondary (duplicates) analyses are done at Skyline. ALS Chemex is used by the project for assay checks of duplicate samples. ALS Chemex has accreditation through ISO 9001:2000 in North America.

15.2 Quality Assurance and Quality Control Protocol

The following protocol was recommended on July 26, 2005 by Kenneth A. Lovstrom, Geochemist, and followed for the Rosemont feasibility study. Shea Clark Smith, Geochemist, assumed guidance for aspects of Quality Assurance (QA) and Quality Control (QC) after January 10, 2006, following the same protocols:

- Standards should be inserted with a frequency of one per 20 drill samples. Standards should be developed from Project matrix and include cut-off, run-of-mine, and high grade concentrations. Inserted standard should estimate interval grades as determined by preliminary logging.

- Duplicate samples should be submitted with a frequency of one per 20 drill samples.

- Blank samples should be submitted with a frequency of one per 40 samples.

- Five percent (5%) of drill samples should be analyzed by a secondary laboratory and include sub-ore and ore-grade intervals. Project standards must be included with a frequency of one per 10 samples.

- Primary and secondary laboratories must be audited annually by an external, independent source with detailed reports completed.

- Primary laboratory data reports must contain internal laboratory quality control data. An original, certified report(s) must be filed with each drillhole.

Seven standard reference materials (SRMs) of various grades of copper, molybdenum, and silver, and a blank that contains metal concentrations at or below the analytical limits of detection, were used to monitor assay accuracy and precision. The range of copper concentrations in these SRMs is 0.01% to 1.95%, while the range of molybdenum values is 0.017% to 0.078%. Silver concentrations range from 2.7 ppm to 7.0 ppm. Full chemical characterization and stability (accuracy and precision) for each of these SRMs were determined by Round Robin analysis at 3 to 7 analytical laboratories, where 5 to 20 aliquots of each material was analyzed. Average (arithmetic mean), standard deviation (SD), and relative standard deviation (%RSD) were determined, and these criteria were used for comparison to SRM values that were returned with drill core sample data. Lovstrom assessed the data for all submittals and determined that, within a few percent; the values for (SD), and relative standard deviation (%RSD) Total Cu (TCu) and Mo were within satisfactory accuracy limits (0% to 7% difference between expected and reported SRM values).

Metal concentrations in drill core from drillholes AR2000 through AR2014 range from 0-5.3% Cu (90% of the concentrations are less than 2%), from 0-0.6% Mo (90% of the concentrations are less than 0.1%), and from 0-70 ppm Ag (90% of the concentrations are less than 20 ppm). SRMs used to monitor these concentration ranges adequately cover 90% of the reported values.

15.3 Skyline Sample Preparation Procedures

Split core samples from a locked core shed at the Rosemont Ranch were delivered daily by Augusta personnel directly to Skyline. At Skyline, the entire sample was crushed using a TM Terminator to produce an 80% pass 10-mesh product. Samples were blended and divided using a two-stage riffle splitter, from which a 300-400 gram aliquot was pulverized to 90% pass 150-mesh product using a TM Max 2 Pulverizer. Wash gravel and sand were used to clean the crushers and pulverizers only when deemed necessary by the operator. Reject material from each sample was saved in locked storage and one of every 20 core samples was used for duplicate analytical testing at a later time.

The SRMs are packaged in small Kraft™ envelopes, which make them easily identified by the laboratory as QA monitors, along side canvas bags of unpulverized core. Since a relatively large number of SRMs were used, each with a different concentration range of Cu, Mo and Ag, these standards were relatively "unknowns" to the analyst. However, both SRMs and blanks would have bypassed crushing and pulverizing stages in the lab, and were thus of limited use as a check on sample preparation processes.

15.4 Skyline Analytical Procedures

Assay pulps were weighed and digested using industry accepted laboratory equipment and procedures. Copper and molybdenum were digested using a three-acid procedure, with copper content being determined by atomic absorption spectrometry (AAS), and molybdenum content being determined by inductively coupled plasma/optical emission spectrometry (ICP/OES). Silver was determined from a different aliquot by aqua regia digestion followed by AAS. Standards and blanks were provided by Augusta personnel and properly inserted into the analytical stream where indicated. Concentrations of copper and molybdenum are reported in percent, while silver is reported in parts per million (ppm).

15.5 QA Results for Drillholes AR2000 – AR2002

Lovstrom reviewed analytical data for submittals related to samples from drillholes AR2000, AR2001, and AR2002, and reported his findings dated August 26, September 15, October 13, and October 14, 2005. Invariably, Lovstrom found that SRMs and Blanks reported within 0% to ± 2% of the expected Cu and Mo value and deemed these data to be accurate and reliable. Duplicate analyses also reported within narrow bounds (-5% to +5% of the original analysis) and Lovstrom also deemed these data to be accurate and reliable.

Assay checks at ALS Chemex were reported by Lovstrom on October 13, 2005. SRMs reported within ±2% of the expected values for Cu and Mo, and duplicate assays similarly showed acceptable differences between Skyline and ALS Chemex of ±2% Cu and ±7% Mo.

Lovstrom reported SRM and duplicates data for drillholes AR2001 and AR2002 on October 14, 2005. The SRM data demonstrate excellent comparison between expected and reported concentrations for Cu of +1% to +3%, with precision of 1% RSD. Data for Mo between expected and reported SRM concentrations show variances of -8% to -11%. From drillhole AR2001, Skyline analyzed five duplicates and was within an acceptable error range for Cu and Mo. Similarly, eight duplicate assays from Skyline for drillhole AR2002 were reported within an acceptable error range for Cu and Mo.

S.C. Smith has reviewed all of this work and concurs with Lovstrom that these SRMs have been acceptable monitors of accuracy and precision for Total Cu and Mo, and that the assay data for core samples are reliable. Silver data for samples may be somewhat less reliable because of low stability in the SRM data, which is more a function of the standard itself than lab performance.

15.6 QA Results for Drillholes AR2003 – AR2014

An SRM provided by Lovstrom was used to monitor laboratory accuracy during analysis of drillholes AR2003 through AR2006, and beginning with AR2005, new standards R1 and R2 were introduced as new monitors. All standards reported within acceptable limits for Cu and Mo, while Ag concentrations displayed wider variability that may be more a function of the standard itself than lab performance.

Round Robin and run-of-assay data have been used to establish a statistical and chemical characterization of standards R1 and R2. Time-series charts have been used to determine the stability of these standards over time during analysis of samples at the primary laboratory (Skyline). R1 contains stable concentrations of 0.48% Cu, 0.025% Mo, and 5.0 ppm Ag. R2 contains stable concentrations of 0.73% Cu, 0.017% Mo, and 6.9 ppm Ag. These values, with variances of only 1-2%, have been consistently reported through assay submittals to Skyline for drillholes AR2005 through AR2014. Blank data reports consistently below the analytical detection limit for Cu and Mo, but at 0.5 ppm Ag with a variance of about 300%, calling into question the validity of the silver data reported in this concentration range.

16. DATA VERIFICATION

This section briefly describes the verification of the drillhole database by comparison to assay certificates, both from previous owners and independent commercial laboratories.

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 drillholes were inspected, representing approximately 14% of the total database. 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.

17. ADJACENT PROPERTIES

Adjacent properties are not included in this PAEE.

18. MINERAL PROCESSING AND METALLURGICAL TESTING

18.1 Metallurgical Test Samples

Metallurgical testing on the Rosemont, reviewed and evaluated for this report, was preformed during two separate campaign periods. The first campaign was conducted by previous owners and is referred to as "historical". This series of test work was done on ore samples which contained chalcopyrite as the primary copper mineral. The historical test work results are summarized in Section 18.2 of this report.

The second test campaign was conducted by Augusta Resources at Mountain States R&D International, Inc. of Vail, Arizona. These series of tests were run on samples containing chalcocite and bornite as the primary copper minerals. This new test work campaign is on-going.

18.2 Historical Metallurgical Testing

The historical metallurgical testing for the project consisted of a test campaign performed in 1974 and 1975. The test samples consisted of half-core samples taken from intervals of selected diamond drill core holes.

Sample composites were selected by ASARCO based on rock type and grade. The samples listed were plotted on a drillhole plan map and were located in the central portion of the Rosemont Deposit.

18.2.1 Grinding

Two types of standard grinding tests were performed on core samples from the Rosemont Deposit. A. R. MacPherson Consultants Ltd. investigated the primary grinding characteristics of the mineralized rock using Aerofall mills and Allis Chalmers performed standard Bond Ball Mill Work index testing on the samples to obtain data for sizing the ball mills.

18.2.2 Primary Autogenous Grinding Tests

The results of the MacPherson test program indicated that the potential ores would perform well using autogenous grinding mills, requiring no grinding media to achieve the required primary reduction. The current flow sheet, however, has assumed that semi-autogenous mills will be used. Three mineralized samples were tested and their resulting autogenous work indexes and actual power requirements are presented in Table 18.1.

Table 18.1 Autogenous Grinding Test Results

Sample	Autogenous Work Index (kWh/ton)	Actual Power Required (kWh/ton)
Sample 1 – High grade garnetized limestone	10.42	10.0
Sample 2 – Low grade garnetized limestone	10.16	9.8
Sample 3 – Quartz monzonite porphyry	20.58	15.5

The MacPherson report indicated that the above mix of potential ore types would be beneficial for autogenous grinding as the high and low grade ores are relatively soft and the quartz monzonite would provide a source of grinding media. This assumption is dependent upon the relative amounts of the three rock types and the period(s) when they will be available for plant feed. Since this information was not available, the option of using autogenous mills was ruled out, but should be evaluated in subsequent studies.

It is recommended that additional testing be performed to assess the variability of the ore and to confirm that the mill selection is correct.

18.2.3 Rod Mill and Ball Mill Grindability Tests

Standard Bond rod and ball mill grindability tests were performed by Allis Chalmers on three composites of potential Rosemont ore taken from drillhole DDH A861. Descriptions of the samples and the resulting rod and ball mill work indices are shown in Table 18.2.

Table 18.2 Rod and Ball Mill Grinding Test Results

Sample	Rod Mill Work Index @ 1.2mm (kWh/ton)	Ball Mill Work Index @ 150 micron (kWh/ton)
Composite No. 1 - Calcium silicated argillite	15.5	14.2
Composite No. 2 - Garnetite	13.9	13.1
Composite No. 3 - Silicated silty limestone	14.7	11.3

The Winters Company, in their 1997 report, selected a weighted-average ball mill work index for the deposit of 14.5 kWh/ton, which is appropriate based on the test results presented in Table 18.2. The grind size selected in the Winters report was 80 percent passing 145 microns.

18.2.4 Flotation

A flotation test program was conducted by Mountain States Research and Development (MSRD) in Tucson, Arizona. Test results from two sequential test periods, September 1974 and

March 1975, were found in the Rosemont Ranch files. The initial set of single-cycle tests were performed in September 1974. Owing to the results of the first test series, a second set of tests were performed in March of 1975 to better define the design criteria. Locked-cycle tests were recommended by MSRD, but no information has been found suggesting that they were never completed.

In the fall of 1974, single-cycle bench-scale rougher flotation tests were performed on a set of six drill core composite samples, taken from intervals of holes DDH 1504 and DDH 1507 by ASARCO. The results of this test are shown in Table 18.3.

Table 18.3 Single-Cycle Bench-Scale Rougher Flotation Test Samples

Sample ID	Number of Samples	Length (ft)	% Total Cu	% AS Cu	% Mo
Composite 1504-3	69	343	0.70	0.07	0.020
Composite 1504-4	15	73	1.10	0.57	0.004
Composite 1504-5	61	305	0.54	0.05	0.008
Composite 1507-6	106	506	0.83	0.04	0.017
Composite 1507-7	33	158	0.39	0.02	0.012
Composite 1507-8	14	65	0.85	0.64	0.006

Standard flotation tests were performed using grind sizes ranging from 40 to 75 percent passing 200 mesh with approximately 80 percent passing 150 microns. A summary of these test results is presented in the Table 18.4. The mass pull for the series of tests ranged from approximately three to six percent of the feed and the primary characteristic affecting flotation performance was found to be the degree of oxidation.

TABLE 18-4 Flotation Results: Metal Recovery to Concentrate

Sample	Rougher Concentrate, (%)	Rougher Recovery, (%)
Composite 1504-3	16.5 - 20.0	90 – 92.5
Composite 1504-4 (Highly Ox)	21.5 – 24	58.6 – 55.6
Composite 1504-5	12.6 – 18.6	82.6 – 88.7
Composite 1507-6	13.5 – 20.4	89.7 – 97.1
Composite 1507-7	15.45 – 16.35	90.9
Composite 1507-8 (Highly Ox)	10.8 – 11.1	18.1 – 19.6

18.3 Augusta Resources Test Work

Metallurgical test work was conducted on composited core samples from Augusta's 2005 core drilling program by Mountain States R&D International, Inc. of Vail Arizona (MSRDI). Core intervals were selected by Augusta. The MSRDI testing program was conducted during the first and second quarter of 2006. Additional delineation testing at MSRDI continues. The following sections summarize the pertinent test work results available and evaluated for this report.

For the flotation testing, individual core sample intervals were combined to provide a bulk composite for testing. Grinding tests were run on individual core sample intervals.

18.3.1 Head Grades for Flotation Test Sample

Sample splits from the bulk composite were submitted for head analyses. These results are summarized in the Table 18.5.

TABLE 18.5 Composite Head Grades

Parameter	Value
Total Copper, %	0.74
Acid Soluble Copper, %	0.104
Cyanide Soluble Copper, %	0.35
Molybdenum, %	0.0175
Iron, %	2.5
Gold, troy ounce/ton	0.002
Silver, troy ounce/ton	0.29

The bulk composite contained 0.104% oxide copper which is equivalent to 14% of the total copper present. Since copper oxide minerals do not report with the flotation concentrate, the expected recoveries are lower than if 100% sulfide ore was processed.

18.3.2 Grinding Test Work

Bond Ball Mill work indices were determined on 10 core samples. The core samples selected were from regions of the deposit, designed to give a spatially representative distribution. The results are presented in Table 18.6.

TABLE 18.6 Bond Grindability Test Work Results

Sample Identification	Bond Ball Mill Index kW hr/ton
194651	16.6
194652	11
194653	11
194654	12.1
194655	12.9
194656	10.5
194657	10.1
194658	9.2
194659	9.4
Average	**11.4**
Max	**16.6**
Min	**9.2**
STDEV	**2.3**
Median	**11.0**

Bond test work indicated that the material is relatively soft with a range of 9.2 to 16.6 kW hr/ton with a median ball mill index of 11.0 kW hr/ton. Sample 194651 is a non ore or waste rock sample and reflect ore grindability. For mill sizing a bond ball mill index of 11.0 kW hr/ton was used.

18.3.3 Flotation

Twelve, preliminary, open circuit rougher flotation tests were performed on a bulk composite created from core samples from the 2005 drilling program. Different reagent schemes and grind sizes were evaluated.

A Box-Binken statistically designed series of experiments were run with varying reagent schemes, slurry pH's and grind sizes on sample splits from the bulk composite. This series was used to provide a statistically sound and optimized set of criteria for flow sheet development. The results from the experimental design series is presented in Table 18.7.

TABLE 18.7 Flotation Results: Rougher Experimental Design

Summary of Box Statistical Study - Rougher Flotation										
					Rougher Conc. %		Recovery, %		Calculated Head	
	pH	Test	Grind P80	Weight %	Cu	Mo	Cu	Mo	Cu	Mo
Reagents 238/343/MIBC/FO	7.8	5	100	2.8	20.6	0.323	77.8	58.0	0.74	0.0155
		9	150	3.3	18.7	0.333	83.6	65.1	0.74	0.0169
		27	200	3.1	18.5	0.442	81.4	59.6	0.70	0.0228
	9.7	4	100	3.1	18.2	0.276	82.3	57.6	0.69	0.0149
		8	150	3.3	19.6	0.274	85.4	59.1	0.75	0.0151
		26	200	3.0	18.9	0.433	82.0	63.3	0.68	0.0203
	10.8	51	100	2.7	19.3	0.329	75.8	57.7	0.70	0.0156
		7	150	2.7	22.7	0.341	86.9	57.2	0.70	0.0159
		14	150	2.6	20.9	0.391	84.2	61.9	0.65	0.0166
		28	200	3.3	18.5	0.383	85.1	64.5	0.71	0.0194
Reagents 8944/238/NaSil/FO	7.8	53	100	2.2	27.6	0.439	76.0	50.4	0.81	0.0193
		15	150	2.8	25.1	0.475	81.0	61.4	0.87	0.0217
		18	200	2.4	24.6	0.52	79.2	59.1	0.75	0.0212
	9.7	54	100	2.2	29.3	0.429	77.6	50.9	0.81	0.0181
		55	150	2.4	28.2	0.452	83.3	61.3	0.82	0.0179
		19	200	2.3	24.9	0.57	79.3	61.1	0.71	0.0211
	10.8	56	100	2.6	27.4	0.376	82.3	53.5	0.86	0.0182
		13	150	2.2	26.2	0.454	80.3	61.8	0.73	0.0164
		20	200	2.6	22.9	0.54	81.9	65.2	0.73	0.0215
Reagents 3501/3501/NaSil/FO	7.8	57	100	3.7	18.2	0.32	84.0	68.5	0.80	0.0171
		58	150	3.7	18.0	0.339	84.3	69.9	0.79	0.0179
		59	200	3.2	19.4	0.375	81.9	69.0	0.75	0.0172
	9.7	49	100	4.5	15.2	0.275	86.1	70.2	0.79	0.0176
		12	150	4.7	13.3	0.275	86.6	68.6	0.72	0.0188
		16	200	5.3	13.5	0.241	87.7	74.8	0.81	0.0169
	10.8	60	100	5.2	11.6	0.182	85.7	58.6	0.70	0.0160
		61	150	5.6	12.6	0.198	88.3	61.5	0.79	0.0179
		62	200	4.9	13.5	0.210	86.9	60.8	0.76	0.0170
Average				**3.3**	**20.3**	**0.364**	**82.7**	**61.8**	**0.75**	**0.01803**
Max				**5.6**	**29.3**	**0.570**	**88.3**	**74.8**	**0.87**	**0.02280**
Min				**2.2**	**11.6**	**0.182**	**75.8**	**50.4**	**0.65**	**0.01490**
STDEV,				**1.0**	**5.0**	**0.103**	**3.5**	**5.8**	**0.06**	**0.00216**
Median				**3.05**	**19.35**	**0.358**	**82.8**	**61.4**	**0.75**	**0.01775**

Rougher concentrate mass pulls averaged 3.3 percent with a range of 2.2 to 5.6 percent. The median mass pull was 3.05 percent. Metal recoveries averaged 82.7 percent and 61.8 percent for copper and molybdenum respectively. Copper concentrate grades averaged 20.3 percent. While, molybdenum content in the rougher concentrate averaged 0.364 percent.

Owing to the presence of acid soluble or oxide copper, flotation recoveries were negatively impacted and are artificially lower when compared to sulfide recoveries. It is expected that copper recoveries from sulfide ore types would typically be in the range of 90 to 95%. For this study, a copper flotation recovery of 89% was used to be conservative.

19. MINERAL RESOURCE ESTIMATES

Only mineral resources are estimated for the Rosemont Deposit in this report. Engineering studies and economic evaluations, while in progress at the time of this writing, have not yet progressed sufficiently to quantify an estimate of mineral reserves.

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

19.1 Model Extents

The mine coordinate system is based on Imperial units (i.e., feet) and was developed by previous property owners. A 25 ft by 25 ft lateral block dimension was selected to better fit the moderating dipping lithologies at Rosemont. Table 19.1 summarizes the limits of the 3D block model expressed in mine coordinates.

Table 19.1 Deposit Model Limits

Direction	Minimum	Maximum	Block Size (ft)	No. of Blocks
X (East)	856,500	863,500	25	280
Y (North)	301,000	308,000	25	280
Z (Elevation)	2,500	6,500	50	80

19.2 Surface Topography

Raw topographic data for the project area was downloaded from the National Elevation Dataset (NED) 1/3 Arc Second database that has been assembled and maintained by the U.S. Geological Survey (USGS). These data provide an approximate 10m resolution using the NAD83 horizontal datum and the NAVD88 vertical datum. Stantec, Inc., working as a consultant to Augusta, converted these elevation data into 10-ft contours covering about 100 square miles in and around the project site. The contours were expressed in longitude and latitude coordinates and elevations in feet. The topographic data were then passed to WLRC as an ArcView® shape file.

The shape file was converted by WLRC into an AutoCAD DXF format. Coordinates were converted into UTM NAD 83 Zone 12 metric values and then to Imperial units. Two National Geodetic Survey control points, Helvetia (PID CG1065) and Huerfano (PID CG1067), have been surveyed in both UTM NAD83 and mine coordinate systems. The Helvetia marker is located along the northern edge of the deposit model, providing a good control point for determining the origin and rotation of the mine coordinate system in terms of UTM NAD83 Imperial values. No adjustments or conversions were made to the elevations.

The following best fit mine origin (0,0) was computed in UTM Imperial coordinates:

Mine system origin - UTM East 852,971.65 feet
UTM North 11,257,354.12 feet

Mine system rotation - 0.4834° (clockwise)

With only two control points, the topographic contours were not rubber-sheeted to fit both markers. Instead, the above origin and rotation provides an exact match in both coordinate systems for the Helvetia marker, but produces a 9.39 ft lateral difference for the Huerfano marker located about four miles to the west-northwest. This difference is likely the result of mine system survey errors between the control points. Comparisons with previous topographic contour maps and visual inspections of the land forms with respect to drillhole locations indicate a good fit for use in a preliminary mineral resource evaluation.

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 that is below topography.

19.3 Drillhole Database

The Rosemont Deposit drillhole database contains collar locations, down-hole deviation surveys, sample assay results and geological information from a recent drilling program by Augusta, and from a series of exploration drilling campaigns conducted by a number of companies in the past, see Table 13.1 of this report. In all, 161 drillholes comprise the database contained within the project area. 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 diamond core, although a small number (less than five percent) were drilled by open hole rotary and reverse circulation techniques.

Stored in the database are 44,496 individual sample values representing approximately 211,436 feet (64,443 meters) of drilling. Each sample interval record contains assayed values for Cu and Mo. Some intervals have assays for recoverable oxide Cu and also for Ag. The reliability of the Ag values, however, has not yet been established and the Ag assays were not considered in this study.

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 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 19-1 and 19-2 in Section 26 of this report. 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 drillholes were inspected, representing approximately 14% of the total database. 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.

A statistical study including frequency distribution histograms for each rock type and lognormal cumulative probability graphs were generated for both Cu and Mo for the deposit as a whole, see Figures 19-3 and 19-4 in Section 26 of this report. 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. The cumulative probability plot of Mo grades exhibits a better behaved population with no high grade outlier segment and, therefore, no grade capping adjustment was made.

19.4 Material Densities

Table 19.2 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 19.2 Rock Types and Bulk Tonnage Factors

Rock/Formation Description	Rock Code	Tonnage Factor (ft^3/ton)
Overburden, unconsolidated	1	14.50
Epitaph	2	12.33
Colina	3	11.81
Earp	4	11.79
Horquilla	5	11.27
Escabrosa	6	11.60
Martin	7	11.77
Quartz Monzonite Porphyry	8	12.26
Andesite	9	11.65
Arkose, Akcg	10	12.15
Limestone, Lscg	11	11.58
Scherrer	12	12.00
Undefined	13	12.00

19.5 Geologic Model

A detailed geologic representation for the deposit was developed in the computer model from drillhole cross-section and level plan maps. Rock type and structural outlines were digitized on 50-ft-interval level plans and loaded into the model blocks. In all, 12 individual rock types were delineated (see Table 19.2). Material not defined from the geologic section and plan maps was assigned a code of 13. The model was checked by plotting out model levels and comparing against the original plan maps. Problem areas from the block tagging algorithm were noted and adjustments/corrections were made.

19.6 Mineralization Controls

In this deposit, all of the rock types are mineralized to some degree. Some lithologies are significantly better hosts due to favourable chemical composition and/or close relationship to feeder structures. For this reason, each rock type was considered as an independent unit for grade estimation purposes. Only grades from drillholes that pierced a particular rock type were use to evaluate that rock type.

19.7 Compositing of Drillhole Data and Statistics

The drillhole sample assay intervals were weight averaged to 50-ft 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 and Mo composite grades. Coefficients of variation for of all rock types were 1.22 for Cu and 0.94 for Mo. These values are very much in line with what one would expect in this type of deposit.

19.8 Variography

Variograms were calculated to determine 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 Cu rock type composites were grouped to provide a clear variogram (see Figure 19-5 in Section 26 of this report) from which parameters could be selected for the block grade estimation equations. A spherical model was fit to the experimental variogram and the following parameters were selected:

Nugget= 0.01517

Sill = 0.18901

Range = 236 feet (72 m)

The orientation of the variogram was at an azimuth of 90 degrees, plunging at a dip of -48 degrees. This is consistent with the measured dip angles of the sedimentary rock formations.

Since the Mo variograms were not sufficiently clear enough to be useable and the relationship between Cu and Mo shows excellent correlation with a coefficient of 0.847, the variogram selected for Cu was used for Mo grade estimation as well.

19.9 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 and Mo 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 is 90 degrees azimuth, -48 degrees dip, with a range of 240 feet (73 m), and the secondary direction is 0 degrees azimuth, 0 degrees dip, with a range of 240 feet. A maximum of nine and a minimum of two composites, with only three composites allowed from any one drillhole, were used in the calculation of any one block grade.

19.10 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 one third of the variogram range (approximately 80 feet) and was estimated by at least three drillholes. A block was considered to be indicated if it was within the variogram range (236 feet) and was estimated by at least two drillholes, or was within 80 feet and less than three drillholes were used for estimation. A block was designated as inferred if it was greater than 236 feet from any drillhole or did not meet the minimum number of drillholes required for the indicated classification.

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 US$2.25/lb and a molybdenum price of US$35.00/lb.

Measured and indicated mineral resource estimates for the Rosemont Deposit are summarized in Tables 19.3 and 19.4, respectively. The combined measured and indicated mineral resource estimates are presented in Table 19.5. Inferred mineral resource estimates are shown in Table 19.6. Imperial units are used in these estimations, where tons refer to short tons (2000 lbs). Cu refers to copper and Mo refers to molybdenum. Copper equivalent (CuEqv) values are based on three-year trailing average prices of US$1.25/lb Cu and US$18.00/lb Mo, with no applied recovery factors.

Table 19.3 Rosemont Deposit: Measured Mineral Resources

Cutoff	Tons (thousands)	% Cu	% Mo	% CuEqv*	lbs Cu (millions)	lbs Mo (millions)	lbs CuEqv* (millions)
0.20% Cu	94,000	0.55	0.015	0.77	1,040	28	1,440
0.25% Cu	87,000	0.58	0.015	0.79	1,000	26	1,380
0.30% Cu	80,000	0.60	0.015	0.82	970	24	1,310

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

Table 19.4 Rosemont Deposit: Indicated Mineral Resources

Cutoff	Tons (thousands)	% Cu	% Mo	% CuEqv*	lbs Cu (millions)	lbs Mo (millions)	lbs CuEqv* (millions)
0.20% Cu	348,000	0.50	0.015	0.72	3,500	104	5,010
0.25% Cu	311,000	0.54	0.016	0.77	3,350	100	4,800
0.30% Cu	277,000	0.57	0.016	0.80	3,160	90	4,450

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

Table 19.5 Rosemont Deposit: Measured and Indicated Mineral Resources

Cutoff	Tons (thousands)	% Cu	% Mo	% CuEqv*	lbs Cu (millions)	lbs Mo (millions)	lbs CuEqv* (millions)
0.20% Cu	442,000	0.51	0.015	0.73	4,540	132	6,450
0.25% Cu	398,000	0.55	0.016	0.78	4,350	126	6,180
0.30% Cu	357,000	0.58	0.016	0.81	4,130	114	5,760

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

Table 19.6 Rosemont Deposit: Inferred Mineral Resources

Cutoff	Tons (thousands)	%Cu	%Mo	%CuEqv*	lbs Cu (millions)	lbs Mo (millions)	lbs CuEqv* (millions)
0.20% Cu	145,000	0.45	0.015	0.67	1,300	43	1,930
0.25% Cu	116,000	0.51	0.016	0.74	1,170	37	1,710
0.30% Cu	96,000	0.56	0.017	0.80	1,070	33	1,540

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

The above estimates include oxide, mixed and sulfide mineralization. At a 0.20% Cu cutoff, the breakdown of the estimated 442 million tons of measured plus indicated mineral resources is approximately 5% oxide, 41% mixed and 54% sulfide. The mixed mineralization contains oxide, sulfide and occasional native copper mineralization. It is currently believed that sulfides comprise the majority of the mixed mineralization.

Silver is also present in this deposit, but at low concentrations. An average silver grade of 0.21 oz/ton (7.2 g/t) was reported in the 2005 drillhole program. However, silver was assayed only sporadically in the historic work; therefore, there is insufficient data to quantify a silver resource at this time. Augusta is planning further silver assay work with the view towards future resource estimation for this mineral. The arithmetic mean of 5,724 recent and historical silver assays for core intervals in the geologic model greater than or equal to 0.2% copper is 0.28 oz Ag/ton (Augusta 2006). Furthermore, recent metallurgical test work showed an average silver content of 0.29 oz Ag/ton in the composite head grades (reference Table 18.5 of this report). Since the spatial distribution of silver throughout the deposit cannot be presently defined due to lack of sufficient data, it is believed that silver should be treated as a by-product credit based on an average content of 0.28 oz Ag/ton of sulfide ore processed for this PAEE.

Rosemont Deposit mineral resources are on patented lands owned or controlled by Augusta. Notwithstanding the existence of a 3% NSR mineral royalty, the estimates of mineral resources are not affected by known legal, title, taxation, socio-economic, marketing, political, or other relevant issues.

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.

The estimates of mineral resources will not be materially affected by mining, metallurgical, infrastructure, or other relevant 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.

19.12 Potential Ore

For purposes of the PAEE, floating cone (FC) evaluations of potentially economic pit limits were conducted using the deposit model and mineral resource base described earlier in this section.

Table 19.7 summarizes the economic and recovery parameters that were incorporated into these analyses.

Table 19.7 Economic Parameters for Floating Cone Evaluations

NSR royalty	3 %
Smelting/refining realizations: Copper Molybdenum	 99 % 98 %
Freight, smelting & refining charges: Copper Molybdenum	 US$ 0.24/lb US$ 0.64/lb
Recoveries: Copper – transition (mixed) Copper – primary sulfide Molybdenum	 85 % 92 % 63 %
Base mining cost Incremental haulage below 5050 ft elev. (pit rim)	US$ 0.62/ton US$ 0.015/ton/bench
Sulfide ore processing cost Oxide ore acid consumption – QMP & Andesite Oxide ore acid consumption – Arkose Cost of acid for leaching (assumes on-site generation) Other oxide processing costs Oxide freight & refining costs	US$ 3.60/ton ore 60 lbs/ton ore 40 lbs/ton ore $ 0.02/lb US$ 0.25/ton ore US$ 0.14/lb Cu
General/administration cost	US$ 0.40/ton ore

The low cost of acid for use in oxide heap leaching is based on the generation of surplus acid from a concentrate leach circuit used to produce refined cathode copper on site. If on-site acid is not generated as a result of changes in the ore processing flowsheet, then the much higher cost of commercially acquired acid will significantly reduce, or possibly eliminate, potential oxide ore reserves.

Overall slope angles used in the FC studies varied by azimuth from the cone vertices and are summarized in Table 19.8. These angles make provisions for in-pit ramps, except for the west wall in the Martin formation, of which the pronounced ridge west of the deposit is composed. Overall slopes in this competent limestone unit were projected at a 52° interramp angle, as ramps are likely to be placed only on the north, east and south walls due to the anticipated open pit development sequence. The eastern walls, particularly the northeast quadrant (0-90° azimuth), were flattened to an overall angle of 38° where the pit wall will consists of mostly arkose – a weak rock type in the Rosemont pit area.

Table 19.8 Floating Cone Overall Slope Angles

Azimuth (degrees)	Slope Angle (degrees)
0	38
90	38
120	42
180	42
200	45
245	45
265	52
320	52
350	38
360	38

The above azimuths and slope angles were loaded into MEDSystem's floating cone program. When slope angles change between two bounding azimuths, the program linearly interpolates the slope angles for the interior azimuths, thus providing a gradual transition. No change in slope angles between bounding azimuths means a fixed slope angle throughout that azimuth range.

A total of eight FC runs were made to test some sensitivities to Cu and Mo prices and to help determine a possible open pit development sequence. As this study is a preliminary assessment consistent with Canadian NI 43-101 standards, inferred material was allowed to be considered as potential ore in these runs along with measured and indicated resources. The results of these floating cone analyses are presented in Table 19.9; however, the estimated potential ore tabulated therein should not be considered to be mineral reserves as defined in NI 43-101.

Table 19.9 Floating Cone Pit Limit Sensitivity Analyses

Metal Prices (US$/lb)		Sulfide Mill Ore (>= $4.00/ton internal NSR cutoff)					Oxide Leach Ore			Waste
					Contained lbs in millions				Contained Cu lbs in millions	
Cu	Mo	Ktons	% Cu	% Mo	Cu	Mo	Ktons	% Cu		Ktons
0.75	5.63	41,000	0.71	0.018	580	15	10,000	0.38	75	148,000
0.80	6.00	150,000	0.65	0.017	1,960	50	20,000	0.30	121	487,000
0.85	6.38	223,000	0.63	0.017	2,810	78	23,000	0.29	134	679,000
0.90	6.75	274,000	0.59	0.017	3,260	95	26,000	0.28	145	748,000
0.95	7.13	301,000	0.57	0.017	3,460	102	30,000	0.26	156	770,000
1.00	7.50	329,000	0.56	0.017	3,670	112	36,000	0.24	175	813,000
1.05	7.50	359,000	0.54	0.017	3,860	122	38,000	0.24	182	842,000
1.37	19.87	532,000	0.45	0.016	4,780	171	59,000	0.20	235	1,126,000

(Table 19.9 includes material classified as inferred. Oxide leach ore internal NSR cutoffs are US$2.35/ton for QMP and andesite and US$1.45/ton for arkose.)

The Washington Group preliminary assessment was initiated by Augusta following an independent resource assessment of Rosemont prepared by WLR Consulting (WLRC), dated April 21, 2006 (Reference 1). The reader is cautioned that this is a preliminary assessment as defined under NI 43-101. It is preliminary in nature and it is based on inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the preliminary assessment will be realized.

Contour plots of the FC shells indicate an initial phase, or starter pit, is approximated by the US$0.75/lb Cu FC shell and is located near the southwest corner of the US$1.37/lb Cu shell. Progressive floating cone expansions show that the pit grows mostly to the east and north.

Initially, seven conceptual mining phases were created to estimate potential ore for the Rosemont preliminary assessment. These fit within the FC pit shell defined by prices of US$1.37/lb Cu and US$19.87/lb Mo – the three-year trailing average prices for these metals. The last phase, however, contained about 77 million tons of potential ore (a significant portion of which is inferred) and 331 million tons of waste, for an incremental stripping ratio of 4.3:1 (tons waste per ton ore). In the interest of discounting some of the inferred material and making the estimate of potential ore a little more conservative, the seventh mining phase was dropped from further consideration in this study. After completion of in-fill drilling and the development of a new deposit model, the economics of the mineralization in this area will be re-evaluated.

The six remaining conceptual mining phases, or pushbacks, roughly approximate the US$1.05/lb Cu FC pit shell and were used to generate potential ore estimates and a preliminary mine production schedule. All of the conceptual phases are manually developed expansions from starter polygons and are not detailed pit designs. While some provisions for in-pit ramps are included in the projected slope angles, no road layouts were developed for any of the phases. The expansion polygons were somewhat smoothed to eliminate noses and notches, however, and incorporated minimum mining width considerations. Such conceptual phases are not sufficient for the definition of mineral reserves, but should be adequate for a scoping-level evaluation that is characteristic of a preliminary assessment.

The first phase (starter pit) will be located towards the southwest corner of the ultimate pit, leaving about a 200- to 250-ft-wide subsequent pushback (Phase 3) to the final limits along the west side. Phase 2 expands the starter pit to the east and north. Phase 3 further extends the pit to the east, south and pushes to the ultimate limits along the west wall. Phases 4, 5, and 6 progressively expand the pit to the north and east, following the orebody down its easterly dip.

Outlines of the conceptual mining phases are illustrated in Figure 19-6 (referenced in Section 26 of this report). At the rim, the pit will be about 6100 ft across north to south, 4600 ft across east to west, and will be about 1500 to 2400 ft deep (the pit bottom elevation is projected at 3350 to 3500 ft).

Table 19.10 summarizes the potential ore estimates by mining phase and includes material from all resource classifications (measured, indicated and inferred). NSR values were computed at prices of US$1.37/lb Cu and US$19.87/lb Mo, using the economic parameters listed in Table 19.7 to define potential ore. Sulfide mill ore was based on an internal NSR cutoff of US$4.00/ton and oxide internal cutoffs varied between US$1.45 and US$2.35/ton, depending on rock type. As inferred material is included in the tabulations, the estimated potential ore listed in Table 19.10 should not be considered to be mineral reserves as defined in Canadian NI 43-101.

Table 19.10 Potential Ore Reserves by Mining Phase

Conceptual Mining Phase	Sulfide Mill Ore (>= $4.00/ton internal NSR cutoff)					Oxide Leach Ore			Waste Ktons
	Ktons	% Cu	% Mo	Contained lbs in millions Cu	Contained lbs in millions Mo	Ktons	% Cu	Contained Cu lbs in millions	
1	43,500	0.52	0.016	460	14.8	24,000	0.20	95.9	68,500
2	45,200	0.46	0.014	410	13.0	14,000	0.19	53.3	49,800
3	96,500	0.41	0.012	800	24.1	12,000	0.20	47.1	167,500
4	95,700	0.44	0.013	850	24.5	5,000	0.20	19.6	150,300
5	76,300	0.51	0.018	770	26.7	3,000	0.21	12.3	142,700
6	74,300	0.51	0.018	760	27.3	1,000	0.28	5.6	180,700
Total	431,500	0.47	0.015	4,050	130.4	59,000	0.20	233.8	759,500

(Table 19.10 includes material classified as inferred. NSRs computed using US$1.37/lb Cu and US$19.87/lb Mo. Oxide leach ore internal NSR cutoffs are US$2.35/ton for QMP and andesite and US$1.45/ton for arkose.)

Total potential sulfide ore is estimated at about 432 million tons grading 0.47% Cu and 0.015% Mo and potential oxide ore is projected at about 59 million tons grading 0.20% Cu. This potential ore contains just over four billion pounds of copper and 130 million pounds of molybdenum. Waste rock is estimated at 759 million tons, for a sulfide stripping ratio of 1.9:1 (tons waste/oxide per ton of potential sulfide ore) and a total potential ore stripping ratio of 1.6:1 (tons waste per ton of potential sulfide/oxide ore).

Table 19.11 below summarizes the potential ore by rock type and also includes mineral resources classified as inferred. Intrusive rock types (quartz monzonite porphyry and andesite) comprise only five percent of the potential sulfide ore.

Table 19.11 Potential Ore By Rock Type

Rock/Formation Description	Ktons	% Cu	% Mo	% of Ore
Potential Sulfide Ore:				
Epitaph	2,500	0.32	0.009	0.6
Colina	83,600	0.65	0.017	19.4
Earp	72,200	0.30	0.016	16.7
Horquilla	200,400	0.52	0.016	46.5
Escabrosa	19,400	0.51	0.009	4.5
Martin	3,600	0.37	0.006	0.8
Quartz Monzonite Porphyry	10,300	0.25	0.016	2.4
Andesite	11,400	0.28	0.005	2.6
Arkose, Akcg	20,900	0.15	0.010	4.8
Limestone, Lscg	7,200	0.18	0.014	1.7
Total Sulfide Ore	431,500	0.47	0.015	100.0
Potential Oxide Ore:				
Quartz Monzonite Porphyry	9,100	0.20	n/a	15.4
Andesite	14,500	0.26	n/a	24.6
Arkose, Akcg	35,400	0.17	n/a	60.0
Total Oxide Ore	59,000	0.20	n/a	100.0

(Includes material classified as inferred.)

All of the potential ore estimates presented in Tables 19.10 and 19.11 are contained within the mineral resource estimates presented in Tables 19.5 and 19.6.

20. OTHER RELEVANT DATA AND INFORMATION

20.1 Process Plant Design

The results of the metallurgical testing program and knowledge of similar ore deposits in the district were used to develop the design criteria and process flow sheets for the Order of Magnitude Study prepared by the Winters Group in 1997. The process simplified flow sheet is included Section 26 of this report.

20.2 Process Description

The process flow sheet selected and plant design represents a conventional flotation concentrator. The process design for the Winters 1997 study was based on that of the Mission concentrator.

20.2.1 Primary Crushing

Primary crushing is achieved with a 60” x 110” gyratory crusher to produce an 8” feed for the grinding circuit.

20.2.2 Grinding

The grinding circuit consists of two 36.5' × 19' EGL semi-autogenous (SAG) mill followed by two 22' diameter by 36.5' ball mills in parallel. The SAG mill discharge passes over a trammel screen and the screen oversize (critical) reports to a pebble crusher before being returned to the SAG mill. Cyclone classification is employed to produce the required particle size distribution of 80% passing 145 microns and density for rougher flotation. A grinding circuit thickener has not been included in this circuit for density control and water recycle. The circuit is quite conventional and appropriate for this ore.

20.2.3 Flotation

The flotation circuit consists of a bank of 11, 4500 cubic foot rougher flotation cells. The concentrate from the first two rougher cells reports directly to a single stage cleaner circuit, consisting to two parallel trains of two 10.8 ft diameter by 38 ft tall column cells. The final cleaner concentrate cleaner cell reports to the concentrate thickener. The final concentrate is then filtered using a ceramic filter to produce a shippable concentrate containing 10 percent moisture.

The middling concentrate from the rougher cells is reground to 80% passing 45 microns in a Vertimill, before being pumped to the cleaner circuit.

The cleaner circuit tailing flows to a bank of eight, 1000 cubic foot scavenger cells. The scavenger concentrate is combined with the rougher middling concentrate in the regrind circuit before reporting to the cleaner cells.

20.2.4 Tailings

The rougher flotation tailing and the scavenger flotation tailing are combined in a tailings thickener for water recovery. The thickener underflow slurry is then pumped to a bank of connected large vacuum belt filters where moisture is reduced from 40% to 50% water to 10% to 15% moisture. The damp tailings are then transported to a combined waste rock and tailings disposal area.

20.2.5 Conclusion

This is a conventional flotation circuit plant design which is based on the results and recommendations from the metallurgical testing program and, the process facilities at the existing Mission Mine.

21. INTERPRETATION AND CONCLUSIONS

- Current information point to a straightforward mining operation with no unusual conditions or requirements pending highwall geotechnical analysis.

- Concentrator flow sheet appears to be viable with respect to sulfide mineralization.

- Heap leaching of oxide mineralization is a potential project upside that requires further evaluation.

- Preliminary laboratory investigations indicate that atmospheric leaching of concentrates may provide economic benefit to the project.

22. RECOMMENDATIONS

Based on the results of this PAEE, the recommendation from the team of qualified persons as well as the experienced persons is to proceed the next phase of the study, either prefeasibility or final feasibility. The following are also recommended:

- Perform geotechnical drilling and analysis to finalize wall slope angles.

- Complete infill drilling to convert inferred resources to indicated and measured resources.

- Update geologic resources and mineralogy, geologic model and mine block model as new information becomes available from ongoing and future drilling programs.

- Perform detailed mine planning and haulage analysis to finalize and optimize mine equipment selection as well as requirements.

- Conduct advanced metallurgical test to optimize processing equipment and plant performance. This should include the treatment of oxide ore to supplement sulfide milling and concentration.

- Conduct condemnation drilling for mill site and support facility locations.

- Complete silver, oxide and potentially gold assays to update mineral resources.

- Initiate acquisition of permits and licenses.

23. REFERENCES

1. Mineral Resources Estimate Revised Technical Report for the Rosemont Deposit Pima County, Arizona, USA,, April 21, 2006. Prepared by: WLR Consulting, Inc.

2. Mineral Resources Estimate, Technical Report for the Rosemont Deposit Pima County, Arizona, USA,, February 15, 2006. Prepared by: WLR Consulting, Inc.

3. Rosemont Project Validation Order-of-Magnitude Study Asarco Incorporated, October 1997. Prepared by Winters Company.

4. Rosemont Copper Project, Arizona Preliminary Development of Flotation Flowsheet Using Selected Composite Sample (Primarily Chalcocite Bornite Mineralization), Draft Report June 2006. Prepared by Mountain States R&D International, Inc.

24. DATE AND SIGNATURES

CERTIFICATE of PRINCIPLE AUTHOR

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

1. I am currently employed as Director of Engineering by:

   Washington Group International
   7800 E. Union Ave., Suite 100
   Denver, CO 80237
   USA

2. I graduated with a Bachelor of Science degree in Mining Engineering from New Mexico Tech in 1979 and a Masters of Science degree in Mining Engineering from University of California at Berkeley in 1981.

3. I am a:

   - Registered Professional Engineer in the State of Texas (No. 55901)
   - Registered Professional Engineer in the State of Montana (No. 8420E)
   - Registered Professional Engineer in the State of New Mexico (No. 14276)
   - Member of the Society for Mining, Metallurgy and Exploration, Inc.

4. I have worked as a mining engineer for 25 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 Principal Author and Qualified Person responsible for the overall preparation of the technical report titled *"Preliminary Assessment and Economic Evaluation for the Rosemont Deposit, Pima County, Arizona, USA"*, dated June 13, 2006, prepared for Augusta Resource Corporation (the "Technical Report") and, in particular, I prepared the mining of Sections 21 (Interpretation and Conclusions) Section 22 (Recommendations) and portions of Section 3 (Summary) of the Technical Report. I oversaw the preparation of Sections 18 (Mineral Processing and Metallurgical Testing), Section 20 (Other Relevant Data and Information), Section 25 (Additional Requirements for technical Reports on Development Properties and Production Properties), and Section 26 (Alternative Case).

7. I have not had prior involvement with the property that is the subject of the Technical Report. I have visited the subject property in May 18, 2006.

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 the 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 13th day of June, 2006.

[John I. Ajie] *[stamped]*

____

Signature of Qualified Person

John I. Ajie
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)
   - Member of the Society for Mining, Metallurgy and Exploration, Inc.

4. I have worked as a mining engineer for 29 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 estimation of potential ore (Subsection 19.12) and the mine production schedule (Subsection 25.1) presented in the technical report titled “*Preliminary Assessment and Economic Evaluation for the Rosemont Deposit, Pima County, Arizona, USA”*, dated June 13, 2006 (the “Technical Report”) relating to the Rosemont property and prepared for Augusta Resource Corporation.

7. I have had prior involvement with the property that is the subject of the Technical Report – specifically, I have prepared two previous technical reports on the subject property dated February 15, 2006 and April 21, 2006 (see references 1 and 2 in Section 23) for Augusta Resource Corporation. I have visited the subject property on August 9, 2005.

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 13th day of June 2006.

[William L. Rose] *[stamped]*

Signature of Qualified Person

William L. Rose
Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, James A. Sturgess, do hereby certify that:

1. I am currently employed as Vice President, Projects and Environment by:

   Augusta Resource Corporation

   4500 Cherry Creek South Drive

   Denver, Colorado 80246

   U.S.A.

2. I graduated with a Bachelor of Science degree in Renewable Natural Resources from the University of California at Davis in 1973 and I graduated with a Master of Science in Ecology from the University of California at Davis in 1976.

3. I am registered as a Certified Environmental Manager in the State of Nevada (No. 1601).

4. I have worked in Environmental Management for over 30 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, professional registration (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 portions of Section 6 (Property Description and Location) of the technical report titled *Preliminary Assessment and Economic Evaluation of the Rosemont Project, Pima County, Arizona, USA* and dated June 13, 2006 (the "Technical Report") relating to the Rosemont property.

7. I have had prior involvement with the property that is the subject of the Technical Report in my role as Senior Associate for Stantec Consulting Inc, which assisted in property due diligence for the party that sold the Rosemont Properties to Augusta, as well as in my role as Vice President Projects and Environment for Augusta, owner of the Rosemont Project. I have visited the subject property repeatedly since 2004.

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. As an officer of Augusta Resource Corporation, I am **not** independent of the issuer according to the guidelines set out 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 13th day of June 2006.

[James A. Sturgess] *[stamped]*

Signature of Qualified Person

James A. Sturgess
Print Name of Qualified Person

25. ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

25.1 Capital Costs

Washington Group developed the capital costs for the mine and processing plant while various consultants provided capital costs for infrastructure, power and water systems.

Washington Group's costs were developed from:

- In-house historical data of similar projects
- Limited budget pricing from vendors for specific major pieces of equipment

The mining equipment capital costs were based on recent new equipment quotes for the initial and replacement capital as well as from recent bid data. The processing plant cost was developed from a factored estimate using the mechanical equipment cost as the basis of the direct field cost. An analysis of the major flowsheet areas was conducted by Washington Group and, on the basis of our recent project experience, factors were applied for each major process facility. The total capital cost for the project is estimated at US$725 million for the Sulfide Concentrator Base Case with an accuracy of +/-40%.

Other Costs compiled in this report have been estimated by others as follows:

- Tailings – Vector Engineering
- Power Supply System – Navigant Consultants
- Water Supply System – Stantec

A summary estimate of the total project capital costs required for the Rosemont project are presented in Table 25.1 and are summarized by major categories. Capital cost estimates for the mine are based on purchasing new equipment and were derived from recent budget quotes and bids as well as factored from similar projects.

Table 25.1 Total Project Capital Costs

Description	Costs, $US M
Direct Costs	
Water Supply System	21
Power Supply System	18
Concentrator and Process Facility	359
Mining Equipment Capital and shop facilities	156
Subtotal Directs	**554**
Contingency @ 15%	**83**
Indirect Costs	
Working Capital, Spare Parts and First Fills	45
Reclamation Guarantee and Bonding	16
Subtotal Indirects	**61**
Sustaining Capital	**28**
TOTAL CAPITAL COSTS	**725**

25.1.1 Mining Capital Cost Estimate

As presently envisioned, open pit mining will be conducted from 50-ft benches using large-scale equipment, including: 12.25-in.-diameter blasthole drills, 50- to 60-cubic-yard (cy) electric rope shovels and 250- to 355-ton off-highway haulage trucks. Interramp slope angles will vary according to rock strength, lithology and structural controls, but are expected to range between 40° and 52°. Where possible, catch benches will be spaced on 100-ft vertical intervals to maximize the effective widths for containing scree.

A. Mine Production Schedule

Sulfide milling is scheduled for 24 hours per day, seven days per week, 360 days per year, at an ore processing rate of 75,000 tons per day (tpd), or 27 million tons per annum. The open pit mine will operate using the same schedule. A provision of five days per year has been made for mill maintenance and weather delays in pit operations. The mine will use four rotating crews, each working 12-hour shifts, to provide continuous operator coverage.

A mine production schedule was developed using the potential ore and waste estimates by bench for each of the six mining phases (see Section 19.12 of this report) and a proprietary scheduling program that simulates open pit mining for a given mill feed rate and cut-off grade policy. The program analyzes preproduction and advancing stripping requirements to maintain continuous ore exposure in the pit, while smoothing total production requirements. Phases are "mined" in sequence from the top bench downward, controlled by sinking rate limitations and geometric constraints (i.e., no undercutting previous pushbacks).

The resulting mine production schedule is presented in Table 25.2. NSR values were computed at prices of US$1.37/lb Cu and US$19.87/lb Mo, using the economic parameters listed in Table 19.7 of this report to define potential ore. Sulfide mill ore was based on an internal NSR cutoff of US$4.00/ton and oxide internal cutoffs varied between US$1.45 and US$2.35/ton, depending on rock type. As inferred material is included in the production schedule tabulations, the estimated potential ore listed in Table 25.1 should not be considered to be mineral reserves as defined in Canadian NI 43-101.

The mill ore tonnages for Year 1 exclude 2,500 ktons of rehandling sulfide ore that would be stockpiled during the preproduction stripping period. The remaining 600 ktons of sulfide ore stockpiled in preproduction represents an average run-of-mine and crushed ore stockpile inventory that will be continuously intermingled with new material and reclaimed over the life of the mine.

Preproduction stripping will require 15 to 21 months for phasing in mine operations, training work crews, constructing access/haul roads, and clearing and grubbing the pit and waste rock storage areas that will be disturbed during the initial years of operation. Peak material handling rates will occur in Years 1-4, averaging about 245,000 tpd of total material before falling off to nearly 225,000 tpd for most of the remaining years. The mine's life is currently projected at 16 years of operation.

Table 25.2 Mine Production Schedule – Potential Ore by Time Period

Time Period	Sulfide Mill Ore (>= $4.00/ton internal NSR cutoff) Ktons	% Cu	% Mo	Contained lbs in millions Cu	Mo	Oxide Leach Ore Ktons	% Cu	Contained Cu lbs in millions	Waste Ktons	Total Ktons	Sulf Ore Strip Ratio	Total Ore Strip Ratio
Preprod	3,100	0.28	0.014	20	0.9	13,500	0.22	58.9	44,500	61,100	18.71	2.68
1	24,500	0.42	0.014	210	7.2	14,600	0.21	61.2	46,500	85,600	2.49	1.19
2	27,000	0.55	0.016	300	8.8	10,300	0.16	33.5	50,800	88,100	2.26	1.36
3	27,000	0.40	0.015	210	7.9	6,200	0.22	26.7	54,900	88,100	2.26	1.65
4	27,000	0.42	0.012	230	6.7	2,500	0.22	10.9	58,600	88,100	2.26	1.99
5	27,000	0.42	0.012	230	6.4	4,400	0.16	14.0	49,300	80,700	1.99	1.57
6	27,000	0.43	0.016	230	8.7	900	0.23	4.2	52,800	80,700	1.99	1.89
7	27,000	0.43	0.012	230	6.4	500	0.18	1.8	53,200	80,700	1.99	1.93
8	27,000	0.43	0.011	230	6.0	2,000	0.15	6.1	51,700	80,700	1.99	1.78
9	27,000	0.49	0.014	260	7.5	400	0.17	1.4	53,300	80,700	1.99	1.95
10	27,000	0.41	0.013	220	7.3	700	0.12	1.6	53,000	80,700	1.99	1.91
11	27,000	0.49	0.014	270	7.4	1,900	0.24	8.9	51,800	80,700	1.99	1.79
12	27,000	0.51	0.017	280	9.4				53,700	80,700	1.99	1.99
13	27,000	0.56	0.017	300	9.4	500	0.11	1.0	53,200	80,700	1.99	1.93
14	27,000	0.41	0.016	220	8.7	600	0.29	3.6	29,200	56,800	1.10	1.06
15	27,000	0.66	0.020	360	10.6				600	27,600	0.02	0.02
16	25,900	0.48	0.022	250	11.3				2,400	28,300	0.09	0.09
Total	431,500	0.47	0.015	4,050	130.4	59,000	0.20	233.8	759,500	1,250,000	1.90	1.55

Disclosure Notice: The above estimates are for an economic assessment that is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that this preliminary assessment will be realized.

(Includes material classified as inferred. NSRs computed using US$1.37/lb Cu and US$19.87/lb Mo. Oxide leach ore internal NSR cutoffs are US$2.35/ton for QMP and andesite and US$1.45/ton for arkose.)

B. Mine Equipment

Haulage profiles were estimated by WLRC for each year in the mine production schedule. Typical profiles were measured for sulfide ore, oxide ore and waste rock. Where production was evenly split between pushbacks for a given ore/waste designation, profiles were measured by pushback as well. Outbound (from loading to dumping points), one-way sulfide ore profiles ranged between 6700 ft in Year 1 to over 16,000 ft in Year 16. Similarly, one-way waste profiles varied between 6300 ft and 24,400 ft.

Using the haulage profiles provided by WLRC and the mine production schedule listed in Table 25.2, Washington Group estimated haul cycle times and selected 355-ton class off-highway haulage trucks for this project. The appropriate loading unit for this size truck is the 50- to 60-cy electric loading shovels. A large front-end loader, 24-cy-class, was added to support the electric shovels and provide flexibility in mobility. Truck requirements fluctuate according to variable haul distances and volume of material handled, but average 15 units for majority of the mine life.

The mine schedule from preproduction through year-16 was assumed to be 24 hours per day, seven days per week, 360 days per year and requires four crews working two 12-hour shifts per day. This schedule and the corresponding productivities of the hauling and loading machines are the basis for the major equipment units presented in Table 25.3 below, as well as estimated unit costs.

Table 25.3 Major Mining Equipment

Equipment	Average No. of Units	Costs, US$ M
Blasthole Drills, 12.25-in.	3	7
Electric Shovels, 60-cy	3	46
Front-End Loader, 24-cy	1	3
Off-Highway Haul Trucks, 355-ton	15	67.5
Crawler Dozers, 850-hp	3	2.5
Crawler Dozers, 570-hp	2	2
Wheel Dozers, 620-hp	2	2
Motor Graders, 500-hp	2	5
Motor Grader, 265-hp	1	1
Water Truck, 30,000-gallon	2	4
Water Trucks, 10,000-gallon	1	1
Admin Bldg, Truck Shop, etc.	1	8
Misc. Capital	1	7
Total Mine Capital		**156**

The drills were selected to match 50-ft bench operation, drill 12.25-in.-diameter holes or larger and meet the production requirement of over 80 million tons per year. Other support equipment, namely dozers, motor graders, rubber tired dozers and water trucks, were selected accordingly to match the size of the operation, as well as the loading and hauling units. Equipment selection is preliminary at this time and requires detailed loader and truck match analysis to determine their final productivities.

25.1.2 Mill Capital Cost Estimate

A detailed breakdown of the capital cost components for the mill are presented below:

A. Plant Equipment

The numbers and type of equipment were based on the process flowsheet. Cost was obtained from vendor's budgetary quotations. For the alternate process plant equipment, cost was derived from historical data of similar installations factored to the size of each individual circuit.

B. Direct Construction Accounts

Includes costs allocated directly to the facility. They include materials delivered to the plant-site and direct craft labor.

C. Installation

Includes labor necessary to install plant equipment

D. Piping

Includes estimated cost of all pipe, valves, fittings, pipe hangers, and supports required for the operation of the facility.

E. Electrical

Includes the estimated material and direct labor costs for the installation of the electrical system beginning with the wiring on the substation side of the disconnect switches on the powerline. Includes the load center, the motor control center power control and wiring

F. Instrumentation

Includes the estimated cost of process instrumentation and controls as required for the indication of liquid flows, pH, and tank levels. This is based on indicating instruments with controls to be performed manually.

G. Structures

Buildings include prefab modular structures erected in place. The process structures (pipe racks, platforms, and walkways) base on medium to light steel completed in shop, painted with touch up as required following erection.

H. Indirect Accounts

Include construction and engineering costs which cannot be allocated directly to the facility. They include man-hours of labor estimated to complete the project.

I. Engineering

Includes engineering costs, design, drafting, material take-off, specification writing, project management, project engineering, procurement, clerical services, data processing, reproduction services and communications. Geotechnical and environmental work is not included in this account.

J. Construction Expense

Includes contractor's field expenses, payroll taxes and insurance, contractor's fee, construction supplies, materials sales tax, temporary facilities and construction equipment.

TABLE 25.4 Mill Capital

Description	Cost (US$, million)
Process Equipment	100
Installation Labor	46
Concrete	21
Piping	28
Structural Steel	28
Instrumentation	10
Insulation	0.5
Electrical	27
Coatings & Sealants	3
Mill Building	15
Total Capital Cost Concentrator	**278.5**
Capital Cost Tailings Filters (Inc Structure)	43
Capital Cost Moly Plant	18
Light Vehicle, Cranes, etc	5
Engineering/Management	15
Total CapEx Processing Facilities	**359.5**

25.2 Operating Costs

The operating costs have been estimated for a plant that processes 75,000 tons of ore or 1,000 tons of concentrate per day, 360 days per year, or a total of 27 million tons of ore per year. Washington Group prepared the mine and processing operating costs for this project. The total operating cost is presented in Table 25.5 by costs centers.

Table 25.5 Unit Operating Cost Summary – Life of Mine Average

Area	Cost US$/Ton-ore
Mining	1.88
Processing	2.51
Admin. Labor	0.10
G&A	0.13
Total, US$/ton	**4.65**

25.2.1 Mining Operating Costs

Mining operating costs were developed from multiple sources; wage and salary surveys, typical benefit packages, current material and supplies cost, unit consumption rates as well as our equipment operating cost experience from other projects. Fuel cost for this estimate is US$2.50 per gallon. All costs have an accuracy of +/- 40%.

Mine Manpower Requirements

This is a large operation with respect to volume; however, it is concentrated in one mine area that grows in surface area and depth as time progresses. Therefore, the number of mine administrative personnel were determined accordingly. The numbers presented in Table 25.6 cover the total number of personnel that can be directly attributable to the mine regardless of work location (i.e., plant or mine).

Table 25.6 Mine Supervision and Technical Personnel

Area	Number
Mine Manager	1
Production	7
Maintenance	10
Safety	1
Business	11
Engineering	9
Dispatch Operators	4
Total	43

The mine craft personnel requirements, presented in Table 25.7 below, were based on the number of major equipment operating units and the mine work schedule.

Table 25.7 Mine Operations and Maintenance Craft Personnel

Area	No. per shift	Total
Production	30	120
Maintenance	15	60
Warehouse	1	4
Total	46	184

This area has a long history of mining and the nearest large town, Tucson, has significant infrastructure as well as the presence of at least two large equipment manufacturers that could support this operation. The mining personnel presented in this section are preliminary at this time and require final equipment selection and matching to project requirements to determine the final personnel requirements.

25.2.2 Process Operating Costs

This section deals only with those operating costs directly incurred in the processing operations and includes the cost of all raw materials, supervision, labor, utilities and supplies required for the operation. Total processing costs are estimated to be US$2.51 per ton of ore. Processing costs are summarized below in Table 25.8.

Table 25.8 Process Operating Costs (US$)

Description	Unit Cost, US$/t	Contribution,%
Manpower	0.20	7.89
Grinding Media	0.87	34.54
Reagents	0.18	7.19
Maintenance Supplies and Fuel	0.18	7.19
Power	1.08	43.20
TOTAL	**2.51**	**100.00**

A. Supervision and Labor

The plant will operate two twelve hour shifts/day, seven days/week, and 365 days/year.

The manpower required for the operation of the concentrator facility is 86 persons, including 9 salaried and 76 hourly waged employees. Wage rates were chosen from local market investigations and range from US$16 to US$22 per hour for operations personnel. Maintenance and Electrician crafts are higher, ranging between US$19 and US$30 per hour. Manpower costs for the concentrator are estimated at US$0.20 per ton. Proposed staffing for the concentrator is summarized in table 25.9 below.

Table 25.9 Concentrator Manpower Costs (US$)

Description	Base Salary ($/yr)	Unsch. o/t ($/a) 8%	Burden ($/yr) 33%	Loaded Salary ($/yr)	Number	Cost ($/yr)	Unit ($/t)
Operations - Salary							
Process Superintendent	130,000		42,900	172,900	1	172,900	0.01
Chief Metallurgist	77,000		25,410	102,410	1	102,410	0.00
Metallurgist	66,000		21,780	87,780	1	87,780	0.00
Mill Clerk	38,500		12,705	51,205	1	51,205	0.00
General foreman	71,500		23,595	95,095	1	95,095	0.00
Metallurgical Technician	40,000		13,200	53,200	1	53,200	0.00
Operations - Hourly							
Shift foreman	62,400	4,992	20,592	87,984	4	351,936	0.01
Crusher Operator	41,600	3,328	13,728	58,656	4	234,624	0.01
Crusher Helper	37,440	2,995	12,355	52,790	6	316,742	0.01
Control room operator	45,760	3,661	15,101	64,522	4	258,086	0.01
Grind operator	41,600	3,328	13,728	58,656	4	234,624	0.01
Mill Helper	35,360	2,829	11,669	49,858	4	199,430	0.01
Copper Circuit Operator	39,520	3,162	13,042	55,723	4	222,893	0.01
Moly Circuit Operator	39,520	3,162	13,042	55,723	4	222,893	0.01
Flotation/reagents	37,440	2,995	12,355	52,790	8	422,323	0.02
Filtration/tailings	37,440	2,995	12,355	52,790	12	633,485	0.02
Maintenance - Salary							
General foreman maintenance	71,500		23,595	95,095	1	95,095	0.00
Electrical foreman	71,500		23,595	95,095	1	95,095	0.00
Maintenance planer	60,500		19,965	80,465	1	80,465	0.00
Maintenance - Hourly							
Electrician	62,400	4,992	20,592	87,984	8	703,872	0.03
Oilers	39,520	3,162	13,042	55,723	2	111,446	0.00
Craftsman	49,920	3,994	16,474	70,387	5	351,936	0.01
Journeyman	45,760	3,661	15,101	64,522	10	645,216	0.02
Apprentice	41,600	3,328	13,728	58,656	4	234,624	0.01
Total					**86**	**5,414,786**	**0.20**

B. Utilities

Electric power charges were derived from the installed operating horsepower with a suitable allowance in the unit cost for peak demand charges. Average power supply costs are estimated at US$0.047 per kilowatt hour. Estimated power for each area of the concentrator is given below in Table 25.10. Estimated mine power consumption is 19 million kWh per year.

Table 25.10 Power Required by Area

Area	Installed Horsepower	Power Utilization, (hp)	Distribution %	Power Utilization (hp) Less Spares	Distribution %
Mining	-	-	0.0	-	0.0
Grinding	81,500	65,200	66.5	65,200	66.5
Tailings Dewatering	13,875	11,100	11.3	11,100	11.3
Concentrate Regrind	12,845	10,276	10.5	10,276	10.5
Flotation	6,868	5,494	5.6	5,494	5.6
Air/Misc.	3,100	2,480	2.5	2,480	2.5
Concentrate Dewatering	2,450	1,960	2.0	1,960	2.0
Crushing	1,350	810	0.8	810	0.8
Reagents	845	676	0.7	676	0.7
TOTAL	**122,833**	**97,996**	**100.0**	**97,996**	**100.0**

Table 25.11 Mineral Processing Costs, US$

Description		Installed	Operating
Operating Power		109,988	87,720
Annual kWh		718,767,180	573,249,305
Power Costs, $/kWh		0.047	0.047
Annual Power Cost		$33,782,057	$26,942,717
Equipment Allowance		10%	10%
Estimated Annual Power Cost		$37,160,263	$29,636,989
Power Costs, $/ton		$1.36	$1.08

C. Materials and Supplies

Annual operating supply consumptions were estimated using laboratory test data, the preliminary mass balance and requirements from other local operations. Maintenance supplies are based on maintenance labor costs. Material prices were either established from quotations by suppliers or furnished by historical data. The consumptions were compared to information and experience from similar existing operations. Grinding media and mill liners are the largest component of the materials cost at US$ 23.7 million per year or US$0.87 per ton of ore.

Table 25.12 Concentrator Operating Supplies (US$)

Item	Total Cost, (US$/year)	Unit Cost, (US$/t)
Grinding Media	23,694,585	0.87
Reagents	4,929,916	0.18
Maintenance & Fuel	7,387,666	0.27
TOTAL SUPPLIES	**36,012,166**	**1.32**

25.2.3 Administration Labor

Administrative labor costs were estimated by Washington Group and given in Table 25.13.

Table 25.13 Administrative Labor Costs

Description	Costs, US$/yr
Administration	654,360
Information Systems	146,300
Materials Management	449,540
Human Resources	142,210
Safety	402,990
Laboratory	299,250
Environmental	239,400
Corporate	239,400
TOTAL	**2,573,550**

25.3 G&A Operating Costs

Table 25.14 General and Administrative Direct Costs

Description	Annual Cost, US$
Environmental Monitoring	610,000
Road Maintenance: Repairs	600,000
Water Supply Operations and Maintenance	510,000
Crew Transport Costs - Bus	250,000
Powerline Maintenance	215,100
Road Maintenance	200,000
Insurance	175,000
Recruiting/Relocation	125,000
Outside Laboratories	100,000
Legal Fees	80,000
Regulatory Compliance	75,000
Small Vehicles	70,000
Mobile Equipment Rentals	60,000
Safety Training Supplies	56,000
Business Travel	50,000
Communications	50,000
Consultants	50,000
Community Relations and Donations	50,000
Laboratory Supplies	36,000
Potable Water Supply	30,000
First Aid Supplies	25,000
Janitorial Services	12,000
Total	**$3,429,100**

25.4 Economic Evaluation

25.4.1 Sulfide Concentrate Production (Base Case)

A financial model was created utilizing the WLRC mine production schedule, the associated diluted metal grades based on the Augusta geological resource, metal recoveries from the Phase I test metallurgical program, capital and operating costs as set out herein and base case metal prices of copper US$1.20/lb, molybdenum US$10.00/lb and silver US$7.50/oz.

The Base Case is for annual mine production to deliver 27.0 million dry tons of run-of mine ore per year to the Cu sulfide process facility; with average annual metal production of 226 million lbs of copper, 5.1 million lbs of molybdenum and 6.7 million ounces of silver. Initial (2008-2011) direct capital for mine, processing, water, power and 15% contingency is estimated at US$636 million. Initial indirect capital for reclamation bonding and working capital is estimated at US$61 million. Sustaining capital is estimated at US$28 million. Project net cash costs after by-product credits are estimated to average US$0.42/lb. Cu over project life.

All figures are stated in United States dollars. Modeling at base case metal prices shows that the project could generate a cumulative net after tax profit of US$1,472 million, a 17% IRR, and a net present value discounted at 8% of US$442 million, over the projected phase one through phase six 16 year mine life.

Other Key Assumptions and Inputs:

General Economics	No price or cost escalation.
	Un-leveraged economics.
	Net cash flow after federal, state and local taxes. The Alternative Minimum Tax (AMT) was not calculated.
Mining	Average Mining Rate = 78 million tons/year or 214,000 tons/day.
	Mining average unit cost = US$.74/ton, includes mine G&A
Sulfide Processing	Sulfide ore processing (milling) rate = 75,000 tons/day
	Sulfide ore average processing unit cost = US$2.50/ton, includes processing G&A
Copper	Cu sulfide grade average = 0.45%
	Cu sulfide recovery = 89%
	Cu concentrate grade = 33%
	Cu concentrate loading and freight to smelter within 100 mile radius = US$10.00/ton

	Cu smelting = US$80.00/ton
	Cu refining = US$.10/lb Cu
	No Cu oxide recovery. Capital and operating cost to recover the Cu oxide ore is not included.
Molybdenum	Mo average grade = .016%
	Mo recovery = 63%
	Mo concentrate grade = 56%
	Mo concentrate loading and freight to roaster = US$31.75/ton
	Mo Leach/Roast = US$0.60/lb Mo
Gold	No Au recovery. Capital and operating cost to recover Au is not included.
Silver	Ag average grade = 9 grams/ton
	Ag recovery = 85%
	Ag refining = US$0.25/oz
Other	Plant-Wide General & Administrative expense = US$6.3 M/yr
	Royalty = 3.00% of net smelter return
	Working Capital = 45 days receivables less 30 days accounts payable less 15 days payroll payable plus 1% of Net PP&E for consumables and 5 days of product inventories. First year full production (2010) working capital = US$44.8 million.

In addition, sensitivity analysis was conducted at standard Trailing and Forward Pricing Cases.

Table 25.15 Primary Base Cases (US$)

Case	Cu Price $/lb	Mo Price $/lb	Ag Price $/oz	IRR %	NPV 5% US$ millions	NPV 8% US$ millions	NPV 10% US $ millions	Net Cost $/lb
Base	1.20	10.00	7.50	17	715	442	306	0.42
Trailing	1.50	20.00	7.50	28	1,481	1,044	827	0.20
Forward	2.14	20.00	7.50	40	2,410	1,774	1,456	0.21

- Trailing Case Prices, as of 5/31/06:
  - Cu - average of 3 years trailing LME cash official prices
  - Mo - average of 3 years trailing price data for molybdenum oxide, F.O.B. North America.
  - Ag – constant US$7.50/oz

- Forward Case Prices, as of 5/31/06:
  - Cu - average of the LME 27 month official forward price and 3 years of trailing LME official cash prices.
  - Mo – same as Trailing Case Prices
  - Ag – same as Trailing Case Prices

The Trailing Price case increases the IRR to 28%, the NPV at 8% to US$1,044 million and decreases the average net cash cost to US$.20lb.

The Forward Price case increases the IRR to 40%, the NPV at 8% to US$1,774 million and decreases the average net cash cost to US$.21/lb.

The potential revenue stream from molybdenum and silver (based upon contained silver metal content); at base-case metal prices, generates by-product revenue of US$.45/lb Cu and covers 52% of copper cash operating costs per pound of copper produced.

The following table provides IRR and NPV sensitivities over a range of Cu and Mo metal prices.

Table 25.16 Additional Price Sensitivities to Base Case

IRR% vs. Copper and Molybdenum Prices (US$)

$1.08/lb Cu	15	16	18	19	20
$1.28/lb Cu	20	21	22	23	24
$1.50/lb Cu	24	25	26	27	29
$1.73/lb Cu	29	30	31	32	33
$1.98/lb Cu	33	34	35	36	37
	$11.56/lb Mo	$13.60/lb Mo	$16.00/lb Mo	$18.40/lb Mo	$21.16/ lb Mo

$ Millions NPV 8% vs. Copper and Molybdenum Prices (US$)

$1.08/lb Cu	345	400	464	526	598
$1.28/lb Cu	569	622	684	746	817
$1.50/lb Cu	827	880	941	1,003	1,074
$1.73/lb Cu	1,085	1,137	1,199	1,260	1,331
$1.98/lb Cu	1,380	1,432	1,494	1,555	1,626
	$11.56/lb Mo	$13.60/lb Mo	$16.00/lb Mo	$18.40/lb Mo	$21.16/ lb Mo

The following table demonstrates the project's sensitivity to increases and decreases in operating costs and capital costs. Note that a 10% increase in operating costs decreases the Base Case IRR by 1% to 16% and decreases the Base Case NPV at 8% by US$73 million to US$369 million. Similarly, a 10% increase in capital costs decreases the Base Case IRR by 1% to 16% and decreases the Base Case NPV 8% by US$48 million to US$394 million.

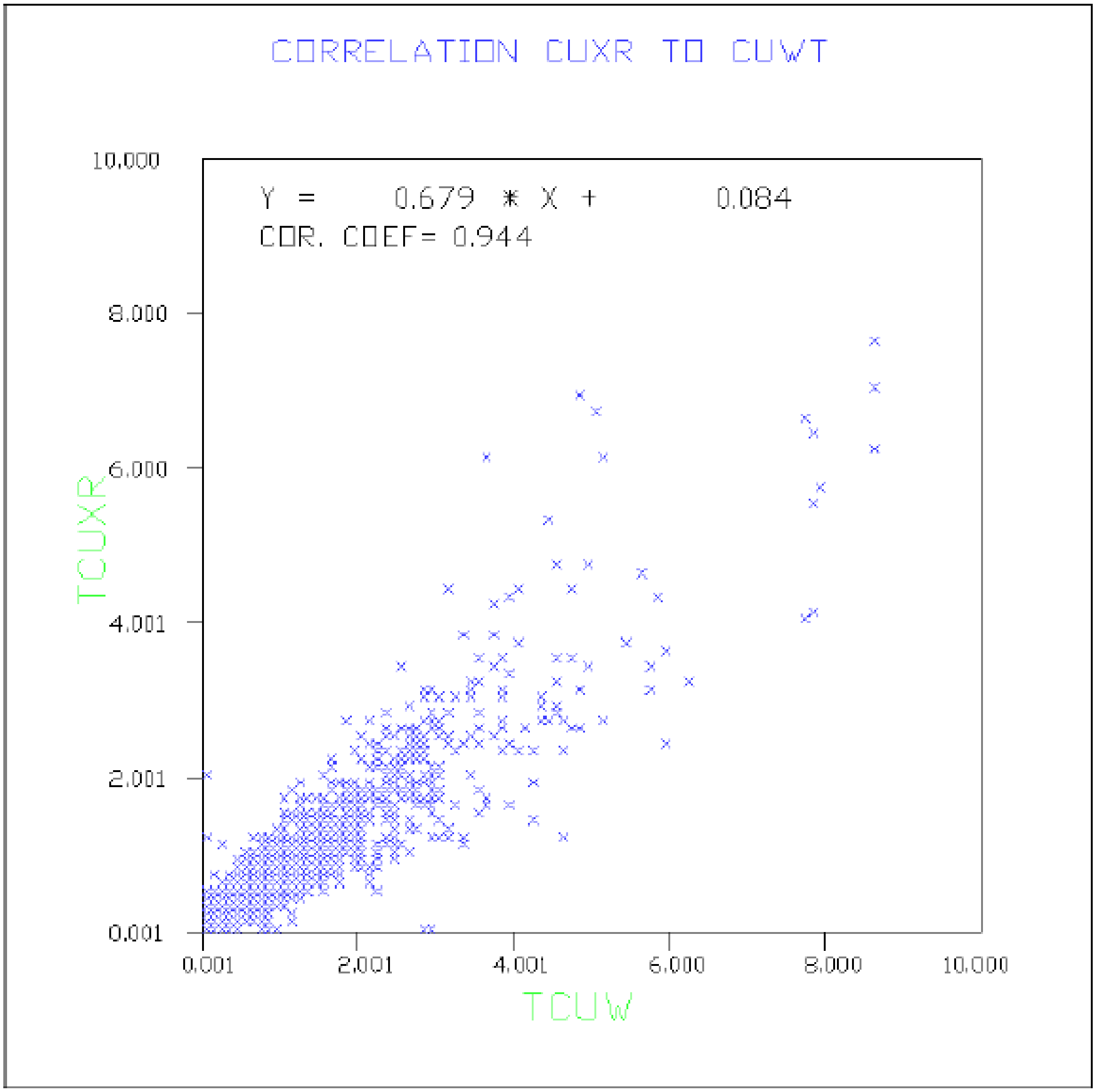
Table 25.17 Operating and Capital Cost Sensitivities to Base Case (US$)

	IRR%	**NPV 5%, $ million**	**NPV 8%, $ million**	**NPV 10%, $ million**	**Net Cash Cost, $/lb**
Base Case: $1.20 Cu, $10.00 Mo, $7.50 Ag	17	715	442	306	0.42
Cost Sensitivity Cases:					
Operating Cost +10%	16	626	369	241	0.48
Operating Cost -10%	19	802	512	368	0.36
Capital Cost +10%	16	669	394	258	0.42
Capital Cost -10%	19	760	488	353	0.42

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

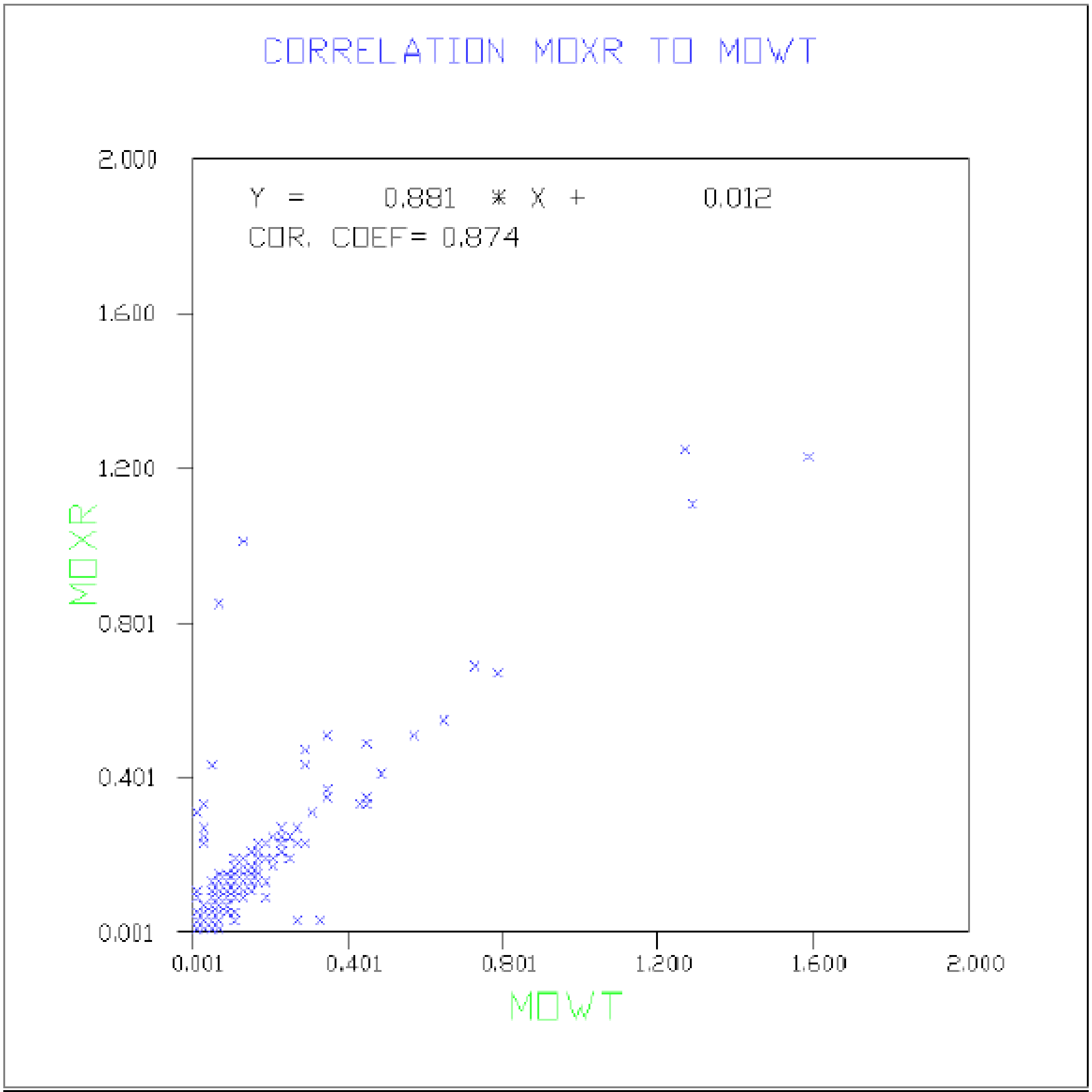
26. ILLUSTRATIONS

Figure 19-1 XRF – Wet Assay Correlation Plot for Cu



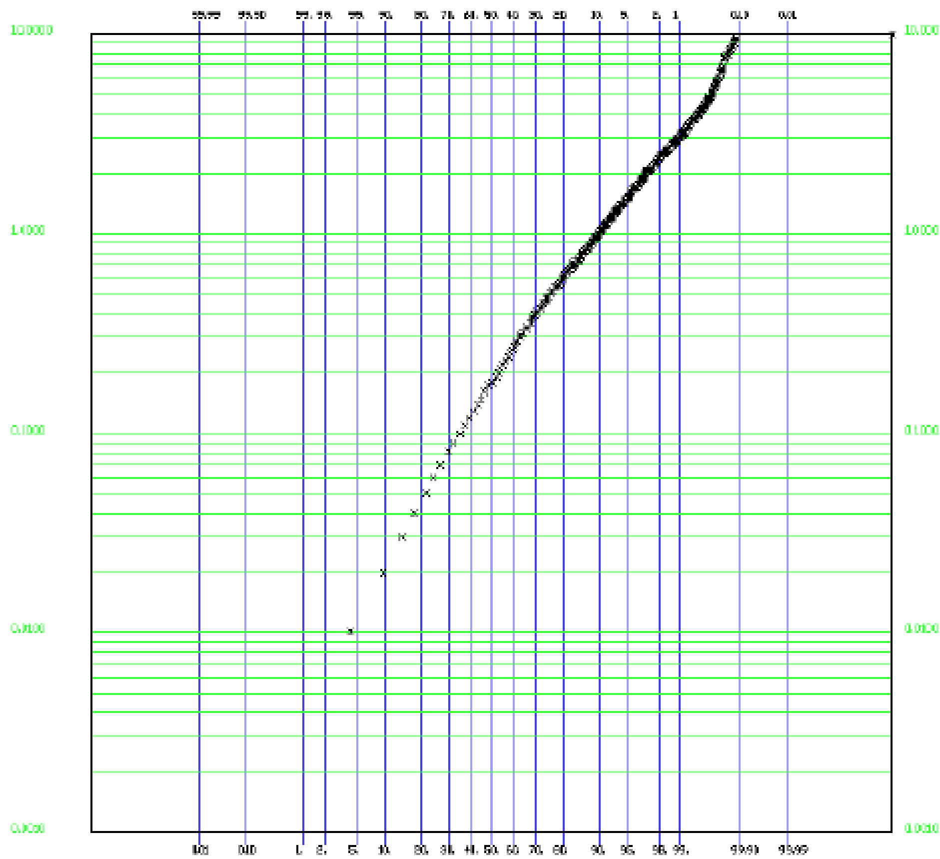
X- MEAN=	0.492	MIN.=	0.000	NO. =	9009
STD DEV=	0.926	MAX.=	24.700	C.V.=	1.882
Y- MEAN=	0.417	MIN.=	0.000	NO. =	9009
STD DEV=	0.665	MAX.=	14.700	C.V.=	1.594

Figure 19-2 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 19-3 Lognormal Cumulative Probability Plot for Cu

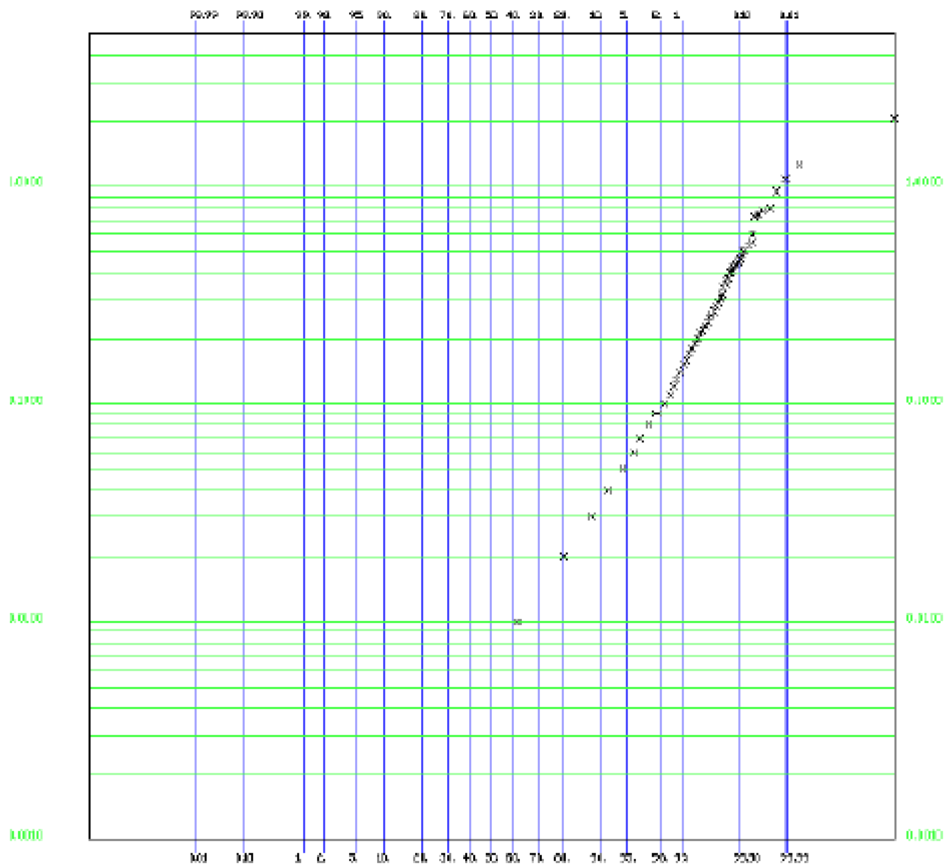


** PROBABILITY DISTRIBUTION PLOT OF TCU **

ITEM	TCU	NATURAL	
NUMBER	30702	NUMBER	30702
MEAN	0.418	MEAN	-1.729
MINIMUM	0.010	MINIMUM	-4.605
MAXIMUM	10.000	MAXIMUM	2.303
VARIANCE	0.504	VARIANCE	1.887
ST.DEV.	0.710	ST.DEV.	1.374

TOTAL DEPOSIT - TCU VALUES

Figure 19-4 Lognormal Cumulative Probability Plot for Mo



** PROBABILITY DISTRIBUTION PLOT OF MO **

ITEM	MO	NATURAL	
NUMBER	17552	NUMBER	17552
MEAN	0.022	MEAN	-4.173
MINIMUM	0.010	MINIMUM	-4.605
MAXIMUM	2.040	MAXIMUM	0.713
VARIANCE	0.002	VARIANCE	0.441
ST.DEV.	0.039	ST.DEV.	0.664

TOTAL DEPOSIT - MO VALUES

Figure 19-5 Variogram of Cu 50 – Ft Composited Values

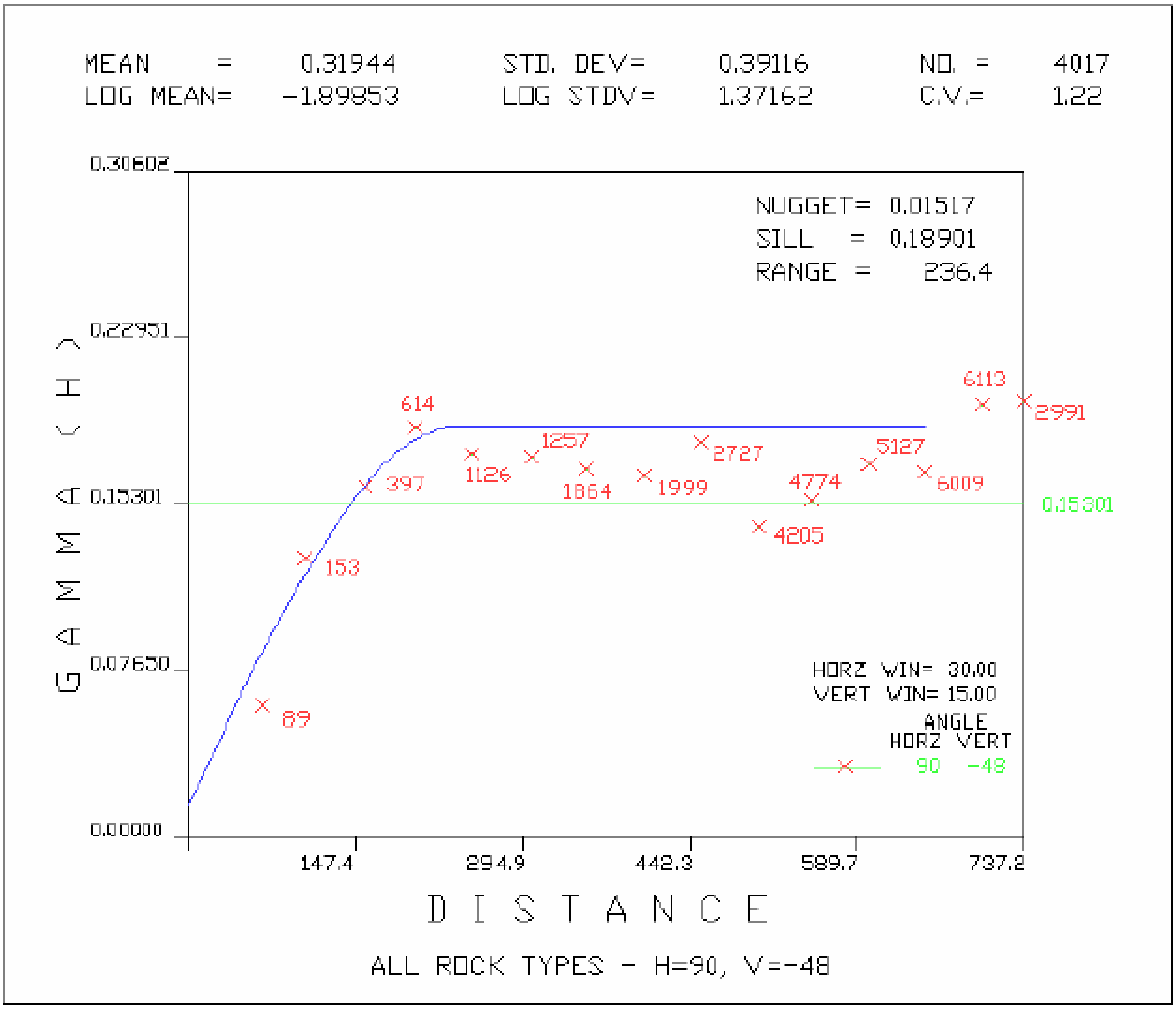
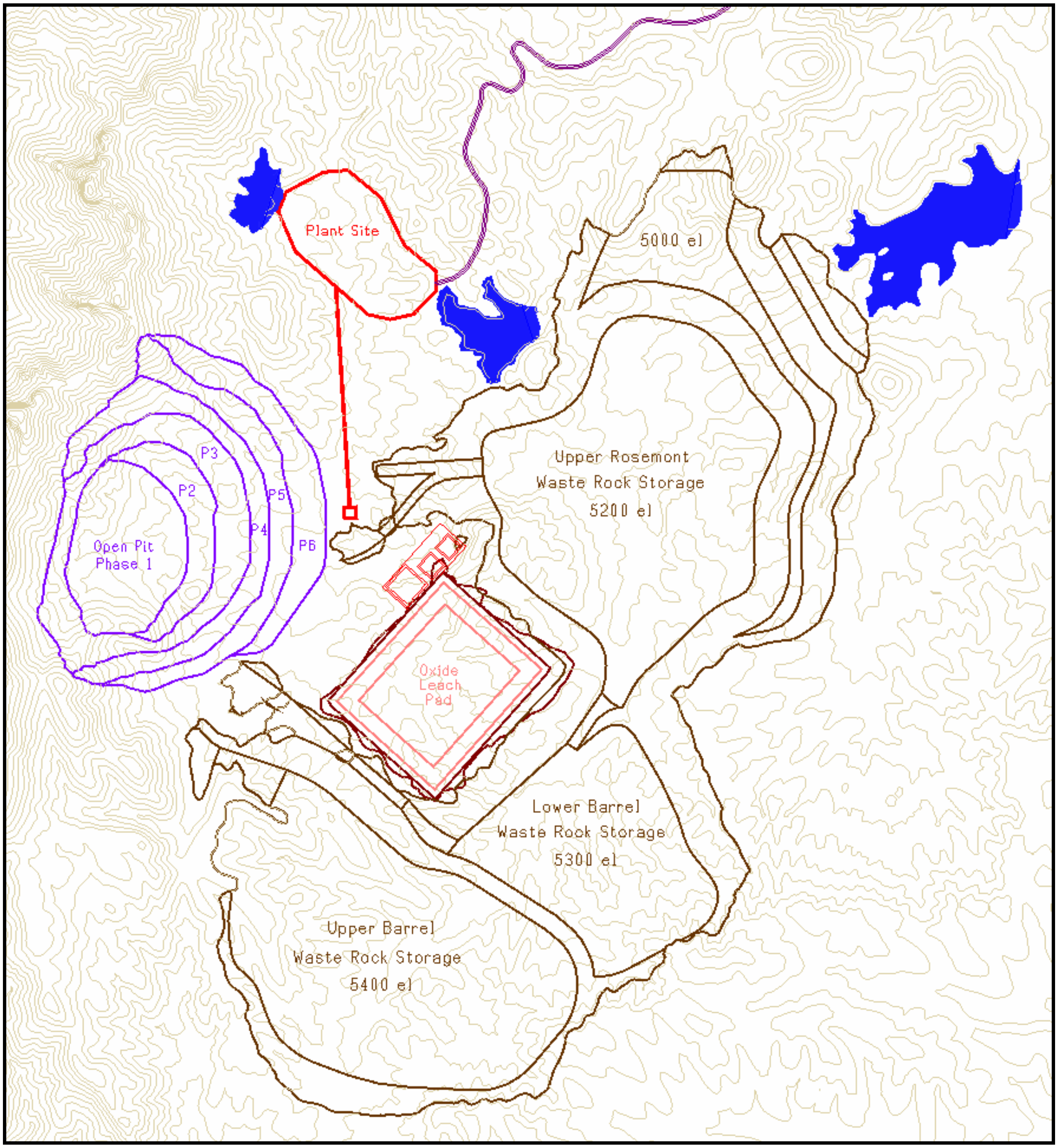
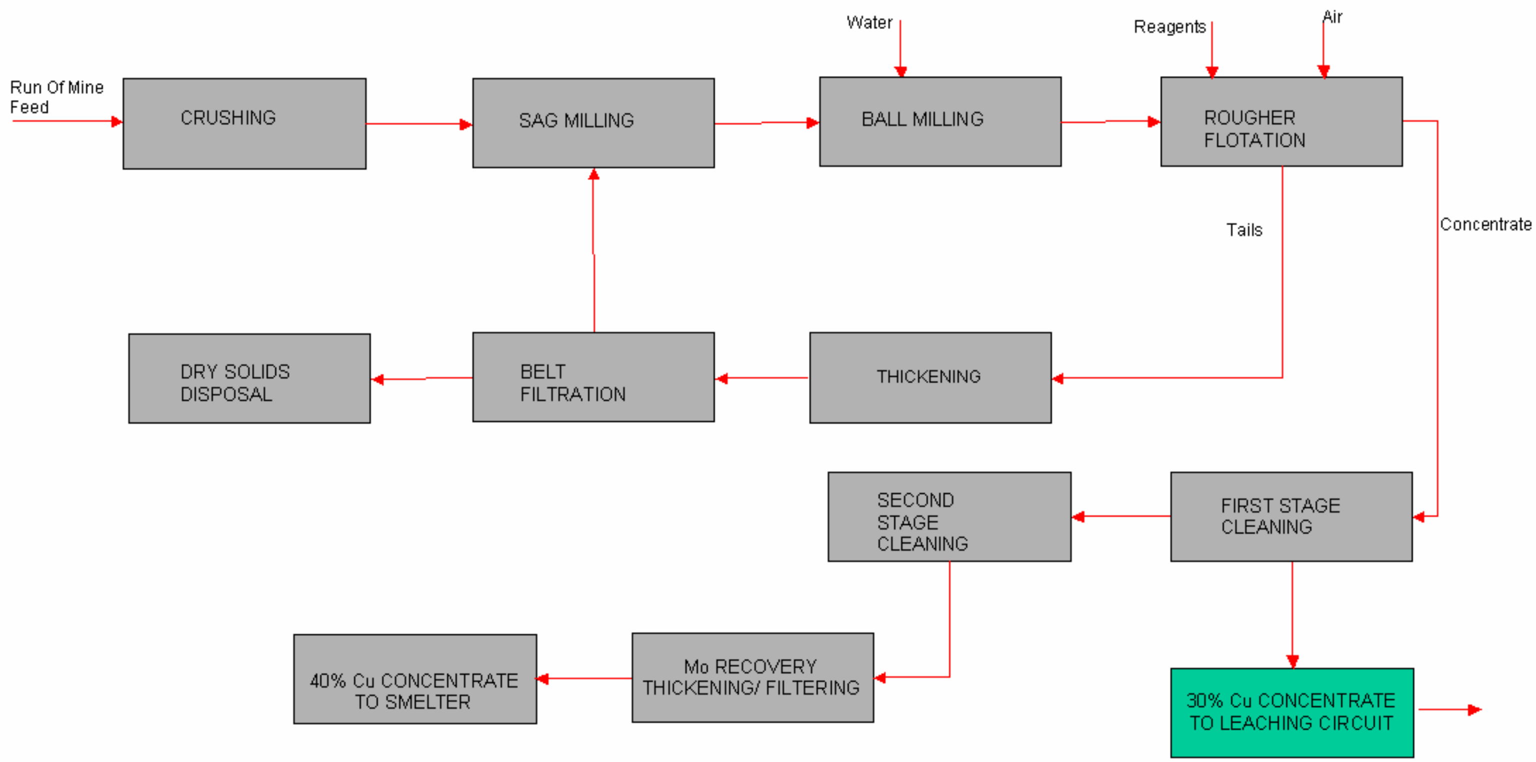


Figure 19-6 Mining Phases and General Site Plan

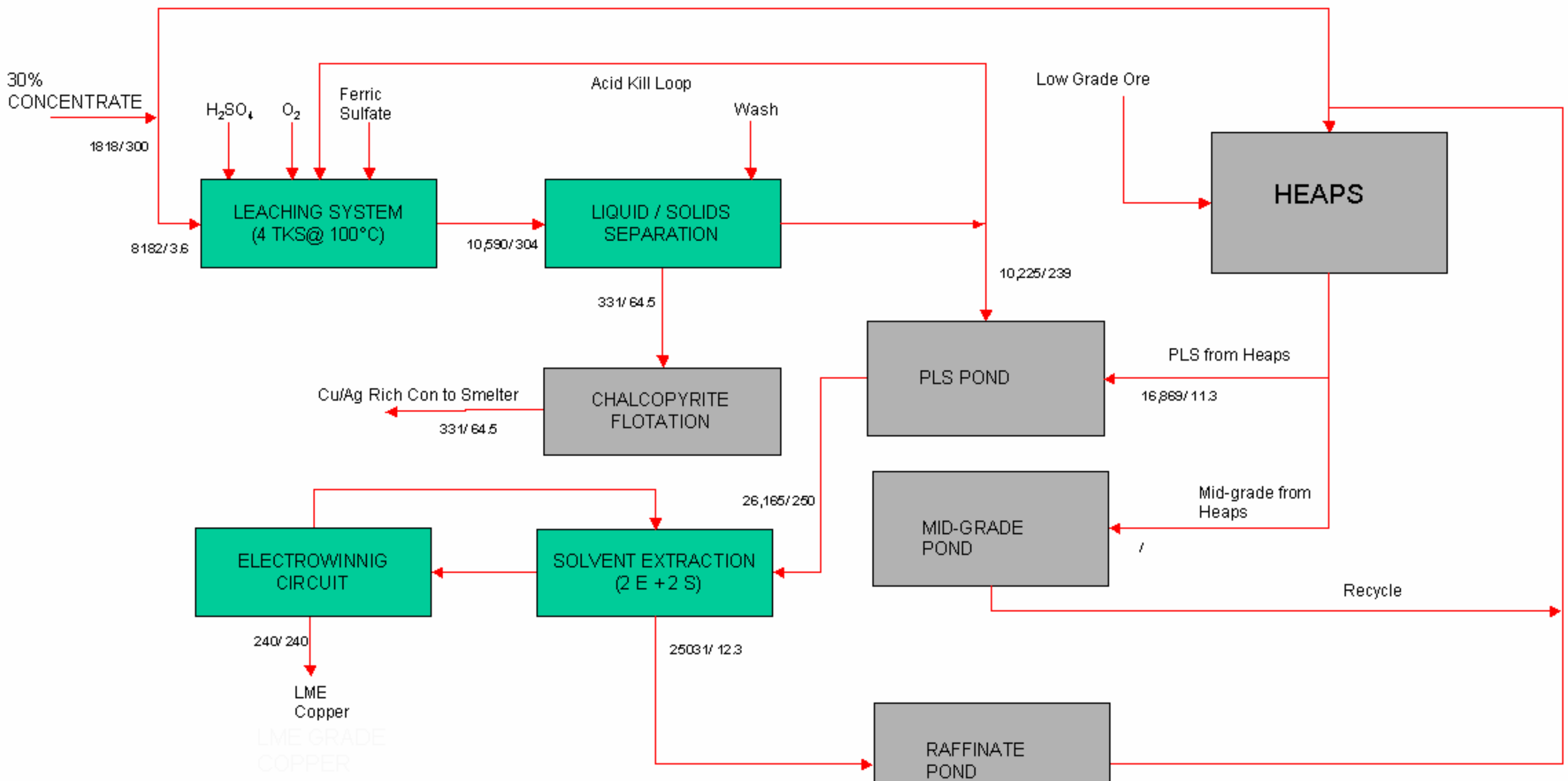


Copper Concentrator
Process Flow Diagram



Legend:
TPD total/TPD Cu

Concentrate Leaching
Process Flow Diagram



27. APPENDIX A

OXIDE HEAP LEACH/ ACID LEACH/ SXEW EVALUATION ALTERNATIVE CASE

Preliminary leach tests on the flotation concentrate indicated that this material is ameanable to acid leaching. It is envisioned that the concentrates would be leached, resulting in a copper rich leach liquor and leach tailings containing copper as chalcopyrite, as well as the precious metals. The copper rich liquor would be further treated by SXEW resulting in the production of cathode copper on site. The presence of an SXEW circuit on-site would also facilitate the treatment of any oxide copper ore by heap leaching methods. These options are considered as a project opportunity and are discussed as follows.

27.1 Economics - Alternative Case

In the Cathode and Concentrate Production Case hereafter called the Alternative Case, the copper oxide ore is leached using run of mine ore on a lined leach pad. The ore is not crushed or agglomerated before placement; however, dozer ripping to loosen material does occur. The pregnant leach solution is then processed through an SX/EW plant. Copper production from the oxide ore adds an average 7 million lbs per year of copper production to the Base Case bringing the average total copper production up to 232 million lbs per year.

In the Alternative Case, 100% of the concentrate is sent to an atmospheric copper leaching circuit. 75% of the copper (bornite and chalcocite) reports to the SX/EW processing plant, leading to an LME electro-won cathode which receives a US$.06/lb premium. The remaining 25% of the copper (chalcopyrite) reports to the concentrate stream and is sent to third parties for smelting and refining. There is no change in sulfide copper production from the Base Case.

Incremental Alternative Case capital to the Base Case is comprised of US$14.0 million for the leach pad, US$134.1 million for the SX/EW and concentrate leach plant, and US$22.2 million for a 15% contingency. Total incremental Alternative Case capital is US$170.3 million.

Initial (2008-2011) direct capital for mine, processing, water, power, leach pad, SX/EW plant and 15% contingency is estimated at US$806 million. Initial indirect capital for reclamation bonding and working capital is estimated at US$66 million. Sustaining capital is estimated at $28 million. Project net cash costs after by-product credits are estimated to average US$0.37/lb Cu over project life.

Other Key Assumptions and Inputs (only differences from the Base Case are listed):

Oxide Ore	Acid consumption: 3 lbs sulfuric acid/1 lb recovered Cu
	US$100/ton delivered acid cost, US$0.15/lb Cu recovered acid cost
	Total Leaching cost = US$0.17/lb Cu recovered
	SX/EW cost = US$0.14/lb Cu recovered

	45% recovery
Concentrate	75% of copper processed through SX/EW plant
	US$0.17/lb Cu Leach/SX/EW cost
	100% of copper contained in concentrate recovered
Other	First year full production (2010) working capital = US$50.1 million

Modeling at Base Case metal prices shows that the project could generate a cumulative net after tax profit of US$1,669 million, a 17% IRR, and a net present value discounted at 8% of US$494 million, over the projected phase one 16 year mine life.

The Alternative Case has a .5% lower IRR than the Base Case; however the NPV @ 8% incrementally increases by US$52 million.

In addition, sensitivity analysis was conducted at standard Trailing and Forward Pricing Cases.

Table 27.1 Primary Alternative Cases

Case	Cu Price $/lb	Mo Price $/lb	Ag Price $/oz	IRR %	NPV 5% US $ millions	NPV 8% US$ millions	NPV 10% US $ millions	Net Cash Cost $/lb
Base	1.20	10.00	7.50	17	807	494	337	0.37
Trailing	1.50	20.00	7.50	26	1,589	1,112	873	0.16
Forward	2.14	20.00	7.50	37	2,554	1,874	1,533	0.17

- Trailing Case Prices: Same as Base Case
- Forward Case Prices: Same as Base Case

The Trailing Price case increases the IRR to 26%, the NPV at 8% to $1,112 million and decreases the average net cash cost to $0.16/lb.

The Forward Price case increases the IRR to 37%, the NPV at 8% to $1,874 million and decreases the average net cash cost to $0.17/lb.

The potential revenue stream from molybdenum and silver (based upon contained silver metal content); at base-case metal prices, generates by-product revenue of $.44/lb Cu and covers 54% of copper cash operating costs per pound of copper produced.

The following table provides IRR and NPV sensitivities over a range of Cu and Mo metal prices.

Table 27.2 Additional Price Sensitivities to Alternative Case

IRR% vs. Copper and Molybdenum Prices

$1.08/lb Cu	15	16	17	18	19
$1.28/lb Cu	19	20	21	22	23
$1.50/lb Cu	23	24	25	26	27
$1.73/lb Cu	27	28	29	29	30
$1.98/lb Cu	31	32	33	34	35
	$11.56/lb Mo	$13.60/lb Mo	$16.00/lb Mo	$18.40/lb Mo	$21.16/ lb Mo

$ Millions NPV 8% vs. Copper and Molybdenum Prices

$1.08/lb Cu	391	446	509	572	644
$1.28/lb Cu	625	678	740	801	872
$1.50/lb Cu	894	947	1,009	1,071	1,141
$1.73/lb Cu	1,163	1,216	1,277	1,339	1,409
$1.98/lb Cu	1,471	1,524	1,585	1,647	1,718
	$11.56/lb Mo	$13.60/lb Mo	$16.00/lb Mo	$18.40/lb Mo	$21.16/ lb Mo

The following table demonstrates the project's sensitivity to increases and decreases in operating costs and capital costs. Note that a 10% increase in operating costs decreases the Base Case IRR by 2% to 15% and decreases the Base Case NPV at 8% by US$89 million to US$405 million. Similarly, a 10% increase in capital costs decreases the Base Case IRR by 2% to 17% and decreases the Base Case NPV 8% by US$60 million to US$434 million.

Table 27.3 Operating and Capital Cost Sensitivities to Alternative Case

	IRR%	NPV 5%, $ million	NPV 8%, $ million	NPV 10%, $ million	Net Cash Cost, $/lb
Base Case: $1.20 Cu, $10.00 Mo, $7.50 Ag	17	807	494	337	0.37
Cost Sensitivity Cases:					
Operating Cost +10%	15	698	405	259	0.44
Operating Cost -10%	18	913	580	413	0.30
Capital Cost +10%	15	749	434	277	0.37
Capital Cost -10%	19	863	552	397	0.37

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