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August 28, 2012

Rosemont Copper Project

NI 43-101 Technical Report
Updated Feasibility Study

Pima County, Arizona, USA

REVISION 0

Prepared For:

AUGUSTA

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DATE AND SIGNATURES PAGE

The Qualified Persons contributing to this report are noted below. The Certificates and Consent forms of the Qualified Persons are located in Appendix A, Certificate of Qualified Persons (QP) and Consent of Authors.

- Dr. Conrad E. Huss, P.E., Ph.D.; Senior Vice President and Chairman of the Board with M3 Engineering & Technology Corporation (M3) in Tucson, Arizona; principal author of this technical report and responsible for Sections 1 through 6, Sections 18 through 22, and Sections 24 through 27 of this report.
- Susan C. Bird, M.Sc., P. Eng.; Senior Associate with Moose Mountain Technical Services (MMTS) in Cranbrook, British Columbia, Canada; responsible for Sections 7 through 12, Section 14, and Section 23 of this report.
- Thomas L. Drielick, P.E.; Senior Vice President with M3 Engineering & Technology Corporation (M3) in Tucson, Arizona; responsible for Section 13 and Section 17 of this report.
- Robert H. Fong, P. Eng.; Principal Mining Engineer with Moose Mountain Technical Services (MMTS) in Cranbrook, British Columbia, Canada; responsible for Section 15 and Sections 16.1 to 16.6 of this report.
- John I. Ajie, P.E.; Vice President of Engineering Civil Construction & Mining Group with URS Energy and Construction in Denver, Colorado; responsible for the mining capital and operating costs in Section 21 of this report.

This Technical Report is current as of August 28, 2012. The effective date of the report is the issue date of the press release, which was July 24, 2012.

(Signed) (Sealed) “Conrad E. Huss” Conrad E. Huss, P.E., Ph.D.	August 28, 2012 Date
(Signed) “Susan C. Bird” Susan C. Bird, M.Sc., P. Eng.	August 28, 2012 Date
(Signed) “Thomas L. Drielick” Thomas L. Drielick, P.E.	August 28, 2012 Date
(Signed) “Robert H. Fong” Robert H. Fong, P. Eng.	August 28, 2012 Date
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TABLE OF CONTENTS

SECTION **PAGE**

ROSEMONT COPPER PROJECT

FORM 43-101F1 TECHNICAL REPORT

LIST OF FIGURES AND ILLUSTRATIONS

FIGURE DESCRIPTION PAGE

LIST OF TABLES

TABLE DESCRIPTION PAGE

LIST OF APPENDICES

1 SUMMARY

1.1 PROPERTY

The Rosemont Copper Project is a planned copper mining development, containing appreciable molybdenum and silver by-products that is being developed by Augusta Resource Corporation (Augusta). The development is situated within the Rosemont Mining District on the northeastern flank of the Santa Rita mountain range and extends into the Helvetia Mining District on the western flank of the range. The Property consists of patented lode claims, unpatented lode claims, and fee lands comprising approximately 19,800 acres (8,012 hectares). Rosemont has also acquired 21 parcels that are more distal from the project area for infrastructure purposes, comprising an additional 300 acres (121 hectares).

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

The copper mineralization of the Rosemont Deposit is primarily sulfide with an overlaying cap of oxide mineralization. The sulfide ore will be mined through conventional open pit mining techniques. Sulfide ore will be processed by crushing, grinding, and flotation to produce a copper concentrate product and a molybdenum concentrate product for market. This property will employ a conventional SAG mill/flotation circuit processing 75,000 short tons per day, equivalently indicated as either stpd or tpd.

1.2 LOCATION

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

1.3 OWNERSHIP

Augusta signed an option agreement on the Rosemont Property in 2005. During the option period, Augusta completed a two-phase drilling program in 2005 and 2006. Augusta completed the purchase of a 100% interest in the property in March 2006. The purchase is subject to a 3% Net Smelter Return (NSR).

Augusta maintains offices in Denver, Colorado, USA, and Vancouver, British Columbia, Canada. The company's common share are traded on the New York Stock Exchange MKT and the Toronto Stock Exchange under the symbol AZC.

1.4 GEOLOGY AND MINERALIZATION

The Rosemont Deposit is typical of the porphyry/skarn copper class of deposits. Similar to many other southwestern USA deposits in this class, Rosemont consists of broad-scale skarn mineralization developed in Paleozoic-aged carbonate sedimentary rocks adjacent to their contact with quartz-latite or quartz-monzonite porphyry intrusive rocks. Broadly disseminated sulfide mineralization occurs primarily in the altered Paleozoic skarn units and to a lesser extent in the altered intrusive units. Near surface weathering has resulted in the oxidation of the sulfides in the overlying Mesozoic units.

1.5 EXPLORATION AND SAMPLING

Exploration of the Rosemont Deposit by previous companies consisted of 179 drill holes for a total of 210,200 feet. Since 2005, Augusta has drilled an additional 87 holes for a total of 132,500 feet. In 2005, Augusta carried out a 15-hole, 27,402-foot diamond drilling program. In 2006, Augusta completed a 40-hole, 68,727-foot diamond drilling program, consisting of resource, geotechnical, and metallurgical holes. Also in 2006, Augusta did extensive resampling and assaying of historic drill holes to fill-in missing data needed for resource calculations. In 2008, Augusta completed a 20-hole, 17,522-foot diamond drilling program, along with the sampling of ten geotechnical holes that had been drilled in 2006, but had not been sampled. Augusta recently completed a 12-hole, 18,649-foot diamond drilling program, along with the additional sampling of core from five older holes. The recent drilling included six holes (7,698 feet) drilled to collect metallurgical test samples, three exploration holes (5,466 feet) drilled to test geophysical targets, and three infill holes (4,711 feet) drilled in support of a revised resource calculation. The results of all of these drilling programs have been used to estimate the mineral resources presented in this report.

The older drilling was conducted by major companies using industry standard procedures of the time and has since been validated by Augusta under the direction of various Qualified Persons. The newer Augusta work has been conducted using standard industry protocols, including Quality Assurance/Quality Control procedures, all under the supervision of Qualified Persons. It is believed that the resulting drill hole database is reliable and can be confidently used in the estimation of the Rosemont resource and reserves.

Additional exploration conducted in 2011 included deep-penetrating induced polarization geophysical surveys (Titan 24). The results identified a number of anomalous responses that may be indicative of potential mineralization. During late 2011/early 2012 the western end of one of the anomalies was partially drill tested, intercepting variable mineralization near the top of the anomaly.

1.6 MINERAL RESOURCE

The mineral resource estimation work was performed by Susan Bird, M.Sc., P. Eng. a Senior Associate at MMTS and an independent Qualified Person under the standards set forth by NI 43-101 (CIM, 2005). The resource is estimated using a 3-dimensional geologic model of all known lithologies and zones to create a block model encompassing the project area. The mineral resource estimates are effective as of July 17, 2012.

Drill hole data including Cu, Mo and Ag grades is incorporated into the modeling by creating 50' bench composites, corresponding to the planned bench height and elevations. Statistical and geostatistical analyses have been used to:

1. determine domain boundaries
2. determine the capping values used to restrict the outlier range of influence during interpolation,
3. determine the rotational and kriging parameters required for interpolation
4. determine appropriate sets of composites to use during interpolation that will preserve the tonnage-grade distribution of the data while allowing internal smoothing to account for dilution

In addition, several validation procedures have been performed on the Rosemont resource model. These checks include a comparison of mean grades as a global grade bias check, a set of swath plots to compare the nearest neighbor (NN) grades to the modeled Cu, Mo, and Ag grades, visual comparisons of drill hole assay and composite data with the modeled grades in section and plan, and verification of the change of support adjustment. Based on the results of this validation, it is the author's opinion that the Rosemont resource model is globally unbiased and is appropriate for use in pit optimization and long range mine planning.

A Lerchs-Grossman (LG) pit shell having a 45-degree slope angle has been applied to the three dimensional block model to ensure reasonable prospects of economic extraction for the reported mineral resources. Metal prices used for the resource pit are $3.50/lb Cu, $15/lb Mo and $20/oz Ag. The mining costs used in the resource pit optimization for ore are $0.777/ton and for waste is $0.882/ton, with processing plus general and administration (G&A) costs of $4.90/ton for sulfide/mixed material and processing costs of $3.03/ton for oxide material. These costs are in line with those developed for use in the mineral reserves.

For the reporting of the in-situ resource by equivalent copper (EqvCu) within the LG pit shell, the metallurgic recoveries, metal prices, and resulting net smelter prices (NSPs) used, are summarized in Table 1-1.

Table 1-1: Base Case Recoveries, Metal Prices and Resulting Net Smelter Prices

Metal	Metal Price	Oxides		Mixed		Sulfide	
		NSP	Recovery	NSP	Recovery	NSP	Recovery
Cu	$2.50 /lb	$2.425 /lb	65%	$2.078 /lb	40%	$2.078 /lb	86%
Mo	$15 /lb	0	0	$13.095 /lb	30%	$13.095 / lb	63%
Ag	$20 /oz	0	0	$17.111 /oz	38%	$17.111/oz	80%

The equivalent copper grades are calculated based on the above information, resulting in the following equations for each metallurgical zone:

Sulfide: $$EqvCu\% = Cu\% + \frac{Mo\% * 0.63 * 13.095}{0.86 * 2.078} + \frac{AgOPT * 0.80 * 17.111}{0.86 * 2.078 * 20}$$

Mixed: $$EqvCu\% = Cu\% + \frac{Mo\% * 0.30 * 13.095}{0.40 * 2.078} + \frac{AgOPT * 0.38 * 17.111}{0.40 * 2.078 * 20}$$

Oxide: $$EqvCu\% = Cu\%$$

The in-situ resource is classified as Measured, Indicated or Inferred corresponding to Canadian National Instrument 43-101 standards (CIM, 2005). The resource by equivalent copper grade for the Rosemont Deposit is summarized in Table 1-2 for Measure, Indicated, Measured+Indicated, and Inferred mineral resources, along with the base case equivalent copper values for each zone (oxide, mixed, sulfide). These cutoffs are sufficient to cover the processing plus G&A costs for the sulfide and mixed material ($4.90/ton) and the processing costs of the oxide material ($3.03/ton), at the expected metallurgical recoveries.

The measured and indicated mineral resource presented here is inclusive of the mineral reserve presented in the Mineral Reserve section. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Due to the uncertainty that may be associated with Inferred mineral resources it cannot be assumed that all or any part of inferred mineral resources will be upgraded to an Indicated or Measured resource.

Table 1-2: Base Case Mineral Resource by Classification and Zone

Class	Zone	Tons (millions)	Cu Eqv (%)	Cu (%)	Mo (%)	Ag (opt)
Measured	Oxide	30.3	0.17	0.17	---	---
	Mixed	13.1	0.64	0.59	0.006	0.05
	Sulfide	334.6	0.56	0.44	0.015	0.12
Indicated	Oxide	33.1	0.16	0.16	---	---
	Mixed	36.8	0.56	0.51	0.007	0.05
	Sulfide	534.7	0.48	0.37	0.014	0.11
Measured+Indicated	Oxide	63.4	0.17	0.17	---	---
	Mixed	50.0	0.58	0.53	0.007	0.05
	Sulfide	869.4	0.51	0.40	0.014	0.11
Inferred	Oxide	1.1	0.15	0.15	---	---
	Mixed	10.1	0.43	0.39	0.006	0.02
	Sulfide	128.5	0.49	0.40	0.013	0.10

Base Case cutoff grades: oxide 0.10%CuEqv, mixed 0.30% CuEqv, sulfide 0.15%CuEqv.

Augusta's 2012 drilling campaign at the Rosemont Deposit has increased both the quantity and confidence level of the estimated mineral resources, which presently totals about 919.3 million tons of measured and indicated, sulfide and mixed mineral resources grading 0.51% CuEqv, 0.41% Cu, 0.014% Mo, and 0.11 ounces per ton Ag, at a 0.15% CuEqv cutoff for sulfide and 0.30% CuEqv cutoff for a minor mixed component. An additional 138.6 million tons of inferred sulfide and mixed mineral resources are estimated at a grade of 0.49% CuEqv, 0.40% Cu, 0.012% Mo, and 0.10 ounces per ton Ag, at the same cutoffs. Sulfide and mixed material can be combined as metallurgical testwork of the mixed material indicates that it can be processed with the sulfide material to produce a concentrate. Augusta's recent drilling program and resource modeling was successful in converting significant tonnages of material previously classified as inferred into measured and indicated resource.

In addition, geologic and metallurgical studies conducted by Augusta have shown the potential for considering the oxide copper mineralization that overlies the sulfide deposit. Estimated measured and indicated oxide mineral resources total 63.4 million tons grading 0.17% Cu, at a 0.10% CuEqv cutoff (for oxide % CuEqv = % Cu). An additional inferred oxide mineral resource of 1.1 million tons grading 0.15% Cu is present, using the same cutoff. Oxide material could potentially be processed by heap leaching to recover the copper.

The classification of currently inferred sulfide and oxide mineral resources can potentially be improved with further drilling. Additional mineral resources may be found in extensions to the north and down-dip of the Rosemont Deposit. Mineralization is also known to occur at Broadtop Butte, which could potentially be added as a satellite development. Further mineralization also occurs in the Copper World and Peach-Elgin deposits on the Rosemont Property.

1.7 MINE RESERVES & MINE PLAN

The Rosemont Deposit is a large tonnage, copper-molybdenum deposit located in close proximity to the surface and amenable to open pit mining methods. The proposed pit operations will be conducted from 50-foot high benches using large-scale mining equipment.

The mine has a 21-year life, with sulfide ore to be delivered to the processing plant at an initial rate of 75,000 tpd. Provisions are included to increase production to 90,000 tons of ore per day (tpd) in year 12 of operations. Seven mining phases have been defined for the extraction sequence for the Rosemont Deposit. The phase development strategy consists of extracting the highest metal grades along with the minimum strip ratios during the initial years to maximize the economic benefits of the ore-body.

The mineral reserve estimates presented in this report were prepared by Mr. Robert Fong, P.Eng., Principal Mining Engineer for Moose Mountain Technical Services. Mr. Fong meets the requirements of an independent Qualified Person under NI 43-101 standards. The mineral reserve estimates are effective as of July 24, 2012.

Proven and probable mineral reserve estimates and waste rock for the Rosemont Deposit are summarized by mining phase in Table 1-3.

Table 1-3: Combined Proven and Probable Mineral Reserves by Phase

Phase	Sulfides >= 4.90 $/ton NSR Cut-off Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Waste Ktons	Total Material Ktons	Strip Ratio
1	61,546	22.38	0.50	0.016	0.14	142,729	204,275	2.32
2	27,169	17.24	0.40	0.011	0.09	84,526	111,695	3.11
3	42,418	19.37	0.40	0.020	0.13	59,553	101,971	1.40
4	42,699	21.54	0.49	0.013	0.14	100,709	143,408	2.36
5	79,845	21.64	0.50	0.013	0.13	156,603	236,448	1.96
6	241,477	17.99	0.42	0.014	0.12	411,973	653,450	1.71
7	172,052	19.16	0.42	0.015	0.11	287,362	459,414	1.67
Total	667,206	19.42	0.44	0.015	0.12	1,243,455	1,910,661	1.86

(NSR values are based on metal prices of $2.50/lb Cu, $15.00/lb Mo and $20.00/oz Ag.)

The pit design reflects an optimum pit at metal price of $1.88 /lb Cu, $11.07 /lb Mo, and $14.87/oz Ag. Proven and probable sulfide mineral reserves within the designed final pit total 667 million tons grading 0.44% Cu, 0.015% Mo and 0.12 oz Ag/ton. There are 1.24 billion tons of waste materials, resulting in a stripping ratio of 1.9:1 (tons waste per ton of ore). Total material in the pit is 1.9 billion tons. Contained metal in the sulfide (proven and probable) mineral reserves is estimated at 5.88 billion pounds of copper, 194 million pounds of molybdenum and 80 million ounces of silver. No mineralized oxide materials are in the ore reserves, they are included with the waste materials.

Nearly 46% of the sulfide mineral reserves in the Rosemont ultimate pit are classified as proven and the remainder (54%) is considered probable. The classifications are based on the exploration drilling in the Rosemont deposit. *All of the mineral reserve estimates reported above are contained in the mineral resource estimates presented in Section 14.*

The Rosemont ultimate pit contains approximately 24 million tons of inferred sulfide mineral resources that are above the $4.90/ton NSR cutoff value for sulfides. These resources are included in the waste estimates presented in Table 1-3. *Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Inferred mineral resources have a great amount of uncertainty as to their existence and as to whether they can be mined economically. It cannot be assumed that all or any part of inferred mineral resources will ever be upgraded.*

1.8 MINERAL PROCESSING & METALLURGICAL TESTING

The earliest existing records of metallurgical testing are from the period 1974 - 1975, at which time grinding and flotation tests were performed. In the first half of 2006, Augusta initiated test work to provide a better understanding of the metallurgy of the Rosemont Deposit and establish the design criteria for the design of a process facility. Additional test work was conducted in 2012 on new drill core obtained from the Augusta drilling program in December 2011.

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

An estimate of metal production in concentrate for the first 21 years of plant operation was prepared from the results of flotation test work. The estimates of annual metal recovery are presented in Table 1-4 .

Table 1-4: Estimated Metal Recovery by Year of Production

Estimated Metal Recovery by Year of Production			
Production Year	Recovery %		
	Cu	Mo	Ag
1	89.8	65.0	77.5
2	89.8	65.0	77.5
3	89.8	65.0	77.5
4	84.1	34.2	72.6
5	84.1	34.2	72.6
6	84.1	34.2	72.6
7	84.1	34.2	72.6
8	90.6	78.7	78.2
9	90.6	78.7	78.2
10	84.8	74.3	73.9
11	82.1	72.2	71.8
12	84.4	73.9	73.5
13	84.0	56.7	73.1
14	85.5	57.2	74.3
15	89.1	58.6	76.9
16	89.1	58.6	76.9
17	89.1	58.6	76.9
18	89.1	58.6	76.9
19	89.1	58.6	76.9
20	89.1	58.6	76.9
21	89.1	58.6	76.9

1.9 RECOVERY METHODS

Sulfide ore will be transported from the mine to the primary crusher by off-highway haulage trucks then conveyed to the concentrator facilities. Copper concentrate produced at the concentrator facility will be loaded into highway haul trucks and transported to a concentrate smelter and metal refinery. Molybdenum concentrate produced at the concentrator facility will be bagged and loaded onto trucks for shipment to market.

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

1.10 ENVIRONMENTAL

Permitting for the Rosemont Copper Project involves federal approvals and requires compliance with the National Environmental Policy Act (NEPA). This in turn requires an Environmental Impact Statement (EIS) and compliance with the Endangered Species Act (ESA) and the National Historic Preservation Act (NHPA). A Mine Plan of Operation was submitted to the US Forest Service on July 11, 2007 to initiate the EIS and start the permitting process. Major federal permits required to construct and begin operation of the Rosemont Project includes an Environmental Impact Statement Record of Decision from the USFS permitting the use of Federal lands and a Clean Water Act (CWA) Section 404 permit for discharge of fill material to on-site washes. Major state permits include an Aquifer Protection Permit, a 401 Certification, and an Arizona Pollution Discharge Elimination System (AZPDES) general storm water permit. The only major local permit required is a Pima County Clean Air Act (CAA) Title V air quality permit. Other permits which do not affect the timeline for project permitting and subsequent start up include explosives permits, nuclear instrumentation licenses, hazardous waste identification, tracking numbers and spill control plans. A list of permits is provided in Section 20.

1.11 ECONOMIC ANALYSIS

1.11.1 Operating Costs

The mine operating costs were derived from equipment hours and cycle times developed by Moose Mountain from their mine plan. Rebuild costs for major equipment were generated from vendor supplied component replacement schedules and URS Energy and Constructions’ data base for similar projects and equipment. Mining costs supplied by others were checked by URS who built the estimate and is the responsible Qualified Person. The average life of mine operating costs for the mining operation is $1.23 per ton mined. These costs include: clearing of vegetation, removal of topsoil, drilling, blasting, loading, hauling, road and dump maintenance, re-grading, mine operations supervision, craft labor and subcontractor costs.

Mill process operating costs for the life of mine average $4.27/ton of mill ore which includes crushing and conveying, grinding and classification, flotation and regrind, concentrate thickening, filtration and dewatering, tailings disposal and mill ancillary services.

The average life of mine operating cost for the supporting facilities and general administrative expenses is $0.59/ton of mill ore. The supporting facilities include laboratory, safety and environmental, accounting, human resources, security and the general manager's office. Also included is an endowment, railcar lease, and CAP water.

The life of mine average site direct operating cost estimate by cost center is shown in Table 1-5 below. All costs are estimated in second quarter 2012 Dollars at an accuracy of ± 10%.

Table 1-5: Summary of Average Life of Mine Operating Costs

	Annual Cost ($000)
Mining	$106,000
Mill Operations & Maintenance	$134,407
Support Facilities and G&A	$18,484
Total	$258,891

1.11.2 Initial Capital Cost

The total capital cost estimate to design, construct and commission the Rosemont facilities is estimated to be $1,060.4 million for the sulfide plant. The estimate includes the direct field cost for constructing the project at $870.6 million as well as $189.8 million for the indirect costs associated with the design engineering, procurement and construction, commissioning, spare parts, contingency, power line gross-up tax, and excludes Owner's cost. All costs are expressed in second quarter 2012 Dollars at an accuracy of ± 10% with no allowance provided for escalation, interest, foreign currency, hedging, or financing during construction.

1.11.3 Financial Analysis

The Rosemont Project economics were prepared using a discounted cash flow model. Costs are in constant second quarter 2012 Dollars with no provisions for escalation. The financial indicators examined for the project included the Net Present Value (NPV), Internal Rate of Return (IRR) and payback period (time in years to recapture the initial capital investment). Annual cash flow projections were estimated over the life of the mine based on capital expenditures, production costs, transportation and treatment charges and sales revenue. The life of the mine is 21 years.

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

Metal sales prices used in the evaluation are listed in Table 1-6.

Table 1-6: Base Case and Historical Metals Prices

	60/40 Weighted Average *	3 Year Historical Average
Copper	\$3.50 / pound	\$3.56 / pound
Molybdenum	\$14.19 / pound	\$15.06 / pound
Silver	\$3.90 / ounce	\$3.90 / ounce
Gold	\$450.00 / ounce	\$450.00 / ounce

**60/40 weighted average of the 36 month historic price and the 24 month futures price forecast*
Silver and gold metal prices are set from the Silver Wheaton agreement

In addition to the above metal sales price cases, a case of long term metal prices was also evaluated. Long term metal prices are shown in Table 1-7 below.

Table 1-7: Long Term Metals Prices

Copper	\$2.62/lb
Molybdenum	\$15.00/lb
Silver	\$3.90/oz
Gold	\$450.00/oz

The after-tax financial results for the three metal pricing scenarios are shown in Table 1-8.

Table 1-8: Financial Indicators (After Tax)

	Base Case (60/40 split)	Historical 36 Months	Long Term Metal Prices
NPV 0%	\$7,257.5	\$7,498.4	\$4,554.4
NPV 5%	\$3,645.8	\$3,776.4	\$2,256.0
NPV 8%	\$2,507.6	\$2,603.1	\$1,529.4
IRR	37.9%	38.8%	30.9%
Payback Years	2.3	2.2	2.4

1.12 CONCLUSIONS & RECOMMENDATIONS

The Rosemont Deposit resource and mineral reserves have increased with the additional drilling campaign in 2012. Metallurgical recoveries improved slightly for copper with the additional metallurgical testing. Metal prices have improved since the 2009 feasibility study update; however, silver and gold prices used in this update are lower reflecting an agreement with Silver Wheaton for a forward sale of gold and silver. The after-tax NPV, IRR, and payback indicators are also improved over the previous 2009 update.

With the improved economic indicators, the Rosemont Copper Project should continue with the design engineering and construction of the facilities as soon as the permitting effort allows.

2 INTRODUCTION

2.1 GENERAL

Augusta Resource Corporation (Augusta) is a base metals company with its corporate office located in Vancouver, British Columbia, Canada, and executive office located in Denver, Colorado. Augusta is focused on advancing its Rosemont Copper Project near Tucson, Arizona. The project is nearing the last stages of permitting with the Record of Decision (ROD) expected to be received in the fourth quarter of 2012 and production to start-up in 2015. Augusta's objective is to build and operate the world-class Rosemont mine and develop a robust portfolio of assets in North America with the focus on organic growth and early stage acquisitions. Augusta trades on the Toronto Stock Exchange and the NYSE MKT under the symbol AZC. Rosemont Copper Company (Rosemont) is a wholly owned subsidiary of Augusta Resource Corporation and will be the operating company for the mine and process facilities.

In 2006, Augusta Resources Corporation retained a number of contractors, including M3 Engineering & Technology Corporation (M3), to provide a review of prior work on the Rosemont Copper Project and prepare technical and cost information to support a bankable level Feasibility Study and Technical Report compliant with the Canadian National Instrument (NI) 43-101 and Form 43-101F1. The Technical Report titled "NI 43-101 Technical Report for the Rosemont Copper Project Feasibility Study, Pima County, Arizona, USA" was issued in August 2007. An update of the 2007 Technical Report, titled "NI 43-101 Technical Report for the Rosemont Copper Project Updated Feasibility Study, Pima County, Arizona, USA", incorporating additional resource information and metallurgical testing, was issued in January 2009 and amended in March 2009.

2.2 PURPOSE OF REPORT

The purpose of this technical report is to present updated mineral resource information and metallurgical testing information completed since the 2009 technical report update. Capital costs, operating costs, and the economic analysis were updated to 2012 costs based on 40% completion of engineering. Basic engineering was completed in November 2010. It is Augusta's intent to continue to develop the Rosemont Copper Project once the Record of Decision (ROD) has been received from the US Forest Service.

2.3 SOURCES OF INFORMATION

This report is based on data supplied by Augusta and Rosemont and information developed during the feasibility study and basic engineering period by M3 and other third party consultants. The source documents are summarized is Section 27.

2.4 CONSULTANTS AND QUALIFIED PERSONS

Augusta retained a number of contractors, including M3 Engineering and Technology Corporation (M3), to provide a review of prior work on the project and prepare technical and cost information to support an updated Feasibility Study and this Technical Report following the outline as defined in Canada National Instrument (NI) 43-101 and in compliance with Form 43-

101F1. Dr. Conrad Huss, P.E. of M3 Engineering and Technology Corporation (M3) is the Principal Author and Qualified Person responsible for the preparation of this report. Dr. Huss has visited the site on numerous occasions prior to this Updated Feasibility Study and is familiar with the site. In addition, the following M3 employees, under the supervision of Dr. Huss, visited the site during the initial Feasibility Study and/or the 2009 Updated Feasibility Study on the dates noted.

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

Other contributing authors and Qualified Persons responsible for preparing this Updated Feasibility Study Technical Report include; Ms. Susan C. Bird, M.Sc., P. Eng., of Moose Mountain Technical Services; Mr. Robert H. Fong, P. Eng., of Moose Mountain Technical Services; Mr. John I. Ajie, P.E., of URS Energy and Construction; and Mr. Thomas L. Drielick, P.E., of M3 Engineering and Technology Corporation.

Augusta retained Moose Mountain Technical Services (MMTS) to develop and oversee the resource estimate, mineral reserve estimate and mining methods. Ms. Susan C. Bird, M.Sc., P.Eng., Senior Associate with MMTS, authored Section 14 – Mineral Resource Estimate and Section 23 – Adjacent Properties. Ms. Bird also reviewed Section 7 – Geological Setting and Mineralization, Section 8 – Deposit Types, Section 9 – Exploration, Section 10 – Drilling, Section 11 – Sample Preparation, Analysis and Security, and Section 12 – Data Verification. These sections were authored by Mr. Mark Stevens of Augusta Resource Corporation and taken from the previous Updated Feasibility Study. Ms. Bird is the Qualified Person responsible for these sections of the current report. Ms. Bird visited the site from January 30, 2012 to February 3, 2012.

Mr. Robert H. Fong, P.Eng., Principal Mining Engineer with Moose Mountain Technical Services is the Qualified Person responsible to estimate and oversee the calculations of the open pit reserves and to develop the Life of Mine (LOM) Mine Plan which includes a Lerchs-Grossman analysis, pit design, mine production schedule, mine access and haul roads and waste rock stockpiles. Mr. Fong authored Section 15 – Mineral Reserve Estimate and Section 16 – Mining Methods (16.1 to 16.6) and is the Qualified Person responsible for these sections of the current report. Mr. Fong visited the project site on November 20, 2008.

Mr. John I. Ajie, P.E. – Vice President of Engineering Civil Construction & Mining Group, with URS Energy and Construction is the Qualified Person responsible for supervising and reviewing the development of the mine capital and operating cost estimate. Mr. Ajie is responsible for the basis for the mine capital and operating costs in Sections 16.7 to 16.14 and mine capital and operating costs in Section 21. Mr. Ajie visited the site on May 18, 2006.

Mr. Thomas L. Drielick, P.E., Senior Vice President of M3 Engineering & Technology Corporation, is the Qualified Person responsible for reviewing the metallurgical test work and establish the process recoveries and process recovery methods. Mr. Drielick visited the site on August 21, 2007.

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

2.5 DEFINITION OF TERMS USED IN THIS REPORT

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

Acronyms and abbreviations used in this report are noted below:

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

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

3 RELIANCE ON OTHER EXPERTS

Mr. Ron Hamagami of URS Energy and Construction provided capital and operating cost information for the mine in Section 21 – Capital and Operating Costs. Mr. Ajie reviewed this work and is the Qualified Person responsible for this section.

Mr. Brian Lindenlaub of WestLand Resources, Incorporated authored Section 18 – Project Infrastructure and Section 20 – Environmental Studies, Permitting and Social or Community Impact. M3 also relied on the information provided by Mr. Robert Loewen who authored Section 19 – Market Studies and Contracts and Mr. Mark Stevens of Augusta Resource Corporation who authored Section 4 – Property Description and Location. These sections were reviewed by M3 and judged to be professionally sound and to industry standards.

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

Mr. Mark Stevens, C.P.G., Vice President of Exploration for Augusta, compiled the drill hole data files and prepared the geology section of this Updated Feasibility Study based on earlier published reports and internal reports (Augusta – 2007-2009). Mr. Stevens has spent time on site on numerous occasions over the last several years. Ms. Susan C. Bird of Moose Mountain Technical Services has review this section and is the independent Qualified Person responsible for this work.

Tetra Tech, Inc. of Golden, Colorado and Tucson, Arizona were responsible for the site geotechnical investigations consisting of a site geotechnical study, a geologic hazards assessment, and a baseline geochemical characterization study. Tetra Tech also provided a site water management plan, waste management plan, initial dry stack tailings facility design, and oxide leach facilities. Tetra Tech provided the design and material quantities for the storm water pond and compliance point dam. M3 estimated the capital cost based on the material quantities. The Tetra Tech reports are referenced in the Technical Report in Section 27.

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

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

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

Stantec Consulting, Inc. of Tucson, Arizona was responsible for the conceptual design of the fresh water pipeline from the well fields to the project site. CDM Smith Inc. of Phoenix, Arizona, was responsible for the final design of the fresh water system and pumping stations, including a water surge analysis for the system. CDM Smith provided the design, the quantity take-offs for construction, and the capital cost estimate for the system.

The primary Qualified Persons responsible for preparing this Technical Report relied on the various reports and documents listed in Section 27. These reports and documents were prepared by technically qualified and professional persons and were found to be generally well organized, to industry standards, and where applicable, the conclusions reached were judged to be professionally sound. It is assumed that the information and explanations given to the Qualified Persons and those assisting the Qualified Persons by the employees of Augusta and third party consultants, who provided the reports referenced in Section 27 during the preparation of this Rosemont Copper Project Updated Feasibility Study and this Technical Report, were essentially complete and correct to the best of each employee's or consultant's knowledge and that no information was intentionally withheld.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Rosemont Property is located approximately 30 miles (50 km) southeast of Tucson, in Pima County, Arizona (Figure 4-1). The Property consists of a comprehensive land package that covers much of the Rosemont and Helvetia Mining Districts, occurring on the eastern and western sides, respectively, of the Santa Rita Range. The lands are under a combination of private ownership by Augusta and Federal ownership. The lands occur within Townships 18 and 19 South, Ranges 15 and 16 East, Gila & Salt River Meridian. The Rosemont Property geographical coordinates are approximately 31º 50’N and 110º 45’W.

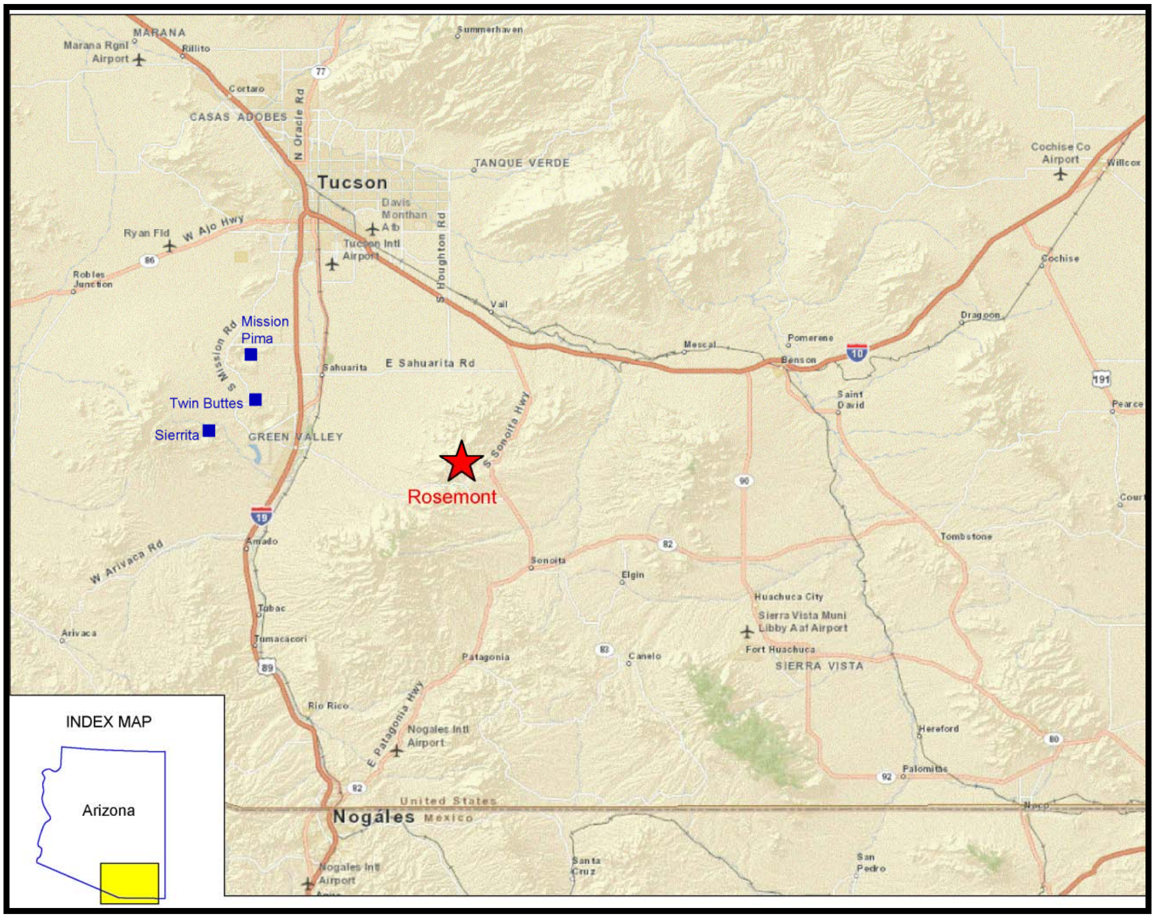

Figure 4-1: Property Location of Rosemont Project

4.2 LAND TENURE

On March 31, 2006, Augusta completed the purchase of a 100% interest in the Property for a total of US$20.8 million and continues to maintain the property in good standing. Augusta retained the legal firm of Fennemore Craig P.C. to handle the legal transfer of the Rosemont Property. Augusta’s land information has come from 2006 property purchase legal documents and has been subject to further validation contracted by Augusta, including a mining claim specialist, Daniel Mead of Tucson, Arizona, and registered mining claim surveyors at Darling Environmental & Surveying, Ltd. of Tucson, Arizona. Darling Environmental & Surveying, Ltd. has conducted an extensive field and office review of the patented and unpatented claims. Fennemore Craig has continued to have legal involvement with land title maintenance.

The Rosemont Property is a combination of fee land, patented mine and mill site claims, and unpatented mine and mill site claims. Taken together, the land position is sufficient to allow mining of the open pit, processing of ore, storage of tailings, disposal of waste rock, and operation of milling equipment. These lands are accessible under the provisions of the Mining Law of 1872, subject to approval from the US Forest Service after the completion of an Environmental Impact Statement (EIS) as per the National Environmental Policy Act (NEPA) process. The EIS process includes interagency consultation on project alternatives and mitigation of environmental impacts. The use of the project surface rights requires obtaining a number of federal, state, and local permits and approvals, some of which are complete, others are in progress.

The core of the Rosemont Property consists of 132 patented lode claims that in total encompass an area of 2,000 acres (809 hectares) as shown in Figure 4-2. Surrounding the patented claims are a contiguous package of 1,060 unpatented lode-mining claims with an aggregate area of more than 16,000 acres (6,475 hectares). Most of the unpatented claims were staked on Federal land administered by the US 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. Associated with the property are 33 parcels of fee (private) land consisting of 1,800 acres (728 hectares). The area covered by the patented claims, unpatented claims and fee lands totals approximately 19,800 acres (8,012 hectares). Rosemont has also acquired 15 parcels that are more distal from the project area that are planned for various infrastructure purposes including, well fields, pump stations, utilities, and ranch operation, comprising an additional 300 acres (121 hectares). A listing of the unpatented claims, patented and fee lands is provided in Appendix B and C, respectively.

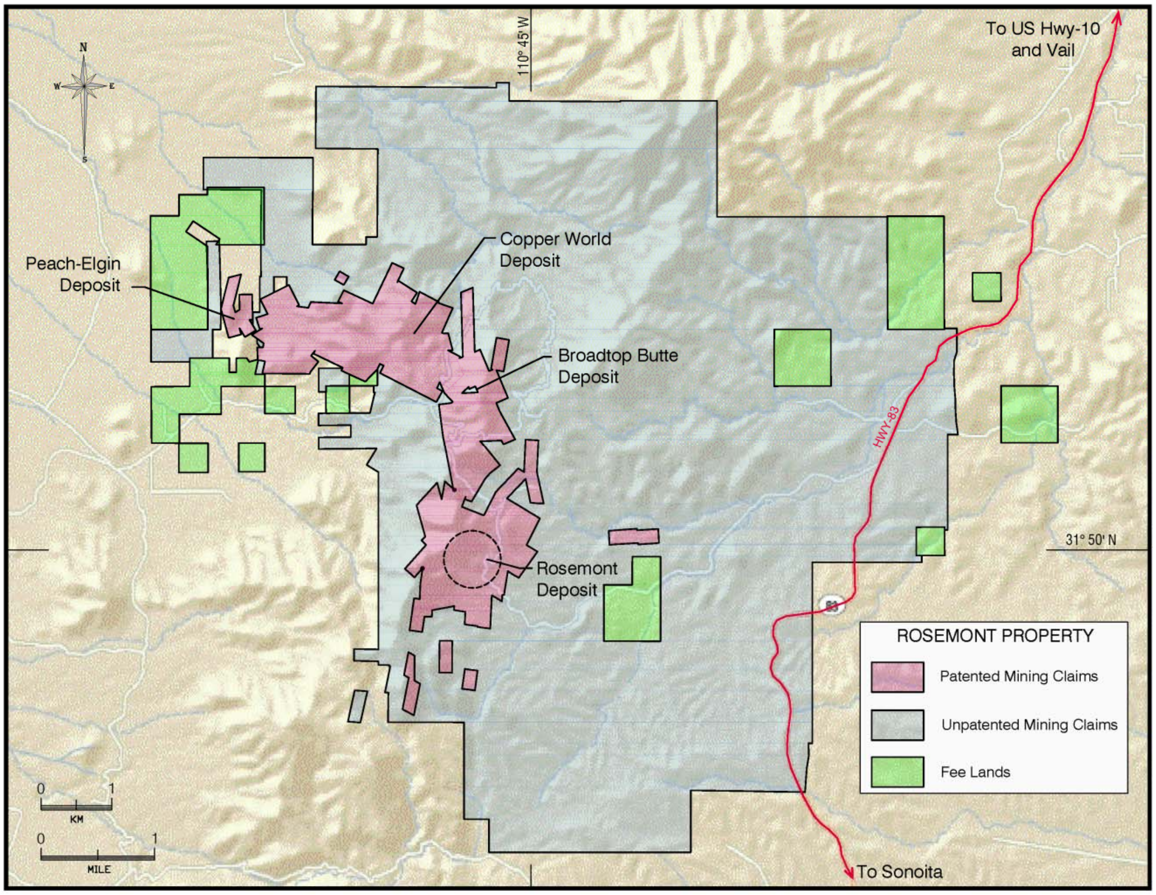

Figure 4-2: Rosemont Property Ownership

The patented lode claims and fee land parcels are both considered to be private lands that provide the owner with both surface and mineral rights. The patented mining claim block, located in the core of the property, is monumented in the field by surveyed brass caps on short pipes cemented into the ground. The fee lands are located by legal description recorded at the Pima County Recorder's Office. The patented claims and fee lands are subject to annual property taxes amounting to a total of approximately $55,000. As long as the property taxes are paid annually on these claims, there is no expiration date.

US Forest Service and Bureau of Land Management lands have had the mineral rights reserved to Augusta on the unpatented lode mining claims that surround the patented claims. Wooden posts and stone cairns mark the unpatented claim corners, end lines and discovery monuments, all of which have been surveyed. The unpatented lode claims have no expiration and are maintained through the payment of annual maintenance fees of $140.00 per claim, for a total of about $150,000, payable to the Bureau of Land Management.

A 3% Net Smelter Return (NSR) royalty applies to the patented claims, most of the unpatented claims, and some of the fee land.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Rosemont Copper Project is located in Pima County, Arizona, approximately 30 miles southeast of Tucson. The project site is accessible from Tucson by going east on Interstate Highway I-10 to Arizona State Route 83 and south on SR-83 approximately 11 miles to the main access road to the plant. The intersection of the proposed primary access road to the plant and SR-83 will be between mile markers 46 and 47 at a point that provides a clear line of site for up to 2,500 feet in each direction. SR-83 will be modified to provide safe ingress and egress from the main access road. Modifications will include a 500 foot long center lane in each direction for accelerating and decelerating. A 220 foot deceleration lane and 500 foot acceleration lane will also be constructed on the southbound lane of SR-83 for safe access into and out of the plant site. The primary access road into the plant is approximately 3.2 miles long and will be paved.

A secondary gravel access road to the plant will be provided from South Santa Rita Road on the west side of the Santa Rita Mountains over the mountain ridge to the plant. Access will be from Interstate Highway I-19 at Sahuarita then east on Santa Rita Road to the start of the west plant access road. The main power transmission line to the plant and the fresh water pipeline will generally follow the alignment along Santa Rita Road and the west access road. The west access road is approximately 4.4 miles long from Helvetia road into the plant and will be used to access the well fields and fresh water booster stations located in the Santa Cruz Valley to the west.

The city of Tucson, Arizona, provides the nearest major railroad and air transport services to support the project.

5.2 CLIMATE

The southern Arizona climate is typical of a semi-arid continental desert with hot summers and temperate winters. The project area is at the north end of the Santa Rita Mountain Range at elevations between 4,550 feet and 5,300 feet above mean sea level (amsl). The higher elevation in the project area results in a milder climate than at the lower elevations across the region. Summer daily high temperatures are above 90 degrees Fahrenheit (˚F) with significant cooling at night. Winter in the project area is typically drier with mild daytime temperatures and overnight temperatures that are typically above freezing. Winter can have occasional low intensity rainstorm patterns that can last for multiple days.

The average annual precipitation in the project area is estimated to be approximately 17.7 inches, based on historical data from eight meteorological stations within a 30 mile radius of the project area. More than half of the annual precipitation occurs during the monsoon season from July through September. The monsoon season is characterized by afternoon thunderstorms that are typically of short duration, but with high-intensity rainfall. The lowest precipitation months are April through June.

Rosemont installed an on-site meteorological station in April, 2006. The station is located at the approximate center of the proposed open pit at an elevation of 5,350 feet amsl. The station monitors site-specific weather data including temperature, precipitation, wind speed, and wind direction. Pan evaporation was added to this station in mid-2008. An annual average pan evaporation of about 71.5 inches was estimated for the Rosemont site based on correlation with data from the University of Arizona and Nogales 6 N weather stations. Pan evaporation rates are approximately 30% higher than open water evaporation due to measurement conditions.

5.3 LOCAL RESOURCES

The Rosemont Copper Project is located 30 miles from the city of Tucson with a population of over 520,000 based on the 2010 census. Tucson is also the county seat for Pima County with a population of approximately one million, which comprises the Tucson Metropolitan Area. Mining has been a part of the Tucson area for decades with three major operating mines within a 75 mile radius; including ASARCO Silver Bell Mine near Marana, the ASARCO Mission Complex near Sahuarita, and the Freeport McMoRan Sierrita Mine near Green Valley. There are sufficient resources in the Tucson area for staffing the Rosemont Copper Project. The cultural and educational facilities provided in the Tucson Metropolitan Area will also attract experienced technical staff into the area. There is also a well-established base of contractors and service providers to the mining industry located in Tucson that provide equipment, materials and supplies, as well as maintenance and repair services to the mining industry.

5.4 INFRASTRUCTURE

The project site is on Arizona State Route 83 (South Sonoita Highway), approximately eleven miles south of Interstate 10 (I-10). This system of state and interstate highways allows quick and convenient access to the site for all major truck deliveries. The majority of the labor and supplies for construction and operations can come from the surrounding areas in Pima, Cochise and Santa Cruz Counties.

The Union Pacific mainline east-west railroad route passes through Tucson, Arizona and generally follows Interstate Highway I-10. The Port of Tucson has rail access from the Union Pacific mainline consisting of a two mile siding complimented by an additional 3,000 foot siding. The siding branches to grade level access, dock level access, intermodal container access, and team track facilities. The Port of Tucson provides transportation and logistics services to businesses in the area and is a registered Foreign Trade Zone. The Port of Tucson is approximately 24 miles from the project site and can trans-load materials and supplies received by rail to trucks for delivery to site.

The Tucson International Airport (TIA) is located approximately 29 miles from the project site and in close proximity to interstate highways I-10 and I-19. TIA provides international air passenger and air freight services to businesses in the area with seven airlines currently providing nonstop service to 15 destinations with connections worldwide. TIA airlines offer over 60 daily departing flights with approximately 6,600 available seats. TIA is designated a US Port of Entry with 24 hour customs and immigration services.

The power supply to the Rosemont mine and production facilities falls within the Tucson Electric Power Company (TEP) and TRICO Electric Cooperative Inc. service territories.

Geographically, the area east of the Rosemont pit, which includes part of the mine and all of the process facilities, falls in the TEP service territory. The area west of the Rosemont pit, which includes the balance of the mine and the fresh water pumping system, falls within the TRICO service territory. Since most of the estimated electrical load for the project is located in the TEP's service territory, TEP will be the electrical utility service provider for the entire facility. A joint venture business arrangement will be established between TEP and TRICO to compensate both service providers. Rosemont Copper Company will receive one electric utility rate and bill for the facilities and the breakdown of revenue between TEP and TRICO will be transparent to the project. This multiple service territory and provider agreement will be submitted to the Arizona Corporation Commission (ACC) for review and final approval prior to implementation.

The connection to the TEP power grid will be at the existing TEP South 345/138 kV substation located northwest of Sahuarita, Arizona, in the Santa Cruz Valley west of the project site. A 138 kV power line will be run east and south to a new switchyard (Toro Switchyard) which will be configured with a ring bus to serve existing TEP transmission lines in addition to the new 138kV transmission line to the project site. The new transmission line will follow an alignment along South Santa Rita Road, crossing the Experimental Range, to Helvetia Road at the western base of the Santa Rita Mountains and then it will follow the western access road over the ridge to the plant site.

The most viable source of water supply for the project is from groundwater from various aquifers in the region. Potential sources for groundwater include the basin-fill deposit aquifers of Cienega Wash drainage basin and /or Davidson Canyon located, east and north of the project area, and basin-fill deposit aquifers of the upper Santa Cruz basin west of the project area. There are bedrock and /or shallow alluvium aquifers on or near the Rosemont Project area; however, they are considered to be insufficient as a primary source of water supply for the project. Since the basin-fill deposit aquifers of Cienega Wash drainage basin and Davidson Canyon basin are considered environmentally sensitive, fresh water for the project will be pumped to the project site from new well fields in the basin-fill deposit aquifer of the upper Santa Cruz basin, which lies west of the Rosemont Copper Project and the Santa Rita Mountains. A 53 acre parcel along South Santa Rita Road near the Santa Rita Experimental Range has been purchased and explored with one test well. The test indicated that this property would support 2 wells at a total flow of approximately 3,000 gpm. Water samples collected from the test well indicated that the quality of the ground water is suitable for potable water. It is estimated that a total of 5 or 6 production wells will meet the water supply needs for the project. Additional well sites are under development.

5.5 PHYSIOGRAPHY

The Rosemont Property is located within the northern portion of the Santa Rita Mountains that form the western edge of the Mexican Highland section of the Basin and Range Physiographic Province of the southwest United States (Wardrop 2005). The Basin and Range physiographic province is characterized by high mountain ranges adjacent to alluvial filled basins. The property occupies flat to mountainous topography in the northeastern and northwestern flanks of the Santa Rita Mountains at a surface elevation ranging from 6,290 to 4,000 feet above sea level.

Vegetation in the project area reflects the climate with the lower slopes of the Santa Rita Mountains dominated by mesquite and grasses while the higher elevations, receiving greater rainfall, support an open cover of oak, pine, juniper and cypress.

6 HISTORY

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

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

Since the cessation of production in 1951, the area stretching from Peach-Elgin (on the northwest) to Rosemont (on the southeast) has seen a progression of exploration campaigns to further evaluate the mineral potential. Churn drilling at Peach-Elgin deposit in 1955 and 1956 by Lewisohn Copper Company began the definition of that deposit. Drilling in 1956 by American Exploration and Mining Company initiated exploration of the Broadtop Butte prospect.

By the late 1950s, Banner Mining Co. had acquired most of the claims in the area and had drilled the discovery hole into the Rosemont Deposit. Anaconda Mining Company subsequently acquired the property in 1963 and carried out a major exploration program that demonstrated Rosemont to be a major porphyry/skarn copper deposit, while also advancing regional exploration, including targets at the Broadtop Butte and Peach-Elgin prospects. In 1973, Anaconda joined with Amax to form the Anamax joint venture. The joint venture continued until 1986 when Anamax sold the entire property to a real estate company during the corporate dissolution of Anaconda. By the end of the Anaconda-Anamax programs, exploration drilling totaled about 300,000 feet (91,000 meters), of which approximately 195,000 feet (59,500 meters) define the Rosemont Deposit.

In 1964, Anaconda produced a historical resource estimate for the Peach-Elgin deposit located in the Helvetia District of the northwest part of the Property. Based on assays from 67 churn and diamond drill holes, the historical estimate identified 14 million tons of sulfide material averaging 0.78% copper and 10 million tons of oxide material averaging 0.72% copper. After calculation of that resource, Anaconda and Anamax drilled approximately 140 additional diamond drill holes, but did not update the 1964 estimate. This historical estimate was not prepared to NI 43-101 requirements, but was made by a reputable major copper company and as such is believed to be reasonable as viewed in a historical context. *Augusta Resource Corporation has not done the work necessary to verify the classification of this resource and is not treating the resource figure as a NI 43-101 defined resource verified by a Qualified Person and, therefore, the resource figures should not be relied upon by investors.*

In 1977, following years of drilling and evaluation, the Anamax Joint Venture commissioned the mining consulting firm of Pincock, Allen & Holt, Inc. (PAH) to estimate a resource for the

Rosemont Deposit. The resulting block modeling calculated a historical geological resource of about 445 million tons of sulfide mineralization at an average grade of 0.54% copper, using a cut-off grade of 0.20% copper. In addition, there were 69 million tons of oxide mineralization at an average grade of 0.45% copper. Subsequent engineering designed a pit based on 40,000 tons per day production rate for a mine life of 20 years. Within the pit design, there were 317 million tons of sulfide mineralization at an average grade of 0.58% copper. In addition, there were 28 million tons of oxide mineralization at an average grade of 0.46% copper. The overall stripping ratio was 3:1 (waste:mineral). The results were described in Pincock, Allen & Holt (1977). This historical estimate was not prepared to NI 43-101 requirements, but was made by a reputable consulting firm and as such is believed to be reasonable as viewed in a historical context. *Augusta Resource Corporation has not done the work necessary to verify the classification of this resource and is not treating the resource figure as a NI 43-101 defined resource verified by a Qualified Person and, therefore, the resource figures should not be relied upon by investors.*

In 1979, Anamax carried out a resource estimate for the Broadtop Butte deposit located about a mile north of the Rosemont Deposit. Based on the assays from approximately 18 widely spaced diamond drill holes, the historical estimate identified 9 million tons averaging 0.77% copper and 0.037% molybdenum. This historical estimate was not prepared to NI 43-101 requirements, but was made by a reputable major copper company and as such is believed to be reasonable as viewed in a historical context. *Augusta Resource Corporation has not done the work necessary to verify the classification of this resource and is not treating the resource figure as a NI 43-101 defined resource verified by a Qualified Person and, therefore, the resource figures should not be relied upon by investors.*

ASARCO purchased the property in 1988, renewed exploration of the Peach-Elgin deposit and initiated engineering studies on Rosemont. ASARCO drilling on Rosemont was limited to 12 diamond drill holes.

ASARCO generated a resource estimate of the Rosemont Deposit that was incorporated into a 1997 consulting report by The Winters Company that comprised an “order of magnitude” mining study of the deposit. The resulting “mineable resource,” contained within a pit limit, totaled nearly 295 million tons at an average grade of 0.67% Cu. The plan was based on a production rate of 30,000 tons per day for a mine life of 28 years. The overall stripping ratio was 3.7:1 (waste:mineral). The results and methodology have also been described in Winters (1997). This historical estimate was not prepared to NI 43-101 requirements, but was made by a reputable consulting firm and as such is believed to be reasonable as viewed in a historical context. *Augusta Resource Corporation has not done the work necessary to verify the classification of this resource and is not treating the above resource figure as a NI 43-101 defined resource verified by a Qualified Person and, therefore, the resource figures should not be relied upon by investors.*

ASARCO sold the entire property to real estate interests in 2004, shortly before the ASARCO takeover by Grupo Mexico S.A. de C.V.

Augusta Resource Corporation involvement with the Rosemont Deposit began in 2005 and was followed shortly thereafter by an option on the Property, with the completion of the purchase occurring on March 31, 2006. During the option period in 2005, Augusta began a program to confirm the results from previous work and completed a Phase I drilling program consisting of 15 core holes. Based on the new Augusta and previous Anaconda drilling, WLR Consulting, Inc. in conjunction with Mine Reserve Associates, Inc. prepared a mineral resource estimate that was presented in an April 21, 2006 report entitled *Mineral Resource Estimate Revised Technical Report For The Rosemont Deposit, Pima County, Arizona, USA*. The resource estimate served as the basis for a June 13, 2006 scoping study by Washington Group entitled *Preliminary Assessment and Economic Evaluation for the Rosemont Deposit, Pima County, Arizona, USA.*

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

Augusta initiated a Feasibility Study with M3 Engineering & Technology Corporation of Tucson, Arizona in the middle of 2006, which was completed in August 2007. The feasibility incorporates the April 2007 resource model, from which a mineable reserve was established, along the economic evaluation of the overall mine development. The findings were presented in an August 2007 report entitled *Rosemont Copper Project Feasibility Study*. The development plan presented in the feasibility study was then incorporated into a Mine Plan Of Operation that was submitted to the United States Forest Service, Coronado National Forest, which initiated the National Environmental Policy Act process for permitting surface use of the Forest Service lands.

Augusta conducted further drilling in 2008. Twenty core holes were drilled to further define the northwestern part of the deposit. In addition, ten previously drilled geotechnical holes from Augusta's 2006 drilling campaign that had not been sampled, were sampled and analyzed. The additional drilling and sampling data was incorporated into a resource estimate that was announced in an October 23, 2008 press release, and was documented by M3 Engineering & Technology Corporation in a January 14, 2009 report entitled *Rosemont Copper Project Updated Feasibility Study*.

In late 2011/early 2012, Augusta completed a 12-hole, 18,649-foot diamond drilling program, and performed metallurgical testing. Drilling included six holes (7,698 feet) to collect metallurgical test samples, three exploration holes (5,466 feet) drilled to test geophysical targets, and three infill holes (4,711 feet), along with the additional sampling of core remaining from five older holes. The updated drill hole database was used to update the resource model in May 2012 and was the basis for the derivation of updated mineral reserves

7 GEOLOGICAL SETTING AND MINERALIZATION

The regional, local and property geology of the Rosemont deposit is complex and has been studied by numerous geologic investigations including those described by McNew (1981), Anzalone (1995), Hardy (1997), and Augusta (2007).

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

The region is characterized by a geologic history that has led to a complex structural character. Compressional tectonism during the Mesozoic and early Cenozoic Laramide Orogeny caused folding and thrust, transverse and reverse faulting, accompanied by extensive calc-alkaline magmatism. Laramide age magmatism, recorded in batholithic and smaller intrusions and their associated volcanic rocks, generated the porphyry copper deposits of the region. At Rosemont, mineralizing quartz monzonite and quartz latite intruded a package of Precambrian intrusive rocks and Paleozoic and Mesozoic sedimentary rocks at the intersection of regional basement structures.

Tertiary extensional tectonism followed the Laramide Orogeny, accompanied by voluminous felsic volcanism. Steeply- to shallowly-dipping normal faults became active during this time, including rotational listric faulting. At Rosemont, it appears that Tertiary faulting has significantly segmented the original deposit, juxtaposing mineralized and unmineralized rocks. The extensional tectonics culminated in the large-scale block faulting that produced the present basin and range geomorphology that is typical throughout southern Arizona.

The generalized regional geology of the Rosemont Property is shown in Figure 7-1. A stratigraphic column of the Rosemont District is presented in Figure 7-2. The local geology of the Rosemont Deposit is shown on a level section in Figure 7-3 and in a vertical section in Figure 7-4.

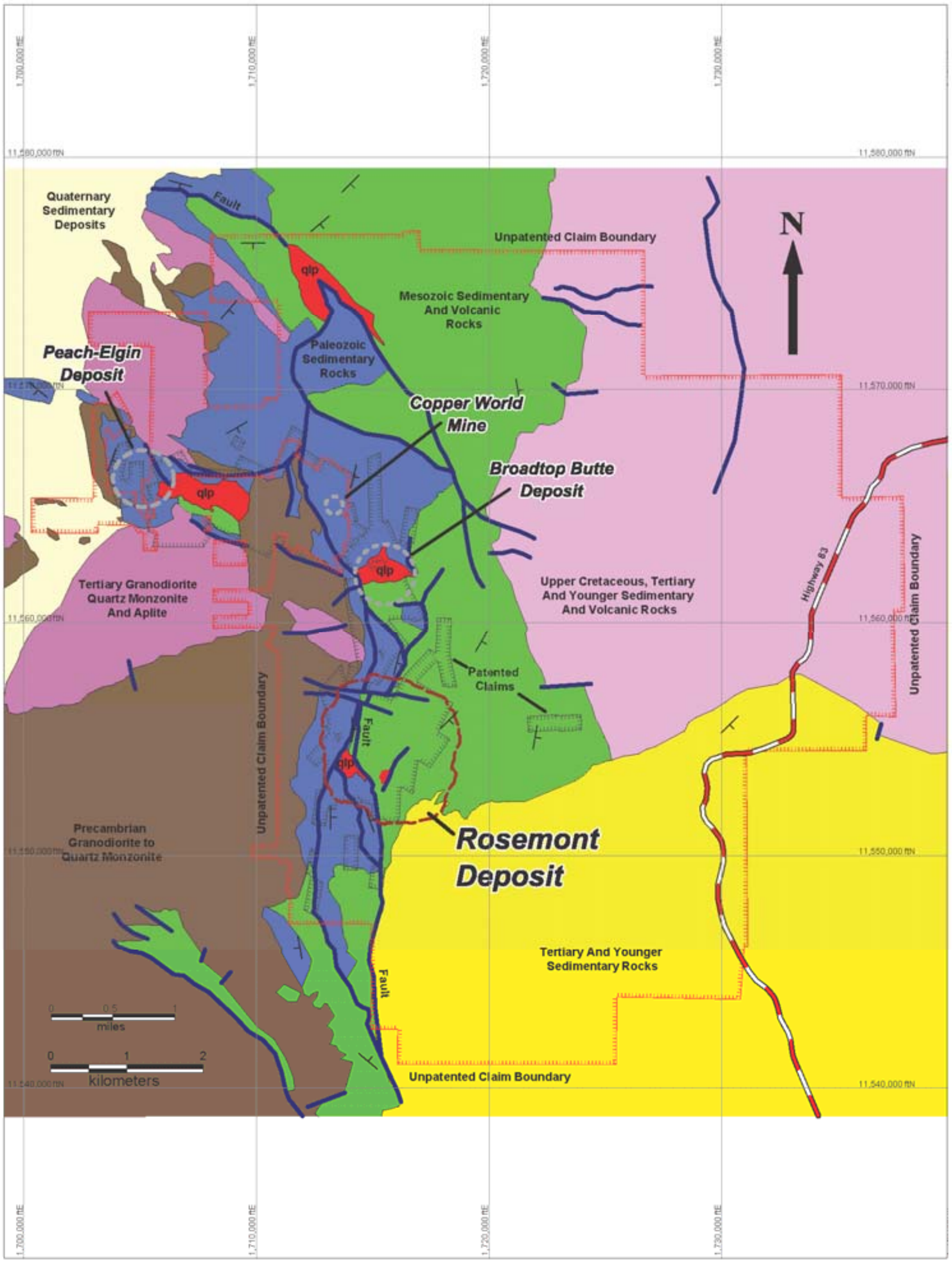

Figure 7-1: Regional Geology of the Rosemont Property

Rosemont District Stratigraphic Column

Era	Period	Formation (lith code)	Thickness (ft)	Section	Lithology	Skarn/Alteration	Mineralization	Relative Copper Content
Cenozoic	Tertiary	Gravel Unit (20)	0-500		Gravel & sand - partially consolidated paleovalley fill.	Not altered.	Not mineralized.	
		Quartz-Feldspar Porphyry (8) Intr.Bx (19)			Quartz monzonite porphyry to quartz latite porphyry intrusive (55.7-56.3 my). Assoc.intr.bx.	Qtz-ksp; qtz-ser-pyr; minor epidote-chlorite	Qtz +/- pyr-cpy-bn-mo veining, commonly oxidized. Mo disseminated and vein controlled.	
Mesozoic	Cretaceous	Willow Canyon Formation (10) Interbedded Andesite (9)	2,200		Interbedded arkosic sandstone, siltstone and conglomerate. Internal andesite flow sequence locally present.	Arkose: wk; ksp, epidote, calcite. Rare qtz veining. Andesite: stronger quartz-chlorite-epidote.	Arkose: wk limonite and secondary Cu mins. Rare qtz-pyr-cpy-bn vns. Andesite: stronger qtz-pyr-cpy-bn veining, commonly oxidized.	
		Glance Conglomerate (11)	0-1500		Limestone conglomerate with clasts of eroded older Paleozoic or Pc lithologies. Locally underlain by "Upper Plate" Concha Ls,+/- Scherrer Qtz,+/- Epitaph Fm. "Flat Fault" at base of sequence.	Weak to locally moderately marblized. Local calc-silicate alteration.	Mineralized locally (rarely).	
Paleozoic	Permian	Rainvalley Fm (18)	0-300		Fossiliferous limestone, dolomite and quartz sandstone	Serpentine-magnetite.	Minor mineralization.	
		Concha Limestone (14)	400-575		Thick-bedded, cherty, fossiliferous limestone	Marblized. Wollastonite and garnet skarns locally.	Secondary Cu and minor sulfides in BT Buttes area	
		Scherrer Fm (12)	720		Quartzite,dolomitic/calcareous, locally x-bedded; dolomite and limestone member. Basal siltstone.	Wk. chlorite in clastics. Garnet skarn after limestone.	Pyr, less common cpy and bn	
		Epitaph Fm (2)	1000		Limestone, dolomite and marl; less quartzite. Distinctive gypsum beds. Basal thick-bedded dolomite. *Gradational lower boundary.*	Strong chlorite-serpentine-magnetite; less garnet-diopside-chlorite skarn; hornfels	Bn-cpy +/- pyr vns and disseminated in skarn	
		Colina Limestone (3)	350		Dark, thick-bedded dolomitic limestone. Fossiliferous. Minor quartzite.	Marblized. Some serpentine-magnetite vns. Some garnet skarn.	Bn-cpy +/- pyr	
		Earp Fm (4)	800		Siltstone, shale, sandstone, chert-pebble conglomerate and limestone. *Grad low bdy*.	Hornfels and garnet-diopside-chlorite skarn.	Qtz-cal-chl and qtz vns with pyr-cpy-bn	
	Pennsylvanian	Horquilla Limestone (5)	1000		Thin- to thick-bedded limestone, siltstone, minor shale. Basal Black Prince Limestone	Skarn, hornfels, and marble. Garnet-pyroxene skarn with lessor chlorite and serpentinite. Local wollastonite.	Qtz and chl-serp-mag vns; bn-cpy-cc +/- pyr in vns and disseminated in skarn. Secondary Cu minerals near faults. Main mineralized formation.	
	Miss.	Escabrosa Limestone (6)	560		Thick-bedded to massive limestone, cherty.	Marble. Serpentine at faults. Garnet-diopside-magnetite at intrusive contact.	Commonly barren. Where altered, strong py-bn-cpy mineralization.	
	Dev.	Martin Formation (7)	400		Thin- to medium bedded dolomite; less limestone, siltstone and sandstone. Faulted contacts.	Marblized; minor chorite and serpentine at faults. Gnt & mixed skarn locally.	Weakly mineralized.	
	Cambrian	Abrigo Formation (13)	740-900		Thin-bedded limestone, siltstone, shale, sandstone.	Marblized. Gnt & mixed skarn locally.	Mneralized locally with py-bn-cpy.	
		Bolsa Quartzite (15)	460		Course-grained, thick-bedded quartzite	Weakly altered, unreactive.	Trace disseminated pyr and cpy	
pC	pC	Continental Granodiorite (16)			Granodiorite porphyry	Weak alteration locally.	Weak mineralization locally	

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

Figure 7-2: Rosemont District Stratigraphic Column

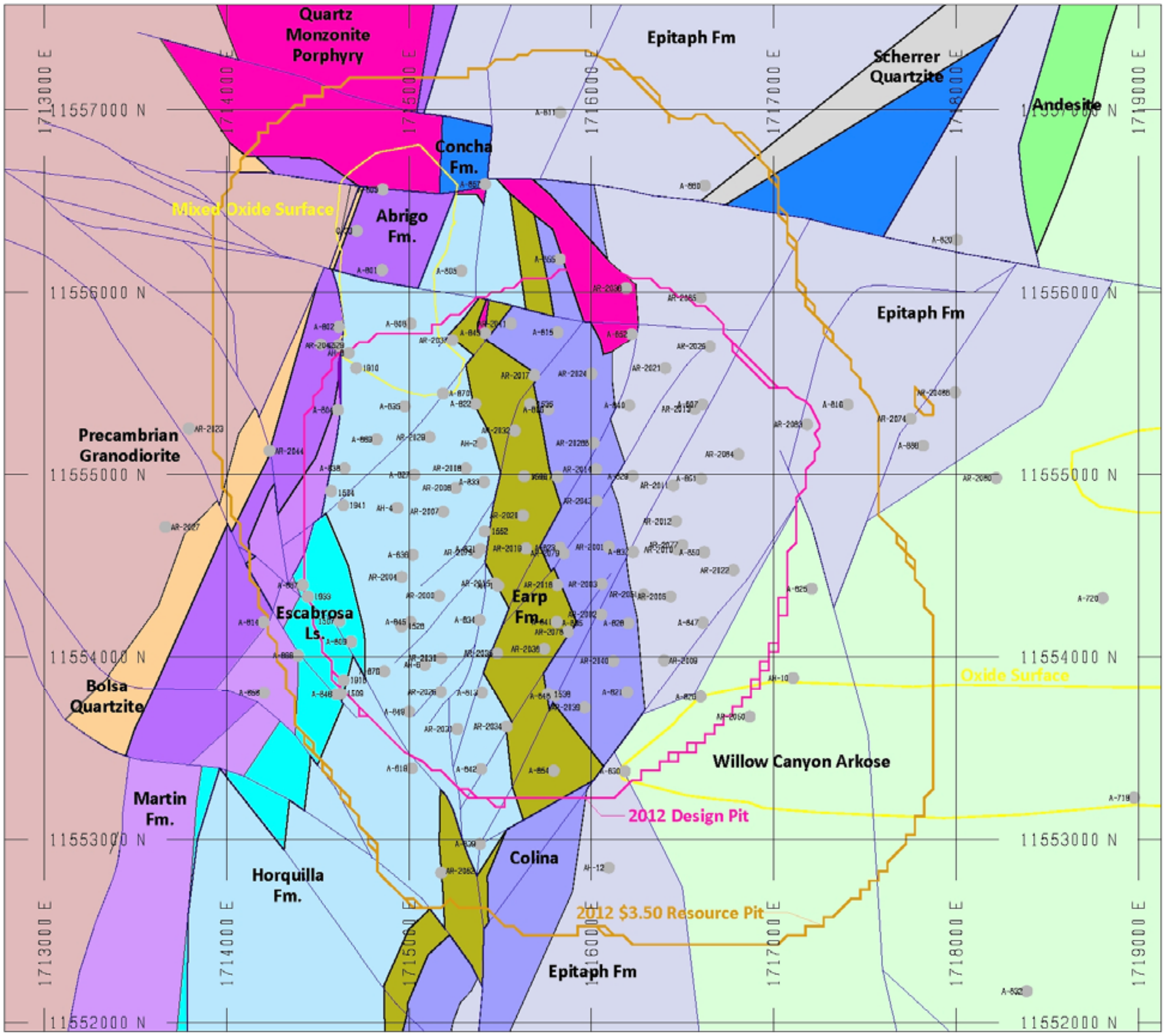

Figure 7-3: Rosemont Deposit Geology – 4,000 Foot Level Plan

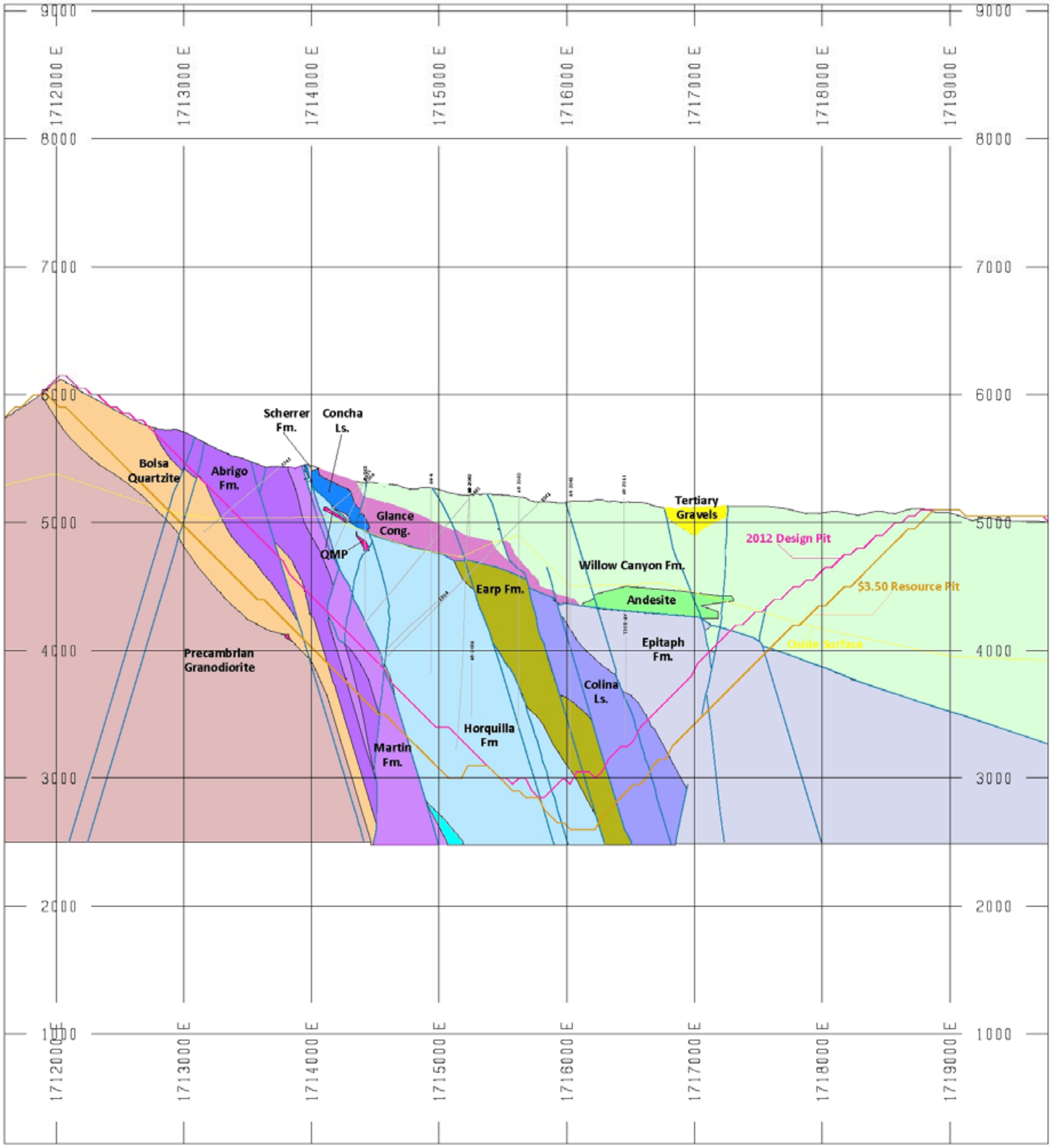

Figure 7-4: Rosemont Deposit Geologic Section – 11,554,825 E

Drilling to date has defined a significant mineral resource approximately 3,500 feet (1,100 meters) in diameter that extends to a depth of at least 2,000 feet (600 meters) below the surface. Post-mineral features partially delimit the defined resource, dividing the deposit into major structural blocks with contrasting intensities of mineralization. The north-trending, steeply dipping Backbone Fault juxtaposes marginally mineralized Precambrian granodiorite and Lower Paleozoic quartzite and limestone to the west against a block of younger, well-mineralized Paleozoic limestone units to the east. The bulk of the copper sulfide resource is contained in the eastern block of the Backbone Fault. Structurally overlying the sulfide resource is a block of

Mesozoic sedimentary and volcanic rocks that contains copper oxide mineralization. These two blocks are separated by the shallowly dipping Low Angle Fault. Other post-mineral features include a deep, gravel-filled Tertiary paleochannel on the south side of the deposit and significant thickness of Cretaceous and Tertiary volcaniclastic material to the northeast of the deposit.

The bulk of the sulfide resource on the east side of the Backbone Fault and below the Low Angle Fault is hosted in a steeply east-dipping package of Paleozoic-age sedimentary rocks that includes the Escabrosa Limestone, Horquilla Limestone, Earp Formation, Colina Limestone, and Epitaph Formation. The Horquilla Limestone is the most significant, accounting for almost half of the sulfide resource. Significant mineralization also occurs in the Earp Formation and Colina Limestone, as well as in the Epitaph Formation. Relatively minor mineralization occurs in the other Paleozoic units. To the south, the mineralization in this block appears to weaken and eventually die out. To the north, mineralization appears to narrow but continues under cover amid complex faulting. Mineralization continues irregularly to the east of the defined resource, beyond the limit of drilling and beneath an increasingly thick block of Mesozoic sediments.

The Mesozoic rocks of the structural block above the Low Angle Fault consist predominantly of arkosic siltstones, sandstones, and conglomerate. Within the arkose are subordinate andesite flows that range from a few tens of feet to several hundred feet thick. Also structurally wedged into the block at the base of the arkose is the Glance Conglomerate, a limestone-cobble conglomerate, and some occurrences of relatively fresh Concha Limestone.

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

The Mesozoic and lesser Paleozoic rocks above the Low Angle Fault are propylitically altered to an assemblage including epidote, chlorite, calcite, and pyrite. Copper mineralization is irregularly developed. The rocks are commonly deeply weathered and limonitic. The original chalcopyrite is typically oxidized to chrysocolla, copper wad and copper carbonates. Supergene chalcocite is locally present.

Silver occurs in minor, but economically significant quantities in the primary sulfide mineralization in the Paleozoic sequence. The silver is associated with the copper mineralization and is typically tied up in the chalcopyrite and bornite mineral grains. The gold content of the deposit is generally very low, but contributes to a production credit.

To the north and northwest of the Rosemont deposit are the Broadtop Butte, Copper World and Peach Elgin deposits. These are hosted by intrusive and skarn-altered Paleozoic rocks similar to those at Rosemont, and all are apparently smaller and more structurally- dissected than the Rosemont Deposit.

8 DEPOSIT TYPES

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

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

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

9 EXPLORATION

Prospecting began in the Rosemont and Helvetia Mining Districts sometime in the middle 1800s and by the 1880s copper production was recorded, which continued sporadically until 1951. By the late 1950s, exploration drilling had resulted in the discovery of the Rosemont Deposit. A succession of major mining companies subsequently conducted exploratory drilling of the Rosemont Deposit and other deposits of the region. Augusta's work on the deposit has consisted largely of verifying older sampling results, in-fill drilling, and geophysics.

In 2011, Rosemont contracted with Quantec Geoscience to conduct a Titan 24 induced polarization (IP) survey over the Rosemont deposit using the proprietary Titan 24 system, which has the ability to penetrate to depths of several hundred meters. Phase 1 of the survey involved running the system on the known Rosemont deposit to characterize the geophysical response over known mineralization. The geophysical results compared well with the results of the drill hole data from the deposit confirming its usefulness for identifying potentially mineralized targets elsewhere on the property. As a result, Rosemont continued with Phase 2 and Phase 3 geophysical surveys, extending the coverage to the east and north of the Rosemont deposit. A total of 32.4 miles (52.2 kilometers) of geophysical lines were run at Rosemont during the year.

The geophysical surveys identified several anomalies with IP responses similar to that of the Rosemont deposit. One encouraging anomaly was located to the northeast of the Rosemont deposit and was partially drill tested by holes AR-2074 and AR-2080. Drill hole AR-2074 was located where the anomaly reached its shallowest extent, approximately 800 feet (244 meters) below the surface. This hole was drilled to a total depth of 3,500 feet (1,070 meters) and tested the full extent of the IP response, with the top of the anomaly consisting of moderate to strong sulfide mineralization. Starting at 820 feet (250 meters), a 125-foot (38 meter) interval contains 0.82 percent copper, 0.025 percent molybdenum, and 0.19 ounces per ton silver.

A second drill hole, AR-2080, was subsequently drilled to test the IP anomaly further to the south. This hole intercepted low-grade mineralization and alteration in the lower part of the Arkose down to the Low Angle Fault, below which it intercepted minor low-grade skarn in the Epitaph. Deeper in the hole was a significant thickness of graphitic limestone in the Colina Limestone. A minor skarn interval at 1,385 feet consisted of a 15 foot (4.6 meters) thick interval grading 0.95% copper, 0.011% molybdenum, and 0.26 ounces per ton silver. Due to site access circumstances, neither of these holes were drilled directly over the center of the anomaly, and therefore tested only the western edge.

Additional information regarding exploration and evaluations performed on the Rosemont Deposit is presented in Section 6 – History and Section 10 – Drilling.

10 DRILLING

Extensive drilling has been conducted at the Rosemont Deposit by several successive property owners. The most recent drilling was by Augusta, with prior drilling campaigns completed by Banner Mining Company, The Anaconda Company, Anamax and ASARCO. Table 10-1 summarizes the drill holes used in the current resource estimate, as they are the holes that are within and adjacent to the deposit. Regional drill holes are archived in a separate data file.

Table 10-1: Rosemont Deposit Drilling Summary

Company	Time Period	Drill Holes		
		Number	Feet	Meters
Banner	1950s-1963	3	4,300	1,311
Anaconda	1963-1973	113	136,838	41,708
Anamax	1973-1986	52	54,350	16,566
ASARCO	1988-2004	11	14,695	4,479
Augusta	2005-2012	87	132,525	40,394
Total		266	342,707	104,457

The drill holes utilized in the database were all drilled using diamond drilling (coring) methods. In some cases the tops of the older holes were drilled using a rock bit to set the collar; in other cases the upper parts of older holes were drilled with rotary drilling, switching to core drilling before intercepting mineralization. A map showing the location of the drill holes is provided in Figure 10-1 along with a general outline of the Rosemont deposit limits. Exploration holes drilled using rotary or older "churn" drill holes were excluded from the resource database.

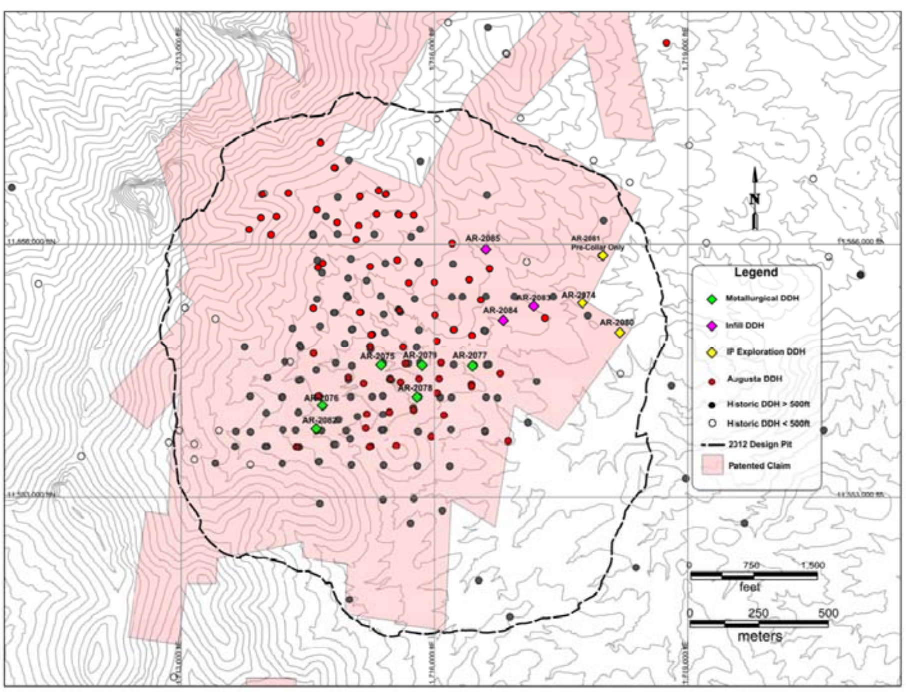

Figure 10-1: Drill Hole Locations in Rosemont Deposit

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

Most of the older Anaconda, Anamax and ASARCO drill core was still available on site or was obtained by Augusta and brought back to the Rosemont Property, where it was systematically re-logged by Augusta personnel to be geologically consistent with the current Augusta drill hole logging. Along with re-logging, this core was also resampled for additional geochemical analyses as described in Section 11 – Sample Preparation, Analyses and Security.

10.1 BANNER MINING COMPANY DRILLING

The first significant core drilling campaign on the Rosemont Property was by the Banner Mining Company, beginning in about 1961. Banner completed primarily shallow diamond drill holes, many of which were subsequently deepened by Anaconda. Three drill holes included in the resource database were shallow holes started by the Banner Mining Company that were significantly deepened during subsequent Anaconda drilling programs. These holes have a combined length of 4,300 feet.

10.2 THE ANACONDA COMPANY DRILLING

Anaconda took over Banner's Rosemont holdings around 1963 and conducted exploration at the Rosemont Deposit and in adjacent mineralized areas. Between the years of 1963 and 1973, they completed 113 diamond drill holes at Rosemont for a total of 136,838 feet. These holes were primarily drilled vertically. Down-hole surveys were conducted during drilling or immediately following drill hole completion for selected holes. Drill hole collars were surveyed by company surveyors. Anaconda drilled approximately 85 percent of the larger N-sized core (1.9-inch diameter) and 15 percent of the smaller B-sized core (1.4-inch diameter). Overall core recovery was more than 85 percent.

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

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

10.3 ASARCO MINING COMPANY DRILLING

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

10.4 AUGUSTA DRILLING

Augusta has conducted diamond drilling in several campaigns, the first starting in the second half of 2005 and continuing into early 2006 (Phase I). The second started in mid-2006 and continued into early 2007 (Phase II). The third started in December 2007 and continued to July 2008 (2008 Drilling). The most recent started in late 2011 and continued into February 2012 (2011/2012 Drilling). In total, Augusta has completed 87 core holes for a total of 132,525 feet (40,394 meters). Of these, 60 drill holes are resource holes to provide infilling of the deposit area, while six were exploration holes outside of the planned pit area, but close enough to be part of the Rosemont deposit database. The remaining 21 Augusta core holes support geotechnical (13) or metallurgical (8) studies. Layne-Christensen, Boart Longyear, and National were the drilling contractors during the Augusta campaigns.

During the 2011-2012 campaign, Augusta drilled 12 holes totaling 18,649 feet (5,684 meters). This included six holes (7,698 feet) drilled to collect metallurgical test samples, 3 exploration holes (5,466 feet) drilled to test a geophysical anomaly, and three infill holes (4,711 feet) drilled in support of a revised resource estimate. Five of the metallurgical test holes and two of the exploration holes were pre-collared with reverse circulation drilling and cased to the top of mineralization. The cored portions of the metallurgical test holes were sampled and assayed for inclusion in the resource database. The infill holes all intercepted significant intervals of copper mineralization and incrementally contributed to the known mineral resources on the northeast edge of the Rosemont deposit. The results of the exploration holes are discussed in the Exploration section of the report. The new drilling was accompanied by the further sampling of five previously drilled holes, the results from which are also incorporated into the new resource model.

Augusta drill holes were usually rock-bitted through overburden, and then drilled with larger HQ-sized core as deeply as possible and finished with NQ-sized core (1.9-inch diameter) when a reduction in core size was required by ground conditions. As described above, in the most recent drilling, some holes were pre-collared to coring depth. Also in the most recent drilling, several holes were collared with PQ-sized core and then reducing to HQ core once through the incompetent, near-surface oxide zone. This practice significantly improved hole stability and allowed most new holes to be completed with HQ. Augusta drill core was approximately 57 percent N-sized (1.9 inch diameter) and about 42 percent being larger H-sized (2.5 inch diameter), with less than 1 percent being smaller B-sized (1.4 inch diameter). Augusta's overall core recovery was approximately 95 percent.

Most of the holes were oriented vertically, although a few of the holes were inclined in order to intercept target blocks from reasonably accessible drill locations. All drill holes were surveyed down-hole with a Reflex EZ-Shot survey instrument that measured inclination/dip and azimuth direction, with readings generally taken every 100 feet down the hole during 2008 and every 200 or 500 feet down the hole during 2005, 2006 and 2011-2012. Phase I drill hole collar locations were surveyed by Putt Surveying of Tucson, Arizona, while all later drilling locations were measured by Darling Environmental & Surveying.

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

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

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

11.1 BANNER, ANACONDA AND ANAMAX SAMPLING AND ANALYSES

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

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

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

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

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

11.2 ASARCO SAMPLING AND ANALYSES

The ASARCO drill core was routinely analyzed for total copper, acid-soluble copper and molybdenum. Oxide zone core does not appear to have been analyzed differently than the sulfide zone core. The core was sampled with preference towards a 10-foot sample length, but longer or shorter intervals were sometimes used. The ASARCO drill core was apparently logged and sampled in much the same style as is described above for the Banner, Anaconda and Anamax core.

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

11.3 AUGUSTA SAMPLING AND ANALYSES

11.3.1 Augusta Core

For the 2005 Phase I and the 2006 Phase II drilling programs, sampling of Augusta drill holes took place at the Rosemont Ranch sampling facility. The 2008 and later drill hole sampling took place at the Hidden Valley Ranch sampling facility. Geotechnical logging was performed on all core drilled by Augusta to systematically quantify Rock Quality Designation (RQD), core recovery, fracture frequency, core hardness, joint condition and large-scale joint expression. Core logging geologists familiar with the project recorded the rock type, alteration, mineralization, and structure. After logging, the geologist assigned and marked the sample intervals and cut-lines directly on the core and on the core box interior with a black marker. Each sample was given a unique, sequenced sample number with the footage noted in a sample tag booklet and in Excel-based spreadsheets. The drill core boxes were then photographed with a digital camera.

Augusta core was sampled at even five-foot intervals, except where massive copper or molybdenum veining, structures or lithologic breaks warranted special investigation through the selection of shorter intervals. Sample intervals would return to footages evenly divisible by five as soon as possible thereafter. This tended to occur in earlier campaigns and was not a practice during more recent campaigns. One exception is the sampling from the 2012 metallurgical holes, which were sampled at 10-foot intervals.

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

Core was analyzed using a geochemical suite that varied depending on whether or not the core retained its primary sulfide mineralization or had been oxidized, similar to the approach described above for Anaconda. In the oxidized zone, the core was routinely analyzed for total and acid-soluble copper. Sulfide zone core was analyzed for total copper, molybdenum and silver. In 2005 and 2006, some core was also analyzed for gold, although that was discontinued when the gold content had been adequately characterized and the cost of additional gold analyses was no longer warranted.

Geochemical analyses for Augusta-drilled core and for the Augusta resampling of the Anaconda, Anamax, and ASARCO core were primarily performed by Skyline Assayers and Laboratories (Skyline) in Tucson, Arizona. During 2005, Skyline was formally known as Actlabs-Skyline and had been owned by ACTLABS (Ancaster, ON, Canada) since 1997. Skyline became independent of ACTLABS in January, 2006. Skyline is accredited in international quality standards through ISO/IEC 17025, with CAN-P-1579 for specific registered tests through the Standards Council of Canada. Skyline is considered to be a reputable and trustworthy facility and is used by a number of major and junior mining companies in the southwest area of the United States.

Augusta had both primary and secondary (duplicates) analyses done at Skyline in 2006 and 2007. ALS Chemex (Vancouver, BC, Canada) analyzed duplicate checks samples in 2005. ALS Chemex has accreditation through ISO 9001:2000 in North America.

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

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

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

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

11.3.2 Banner, Anaconda, Anamax, and ASARCO Core Resampling and Analysis

Augusta extensively resampled core drilled by Anaconda, Anamax, and ASARCO, most of which was available, to fill-in missing analytical information and to validate the older analyses. Resampling of pre-Augusta drill holes took place at the Hidden Valley Ranch sampling facility in 2006. Augusta geologists identified intervals requiring additional (infill) analyses by referring to the previous logging and analyses for the core. New Augusta sample intervals were assigned unique, sequenced sample numbers from sample tag books in which hole identification and interval footage were recorded, and this information was recorded in an Excel-based spreadsheets. The core boxes were carefully photographed using a digital camera, and the photos were inspected and archived before samples were collected. The assigned intervals were measured and collected by sampling technicians, taking the entire remaining core with the exception of some small, representative archive samples. The individual samples were placed in plastic sample bags marked with the new sample number and the paper tags from the sample books were placed in the sample bags before the bags were sealed with plastic ties.

Whenever possible, the sample intervals for additional analyses conformed to pre-existing sample intervals, allowing the old and new data to be easily combined and compared. Augusta required all samples to be seven feet or shorter. If only intermittent samples had previously been collected (i.e., a five-foot sample every 20-30 feet), unsampled original intervals were divided into multiple new sample intervals of approximately five feet in length, preserving the starting and ending footages of the original sample intervals. If core was missing, either lost or previously taken for metallurgical work, Augusta sample intervals were aligned to reflect the missing core intervals.

Oxide zone intervals were analyzed for both total and acid-soluble copper if total copper was estimated to be >0.1% Cu, but acid soluble copper data were not available. All sulfide zone drill core from within the deposit area that had not been analyzed for both total copper and molybdenum was sampled and analyzed to provide complete, continuous copper and molybdenum data.

Silver and to a minor extent gold were analyzed for drill hole intervals that were missing these values in the historic data. Sulfide zones for which previous copper analyses indicated an interval contained greater than 0.2% Cu was sampled by means of a composite representing a 50-foot continuous interval length. Gold analyses were discontinued late in 2006 after the gold mineralization was sufficiently characterized. For the purposes of silver and gold analyses, the composite sample intervals were combined into length-weighted 50-foot sample before analysis, thereby reducing the total number of samples. This compositing was performed on pulp samples at the analytical laboratory using relative weight contributions for each component sample calculated by Augusta geologists.

The analytical procedure for the core resampling program were the same as described above for the Augusta drill holes.

11.4 SAMPLE HANDLING AND SECURITY

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

For the new Augusta drilling program, the drilling contractors kept the core in a secure area next to the drill rig before delivering it to the Rosemont Ranch (2005, 2006) or Hidden Valley (2008-2012) sampling facility, approximately three miles from the drilling area. At the Rosemont Ranch facility in 2005 and 2006 and subsequently at Hidden Valley in 2008, samples were logged, marked, cut and placed in sample bags by geologists and helpers contracted by Augusta. At both locations, for programs through 2008, the samples were kept in locked storage units on site until they could be transported to the analytical laboratory in Tucson. The logging and sampling areas were kept under closed-circuit video surveillance to provide a record of the personnel that had accessed the logging and sampling areas. Additional security was afforded by ranch personnel that oversaw the premises at night. For the 2011-2012 drilling, the locked storage units and video surveillance were superseded by 24 hour-per-day private security guards. No core handling or core security issues were experienced during the drilling or sampling programs.

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

11.5 QUALITY ASSURANCE AND QUALITY CONTROL

11.5.1 Historic Protocols

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

11.5.2 Augusta Protocols

Rosemont verified the accuracy and precision of its geochemical analyses by inserting standards of known metal content in the sample stream at periodic intervals and by reanalyzing approximately 5% of all samples to check the repeatability of results. Rosemont's QA/QC protocol was provided initially by Kenneth A. Lovstrom (deceased), Geochemist. After January 2006, the protocol was subsequently directed by Shea Clark Smith, Geochemist and Principal of Minerals Exploration & Environmental Geochemistry. Details of Rosemont's QA/QC procedure are as follows:

1. Standards were submitted with a frequency of one per 20 samples. The inserted standards were chosen to be similar in grade to the drill holes samples that they accompanied whenever possible.
2. Blank samples were submitted with a frequency of one per 40 samples.
3. Approximately 5% of all samples were reanalyzed in what was called the Check Assay Program.

As part of the protocol, whenever standards or blanks returned from the laboratory with values significantly different from what was expected, the standard or blank pulp was resubmitted to the laboratory along with two samples that occurred on either side of the questionable standard or blank in the sample stream. In most cases this process validated the initial analyses. If not, the entire job was rerun, which only occurred in a couple of rare instances since 2005.

In addition to Rosemont's standards and repeat analyses, further QA/QC was provided by the results from other standards inserted into the sample stream by the assay lab, Skyline Assayers and Laboratories, Tucson, Arizona (Skyline). The results from those standards are reported on Rosemont's assay certificates.

11.5.3 External Augusta Standard Reference Materials

Since 2005, Rosemont has used 14 standard reference materials (SRM) incorporating a range of copper, molybdenum, and silver concentrations that approximate the range of metal values encountered in Rosemont's analytical samples. The SRM used in 2005 were KM-5, GRS-3, GRS-4, OC-43, OC-48, R1, and R2. Of these, KM-5, GRS-3, GRS-4, OC-43, OC-48 were developed by Mr. Lovstrom and the detailed analytical results on which their Certified Values are based are no longer available (they were only used minimally).

A new suite of SRM was created specifically for the Rosemont Deposit in 2005 and 2006 and includes: R1, R2, R4A, R4B, R4C, R4D, R4E, R4F, and R4G. These were prepared at MEG Labs (Carson City, NV) from naturally mineralized rock that had been collected at the Rosemont Deposit. The metal values for these standards were established by a round robin analytical program, compiled from a minimum of 25 samples of each SRM that had been sent to 5 or more laboratories. The average values and standard deviations calculated from the round robin program establish MEG Labs Certified Values for the R1, R2, and R4 suite of standards. It is noted that there is a good balance between the known copper grades of the R-series SRMs and the average economic metal concentration in the drill samples being run with these SRMs.

For Rosemont samples, the performance of mineralized SRM in the analytical stream is good to excellent for copper and molybdenum. Figure 11-1 shows the copper results for four of the more widely used external standards. Table 11-1 summarizes the performance of each of the standards used during the various sampling campaigns. It is noted that for little-used (2005 only/limited number of samples) Lovstrom standards GRS-3 and GRS-4, the reported total copper concentrations are up to ten percent lower than expected for copper. The issues with these SRMs were quickly identified and their use was terminated.

For the most widely-used standards, copper values returned are typically within about +/-0.01% Cu of the expected value. No significant, systematic bias in copper values is apparent across the range of results. These results show that copper, by far the most important metal in the deposit, has reliable assays.

Molybdenum results are good, with molybdenum from mineralized SRM generally reported within +/- 0.002% Mo of the established value for most standards used. In the low-grade range of molybdenum commonly found in the deposit, a relatively larger percent variability is commonly observed. Generally, the analytical laboratory is reporting acceptable levels of molybdenum, with a slight tendency to report lower than the expected SRM concentrations of molybdenum.

Silver results are fair to good, with silver from mineralized SRMs generally reported within +/- 0.015 opt Ag of the established value for most standards used. The results for silver, a minor economic constituent of the deposit, are influenced by the low-grade range typically being considered. In this low-grade range, the relative percent variability is higher, particularly as values get closer to the analytical detection level. Silver concentrations from the analytical laboratory are within acceptable levels of silver, but there is a tendency to sometimes report lower than expected SRM concentrations. The analysis of Ag at the level of several parts per million or less is inherently difficult, affected by issues such as sample-dependent correction requirements, sample homogeneity, and reliability of sample digestion.

The performance of the SRMs in the analytical stream was acceptable for the three economic metals under consideration.

11.5.4 Internal Skyline Laboratory Standard Reference Materials

Internal standards used by Skyline indicate that accuracy is within tight tolerances of 0.00% to 0.03% for copper, 0.000% to 0.001% for molybdenum, and 0.000 to 0.015 opt for silver. Copper shows no systematic bias.

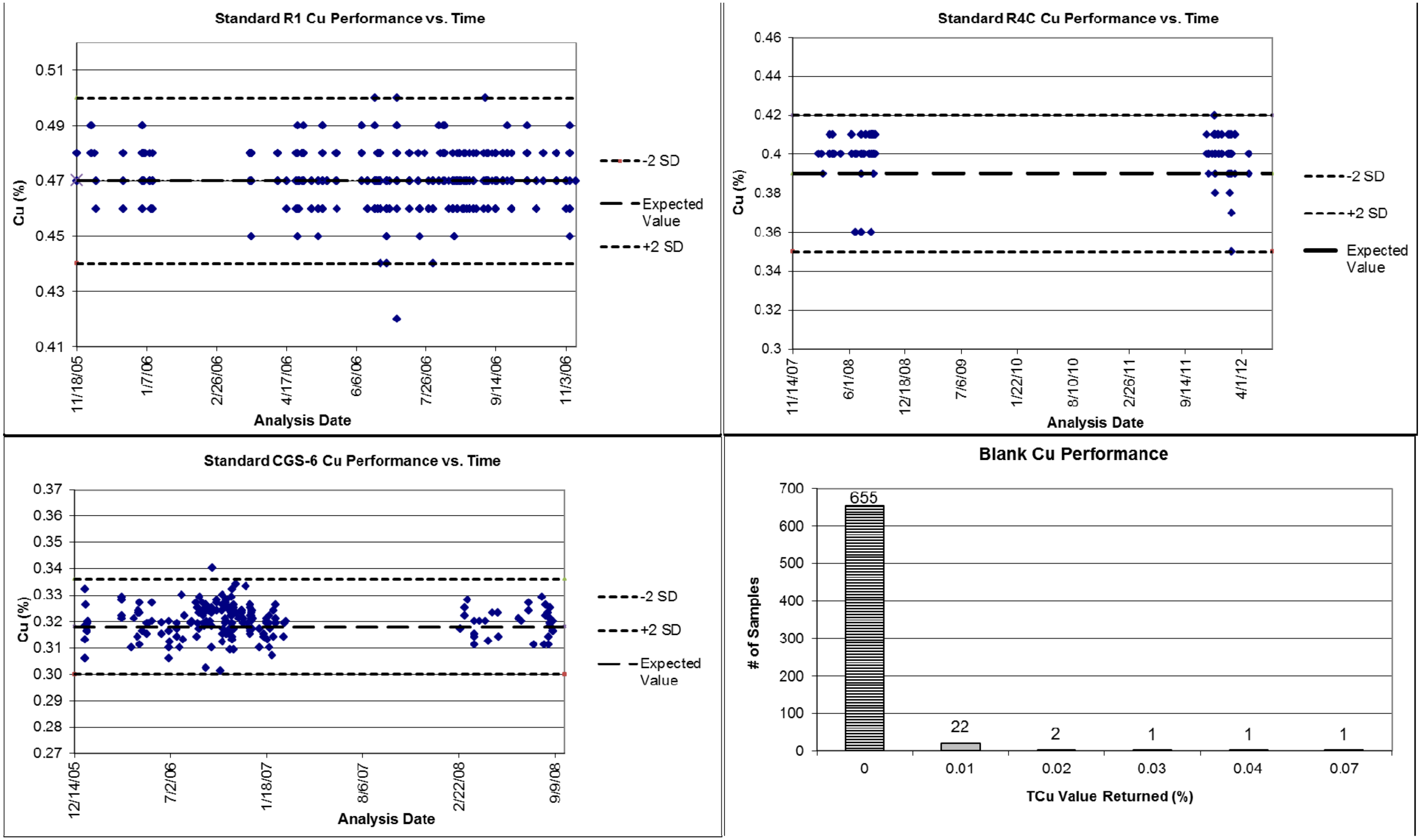

Figure 11-1: QA/QC Standard and Blank Performance

Table 11-1: Rosemont and Skyline Standards Qa/Qc Results

Stan ID	N[4]	Cu (%)				Mo (%)				Ag (opt)			
		Expected	Found	% Var	Notes[2]	Expected	Found	% Var	Notes[2]	Expected	Found	% Var	Notes[2]
Rosemont Standards													
R1	462	0.47	0.47	0	1 rej	0.025	0.025	0	6 rej	0.149	0.149	0	6 rej
R2	278	0.72	0.71	-1		0.017	0.018	6		0.207	0.204	-1	
R4A	2	1.43	1.40	-2		0.032	0.031	-3					
R4B	54	0.57	0.57	0	1 rej	0.030	0.029	-3	1 rej	0.114	0.110	-3	1 rej
R4C	133	0.39	0.40	3		0.033	0.031	-6		0.090	0.072	-20	
R4D	89	0.30	0.30	0		0.018	0.018	0		0.069	0.059	-14	
R4E	135	0.22	0.21	-5	1 rej	0.011	0.009	-18		0.051	0.037	-27	1 rej
R4F	106	0.14	0.14	0	3 rej	0.010	0.009	-10		0.041	0.029	-30	
R4G	22	0.07	0.07	0		0.016	0.014	-13		0.035	0.016	-54	
Blank	792	27/655 >dl[5], those average 0.01%				74/536 >dl, those average 0.002%				269/508 >dl, those average 0.010 opt			
Lovstrom Standards[3]													
KM-5	90	0.99	1.01	2									
GRS-3	19	1.23	1.12	-9									
GRS-4	18	2.02	1.9	-6									
OC-43	25					0.035	0.034	-3					
OC-48	21					0.078	0.074	-5					
Skyline Standards[1]													
CGS-2	255	1.18	1.16	-2	1 rej								
CGS-3	271	0.65	0.65	0	1 rej								
CGS-4	184	1.95	1.93	-1	1 rej								
CGS-6	216	0.32	0.32	0	1 rej								
GXR-1	121									0.905	0.867	-4	
GXR-2	121									0.496	0.481	-3	
GXR-4	118									0.117	0.106	-9	
HV-2	295	0.57	0.58	2		0.048	0.047	-2					
CGS-25	25	2.19	2.16	-1									
CM-1	53	0.85	0.85	0		0.076	0.076	0					
CM-2	33	1.01	1.00	-1		0.029	0.028	-3					
CM-8	52	0.36	0.37	2		0.016	0.016	0					
Cu-121	15	0.97				0.042				0.964	0.975	1	
Cu-122	34	0.79	0.77	-3		0.076	0.077	1		2.132	2.143	1	
Cu-123	78	0.49				0.051				1.256	1.275	2	

1. These standards are run by Skyline as part of their ongoing Qa/Qc program and are reported at the request of the client. Standards GXR-1, 2, and 4 are only pertain to Ag composite samples from the core reassay program (2006-2007), whereas the other Skyline standards reflect a broader scope of analytical work.

2. Reject ("rej") samples are those that are removed from the population of standards for the purposes of average calculations and charting. They are interpreted to have been physically-switched or misidentified by either Rosemont or Skyline personnel.

3. Lovstrom standards were used in 2005 only. They were provided by Ken Lovstrom, geochemist (deceased). Documentation of their metals content is lacking.

4. N is the number of times a standard was used in the Qa/Qc program.

5. dl is the detection limit for the analytical analysis.

11.5.5 Blank Samples

Blanks, made of barren quartz sand, do not contain metal concentrations above the limit of detection. These were submitted with drill cuttings with a frequency of one per 40 drill samples. Blanks test the laboratory performance at the limit of detection and can reveal problems with contamination between samples.

Out of 655 copper assays reported for the blanks, only 27 (4%) reported concentrations for copper above the limit of detection (0.01% Cu), and 22 of those 27 values were at the threshold limit of 0.01% Cu. The highest copper value reported for a blank was 0.07% Cu.

For molybdenum, 74 of 536 (14.0%) values reported for the blank were above the limit of detection, and 64 of the 74 values were at the threshold value of 0.001% Mo. The highest value reported for a blank was 0.013% Mo, and the average of non-blank values was 0.002% Mo.

For silver, there were 269 out of 508 (53%) values reported for the blank that were above the limit of detection, of which 77 values were at the threshold value of 0.003 opt Ag (0.1 ppm Ag). The average concentration of those samples that reported above the detection limit is about 0.010 opt Ag (0.34 ppm). These silver results further demonstrate the variability of very low-level (non-economic) Ag analyses. It is noted that the performance of the blanks at Skyline has improved with Rosemont QA/QC discussions that have allowed for improvement to the analytical procedures.

The performance of the blanks in the analytical stream was acceptable for the three economic metals under consideration. The incidence of blanks reporting metals values was very minor, with the exception of silver, where economically insignificant low-grade values were sometimes reported.

11.5.6 Check Assays (Pulp Rerun Analysis)

Approximately 5% of the samples (new drilling and resampling by Augusta) were resubmitted to Skyline Assayers and Laboratories of Tucson, Arizona, at the end of each drilling campaign in what was called the Check Assay Program. Samples consisted of the originally prepared pulp material that was resubmitted for analysis. All samples analyzed were total copper, acid-soluble copper, molybdenum and silver.

A suite of standards described below accompanied each analytical batch in the laboratory during the Check Assay Program. In each batch there are 16 core samples for reanalysis, accompanied by up to four Augusta standards and as many as five Skyline internal standards. In addition to the inclusion of standards, further QA/QC validation of the check assays is provided by Skyline's reporting of repeat AA readings for the first and last of Augusta's 20 samples and standards in each analytical batch.

Generally, results for total copper compare quite precisely and with no significant bias from values being systematically higher or lower than original values. Considering samples with copper contents greater than the detection limit, 91 percent of the samples have comparison assays that are within 10% of each other. This provides good confidence in the repeatability of the copper analyses. The summarized results are presented in the attached Check Assay Summary Table shown below. The check assay results are summarized in Figure 11-2 and in Table 11-2.

Molybdenum results show somewhat greater deviation (lower precision) with 61 percent of the samples returning comparison assays within 10% of each other. Again, there is no significant bias indicated by the average difference between the original and check assay. It is noted that the molybdenum grades being analyzed are very low and the associated broader variability observed is typical in these lower grade ranges.

Silver comparisons indicate less precision, with 44 percent of the samples returning comparison assays within 10% of each other. As in the other elements, there is no significant bias for the check value to be higher or lower than the original values. It is noted that the silver grades being analyzed are very low and approach threshold limits, were broader variability would be expected. A number of silver checks are different enough to indicate possible sample switches. That issue has not introduced a bias in data.

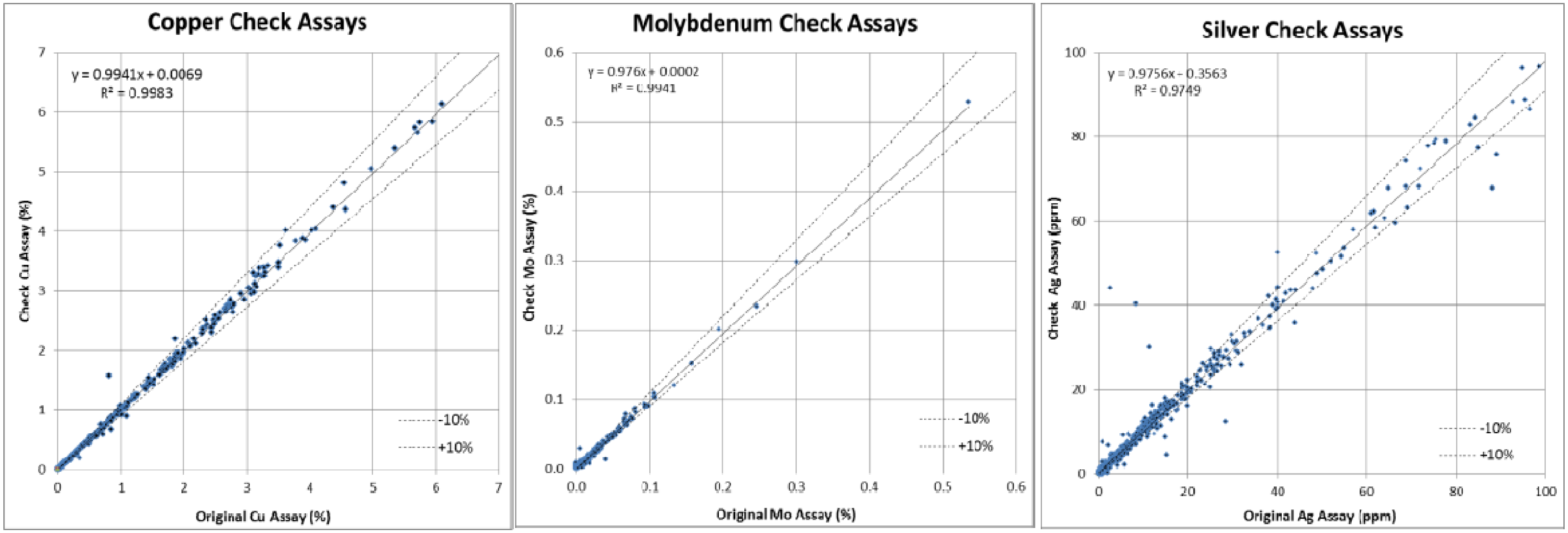

Figure 11-2: Check Assay Performance

Table 11-2: Check Assay Performance Summary

Augusta Resource Rosemont Project

Check Assay Summary Table[1]

Copper Check Assay Summary

Cu > Detection Limit (N=572)	% Variance[2]	≤5	≤10	≤25	≤100
	n	462	523	552	572
	% of N	81	91	97	100
Original Assay-Check Assay: Average Value = 0.00% Cu					

Molybdenum Check Assay Summary

Mo > Detection Limit (N=429)	% Variance[2]	≤5	≤10	≤25	≤100
	n	191	260	350	429
	% of N	45	61	82	100
Original Assay-Check Assay: Average Value =0.000% Mo					

Silver Check Assay Summary

Ag > Detection Limit (N=732)	% Variance[2]	≤5	≤10	≤25	≤100
	n	188	321	525	732
	% of N	26	44	72	100
Original Assay-Check Assay: Average Value = -0.1 opt Ag					

Notes:

1. 5% of 2005-2006 samples from Augusta's AR drill holes and the Anaconda/Anamax/Asarco drill hole resample program were submitted for check analyses.

2. % Variance = Absolute value of 100*((Origninal Assay-Check Assay)/Original Assay); % variance is always positive in this table.

11.6 QA/QC SUMMARY

The analytical QA/QC program demonstrated that the copper, representing approximately 80% of the deposit value, is well behaved based on sample QA/QC work. Molybdenum and silver, accounting for approximately 15 percent and 5 percent of the deposit value, respectively, experience a little more variability, which can largely be attributed to the low concentration levels of these metals in the samples that are analyzed. Overall, all three metals are considered reliable for resource estimation work. The most recent drilling program results are similar to the overall QA/QC results obtained since 2005 by the various Augusta sampling campaigns.

12 DATA VERIFICATION

Augusta took a number of steps to verify the results of earlier exploration results by other companies. These previous efforts were conducted by recognized major companies and it is believed their work was conducted to industry standards at that time. Augusta's own work was conducted with appropriate sampling handling and QA/QC measures to ensure that resulting data were reliable. Quality control measures for sample assaying are described in detail in Section 11.

12.1 TEN HOLE RESAMPLING PROGRAM

Anaconda, and to a lesser extent Anamax, and to a minor extent ASARCO generated a portion of the geochemical data in the resource database. Augusta performed significant resampling and assaying of the older drill holes to fill in missing data, but typically did not generate replacement data. In order to directly validate the old data with comparable values from Skyline Laboratories, Augusta reanalyzed 10 historic drill holes (5 Anaconda (A-xxx), 4 Anamax (xxxx) and 1 ASARCO (AH-x)) in their entirety. The remaining ½ split of core material from the 10 historic holes was collected in sample intervals corresponding to the original sample intervals and assayed for Cu, Mo and Ag. For silver, several of the historic holes did not have previous analyses to compare with the new values. The results are tabulated in Table 12-1.

Table 12-1: Ten Hole Resampling Program Summary

	Cu (%)		Mo (%)		Ag (opt)	
	Old Hole	Augusta	Old Hole	Augusta	Old Hole	Augusta
All Hole Average	0.50	0.48	0.022	0.017	0.31	0.20
Individual Drill Hole Comparisons						
A-804	0.45	0.43	0.007	0.005	---	---
A-813	0.51	0.52	0.033	0.028	---	---
A-821	0.57	0.53	0.038	0.029	---	---
A-834	0.48	0.45	0.018	0.018	---	---
A-858	0.35	0.34	0.018	0.016	---	---
1485	0.43	0.38	0.017	0.005		
1508	0.94	0.90	0.024	0.023	0.31	0.23
1916	0.39	0.39	0.026	0.015	0.31	0.17
1917	0.23	0.25	0.023	0.012	0.30	0.13
AH-4	0.37	0.38	0.009	0.009	---	---

Generally, new (Rosemont) values for total copper are quite similar to the old (Anaconda, Anamax, ASARCO) values. Overall, the averages for all comparison intervals are 0.50% Cu (old) vs. 0.48% Cu (new). It is believed that the small amount of difference is due to variability in the distribution of mineral grains in the core.

Molybdenum results show more variability between old and new values than do the copper results, with a tendency for old molybdenum values to be higher than new molybdenum values. The difference is somewhat attributable to the presence in the database of old (1964-1983) low-grade, x-ray analyses from Anaconda and Anamax laboratories. Typically, samples were screened with the x-ray technique, and only run with the more accurate wet chemical technique if the x-ray results indicated a value greater than 0.020% Mo or 0.20% Cu.

A statistical study was conducted by MRA (2006) on all available data to determine the correlation coefficient between XRF and wet chemical values for both Cu and Mo. The study showed excellent agreement with correlation coefficients of 0.944 for Cu and 0.874 for Mo. MRA concluded that these results indicate that the lower grade XRF values would be valid for use in grade estimation in the model for both Cu and Mo. It is noted that the XRF values would now usually only remain in the database for lower grade intervals of limited relevance to the project viability.

Silver results also show greater variation between old and new values than do the copper results, although only three holes can be compared because of the lack of historical silver values for these particular holes. It is noted that the combination of Augusta's own drilling and Augusta's resampling of older core where silver was missing, has resulted in a majority of the silver data in the current resource database to be the newer Skyline analyses.

12.2 ADJACENT (METALLURGICAL) COMPARISON HOLES

To further compare historic data to newer Augusta data, the results from metallurgical holes drilled by Augusta in 2011 were compared to adjacent historic holes. The holes were not necessarily twins, as they had separation distances that ranged from 13 to 39 feet. The metallurgical holes were assayed as are typical resource drill holes, and provide 6 pairs of adjacent drill holes for comparison of the metal contents of the historic holes. Table 12-2 compares the adjacent hole metal contents.

Table 12-2: Adjacent Drill Hole Comparison Summary

Drill Holes Compared	Interval	Thickness	Cu (%)	Mo (%)	Ag (opt)
Average Met Hole Value:			**0.656**	0.015	**0.19**
Average Historic Hole Value:			0.633	**0.018**	0.19
AR-2075 (met hole)	490'-1590'	1100'	**0.63**	0.017	**0.20**
A-831	507'-1592'	1085'	0.55	**0.019**	0.17*
(25' collar spacing)					
AR-2076 (met hole)	320'-1100'	780'	0.48	**0.016**	0.18
A-809	305.5'-1126'	820.5'	**0.55**	0.012*	**0.26***
(17' collar spacing)					
AR-2078 (met hole)	720'-1350'	630'	0.52	0.011	0.12
A-841	736'-1350'	614'	**0.65**	**0.020**	**0.12**
(18' collar spacing)					
AR-2079 (met hole)	740'-1490'	750'	**0.99**	0.013	**0.22**
A-823	740'-1491'	751'	0.71	**0.025**	0.17*
(39' collar spacing)					
AR-2082 (met hole)	228-810	582'	**0.57**	0.011	**0.24**
A-846	228-813	585'	0.45	**0.016**	0.18*
(13' collar spacing)					
AR-2077 (met hole)	810'-1320'	510'	0.75	**0.020**	0.20
AR-2010	818'-1323'	505'	**0.89**	0.018	**0.27**
(22' collar spacing)					
* historic data incomplete					

The average copper grade of all metallurgical drill holes is 0.656% Cu, compared to 0.633% Cu for the average of all adjacent drill holes. For most comparison pairs, copper values show only minor variability. It is believed that the small amount of difference is due to natural geologic variability that exists over the 13 to 39-foot spacing between the holes. No bias is indicated, as there was no systematic difference between the older hole and the Augusta hole when comparing the grade pairs.

The average molybdenum grade of all metallurgical holes is 0.015% Mo, compared to 0.018% Mo for the adjacent drill holes. For most comparison pairs, molybdenum values show moderate variability, most of which can be attributed to natural geologic variability between the holes. There is also no bias indicated when comparing the grade pairs.

Average silver values show negligible overall differences in the overall averaged values for all comparison pairs. Both metallurgical holes and adjacent holes contain 0.19 opt Ag on average. Individual pairs, however, show moderate variability, some of which is attributable to the low silver levels being analyzed for and some being natural geologic variability. Comparison of the grade pairs does not indicate a bias.

This comparison of the metallurgical drill holes with adjacent drill holes contributes to the validation of the three metals of economic interest. Overall, the grades are comparable and in all cases are considered to reasonably reflect the grades of the Rosemont deposit.

12.3 DRILL HOLE DATA ENTRY VALIDATION

In order to assess the integrity of the data entry in the drill hole database, the database has been inspected by MRA after the completion the 2005, 2006 and 2008 drilling programs. MMTS inspected the more recent data entry related to the 2011 drilling. 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. The number of data errors found was minimal from all of these data entry checks and some required relatively insignificant changes. From this it was concluded that the data entry into the drill hole database was reliable for use in the resource modeling.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

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

The samples tested both as deposit composites and individual variability samples and are considered to fairly represent the deposit. Details of the individual samples used to make up composites are described in the individual test reports.

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

- Horquilla
- Earp
- Colina
- Other including Epitaph and Escabrosa

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

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

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

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

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

The comminution test program consisted of:

- JK Drop-weight and Abrasion Test

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

Grinding mill sizing parameters were provided to mill manufacturers for use in their mill sizing methods. The mill sizing parameters are shown in Table 13-1.

Table 13-1: Grinding Mill Sizing Parameters

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

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

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

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

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

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

During 2008, flotation tests were conducted at MSRDI on composite samples of five individual rock lithology samples and one composite sample representing the ore expected to be processed during the first three years of process plant operation. The test program was designed to examine the process of producing molybdenite concentrate. The bulk (copper-molybdenite) flotation concentrate from Horquilla ore produced a molybdenite concentrate grading 52.7% molybdenum with a 93% molybdenum recovery from bulk concentrate. The results of testing the other samples indicate lower molybdenite concentrate grades and with variable molybdenite recovery from the bulk concentrate with the procedure used. The results of the testing are presented in Table 13-2.

Table 13-2: Molybdenite Flotation

Molybdenite Flotation				
	Concentrate Assay %			Recovery %
Sample ID	Cu	Mo	Insol	Mo
Horquilla	0.44	52.7	1.8	93.0
Colina	0.70	26.5	16.9	96.5
Earp	0.50	42.8	6.5	93.0
Epitaph	0.30	39.3	17.5	55.7
Escabrosa	0.50	27.9	25.8	84.8
1-3 Yr Composite	0.06	41.6	13.5	96.5

In 2012, a metallurgical test program was designed to prepare composite samples representing four periods of mine production and test them by bench scale test procedures. The test procedures followed the treatment methods proposed for the process plant. The metallurgical test composite samples were prepared from half-core drill hole segments from six holes drilled in late 2011. The drill core segments were selected so that the composite samples would have the ore grade, lithology, and spatial characteristics of ore predicted to be produced during the mine operating periods of years 1 through 3, years 4 through 7, years 8 through 12, and years 13 through 21.

The composition of the composite samples by lithology is shown in Table 13-3.

Table 13-3: Lithology of Composite Samples

Lithology	Composite Samples Representing Mine Production Years			
	1 through 3	4 through 7	8 through 12	13 through 21
Epitaph	-	-	10%	16%
Colina	-	11%	17%	25%
Earp	16%	28%	23%	16%
Horquilla	84%	61%	50%	43%

The result of closed circuit flotation tests on the year 1 through year 3 composite sample indicates a final copper recovery of 87.9% and a molybdenum recovery of 62% in a final bulk concentrate grading 41% copper, 1.02% molybdenum, and 502 ppm silver.

The result of closed circuit flotation tests on the year 4 through year 7 composite sample indicates a final copper recovery of 81.2% and a molybdenum recovery of 2.5% in a final bulk concentrate grading 44% copper and 0.047% molybdenum. (The anomalous value for the molybdenum recovery was checked by re-testing the same composite sample and the results were a somewhat improved rougher concentrate molybdenum recovery (40 to 60%) but not as good as previously tested composite samples. Previous results from testing samples containing the Colina ore type indicated that lower molybdenum recovery was to be expected. The cause is not specifically known at this time.)

The result of closed circuit flotation tests on the year 8 through year 12 composite sample indicates a final copper recovery of 92% and a molybdenum recovery of 84% in a final bulk concentrate grading 28% copper and 1.22% molybdenum. Additional analysis of the concentrate produced in the test work indicates that the concentrate contained low amounts of contaminates such as arsenic (<80 g/t) and mercury (0.8 g/t) and contained payable amounts of gold (1.91 g/t) and silver (294 g/t).

The result of closed circuit flotation tests on the year 13 through year 21 composite sample indicates a final copper recovery of 75.8% and a molybdenum recovery of 31.1% in a final bulk concentrate grading 36% copper, and 0.56% molybdenum. The core submitted for the year 13 to year 21 composite contained a higher percentage of oxidized material than will be mined, which resulted in lower metal recovery. A second composite sample was compiled with less oxidized material. Results of the second sample closed cycle test indicates copper recovery of 91.4% and molybdenum recovery of 66% in a final bulk concentrate containing 37% copper and 0.84% molybdenum.

An estimate of metal production in concentrate for the first 21 years of plant operation was prepared from the results of flotation test work performed by MSRDI in 2009 and MSRDI, G&T, and SGS in 2012. The 2012 work indicated the bulk concentrate production that could be expected by treating the expected ore composition for the operating years 1 through 21. The

2009 work indicated the separation efficiency that could be expected from treating bulk (copper-molybdenite) concentrate to produce a molybdenite concentrate and a final copper concentrate.

Graphical analysis determined the bulk concentrate copper and molybdenite recovery that can be expected when the concentrate grade is fixed at 30% copper. It has been estimated that the recovery of molybdenite to a molybdenite concentrate separated from the bulk concentrate will be 95%. The molybdenite production was then calculated by applying the separation efficiency factor (95%) to the estimated annual production of molybdenite in the bulk concentrate as determined by locked cycle flotation testing of the composite samples.

Silver recovery was determined by submitting the flotation products from locked cycle flotation testing of the composite samples to fire assay procedures to determine the silver contents of each. The average silver recovery by annual periods was than estimated from the results of test and assays.

The estimates of annual metal recovery are presented in Table 13-4.

Table 13-4: Estimated Metal Recovery by Year of Production

Estimated Metal Recovery by Year of Production			
	Recovery %		
Production Year	Cu	Mo	Ag
1	89.8	65.0	77.5
2	89.8	65.0	77.5
3	89.8	65.0	77.5
4	84.1	34.2	72.6
5	84.1	34.2	72.6
6	84.1	34.2	72.6
7	84.1	34.2	72.6
8	90.6	78.7	78.2
9	90.6	78.7	78.2
10	84.8	74.3	73.9
11	82.1	72.2	71.8
12	84.4	73.9	73.5
13	84.0	56.7	73.1
14	85.5	57.2	74.3
15	89.1	58.6	76.9
16	89.1	58.6	76.9
17	89.1	58.6	76.9
18	89.1	58.6	76.9
19	89.1	58.6	76.9
20	89.1	58.6	76.9
21	89.1	58.6	76.9

Reagent consumption rates for the full scale plant operation have been estimated from test results. The estimated reagent consumption rates for sulfide ore processing are shown in Table 13-5.

Table 13-5: Estimated Reagent Consumption Rates

Estimated Reagent Consumption Rates	
Item	Rate lbs/ton ore
Copper Circuit	
Aero Promoter 8944	0.04
Collector, C-7	0.098
Frother, Methyl Isobutyl Carbonal (MIBC)	0.026
Lime	1.797
Sodium Meta Silicate	0.14
#2 Diesel Fuel	0.026
Item	Rate lbs/ton Copper-Moly Concentrate
Molybdenite Circuit	
Sodium Hydrosulfide	934.4
Sodium Meta Silicate	25.4
#2 Diesel Fuel	15.2
Methyl Isobutyl Carbonal (MIBC)	15.2
Flomin D-910	88.9

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

14 MINERAL RESOURCE ESTIMATES

The updated Mineral Resource Estimate for the Rosemont deposit is prepared by Susan C. Bird of Mouse Mountain Technical Services (MMTS). This represents an update from the 2008 resource estimate (WLR Consulting, 2008) based on drilling and sample results up to March 2012, as well as updated geology.

The resource model is built using MineSight®, an industry standard in geologic modeling and mine planning software. The three dimensional block model has block dimensions of 50’x50’x50’ to reflect both the drill spacing and the bench height.

The block model limits, based on Imperial coordinates converted from UTM NAD 83 are: 1,710,000 to 1,722,000 East; 11,550,000 to 11,560,000 North; and 2,500 foot to 6,500 foot elevation. The volume modeled covers the extent of the main mineralized zone, as well as all pit limits tested.

The geology is separated into domains based on lithology and also into three metallurgical zones (oxide, mixed, and sulfide) based on interpretation of the assay data.

Statistical analyses (cumulative probability plots, histograms, contact plots and classic statistical values) of the assay and composite data are used to confirm the domain selection and to decide if capping is necessary within each zone and domain. Assay data is composited into 50’ intervals. The composites are used to create correlograms for Cu, Mo and Ag grades using the MSDA module of MineSight®, thus establishing rotation and search parameters for the block model interpolation. The composites used during interpolation are limited by both zone and domain. The resource is then classified as Measured, Indicated or Inferred based on variogram parameters and in accordance with the CIM Definition Standards (CIM, 2005).

Validation of the model is completed by comparison of the Ordinary Kriged (OK) values with both Inverse Distance Squared (ID2) and Nearest Neighbor (NN) interpolated block values, by the use of swath plots and grade tonnage curves. A visual inspection in section and plan throughout the deposit was performed to compare the modeled grades with the assay data. In addition, average grades for each data set were compared.

14.1 DRILL HOLE DATABASE

Additional data since the 2008 resource estimate includes assays from 11 of the 12 drill holes recently completed by Augusta, as well as the additional sampling from five previously drilled exploration holes. One exploration drill hole, AR-2080, had not yet been assayed at the time of the model build. There are a total of 258 holes within the block model limits (of the 266 holes in the database), for a total number of intervals of 58,281with assayed Cu values.

All mineralized zones for every hole were drilled using diamond core, with less than five percent started by open-hole rotary techniques.

14.1.1 Assay Statistics and Domain Definition

Assay statistics for each lithologic unit have been used to determine the domains for interpolation. Changes to the lithology since 2008 include the separation of all units above and below the Low Angle Fault (LAF). Lithologic boundaries have been re-interpreted where necessary, requiring minor changes due to additional drilling information. Coding of both lithology and domain to the model is done using 3D wireframe solids created in MineSight for each unit.

The assay data, by lithology, was analyzed to determine appropriate hard boundaries required to form domains for the interpolation. Assay means by domain, contact plots, as well as geologic knowledge on the location of major faults all contributed to the final domain assignments.

Changes to the domain boundaries from the 2008 model include:

1. Combining lithologies Colina and Epitaph below the LAF
2. Combining lithologies Glance, Scherrer, Concha and Epitaph above the LAF
3. Combining Martin and Martin West
4. Separating the Horquilla and Earp formations

The resulting combinations of lithologies for each domain are summarized in Table 14-1.

Table 14-1: Domain Definition based on Lithology

Domain	Lithologic Units	Description
1	1	Overburden
2	2,3	Epitaph and Colina - below LAF
4	4	Earp
5	5	Horquilla,
6	6	Escabrosa
7	7,18	Martin, Martin West
8	8	QMP
9	9	Andesite
10	10	Arkose
11	11,22,212,214	Glance, Epitaph, Scherrer, Concha - above LAF
12	12,14	Scherrer, Concha - below LAF
13	13	Abrigo
15	15	Bolsa
16	16	Granodiorite
17	17	Epitaph North
21	21	Gravel

The resulting assay statistics for total copper (TCu) by domain and zone are summarized in the tables 14-2 through 14-5, and for Mo in tables 14-6 through 14-9. Examination of these tables indicates that within the sulfide and mixed zones, domains 2, 4, 5, 6, and 8 are the primary ore bearing domains. Within the oxides, only domains 8 through 10 contain lithology conducive to leaching. Of these, domains 8 and 9 are both significant leach oxide hosts.

Table 14-2: Assay Statistics of TCu for Domains 8 through 10 – Oxides

	Domain		
Parameter	**8**	**9**	**10**
Num Samples	915	837	8794
Num Missing Samples	0	0	0
TCu (%) Min	0	0	0
TCu (%) Max	11.900	3.890	3.500
TCu (%) - Weighted Mean	0.175	0.136	0.044
TCu (%) - Weighted SD	0.533	0.296	0.122
TCu (%) - Weighted Var	0.284	0.088	0.015
TCu (%) - Weighted CV	3.051	2.185	2.8

Table 14-3: Assay Statistics of TCu for Domains 1 through 9 – Sulfides

	Domain							
Parameter	**1**	**2**	**4**	**5**	**6**	**7**	**8**	**9**
Num Samples	0	9352	5384	14178	1846	869	1030	1789
TCu (%) Min	-	0	0	0	0	0	0	0
TCu (%) Max	-	14.500	32.200	15.390	24.700	3.260	15.880	10.300
TCu (%) - Weighted Mean	-	0.423	0.273	0.473	0.321	0.052	0.328	0.149
TCu (%) - Weighted SD	-	0.636	0.356	0.7	1.535	0.205	0.85	0.428
TCu (%) - Weighted Var	-	0.405	0.127	0.49	2.355	0.042	0.723	0.183
TCu (%) - Weighted CV	-	1.504	1.305	1.48	4.785	3.967	2.592	2.864

Table 14-4: Assay Statistics of TCu for Domains 10 through 21 – Sulfides

	Domain							
Parameter	**10**	**11**	**12**	**13**	**15**	**16**	**17**	**21**
Num Samples	3082	1388	325	1059	374	193	518	11
TCu (%) Min	0	0	0	0	0	0	0	0
TCu (%) Max	3.140	3.650	1.500	16.520	1.510	2.210	6.880	0.040
TCu (%) - Weighted Mean	0.074	0.072	0.055	0.285	0.09	0.083	0.139	0.001
TCu (%) - Weighted SD	0.207	0.272	0.158	0.741	0.16	0.162	0.356	0.006
TCu (%) - Weighted Var	0.043	0.074	0.025	0.55	0.026	0.026	0.127	0
TCu (%) - Weighted CV	2.8	3.783	2.857	2.603	1.772	1.942	2.567	5.102

Table 14-5: Assay Statistics of TCu for Domains 1 through 21 – Mixed

	Domain						
Parameter	**4**	**5**	**8**	**10**	**11**	**12**	**13**
Num Samples	12	1119	190	6	4	68	581
TCu (%) Min	0.15	0	0	0	0.23	0	0
TCu (%) Max	0.450	5.050	1.420	0.340	0.550	0.510	2.360
TCu (%) - Weighted Mean	0.32	0.635	0.215	0.118	0.335	0.191	0.444
TCu (%) - Weighted SD	0.099	0.475	0.222	0.138	0.129	0.08	0.349
TCu (%) - Weighted Var	0.01	0.226	0.049	0.019	0.017	0.006	0.122
TCu (%) - Weighted CV	0.309	0.748	1.031	1.165	0.385	0.418	0.786

Table 14-6: Assay Statistics of Mo for Domains 8 through 10 – Oxides

	Domain		
Parameter	**8**	**9**	**10**
Num Samples	915	837	8794
TCu (%) Min	0	0	0
TCu (%) Max	0.144	0.195	0.105
Mo (%) - Weighted Mean	0.006	0.0022	0.0013
Mo (%) - Weighted SD	0.0086	0.0079	0.0039
Mo (%) - Weighted Var	0.0001	0.0001	0
Mo (%) - Weighted CV	1.4345	3.5698	3.1272

Table 14-7: Assay Statistics of Mo for Domains 1 through 9 – Sulfides

	Domain							
Parameter	**1**	**2**	**4**	**5**	**6**	**7**	**8**	**9**
Num Samples	0	9352	5384	14178	1846	869	1030	1789
TCu (%) Min	-	0	0	0	0	0	0	0
TCu (%) Max	-	5.966	1.590	2.660	0.350	0.040	0.260	0.030
Mo (%) - Weighted Mean	-	0.0127	0.0144	0.0157	0.0056	0.0028	0.0164	0.0019
Mo (%) - Weighted SD	-	0.0668	0.0312	0.0463	0.0135	0.0047	0.0254	0.004
Mo (%) - Weighted Var	-	0.0045	0.001	0.0021	0.0002	0	0.0006	0
Mo (%) - Weighted CV	-	5.2528	2.1642	2.9568	2.4346	1.6665	1.5514	2.1133

Table 14-8: Assay Statistics of Mo for Domains 10 through 21 – Sulfides

	Domain							
Parameter	**10**	**11**	**12**	**13**	**15**	**16**	**17**	**21**
Num Samples	3082	1388	325	1059	374	193	518	11
TCu (%) Min	0	0	0	0	0	0	0	0
TCu (%) Max	0.132	0.302	0.126	0.430	0.155	0.030	0.170	0.006
Mo (%) - Weighted Mean	0.0016	0.0024	0.0044	0.0057	0.0021	0.0025	0.0059	0.0001
Mo (%) - Weighted SD	0.0045	0.0088	0.008	0.0201	0.0076	0.0042	0.0164	0.0006
Mo (%) - Weighted Var	0	0.0001	0.0001	0.0004	0.0001	0	0.0003	0
Mo (%) - Weighted CV	2.8479	3.639	1.8276	3.5339	3.6475	1.6447	2.7791	6.0051

Table 14-9: Assay Statistics of Mo for Domains 1 through 21 – Mixed

	Domain						
Parameter	**4**	**5**	**8**	**10**	**11**	**12**	**13**
Num Samples	12	1119	190	6	4	68	581
TCu (%) Min	0.001	0	0	0	0	0	0
TCu (%) Max	0.011	0.560	0.050	0.114	0.001	0.012	0.111
Mo (%) - Weighted Mean	0.0047	0.0061	0.0048	0.0101	0.0005	0.0046	0.0033
Mo (%) - Weighted SD	0.003	0.0125	0.0053	0.0305	0.0005	0.0029	0.0123
Mo (%) - Weighted Var	0	0.0002	0	0.0009	0	0	0.0002
Mo (%) - Weighted CV	0.6508	2.0427	1.1059	3.0061	1	0.6358	3.7812

14.1.2 Compositing of Drill Hole Data

Compositing is done by 50’ bench composites in order to correspond to the planned bench height and elevations. The domains are coded to the composites by a majority code using the model block codes. The zones are coded using 3D surfaces corresponding to the bottom of each zone layer.

Composites are used to determine capping of metal values during the interpolations. Cumulative Probability Plots (CPP) are created for each metal and domain in the sulfide zone to determine that lognormal distribution applies, and to aid in selection of the capping required. In the mixed and oxide zone, each metal is plotted for all domains together due to the lack of data. Resulting capping values are given in Table 14-10.

Figures 14-1 and 14-2 are CPP plots for each of the three modeled metals (Cu, Mo and Ag) in the oxide and mixed zones respectively. Figures 14-3 through 14-5 are CPP plots of the domains requiring capping in the sulfide zone for Cu, Mo and Ag respectively.

Table 14-10: Capping Value of Composites

Zone	Domain	Capping Threshold		
		TCu (%)	Mo (%)	Ag (opt)
Sulfides	2	2	---	0.6
	4	1.7	---	---
	5	---	0.2	---
	6	---	---	---
	7	1.7	---	1
	8	1.5	---	0.8
	9	1.2	---	0.3
	10	0.5	---	0.2
	11	---	---	---
	12	---	---	---
	13	---	---	1.2
	15	---	---	---
	16	---	---	---
	17	---	---	---
	21	---	---	---
Mixed	all	---	---	0.2
Oxides	all	1.2	0.35	0.5

For grade values above the capping limits, the search distance for use of the value during interpolation is restricted to 50 ft. Beyond this distance, the capped value of the composite is used for interpolation.

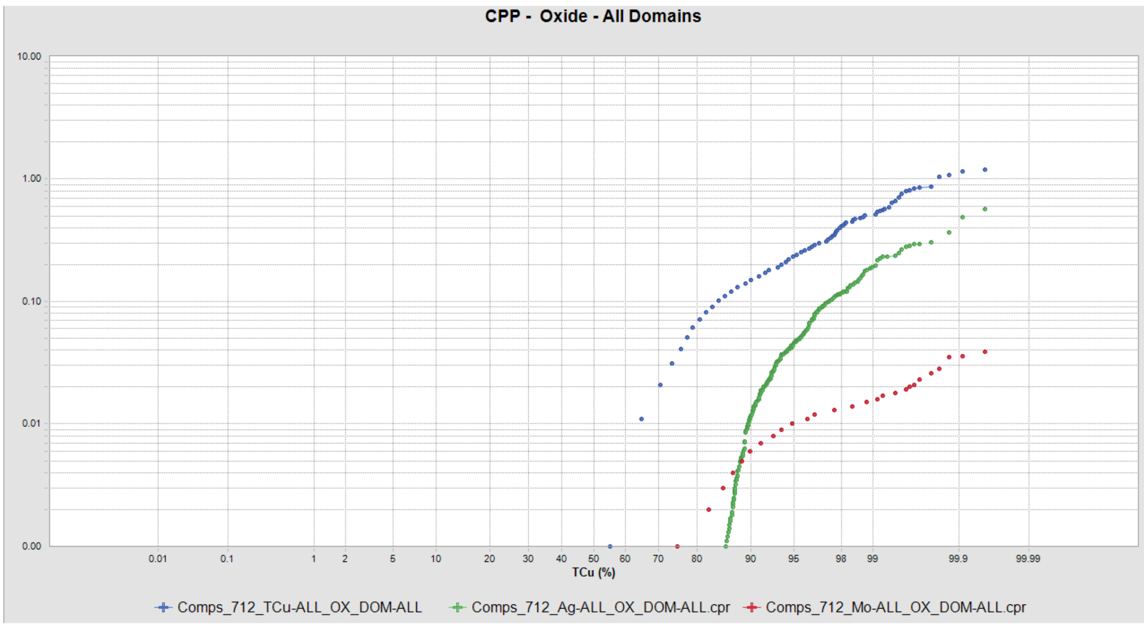

Figure 14-1: CPP Plots of each Metal for All Domains – Oxide Zone

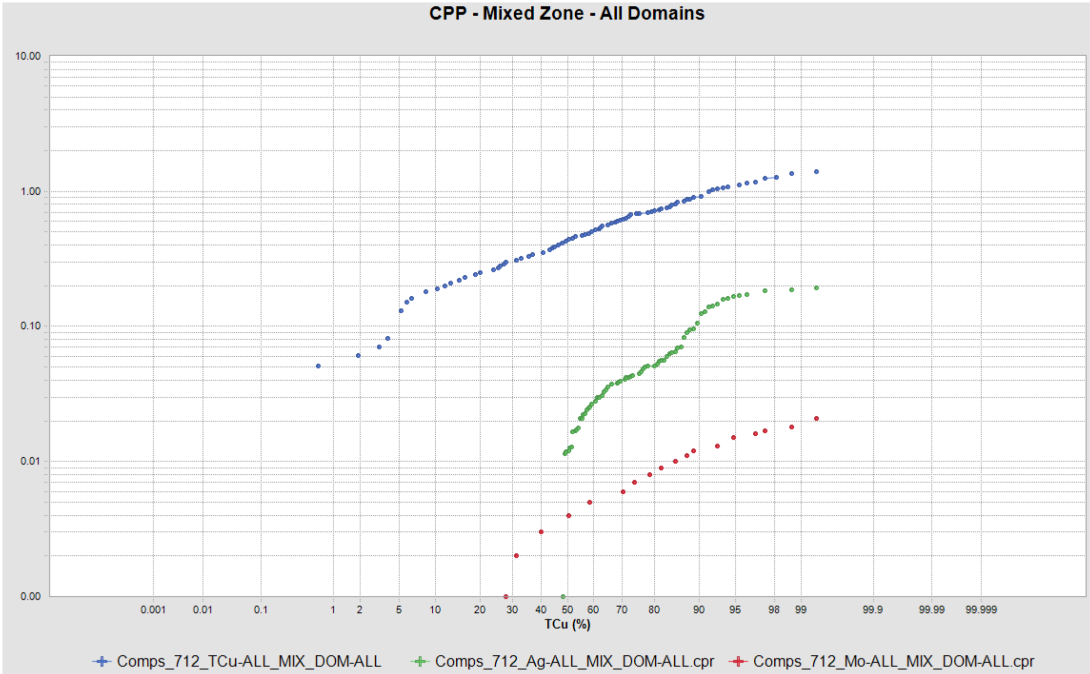

Figure 14-2: CPP Plots of each Metal for All Domains – Mixed Zone

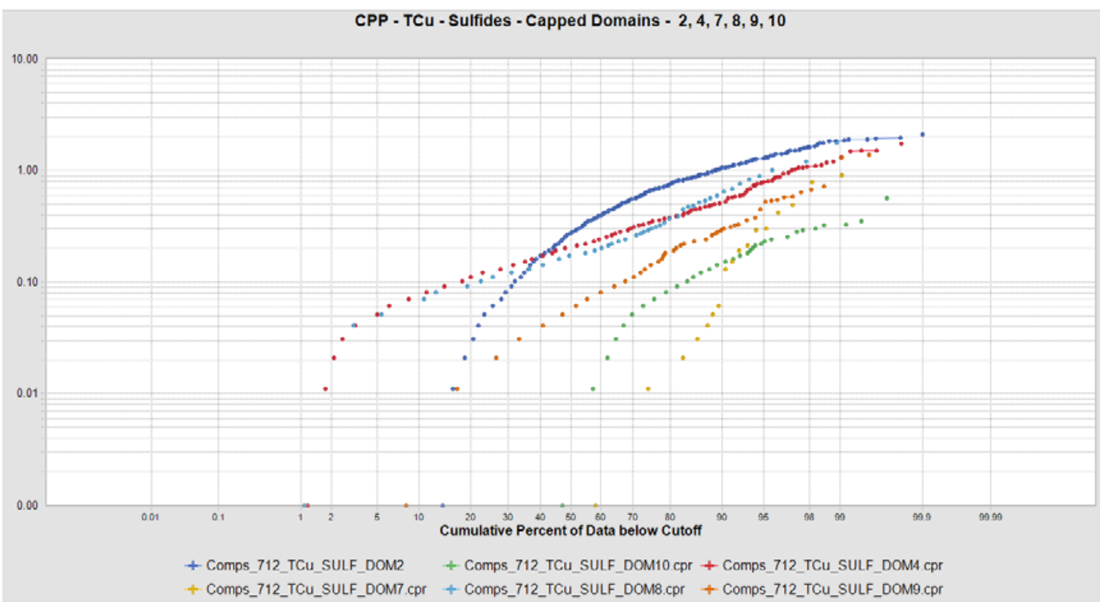

Figure 14-3: CPP Plots of TCu for Sulfide Domains requiring Capping

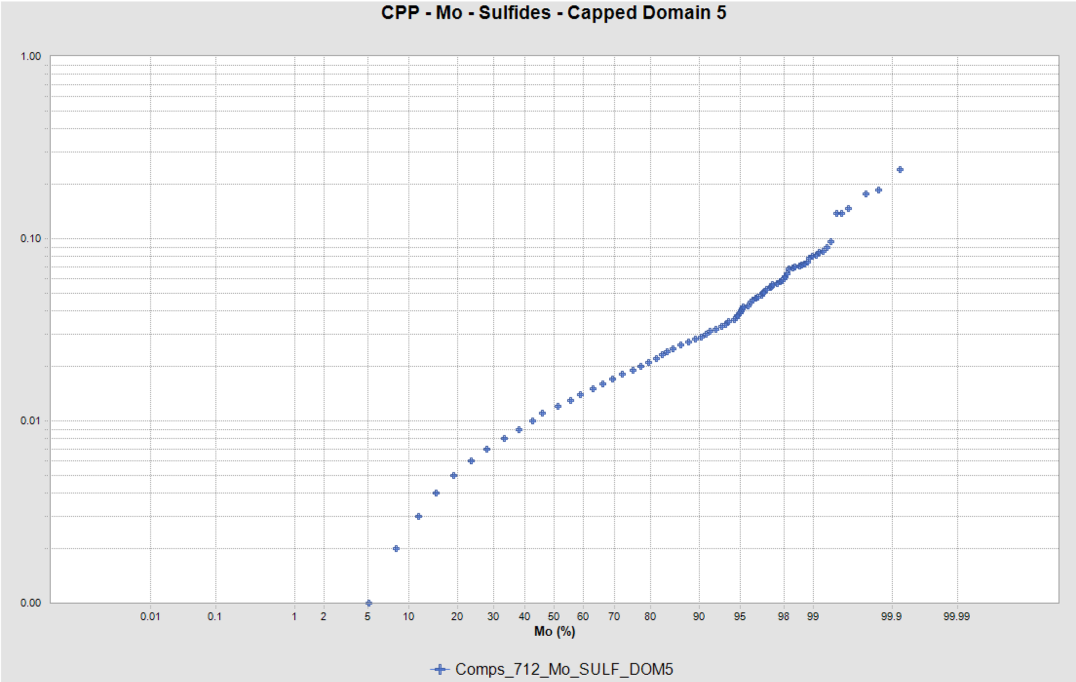

Figure 14-4: CPP Plots of Mo for Sulfide Domains requiring Capping

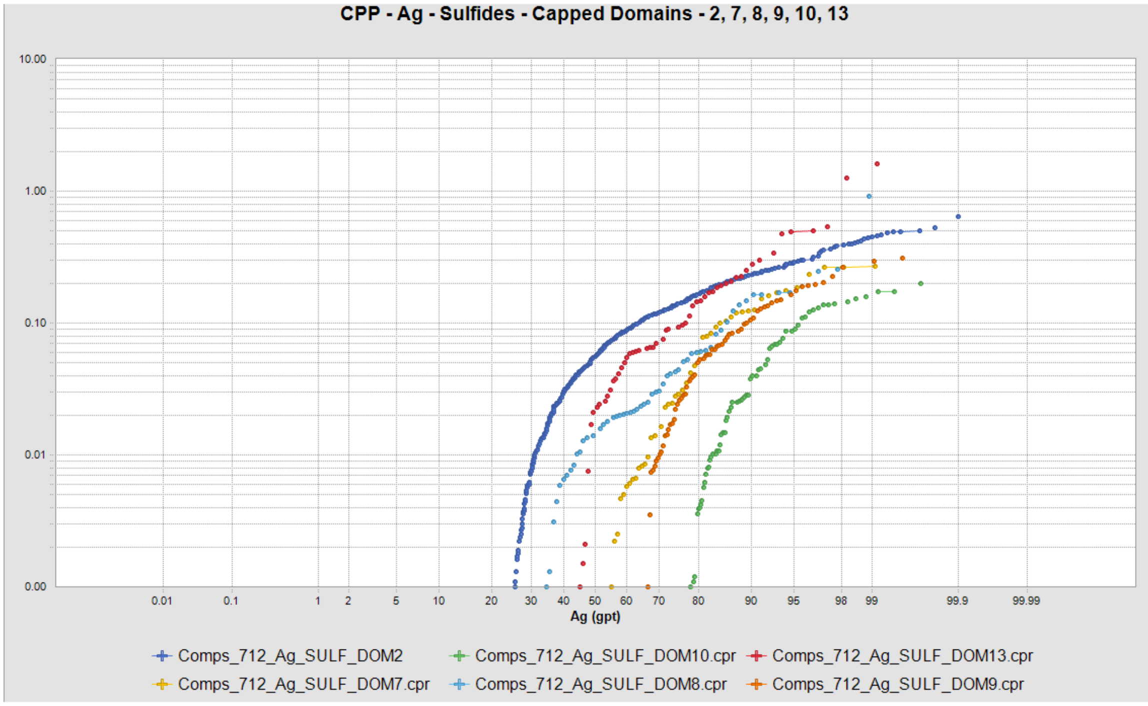

Figure 14-5: CPP Plots of Ag for Sulfide Domains Requiring Capping

14.2 VARIOGRAPHY

In order to determine the directional properties of each domain, correlograms have been created for all three metals in each domain and zone. It was found that there is insufficient data to produce reliable correlograms by domain separately. Therefore, within the sulfide zone, the primary mineralized domains – domains 2, 4, and 5 are used to select the rotational and kriging parameters for all domains except the quartz monzonite porphyry (QMP) domain.

The lithologic units within the Rosemont deposit area strike generally approximately N-S and dip moderately to the east, as is evident in the geologic section of Figure 7-4. A spherical model is used to obtain the best fit in all cases. Variography adheres to the bedding, as is indicated in the summary of parameters listed in Table 14-11. The major axis of the spheroid plunges down-dip of the formations, with the minor axis plunging to the north at 15 degrees. The major and minor axes correlograms and corresponding spherical models are illustrated in Figures 14-6 and 14-7 for TCu.

The QMP intrusions did not indicate any directional preference, and therefore omni-directional correlograms are determined to be appropriate for this domain for each metal and zone.

Table 14-11: Variogram Parameters

Domains	Zone	Metal	Rotation (GSLIB-MS)		Axis	Range 1 (ft)	Range 2 (ft)	Nugget	Total Sill	Sill1	Sill2
All except QMP	All	TCu	ROT	110	Major	450		0.4	1	0.6	0
			DIPN	-55	Minor	400					
			DIPE	15	Vert	300					
		Mo	ROT	110	Major	300		0.4	1	0.6	0
			DIPN	-55	Minor	300					
			DIPE	10	Vert	150					
		Ag	ROT	110	Major	300	1500	0.4	1	0.42	0.18
			DIPN	-55	Minor	300	1000				
			DIPE	10	Vert	100*	300				
QMP	All	TCu	Omni-Directional			400		0.4	1	0.6	
		Mo				300		0.2	1	0.8	
		Ag				300		0.3	1	0.7	

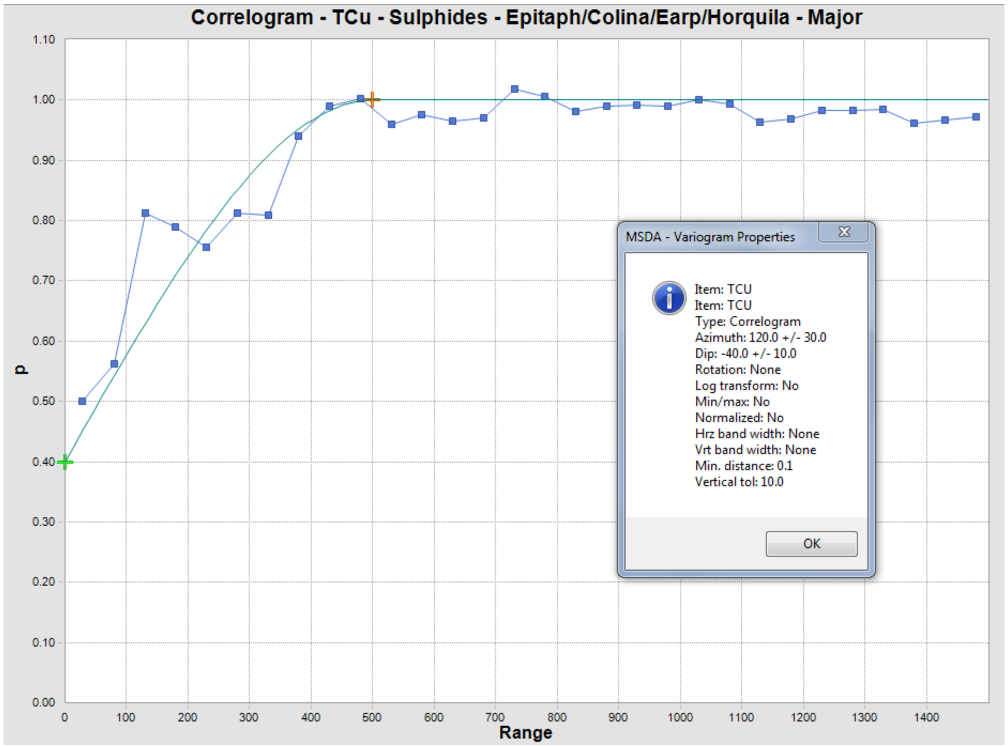

Figure 14-6: Major Axis Correlogram – TCu

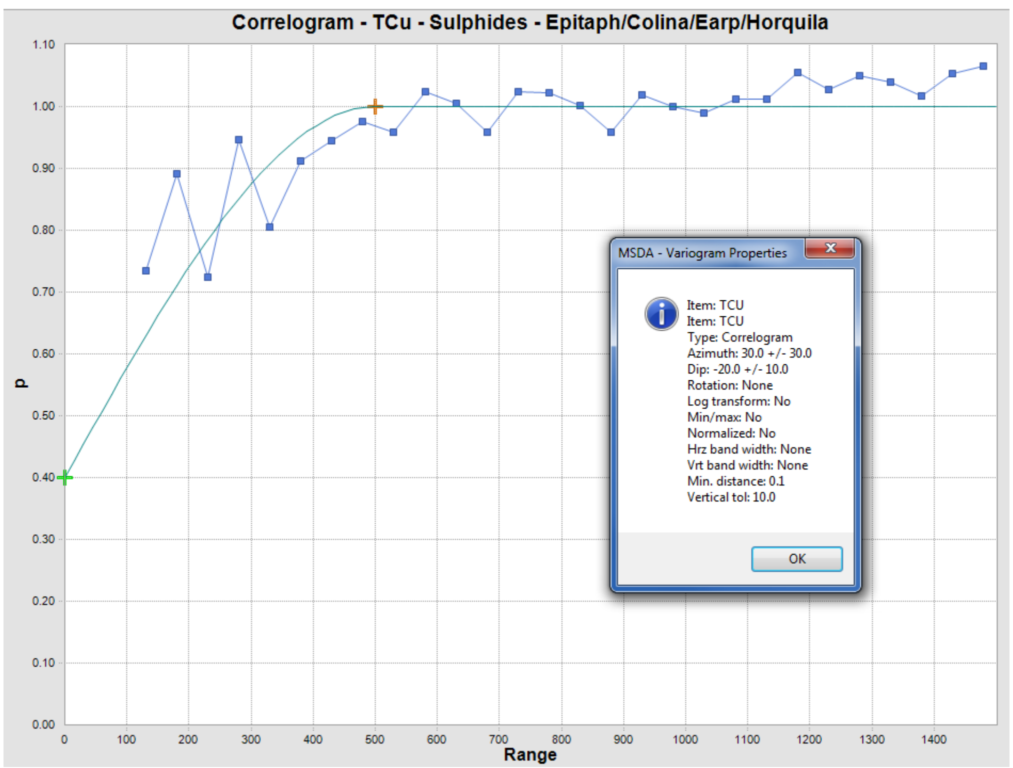

Figure 14-7: Minor Axis Correlogram - TCu

14.3 BLOCK MODEL INTERPOLATION AND RESOURCE CLASSIFICATION

The block model built for the Rosemont deposit has the dimensions summarized in Table 14-12.

Table 14-12: Block Model Dimensions

Direction	Minimum (ft)	Maximum (ft)	Block Dimension (ft)	# of Blocks
Easting	1,710,000	1,722,000	50	240
Northing	11,550,000	11,560,000	50	200
Elevation	2,500	6,500	50	80

The block model is coded according to domain, based on wire-frame solids and zone surfaces using oxide and transition surfaces derived from the drill hole database information, including the acid soluble copper content.

The topographic surface is based on an aerial survey flown by Cooper Aerial Surveys Company of Tucson, Arizona in the summer of 2006. The vertical datum is based on the NAVD 88 standard. Cooper provided electronic files with elevation data on 10-foot contour intervals covering the project area. The percent of the block below topography is also calculated into the model blocks.

Specific gravity values are based on 392 measurements by Skyline Laboratories based on the differential of the weight in air and the weight in water. Table 14-13 summarizes the tonnage factors coded to the block model.

Table 14-13: Specific Gravity by Lithology

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

The interpolation is completed using ordinary kriging (OK) in 2 passes with search parameters based on the variogram parameters. The first pass maximum search distances are equal to ½ the range of the variograms, and the second pass has a maximum search distance equal to the full range. Restrictions on the search distances and number of composites used in each pass are given in Table 14-14 and Table 14-15 below. The selection of a composite for interpolation is also restricted by both the domain and the zone codes, which are required to match the block model codes.

Table 14-14: Interpolation Search Parameters

Domains	Zone	Metal	Rotation (GSLIB-MS)		Axis	Distance (ft) 1st Pass	Distance (ft) 2nd Pass
All Except QMP	all	TCu	ROT	110	Major	225	450
			DIPN	-55	Minor	200	400
			DIPE	15	Vert	150	300
		Mo	ROT	110	Major	150	450
			DIPN	-55	Minor	150	400
			DIPE	10	Vert	75	300
		Ag	ROT	110	Major	150	450
			DIPN	-55	Minor	150	400
			DIPE	10	Vert	50	300
QMP	all	TCu	Omni-Directional			200	400
		Mo				150	400
		Ag				150	400

Table 14-15: Interpolation Composite Restrictions

Metal	Interpolation Pass	Min. # Comps	Max # Comps	Max Comps/DH	Max Comps/Quad
TCu	1	3	8	2	2
	2	1	8	1	4
Mo	1	3	12	2	2
	2	1	12	1	4
Ag	1	3	8	2	2
	2	1	6	1	4

14.3.1 Resource Classification

Classification of the resource into Measured, Indicated, and Inferred is based on the variogram parameters and restrictions on the number of composites and drill holes used in each pass of the interpolation. The resource is classified as Measured or Indicated according to the distances and composites numbers summarized in Table 14-16, with domain boundaries not honored for the purposes of classification. Inferred blocks are defined as all blocks with grades interpolated that do not meet the measured or indicated constraints.

The definition of Indicated and Inferred used to classify the resource is in accordance with that of the CIM Definition Standards (CIM, 2005).

Table 14-16: Interpolation Composite Restrictions

Class	Minimum # Comps	Maximum Distance (ft)
Measured	3	100
	1	50
Indicated	2	320
Inferred	1	450 (400 for QMP)

14.4 BLOCK MODEL VALIDATION

Validation of the model is completed by comparison of mean grades, swath plots, grade-tonnage curve comparisons, and visual inspection in section and plan across the extent of the model.

14.4.1 Comparison of Mean Grades

The following Table compares the block model interpolated values for the kriged grades (OK) and Nearest Neighbor (NN) for blocks within the resource pit. For the oxide, only the domains with leach potential are compared (domains 8-10). For the Sulfide zone, all domains are used in the comparison.

The Nearest Neighbor interpolation is essentially the composite data, de-clustered to remove any bias in the drilling locations. Each metal and zone indicates good correlation between the OK grade and the de-clustered composite data.

Table 14-17: Comparison of OK grades with NN Grades within Resource Pit

Zone	Parameter	CUOK (%)	CUNN (%)	Diff (%)	MOOK (%)	MONN (%)	Diff (%)	AGOK (opt)	AGNN (opt)	Diff (%)
Oxide	Samples	30815	30815	na	30815	30815	na	30815	30815	na
	Missing Samples	199	199	na	199	199	na	199	199	na
	Weighted Mean	0.047	0.049	-4.1%	0.0013	0.0012	4.2%	0.0076	0.0071	7.0%
Sulfide	Samples	85789	85789	na	85789	85789	na	85789	85789	na
	Missing Samples	614	614	na	614	614	na	614	614	na
	Weighted Mean	0.351	0.338	3.8%	0.0134	0.013	3.1%	0.0971	0.091	6.7%

14.4.2 Grade-Tonnage Curves

Grade-tonnage curves are used to compare distribution data of the interpolated OK grades with the Nearest Neighbor distributions and the Nearest Neighbor Corrected (NNC). The NNC model is used in order to correct for the change of sample size from the composite to the block size of 50’x50’x50’. The Indirect Lognormal Correction that has been used is based on the variogram parameters, the block size, the mean grades, and the Coefficient of Variation of the NN grades.

Figures 14-8 through 14-16 illustrate the grade-tonnage curves for each metal and zone. For the oxide zone, only domains 8 through 10 are plotted, as they are the only leachable veins. Each plot indicates that the OK grades are slightly lower than the corrected NN (NNC) grades throughout the grade distribution, and particularly at the higher end of the curve. This is to be expected, and accounts for the internal smoothing of the model during interpolation of the OK grades. Search parameters were chosen in an iterative procedure in order to create tonnage-grade curves that correspond to the necessary amount of dilution expected. Average copper equivalent values for sulfides/mixed material above a cutoff grade of 0.20 percent indicate a copper grade that is 2 percent lower for the OK interpolation compared to the NNC interpolation. Because of the modeling procedure, no additional dilution is required when reporting the resource and reserves.

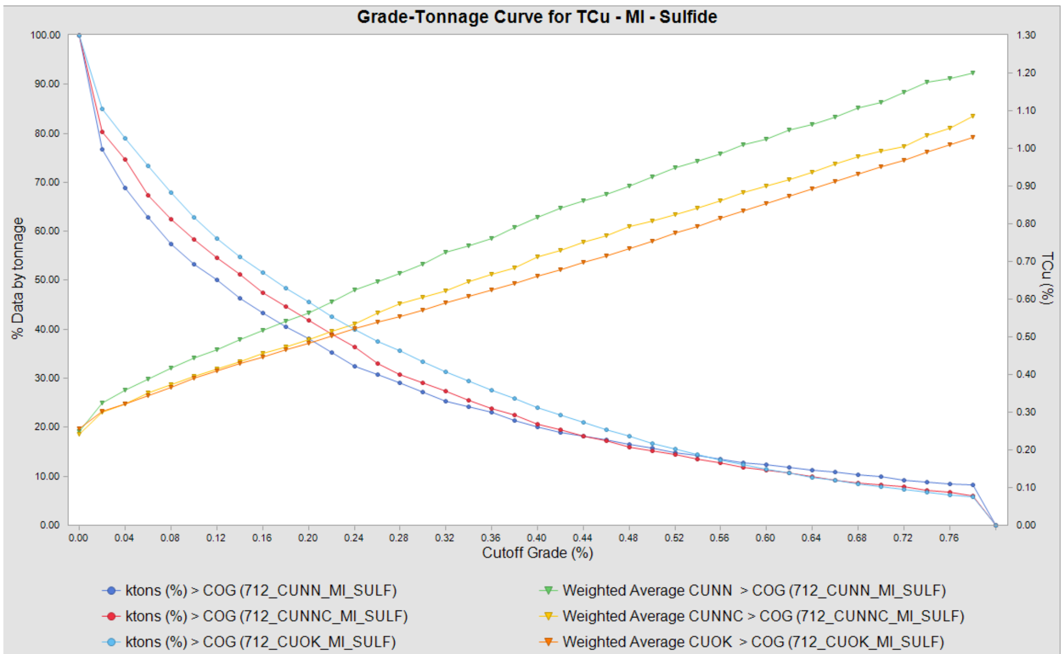

Figure 14-8: Grade-Tonnage Curve for TCu-Sulfide Zone

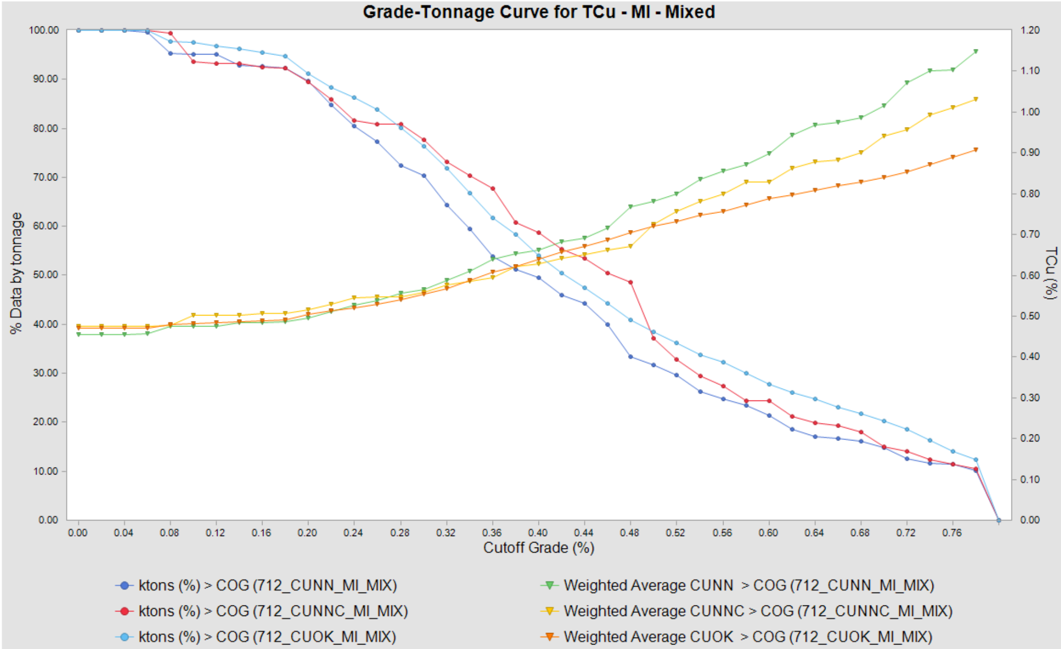

Figure 14-9: Grade-Tonnage Curve for TCu-Mixed Zone

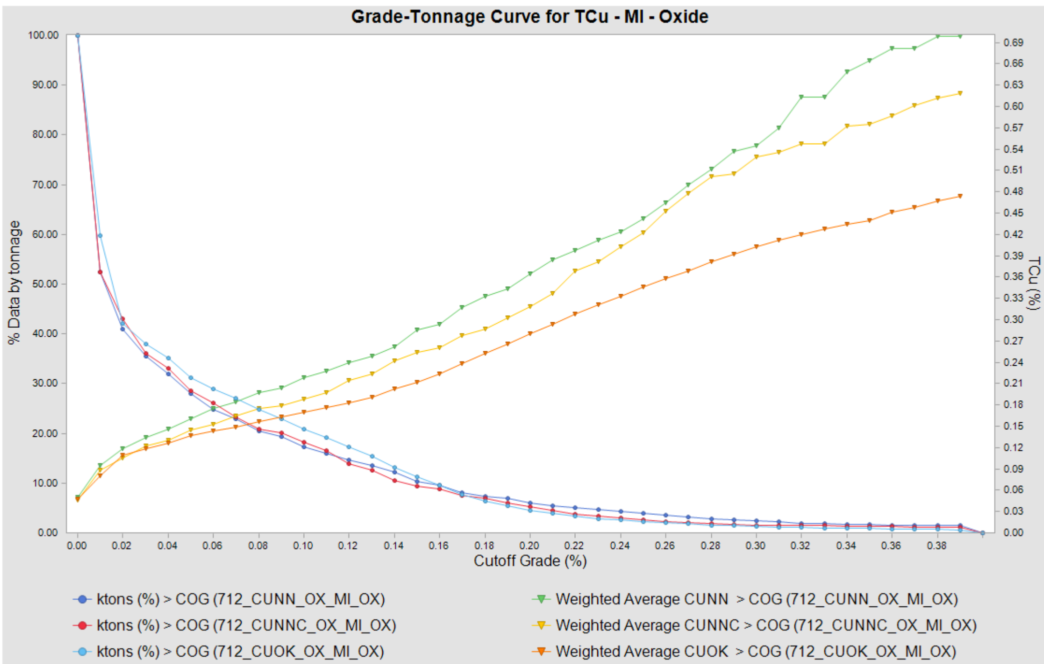

Figure 14-10: Grade-Tonnage Curve for TCu-Oxide Zone

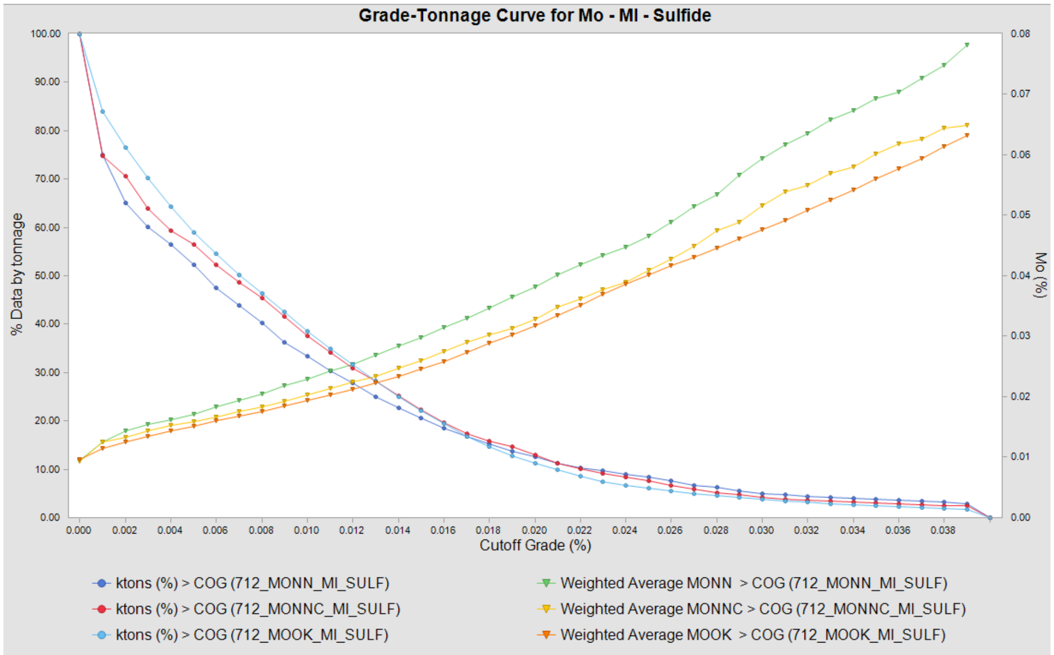

Figure 14-11: Grade Tonnage Curve for Mo-Sulfide Zone

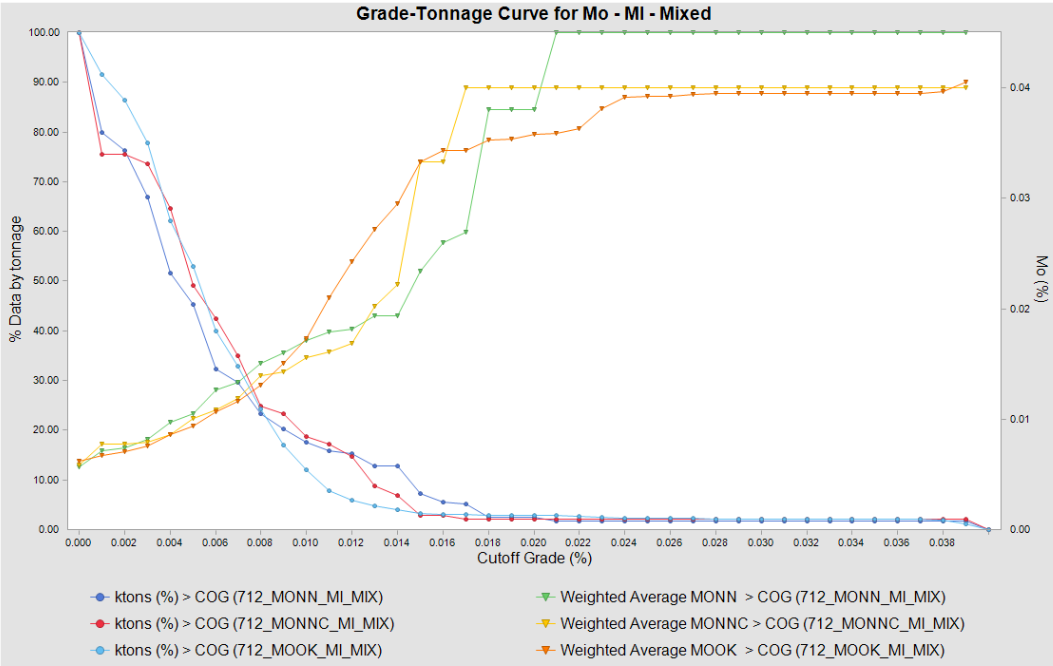

Figure 14-12: Grade-Tonnage Curve for Mo-Mixed Zone

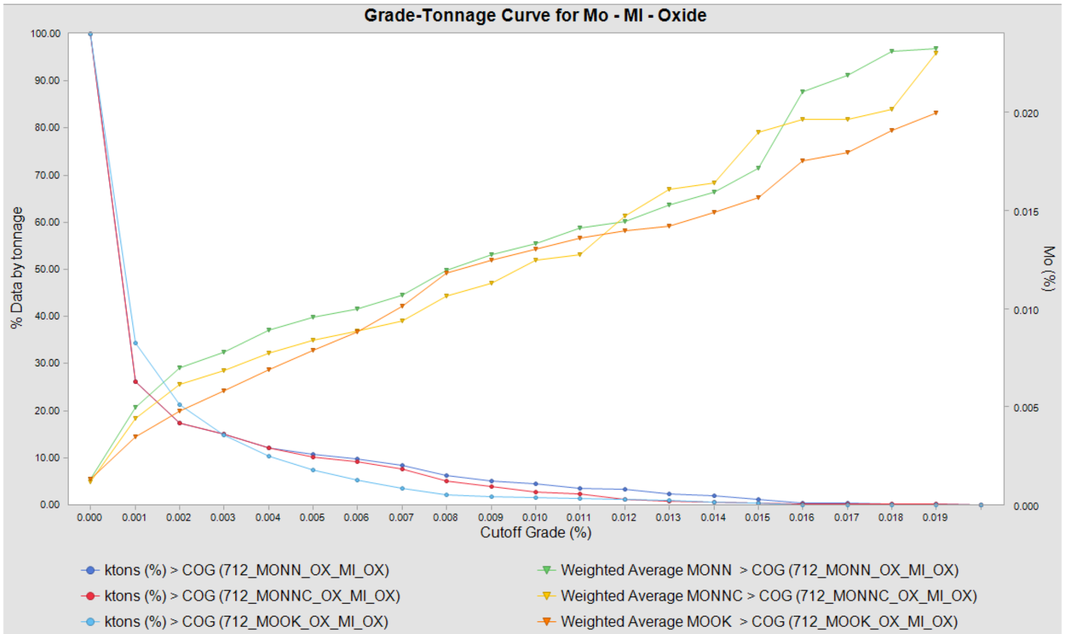

Figure 14-13: Grade-Tonnage Curve for Mo-Oxide Zone

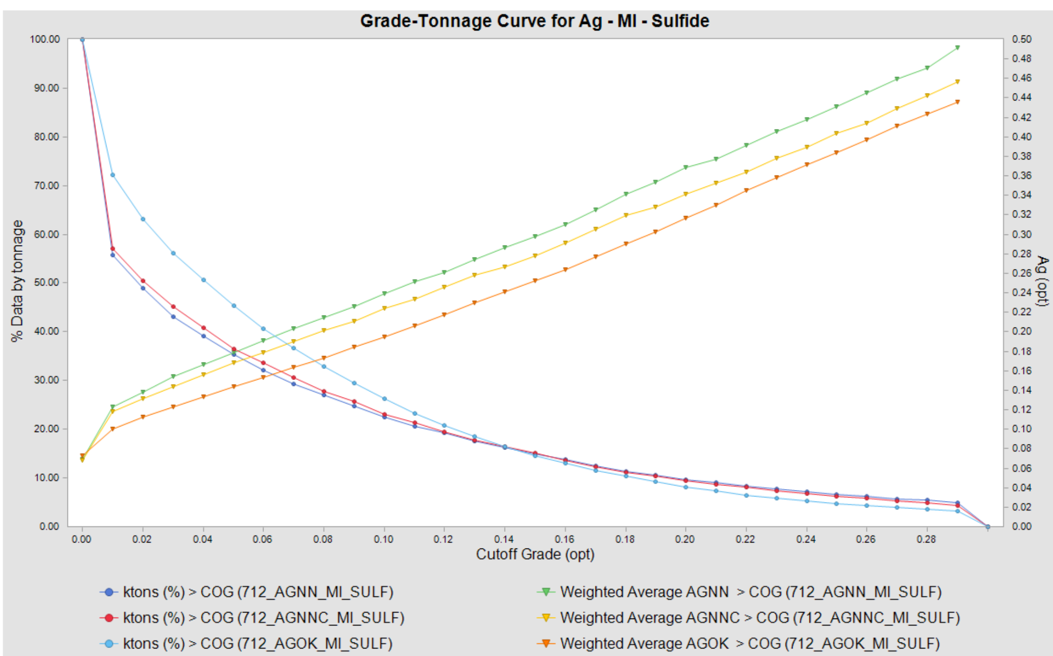

Figure 14-14: Grade-Tonnage Curve for Ag-Sulfide Zone

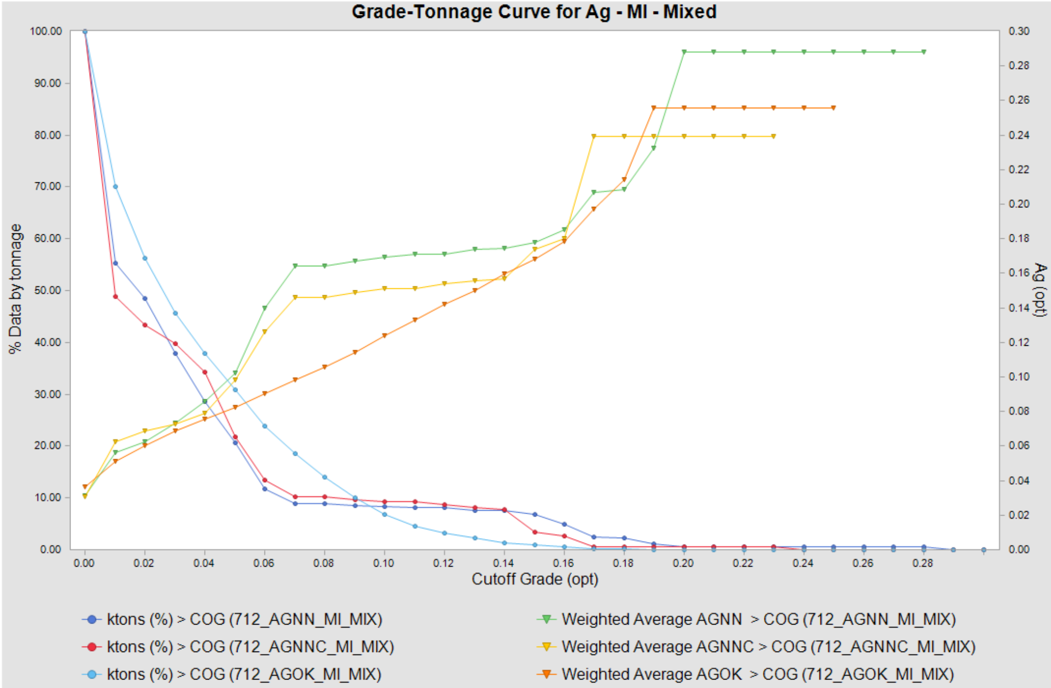

Figure 14-15: Grade-Tonnage Curve for Ag-Mixed Zone

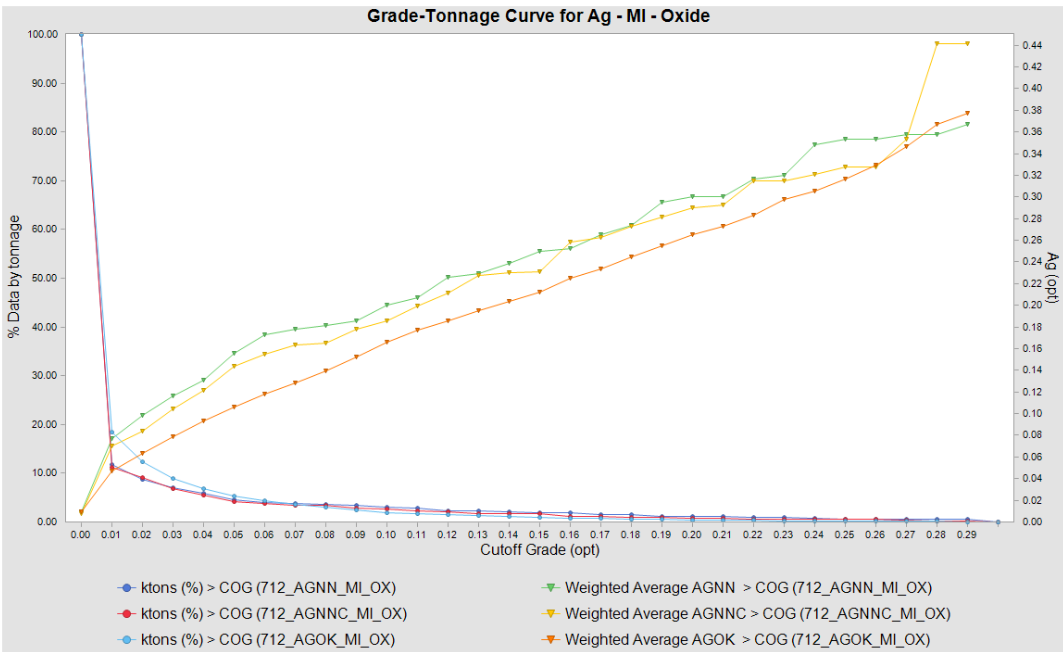

Figure 14-16: Grade-Tonnage Curve for Ag-Oxide Zone

14.4.3 Swath Plots

Swath plots through the block model are created in the N-S, E-W, and vertical directions for the three metals, in order to compare the ordinary kriged (OK) grades to those of the Nearest Neighbor (NN) model, which acts as a proxy for the de-clustered data. Swath plots of the inverse distance squared (ID) interpolation are also plotted as a further check. These are illustrated in Figures 14-17 through 14-20. Each “swath” is created as an average of the block grades for each direction shown, with steps of 100 feet along the easting and northing plots, and 50 feet along the vertical plots. The bar graph of the block tonnage provides an indication of the location of the majority of the data. The swath plots do not indicate any global bias in the OK grade values and show good correlation with the NN and ID grades throughout the main body of the data.

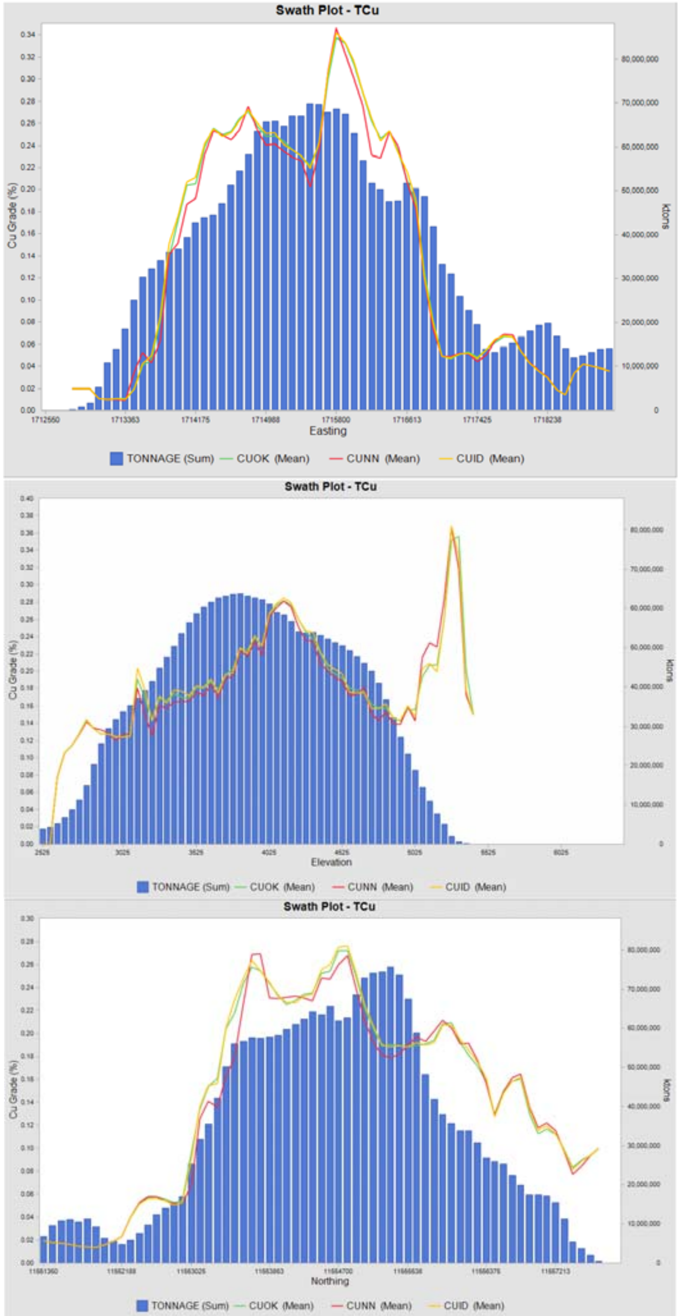

Figure 14-17: Swath Plots for TCu-Sulfide Zone

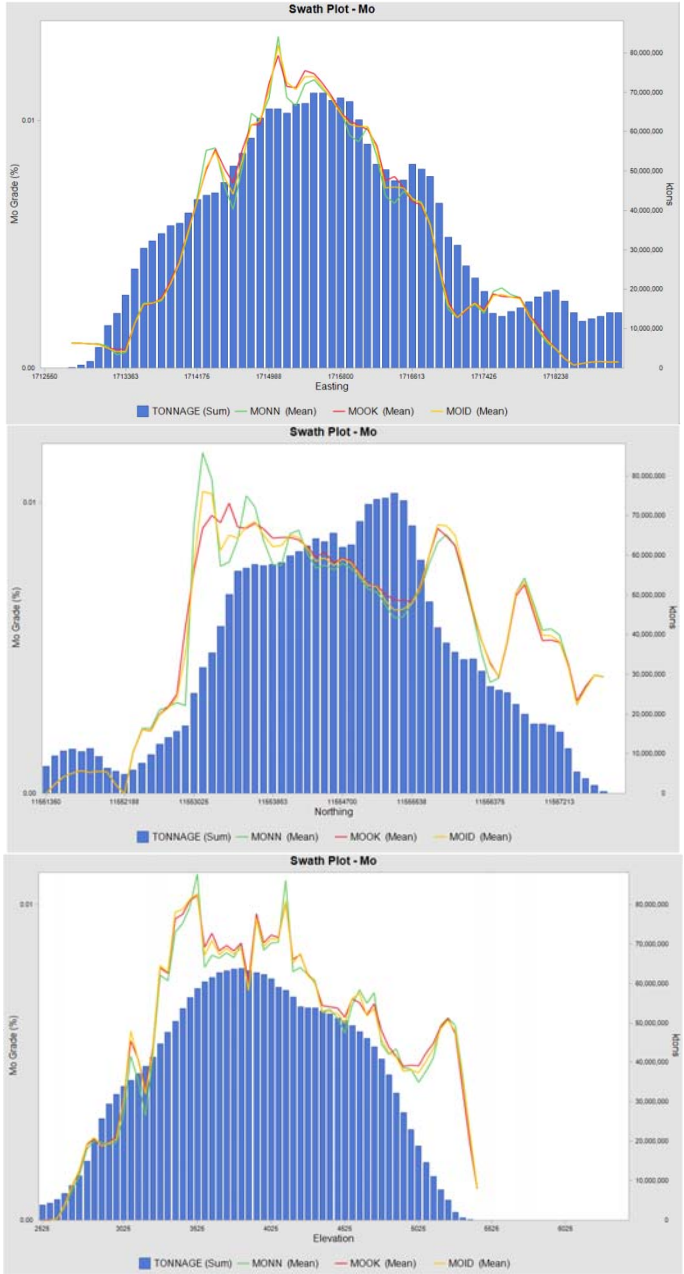

Figure 14-18: Swath Plots for Mo – Sulfide Zone

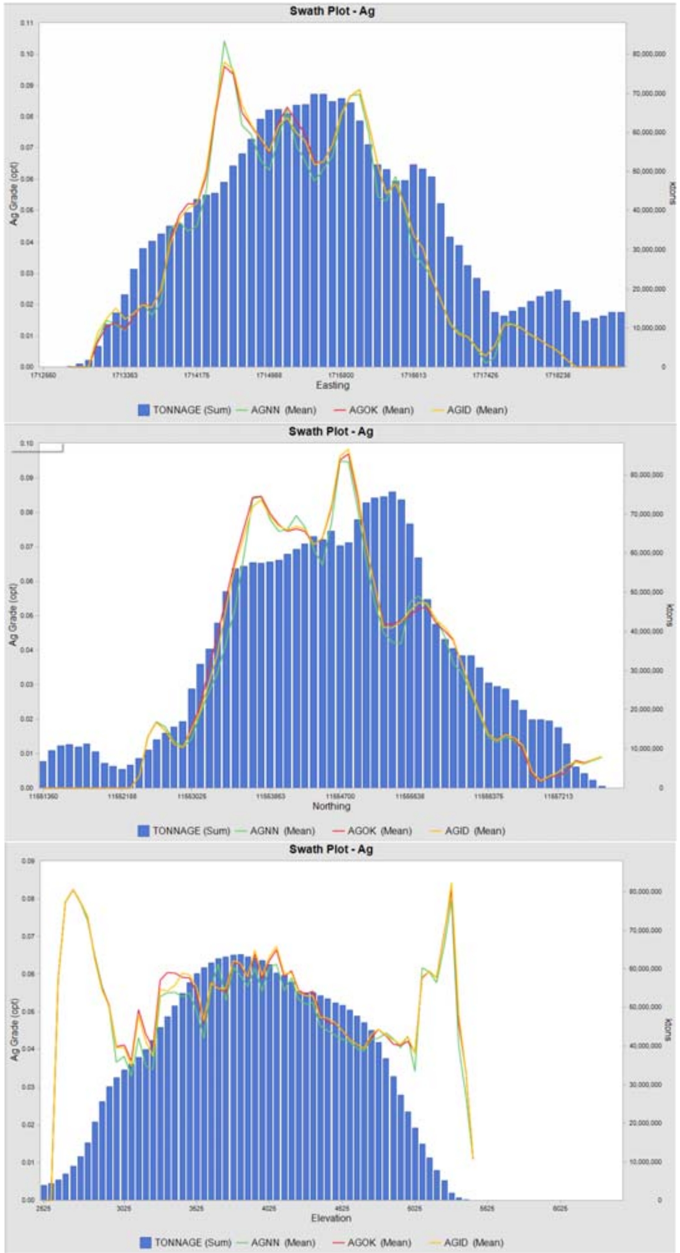

Figure 14-19: Swath Plots for Ag – Sulfide Zone

14.4.4 Visual Validation

A series of E-W, N-S sections and plans (every 100') have been used to inspect the ordinary kriged block model grades with the drill hole data. Figures 14-20 through 14-22 give examples of this comparison for the E-W section at 11,554,825N, for Cu, Mo and Ag grades respectively. The drill hole projection is 100' from the section. Figures 14-23 through 14-25 are plans of the kriged Cu, Mo, and Ag grade, respectively, along with the composite Cu grades at the 4,000 foot bench elevations. The following Figures include the zone surfaces for the bottom of the oxide and mixed zones as well as the resource and ultimate pit outline.

Plots throughout the model confirmed that the block model grades corresponded very well with the assayed grades.

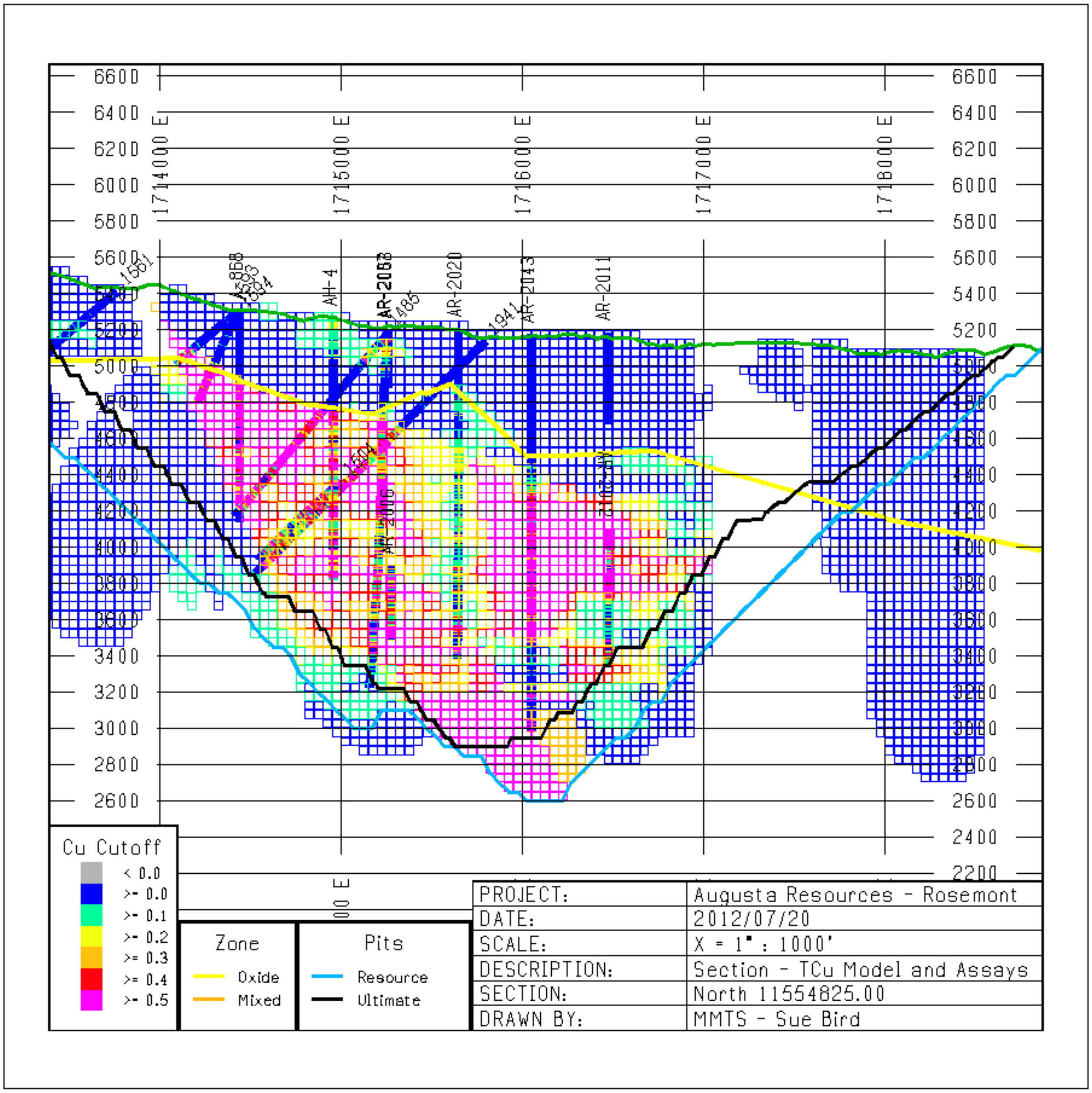

Figure 14-20: E-W Section at 11,554,825N of OK Model and Assays – TCu Grades

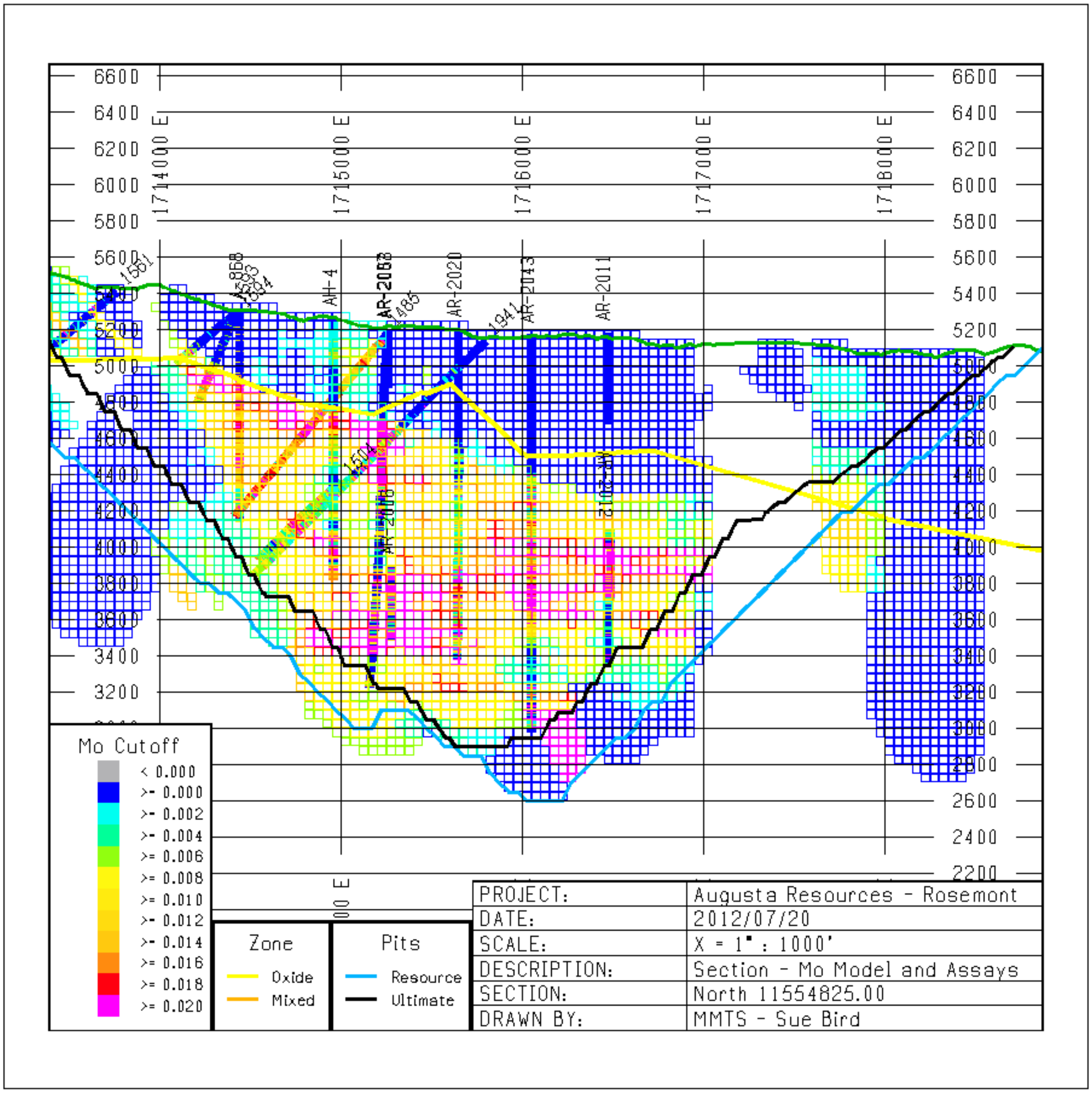

Figure 14-21: E-W Section at 11,554,825N of OK Model and Assays – Mo Grades

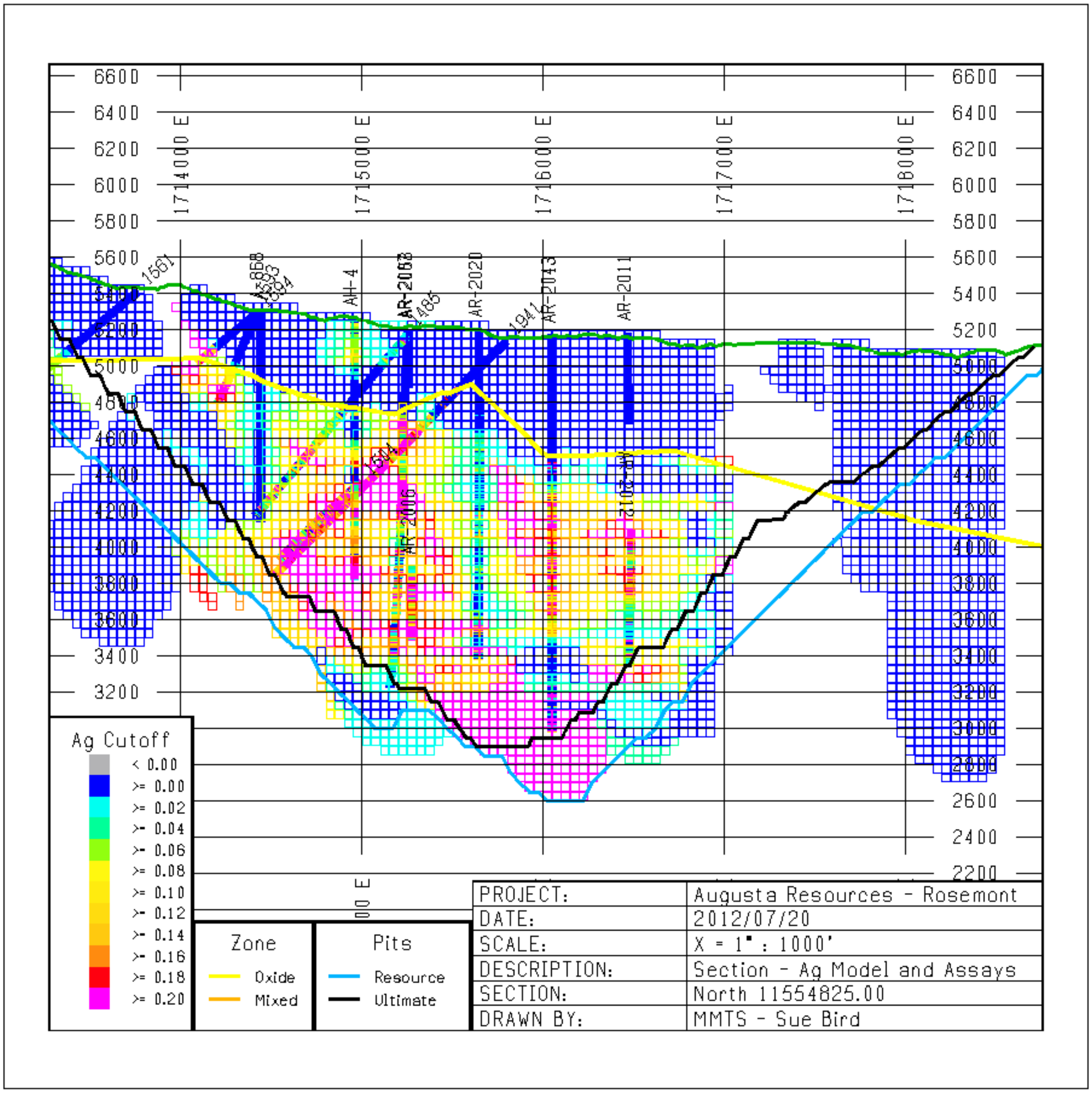

Figure 14-22: E-W Section at 11,554,825N of OK Model and Assays – Ag Grades

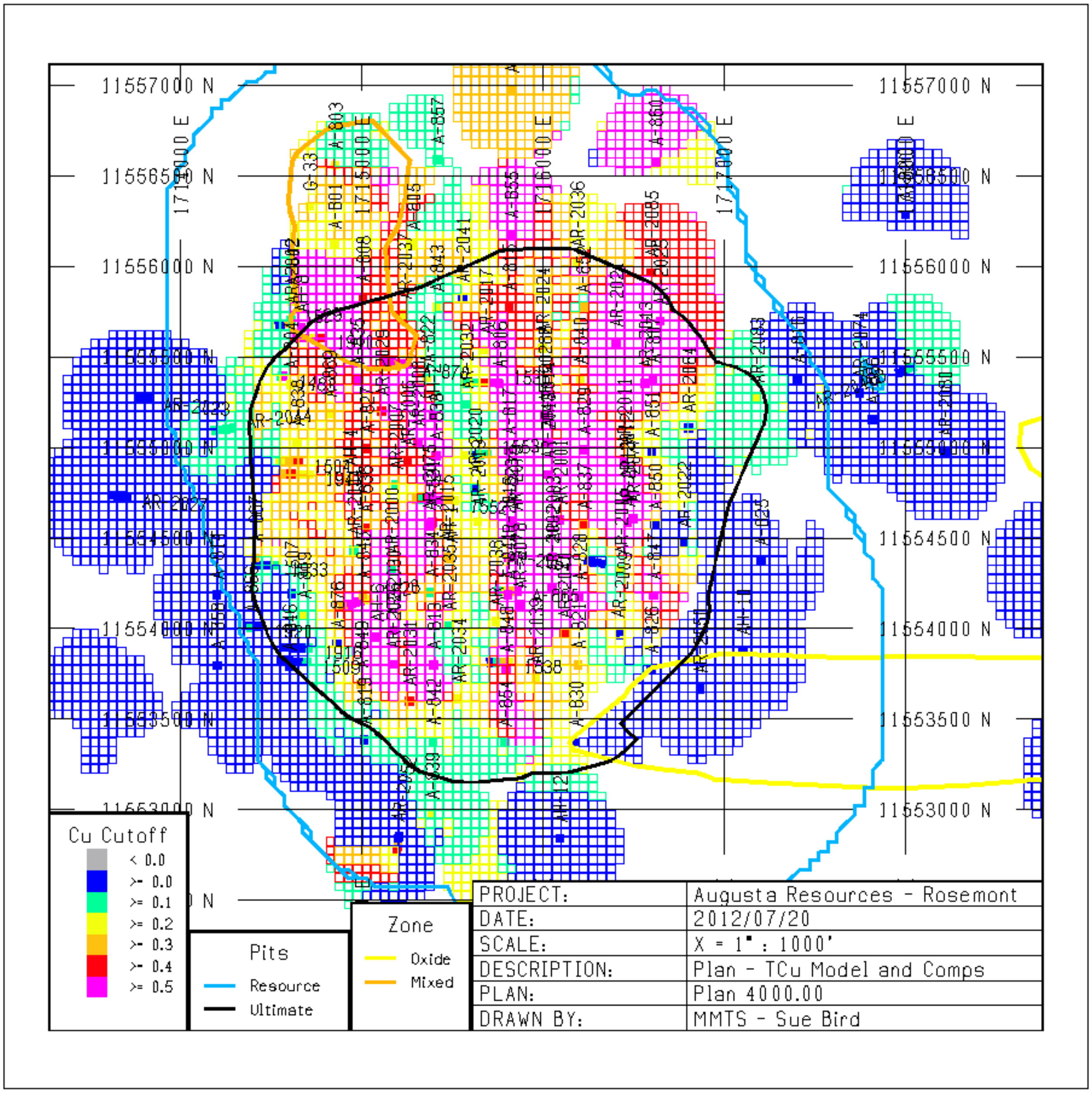

Figure 14-23: Plan at 4000' of OK Model and Assays – TCu Grades

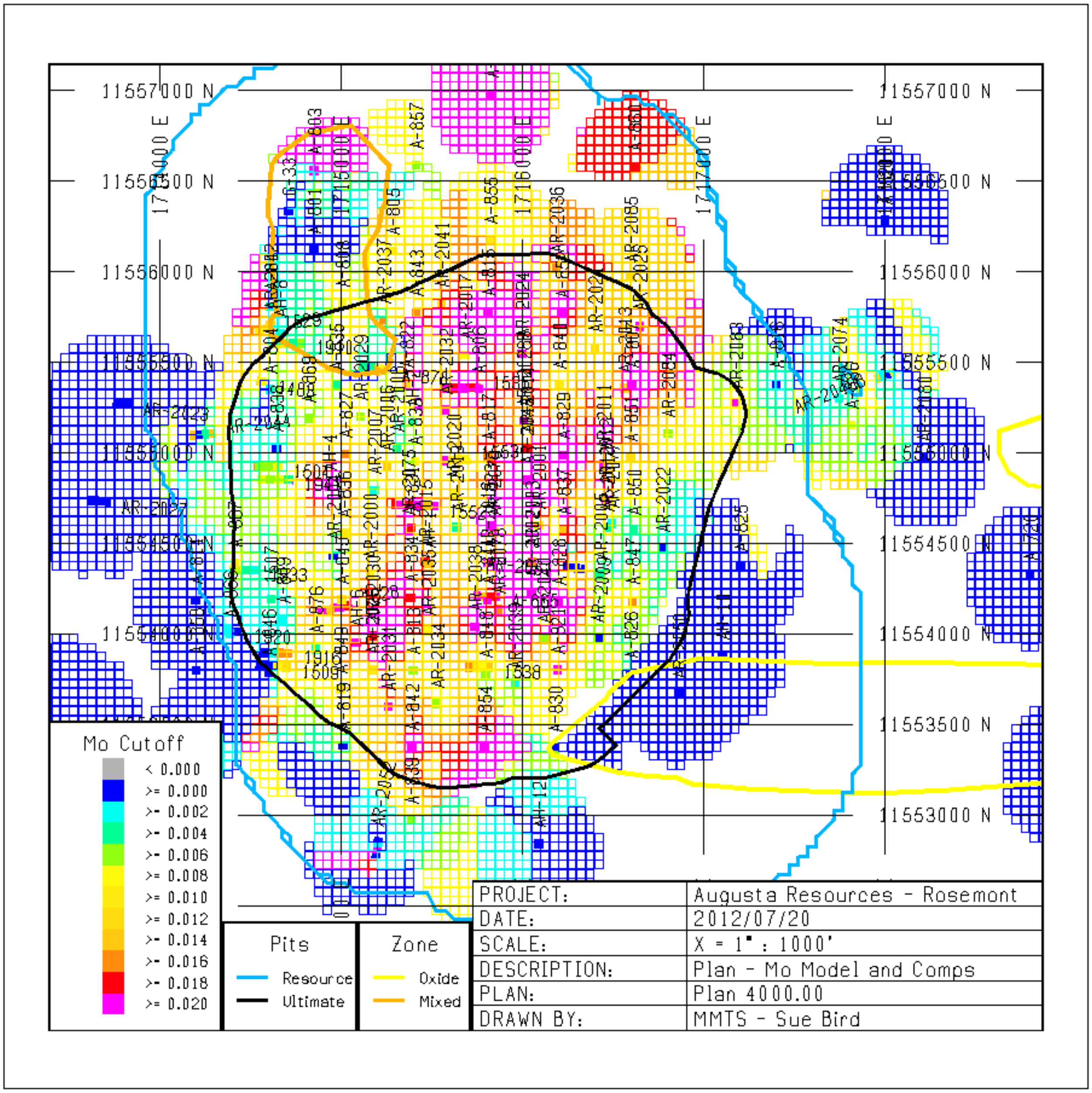

Figure 14-24: Plan at 4000' of OK Model and Assays – Mo Grades

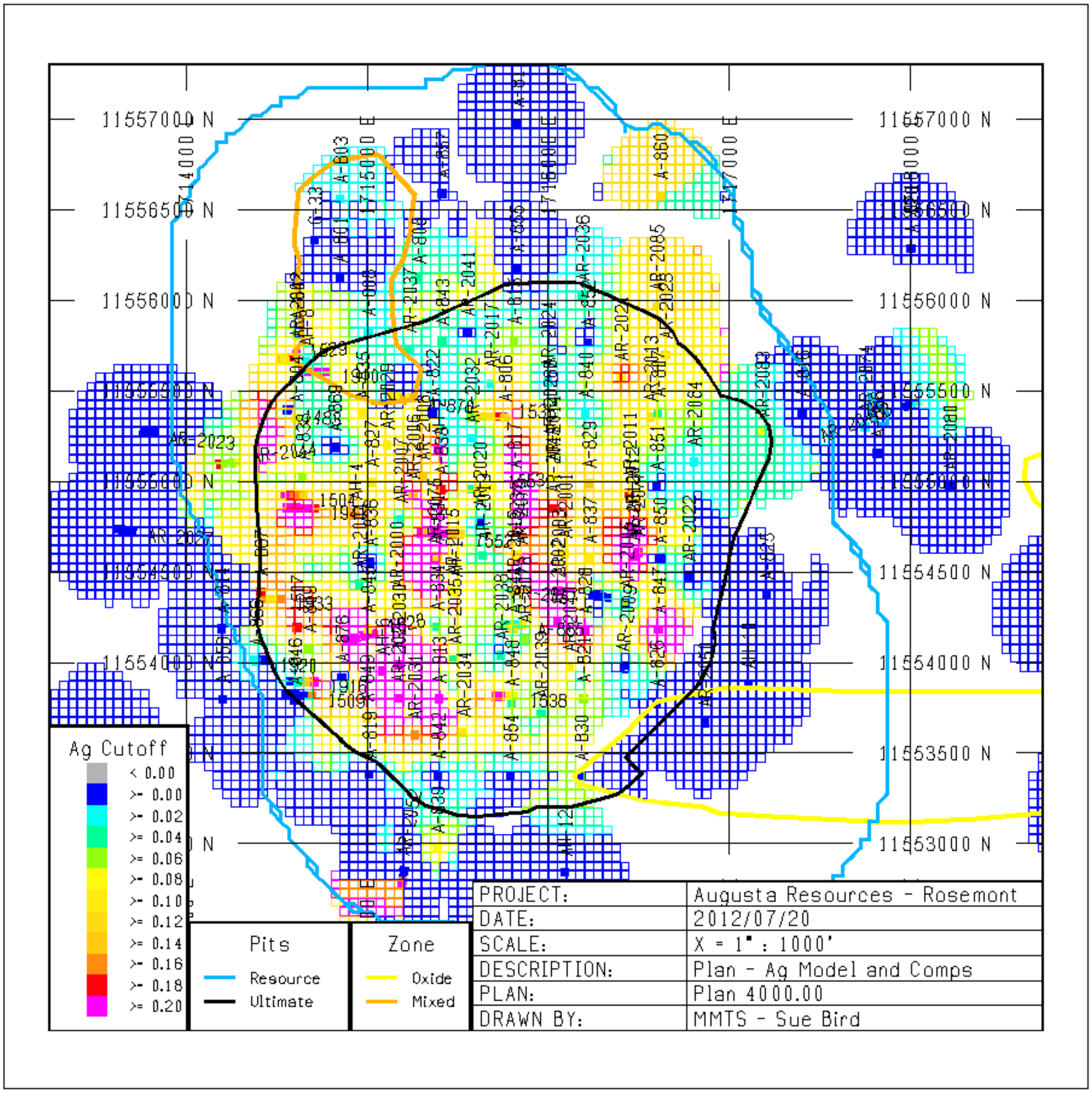

Figure 14-25: Plan at 4000' of OK Model and Assays – Ag Grades

14.5 IN SITU MINERAL RESOURCE ESTIMATE

The mineral resource estimation work was performed by Susan C. Bird, M.Sc., P.Eng. a Senior Associate at MMTS and an independent Qualified Person under the standards set forth by NI 43-101. The final resource model was from May 25, 2012 and was referred to as the "712 model".

A Lerchs-Grossman (LG) pit shell having a 45 degree slope angle has been applied to the three dimensional block model to ensure reasonable prospects of economic extraction for the reported mineral resources. Metal prices used for the resource pit are \$3.50/lb Cu, \$15/lb Mo and \$20/oz Ag. The resource pit optimization was based on mining costs of \$0.777/ton of mineralized material and $0.882/ton of waste material. For sulfide/mixed material a processing cost of $4.20/ton of mineralized material and a general and administrative (G&A) cost of $0.70/ton of mineralized material, for a total of $4.90/ton, was used. For oxide material a processing cost of $3.03/ton of mineralized material was used. These costs are in line with those developed for use in the mineral reserves.

For the reporting of the in-situ resource by equivalent copper (EqvCu) within the LG pit shell, the metallurgic recoveries, metal prices, and resulting net smelter prices (NSPs) used, are summarized in Table 14-18.

Table 14-18: Base Case Recoveries, Metal Prices and Resulting Net Smelter Prices

Metal	Metal Price	Oxides		Mixed		Sulfide	
		NSP	Recovery	NSP	Recovery	NSP	Recovery
Cu	$2.50 /lb	$2.425 /lb	65%	$2.078 /lb	40%	$2.078 /lb	86%
Mo	$15 /lb	0	0	$13.095 /lb	30%	$13.095 / lb	63%
Ag	$20 /oz	0	0	$17.111 /oz	38%	$17.111/oz	80%

The equivalent copper grades are calculated based on the above information, resulting in the following equations for each metallurgical zone:

Sulfide:

$$EqvCu\% = Cu\% + \frac{(Mo\% * 0.63 * 13.095)}{(0.86 * 2.078)} + \frac{(AgOPT * 0.80 * 17.111)}{(0.86 * 2.078 * 20)}$$

Mixed:

$$EqvCu\% = Cu\% + \frac{(Mo\% * 0.30 * 13.095)}{(0.40 * 2.078)} + \frac{(AgOPT * 0.38 * 17.111)}{(0.40 * 2.078 * 20)}$$

Oxide:

$$EqvCu\% = Cu\%$$

The in situ resource is classified as Measured, Indicated or Inferred corresponding to Canadian National Instrument 43-101 standards (CIM, 2005). The resource by equivalent copper grade for the Rosemont deposit is summarized in Tables 14-18 through 14-22, for Measure, Indicated, Measured+Indicated, and Inferred mineral resources respectively. The tables present a range of cutoffs, of which the base case equivalent copper values for each zone are highlighted in each table. These cutoffs are sufficient to cover the processing plus G&A costs for the sulfide and mixed material ($4.90/ton) and the processing costs of the oxide material ($3.03/ton), at the expected metallurgical recoveries.

The measured and indicated mineral resource presented here is inclusive of the mineral reserve presented in the Mineral Reserve section. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Due to the uncertainty that may be associated with Inferred mineral resources it cannot be assumed that all or any part of inferred mineral resources will be upgraded to an Indicated or Measured resource.

Table 14-19: Measured Resource by Cu Equivalent Grade

Zone	Cutoff (Cu Eqv %)	In situ tons (ktons)	In Situ Grades			
			Cu Eqv (%)	Cu (%)	Mo (%)	Ag (opt)
Oxide	0.05	40,865	0.144	0.144	---	---
	0.10	30,289	0.170	0.170	---	---
	0.15	16,806	0.208	0.208	---	---
	0.20	6,662	0.273	0.273	---	---
	0.25	3,337	0.328	0.328	---	---
	0.30	1,688	0.386	0.386	---	---
Mixed	0.10	14,997	0.589	0.545	0.005	0.047
	0.15	14,824	0.594	0.550	0.005	0.048
	0.20	14,621	0.600	0.556	0.005	0.048
	0.25	14,080	0.615	0.570	0.006	0.050
	0.30	13,125	0.640	0.593	0.006	0.053
Sulfide	0.10	365,279	0.519	0.411	0.014	0.116
	0.15	334,619	0.556	0.440	0.015	0.124
	0.20	309,123	0.588	0.466	0.016	0.130
	0.25	282,030	0.623	0.496	0.016	0.137
	0.30	254,867	0.661	0.528	0.017	0.144

Table 14-20: Indicated Resource by Cu Equivalent Grade

Zone	Cutoff (Cu Eqv%)	In situ tons (ktons)	In Situ Grades			
			Cu Eqv (%)	Cu (%)	Mo (%)	Ag (opt)
Oxide	0.05	48,019	0.134	0.134	---	---
	0.10	33,122	0.163	0.163	---	---
	0.15	17,391	0.201	0.201	---	---
	0.20	6,380	0.262	0.262	---	---
	0.25	2,348	0.342	0.342	---	---
	0.30	1,156	0.422	0.422	---	---
Mixed	0.10	43,663	0.509	0.464	0.006	0.039
	0.15	43,328	0.512	0.466	0.006	0.040
	0.20	42,302	0.520	0.474	0.007	0.040
	0.25	40,321	0.535	0.487	0.007	0.042
	0.30	36,834	0.560	0.509	0.007	0.045
Sulfide	0.10	623,039	0.428	0.332	0.013	0.095
	0.15	534,735	0.479	0.373	0.014	0.105
	0.20	468,463	0.523	0.409	0.015	0.114
	0.25	409,461	0.566	0.446	0.016	0.123
	0.30	358,153	0.609	0.481	0.017	0.132

Table 14-21: Measured + Indicated Resource by Cu Equivalent Grade

Zone	Cutoff (Cu Eqv %)	In situ tons (ktons)	In Situ Grades			
			Cu Eqv (%)	Cu (%)	Mo (%)	Ag (opt)
Oxide	0.05	88,884	0.139	0.139	---	---
	0.10	63,411	0.166	0.166	---	---
	0.15	34,197	0.204	0.204	---	---
	0.20	13,042	0.268	0.268	---	---
	0.25	5,685	0.334	0.334	---	---
	0.30	2,844	0.401	0.401	---	---
Mixed	0.10	58,660	0.529	0.485	0.006	0.041
	0.15	58,152	0.533	0.487	0.006	0.042
	0.20	56,923	0.541	0.495	0.006	0.042
	0.25	54,401	0.556	0.508	0.006	0.044
	0.30	49,959	0.581	0.531	0.007	0.047
Sulfide	0.10	988,318	0.462	0.361	0.013	0.103
	0.15	869,354	0.509	0.399	0.014	0.112
	0.20	777,586	0.549	0.432	0.015	0.120
	0.25	691,491	0.589	0.466	0.016	0.129
	0.30	613,020	0.631	0.501	0.017	0.137

Table 14-22: Inferred Resource by Cu Equivalent Grade

Zone	Cutoff (Cu Eqv %)	In situ tons (ktons)	In Situ Grades			
			Cu Eqv (%)	Cu (%)	Mo (%)	Ag (opt)
Oxide	0.05	27,123	0.069	0.069	---	---
	0.10	1,146	0.152	0.152	---	---
	0.15	728	0.167	0.167	---	---
	0.20	---	---	---	---	---
	0.25	---	---	---	---	---
	0.30	---	---	---	---	---
Mixed	0.10	17,094	0.345	0.313	0.005	0.020
	0.15	17,084	0.345	0.313	0.005	0.020
	0.20	16,797	0.348	0.316	0.005	0.021
	0.25	11,727	0.403	0.368	0.006	0.021
	0.30	10,108	0.426	0.388	0.006	0.022
Sulfide	0.10	154,600	0.430	0.345	0.011	0.090
	0.15	128,488	0.494	0.397	0.013	0.104
	0.20	102,116	0.579	0.465	0.014	0.124
	0.25	88,512	0.634	0.510	0.015	0.139
	0.30	80,812	0.669	0.539	0.016	0.147

Augusta's 2012 drilling campaign at the Rosemont deposit has increased both the quantity and confidence level of the estimated mineral resources, which presently totals about 919.3 million tons of measured and indicated, sulfide and mixed mineral resources grading 0.51% CuEqv, 0.41% Cu, 0.014% Mo, and 0.11 ounces per ton Ag, at a 0.15% CuEqv cutoff for sulfide and 0.30% CuEqv cutoff for a minor mixed component. An additional 138.6 million tons of inferred sulfide and mixed mineral resources are estimated at a grade of 0.49% CuEqv, 0.40% Cu, 0.012% Mo, and 0.10 ounces per ton Ag, at the same cutoffs. Sulfide and mixed material can be combined as metallurgical test work of the mixed material indicates that it can be processed with the sulfide material to produce a concentrate. Augusta's recent drilling program and resource modeling was successful in converting significant tonnages of material previously classified as inferred into measured and indicated resource.

In addition, geologic and metallurgical studies conducted by Augusta have shown the potential for considering the oxide copper mineralization that overlies the sulfide deposit. Estimated measured and indicated oxide mineral resources total 63.4 million tons grading 0.17% Cu, at a 0.10% CuEqv cutoff (for oxide % CuEqv = % Cu). An additional inferred oxide mineral resource of 1.1 million tons grading 0.15% Cu is present, using the same cutoff. Oxide material could potentially be processed by heap leaching, to recover the copper.

14.6 ADDITIONAL MINERAL RESOURCE POTENTIAL

The classification of currently inferred sulfide and oxide mineral resources can potentially be improved with further drilling. Additional mineral resources may be found in extensions to the north and down-dip of the Rosemont Deposit. Mineralization is also known to occur at Broadtop Butte, which could potentially be added as a satellite development. Further mineralization also occurs in the Copper World and Peach-Elgin deposits on the Rosemont Property. The mineralized areas at Broadtop Butte, Copper World and Peach-Elgin are characterized by related styles of mineralization and have formed through common geologic processes as the Rosemont Deposit. Historic drilling by Anaconda, Anamax, and ASARCO intercepted significant copper grades in what are commonly widely spaced holes. These areas warrant further exploration and have the potential to add to the mineral resource base.

15 MINERAL RESERVE ESTIMATES

15.1 MINING OVERVIEW

The Rosemont Deposit is a large tonnage, skarn-hosted, porphyry-intruded, copper-molybdenum deposit located in close proximity to the surface and is amenable to open pit mining methods. The proposed pit operations will be conducted from 50-foot-high benches using large-scale equipment, including: 12.25-inch-diameter rotary blasthole drills, 60-cu-yd class electric mining shovels, 25 and 36-cu-yd front-end loaders, 46–cu-yd hydraulic shovel, 260 ton off-highway haul trucks, 580- to 850-hp crawler dozers, 500-hp rubber-tired dozers, 270- to 500-hp motor graders and 30,000-gallon off-highway water trucks.

The mine has a 21-year life, with sulfide ore to be delivered to the processing plant at an initial rate of 75,000 tons per day (tpd). Provisions are included to increase tonnage to 90,000 tons of ore per day in Year 12 of operations. Mine operations are scheduled for 24 hours per day, 365 days per year.

A pre-production period of 1.75 years, or 22 months, will be required to ensure ore is readily available at mill start-up. During this pre-production period, approximately 99 million tons of waste will be stripped and 6 million tons of ore will be moved to the ore stockpile. After mill start-up, an average of 143,000 tpd of waste rock must be removed to maintain adequate ore supplies for continuous plant operations, bringing the total daily material production from the open pit to about 225,000 tons.

All mineral reserve estimates and mine plans are based on the deposit model described in Section 14. Consistent with industry standards for feasibility-grade analyses, mineral reserves are based on only ore-grade material classified as proven and probable; all inferred mineral resources are treated as waste. Imperial units of measurement are used throughout the mine plans. Tons refer to short tons (2000 pounds) and "ktons" refer to tons x 1000.

15.2 GEOTECHNICAL RECOMMENDATIONS

Call & Nicholas, Inc. (CNI) was contracted by Augusta in 2012 to provide an update of their geotechnical recommendations for slope angles for the open pit development of the Rosemont Deposit. The current and previous work included geologic and geotechnical mapping, drilling, rock strength testing and slope stability analysis to determine pit slope design criteria that is consistent with industry norms for safety and cost effectiveness. CNI provided a report in February 2008 - Feasibility-Level Geotechnical Study For The Rosemont Deposit, and subsequently updated the report with a memorandum letter – Preliminary Findings from Slope Stability Review, June 19, 2009. CNI provided updated pit slope recommendations in 2012, based on their recent assessment. CNI also issued a memorandum letter– Slope Angle for Planned Open Pit Mine South Wall Tertiary Gravel Slope, July 20, 2012, updating the pit slope recommendation in the tertiary gravels on the south highwall. The following paragraphs in Section 15.2 are authored by CNI.

The slope design parameters are a function of rock type (lithology), rock strength, wall height, faulting, and bedding and/or structure orientations. CNI subdivided the pit into 14 design sectors, which are illustrated in Figure 15-1.

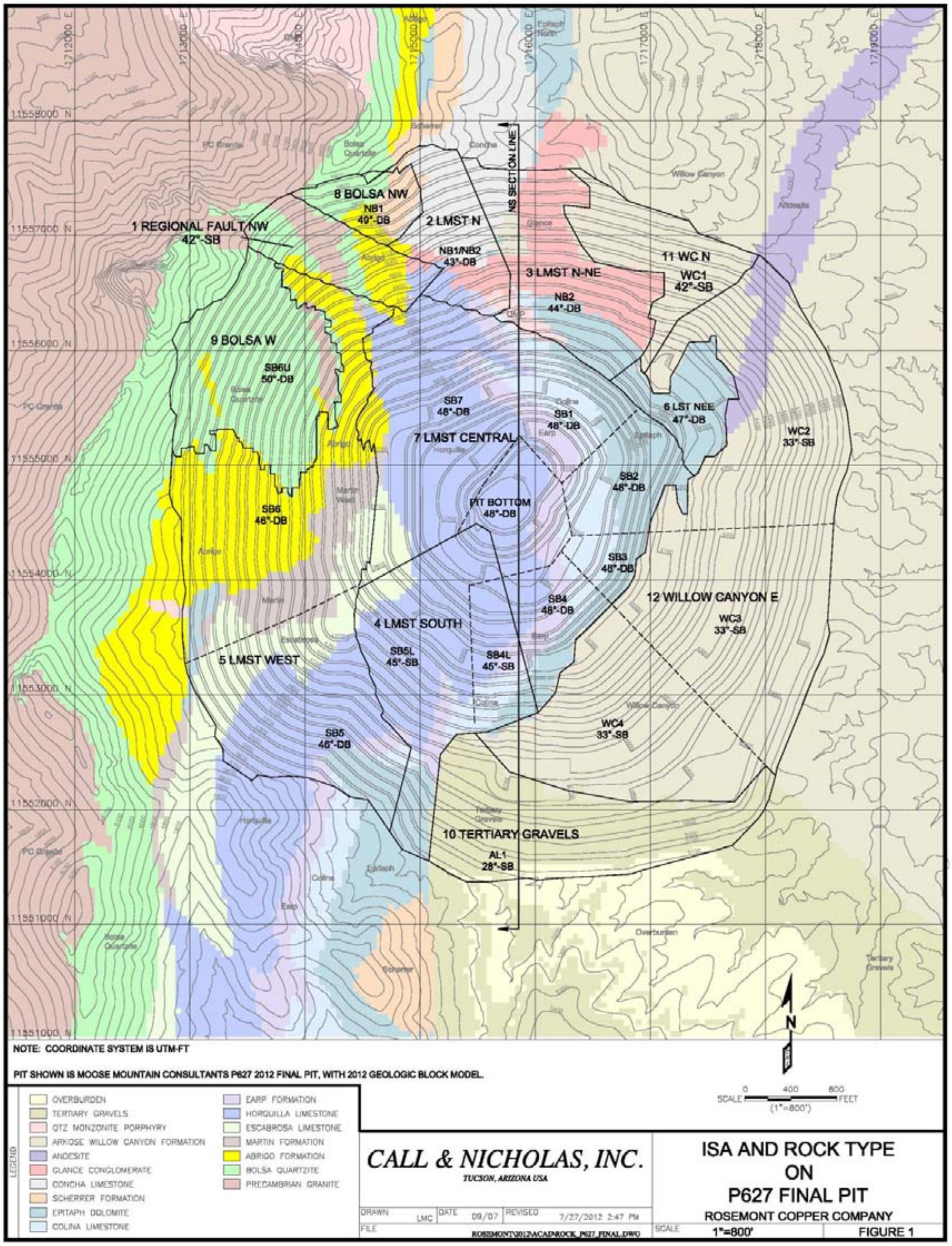

Figure 15-1: Pit Slope Design Sectors and Maximum Slope Angles

Table 15-1 below summarizes by design sector the slope angle recommendations used for all mining phase/pit plans presented in this feasibility study.

Table 15-1: Pit Slope Angle Recommendations

Design Sector	Sector Location	Slope Angles		Bench Face Angle	Bench Height (ft)
		Interramp	Overall		
1	Regional Fault – NW	42°	-	58°	50
2	Limestone – North	43°	-	60°	100
3	Limestone – N to NE	44°	-	66°	100
4	Limestone – South	45°	-	65°	100
5	Limestone – West	46°	-	66°	100
6	Limestone – ENE	47°	-	66°	100
7	Limestone – Central	48°	-	68°	100
8	Bolsa – NW	49°	-	68°	100
9	Bolsa - West	50°	-	68°	100
10	Tertiary Gravels	-	35°	-	50
11	Willow Canyon Frm	-	42°	-	50
12	Willow Canyon Frm	-	33°	-	50
13	Willow Canyon Frm	-	35°	-	50

Generally, slopes in the limestones/skarns and the Bolsa formation can be double-benched (i.e., a catch bench every 100 vertical feet) and are designed using interramp guidelines. An exception to this is Sector 1, which is in proximity to a strong regional fault and where single benching on 50-foot intervals is recommended.

All alluvium/overburden and arkose (Willow Canyon Formation) slopes should be single-benched and are limited to overall angles that are functions of wall height and groundwater levels. At the time of CNI's analysis, the groundwater elevation was estimated to be around 4,400 feet in the immediate vicinity of the proposed open pit. Consequently, interramp slope angle versus slope height graphs were developed for pit walls above and below this elevation. Figure 15-2 presents the interramp slope graph for arkose. These interramp angles were applied mostly to the design of the internal mining phases as CNI's recommendations in Table 15-1 already incorporated slope height allowances for the ultimate pit.

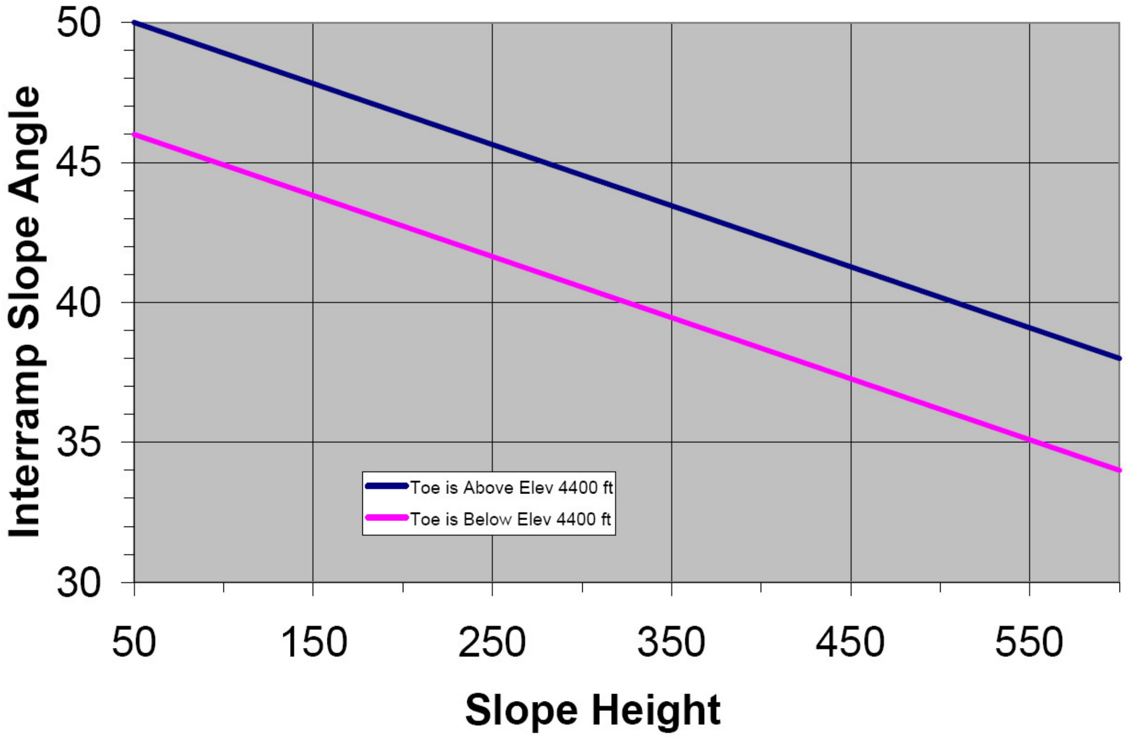

Figure 15-2: Arkose Interramp Slope Angles

15.3 PIT OPTIMIZATION

Processing plans call for the treatment of mostly sulfide ores at a milling rate of 75,000 tpd and increased later on for expansions and increased plant operating availabilities. The mill will produce two concentrates for shipment to off-site smelters or roasters: a copper concentrate that will include recoverable silver and a molybdenum disulfide concentrate.

Lerchs-Grossman analyses were conducted using the Rosemont Deposit model (described in Section 14) to determine the ultimate pit limits and best extraction sequence for open pit mine design. Only mineral resources classified as measured or indicated were considered as potential ore in the Lerchs-Grossman analyses; all inferred resources were treated as waste.

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

15.3.1 Metallurgical Recoveries

Metal recoveries were derived from metallurgical test work conducted by Mountain States Research and Development, Inc. (MSRDI), SGS (Vancouver division) and G&T Metallurgical Laboratories (G&T) in Kamloops, British Columbia. These tests included: grinding and flotation testwork. The metallurgical test work is described more fully in Section 13.

Based on preliminary results early in this test work, Table 15-2 presents the metallurgical recoveries used in the Lerchs-Grossman evaluations and subsequent mineral reserve estimation. Only the three primary metals – copper, molybdenum and silver – were modeled and used in the revenue calculations. No recovery of molybdenum and silver from oxide ore is projected.

Table 15-2: Metallurgical Recoveries Used in Lerchs-Grossman Evaluations

Metal	Oxide Ore	Sulfide Ore	Mixed Sulfide Ore
Copper	65 %	86 %	40%
Molybdenum	-	63 %	30%
Silver	-	80 %	38%

15.3.2 Economic Parameters

Table 15-3 summarizes the economic parameters and offsite costs used in the base-case Lerchs-Grossman evaluations of the Rosemont Deposit.

Table 15-3: Base-case Lerchs-Grossman Economic Parameters

Metal Prices:	
Copper (Cu)	$ 2.50 / lb Cu
Molybdenum (Mo)	$ 15.00 / lb Mo
Silver (Ag)	$ 20.00 / troy oz
Operating Costs (excl oxide leaching):	
Base ore mining	$ 0.777 / ton
Base waste mining	$ 0.882 / ton
Incremental haulage (below pit rim at 5050 ft elevation)	$ 0.028 / ton / bench
Sulfide ore milling & flotation	$ 4.20 / ton ore
General/administration	$ 0.70 / ton ore
Oxide Copper Ore Processing:	
Cu oxide freight & refining	$ 0.00 / lb Cu
Acid consumption	28.6 lbs acid / ton ore
Cost of acid	$ 0.07 / lb acid
Oxide Ore Process Cost	$ 3.03 / ton ore
Copper Concentrate Processing:	
Cu grade in concentrate	30 %
Cu realization	96.5 %
Cu concentrate transportation	$ 75.00 / dry ton
Cu concentrate treatment	$ 55.00 / dry ton
Cu refining	$ 0.055 / lb Cu
Ag realization	90.0 %
Ag refining	$ 0.40 / troy oz Ag
Molybdenum Concentrate Processing:	
Mo grade in concentrate	50 %
Mo realization	90.0 %
Mo concentrate transportation	$ 0.00 / dry ton
Mo treatment & refining	$ 0.00 / lb Mo
NSR royalty	3 %

The base input mining costs are estimated from the results derived from the 2009 Feasibility Study (staff salaries in 2009 mining costs are excluded, and included with the G & A costs for this study). When applied along with the increment bench costs to the material contained within the base-case Lerchs-Grossman pit shell, the average mining cost is nearly $1.11 per ton of material. Mining costs near the pit bottom – below 3750 level, will exceed $1.68 per ton in 2012 US Dollars.

Consistent with current market conditions, no price participation charges are included in the concentrate processing costs. NSR values are computed using the parameters in Table 15-2 and Table 15-3, and are incorporated into the following formula for sulfide ore:

NSR, \$/ton = [(Net CuPrice * CuGrade * CuRec)
+ (Net MoPrice * MoGrade * MoRec)
+ (Net AgPrice * AgGrade * AgRec)]

Where:	Net CuPrice	=	Cu price in \$/lb net of offsite costs
	Net MoPrice	=	Mo price in \$/lb net of offsite costs
	Net AgPrice	=	Ag price in \$/oz net of offsite costs
	CuGrade	=	Interpolated block Cu grade expressed in %
	MoGrade	=	Interpolated block Mo grade expressed in %
	AgGrade	=	Interpolated block Ag grade expressed in oz/ton
	CuRec	=	Cu process recovery
	MoRec	=	Mo process recovery
	AgRec	=	Ag process recovery

For the Base Case metal prices:

NSR(sulf), \$/ton = (\$2.078/lb * CuGrade/100 * 86% * 2000 lb/ton) + (\$13.095/lb * MoGrade /100 * 63% * 2000 lb/ton) + (\$17.111/oz * AgGrade * 80%)

NSR(mixed), \$/ton = (\$2.078/lb * CuGrade/100 * 40% * 2000 lb/ton) + (\$13.095/lb * MoGrade /100 * 30% * 2000 lb/ton) + (\$17.111/oz * AgGrade * 38%)

Similarly, the NSR formula for oxide ore is:

NSR(ox), \$/ton = (NetCuPrice *CuGrade * (1-Royalty) * CuRec

= (\$2.50 * CuGrade/100 * (1-3%) * 65% * 2000 lb/ton

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

15.3.3 Slope Angles

Overall slope angles used on the Lerchs-Grossman evaluations were derived from the geotechnical recommendations made by CNI for pit slope designs. The overall slopes were adjusted to accommodate CNI's recommended slope angles and the anticipated placement of internal haulage ramps along the pit walls in certain design sectors. CNI provide slopes angles for each model block, and a slope code was assigned to the block representing each of the pit slopes. The slope codes and pit slopes are then read as input to the Lerchs-Grossman analysis. The resulting overall slope angles are summarized in Table 15-4.

Table 15-4: Overall Slope Angles Used in Lerchs-Grossman Analyses

Slope Code in Block Model	Slope Angle
1	28°
2	33°
3	35°
4	42°
5	43°
6	44°
7	45°
8	46°
9	47°
10	48°
11	49°
12	50°

15.3.4 Lerchs-Grossman Analyses

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

The base-case Lerchs-Grossman pit shell is defined by the recoveries and economic parameters listed in Table 15-2 and Table 15-3, respectively. The metal prices of $2.50 /lb Cu, $15.00 /lb Mo and $20.00 /oz Ag are below a three-year trailing average. This pit shell contains about 755 million tons of measured and indicated sulfide mineral resources above an internal NSR cutoff of $4.90/ton and approximately 62 million tons of measured and indicated oxide mineral resources above a $3.03 /ton NSR cutoff. The resulting stripping ratio is about 1.8:1 (tons waste per ton of ore). However, this is not the pit shell selected for design. The current design for the tailings facilities has a limited capacity for approximately 680 million tons of ore feed, and a pit design around the base-case economic pit shell would produce tailings that will exceed the capacity of the storage facility. Therefore the selected pit shell has to contain measured and indicated sulfide mineral resources less than 680 million tons, and is picked from a set of simulations resulting from a sensitivity analysis.

Additional Lerchs-Grossman runs were made to evaluate sensitivities to metal prices and to mine operating costs. These sensitivities were generally conducted in 5% increments to +30% and –50% of the base case parameters. Table 15-5 and Table 15-6 present the results of the Lerchs-Grossman price and cost sensitivity analyses, respectively.

The pit shell that yields nearest to 680 million tons of measured and indicated mineral resource is one that is simulated with metal prices 25% less than the base case. The copper price used to generate this case is $1.88 /lb. It is selected as the basis for the ultimate pit design, and is approximately 9% smaller than the optimum economic pit.

The selected Lerchs-Grossman pit shell compared to the base-case pit shell is shown in plan in Figure 15-3 and in cross section in Figure 15-4.

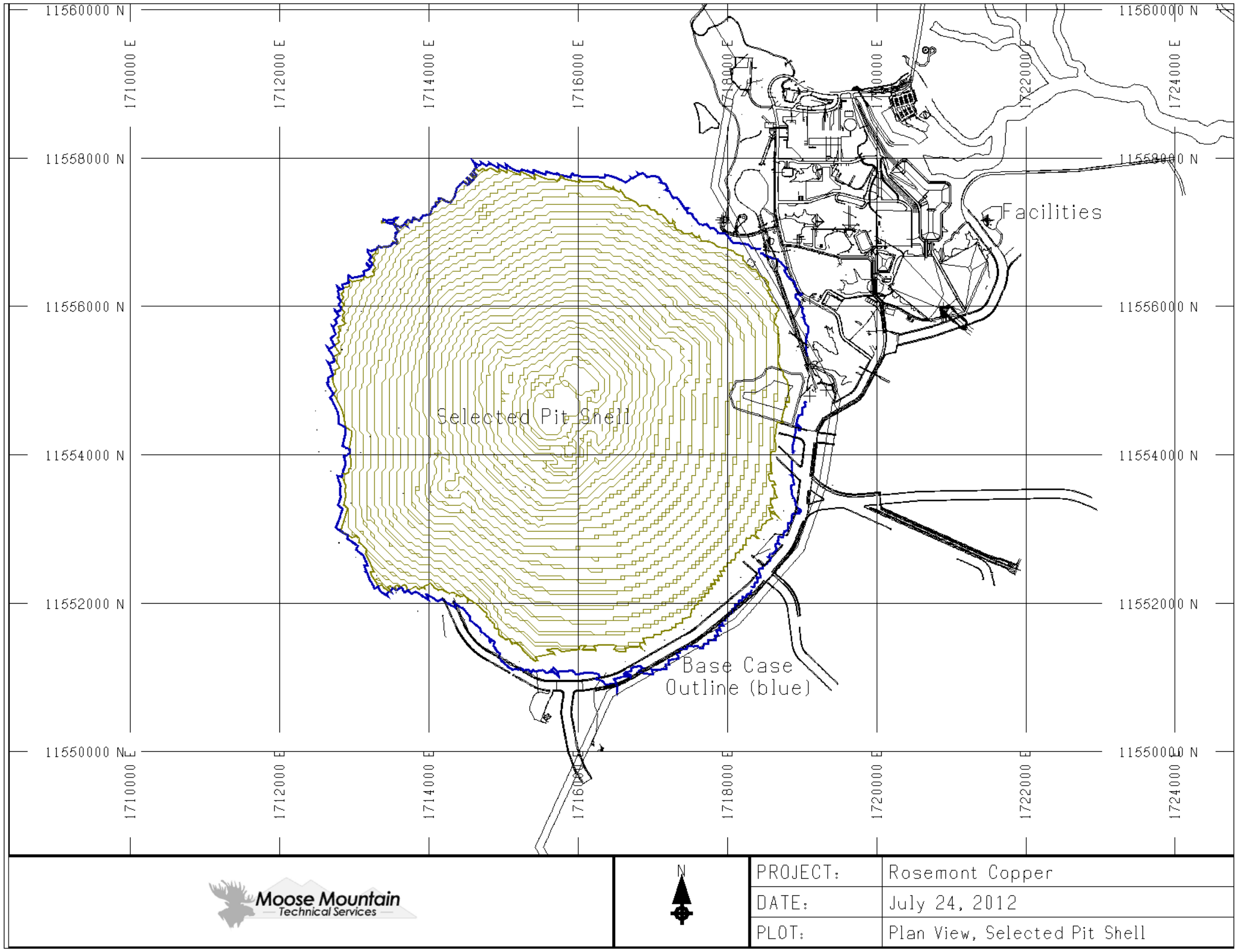

Figure 15-3: Plan View Contours of Selected Lerchs-Grossman Pit Shell

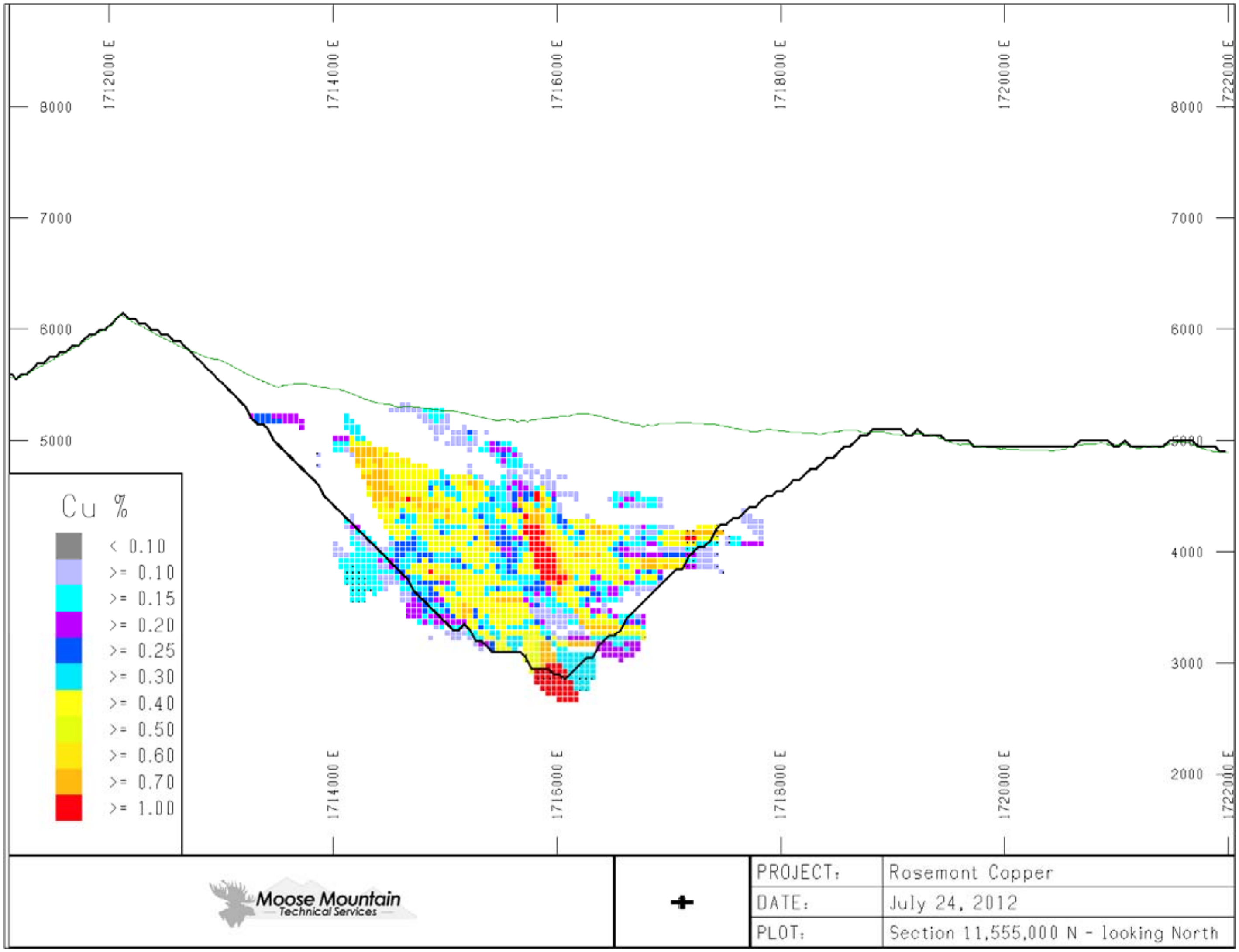

Figure 15-4: East West Section View of Selected Lerchs-Grossman Pit Shell

Table 15-5: Lerchs-Grossman Results – Metal Price Sensitivities

	Prices			Internal Cutoffs		Sulfide Mineral Resources* Above Internal NSR Cutoffs					Oxide Mineral Resources* Above Internal NSR Cutoffs			Waste	Total	Strip
Sensitivity	Cu $/lb	Mo $/lb	Ag $/oz	Sulfide	Oxide	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %	Ktons	Ktons	Ratio
+30%	3.25	19.50	26.00	4.90	3.03	785,495	18.55	0.42	0.014	0.12	67,391	5.40	0.17	1,628,490	2,481,376	1.91
+20%	3.00	18.00	24.00	4.90	3.03	779,526	18.62	0.42	0.014	0.12	67,329	5.40	0.17	1,601,579	2,448,434	1.89
+15%	2.88	17.25	23.00	4.90	3.03	776,396	18.66	0.42	0.014	0.12	67,318	5.40	0.17	1,586,457	2,430,171	1.88
+10%	2.75	16.50	22.00	4.90	3.03	768,862	18.74	0.42	0.014	0.12	67,308	5.40	0.17	1,549,866	2,386,036	1.85
+5%	2.63	15.75	21.00	4.90	3.03	762,080	18.82	0.43	0.014	0.12	67,298	5.40	0.17	1,525,459	2,354,837	1.84
Base	**2.50**	**15.00**	**20.00**	**4.90**	**3.03**	**755,360**	**18.88**	**0.43**	**0.014**	**0.12**	**67,277**	**5.40**	**0.17**	**1,494,128**	**2,316,765**	**1.82**
-5%	2.38	14.25	19.00	4.90	3.03	747,728	18.95	0.43	0.014	0.12	67,277	5.40	0.17	1,458,514	2,273,519	1.79
-10%	2.25	13.50	18.00	4.90	3.03	735,512	19.03	0.43	0.014	0.12	67,277	5.40	0.17	1,397,931	2,200,720	1.74
-15%	2.13	12.75	17.00	4.90	3.03	717,206	19.16	0.43	0.014	0.12	67,277	5.40	0.17	1,316,674	2,101,157	1.68
-20%	2.00	12.00	16.00	4.90	3.03	707,591	19.28	0.44	0.015	0.12	67,266	5.40	0.17	1,296,982	2,071,839	1.67
-25%	1.88	11.25	15.00	4.90	3.03	687,923	19.38	0.44	0.015	0.12	67,256	5.40	0.17	1,204,093	1,959,272	1.59
-30%	1.75	10.50	14.00	4.90	3.03	662,867	19.58	0.45	0.015	0.12	67,227	5.40	0.17	1,131,895	1,861,989	1.55
-35%	1.63	9.75	13.00	4.90	3.03	638,135	19.76	0.45	0.015	0.12	67,205	5.40	0.17	1,066,663	1,772,003	1.51
-40%	1.50	9.00	12.00	4.90	3.03	597,389	20.00	0.46	0.015	0.12	67,154	5.40	0.17	959,114	1,623,657	1.44
-45%	1.38	8.25	11.00	4.90	3.03	489,084	20.68	0.47	0.015	0.13	67,099	5.40	0.17	736,291	1,292,474	1.32
-50%	1.25	7.50	10.00	4.90	3.03	417,535	21.19	0.48	0.015	0.13	66,311	5.41	0.17	616,534	1,100,380	1.27

* Only measured and indicated mineral resources are reported above; all inferred mineral resources are treated as waste.

Table 15-6: Lerchs-Grossman Results – Cost Sensitivities

	Prices			Internal NSR Cutoffs		Sulfide Mineral Resources* Above Internal NSR Cutoffs					Oxide Mineral Resources* Above Internal NSR Cutoffs			Waste	Total	Strip
Sensitivity	Cu $/lb	Mo $/lb	Ag $/oz	Sulfide	Oxide	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %	Ktons	Ktons	Ratio
+50%	2.50	15.00	20.00	4.90	3.03	725,441	19.07	0.43	0.014	0.12	67,277	5.40	0.17	1,336,228	2,128,946	1.69
+40%	2.50	15.00	20.00	4.90	3.03	735,814	19.01	0.43	0.014	0.12	67,277	5.40	0.17	1,390,396	2,193,487	1.73
+30%	2.50	15.00	20.00	4.90	3.03	737,969	19.00	0.43	0.014	0.12	67,277	5.40	0.17	1,399,143	2,204,389	1.74
+20%	2.50	15.00	20.00	4.90	3.03	746,815	18.93	0.43	0.014	0.12	67,277	5.40	0.17	1,443,712	2,257,804	1.77
+15%	2.50	15.00	20.00	4.90	3.03	750,421	18.91	0.43	0.014	0.12	67,277	5.40	0.17	1,462,073	2,279,771	1.79
+10%	2.50	15.00	20.00	4.90	3.03	750,617	18.91	0.43	0.014	0.12	67,277	5.40	0.17	1,462,245	2,280,139	1.79
+5%	2.50	15.00	20.00	4.90	3.03	752,313	18.90	0.43	0.014	0.12	67,277	5.40	0.17	1,476,334	2,295,924	1.80
Base	**2.50**	**15.00**	**20.00**	**4.90**	**3.03**	**755,360**	**18.88**	**0.43**	**0.014**	**0.12**	**67,277**	**5.40**	**0.17**	**1,494,128**	**2,316,765**	**1.82**
-5%	2.50	15.00	20.00	4.90	3.03	760,160	18.84	0.43	0.014	0.12	67,277	5.40	0.17	1,517,840	2,345,277	1.83
-10%	2.50	15.00	20.00	4.90	3.03	760,510	18.83	0.43	0.014	0.12	67,287	5.40	0.17	1,518,915	2,346,712	1.83
-15%	2.50	15.00	20.00	4.90	3.03	761,749	18.82	0.43	0.014	0.12	67,308	5.40	0.17	1,525,957	2,355,014	1.84
-20%	2.50	15.00	20.00	4.90	3.03	764,873	18.79	0.43	0.014	0.12	67,308	5.40	0.17	1,543,290	2,375,471	1.85
-25%	2.50	15.00	20.00	4.90	3.03	766,943	18.78	0.42	0.014	0.12	67,308	5.40	0.17	1,559,164	2,393,415	1.87

* Only measured and indicated mineral resources are reported above; all inferred mineral resources are treated as waste.

The estimates presented in Tables 15-5 and 15-6 should not be confused with mineral reserves, which are based on open pit designs that incorporate access, operating, geotechnical and other criteria in addition to economic constraints. The Lerchs-Grossman results should not be relied upon, but do provide an indication of potential mineral reserves that must be validated by proper designs. *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*

A 20% increase in metal prices boosts the ore-grade measured and indicated mineral resources by only 3%, while a 20% decrease in prices reduces them by about 6%. Figure 15-5 is a graphical presentation of the sensitivity to metal prices. Similarly, a 20% mine operating cost increase lowers ore-grade mineral resources by 1% and a 20% cost decrease adds to these resources also by about 1% in tonnage. If mine operating costs increase as much as 50%, there is only a 4% decrease in the contained resource. The graph in Figure 15-6 shows the sensitivity to mine operating costs.

In summary, the potential recoverable open pit resource is not very sensitive to metal prices until they drop below 40% less than the base case. It is even less sensitive to changes in mine operating costs. Upside pit expansion is impacted more as a result of the easterly dipping mineralized beds and the resulting rapidly increasing incremental stripping ratios.

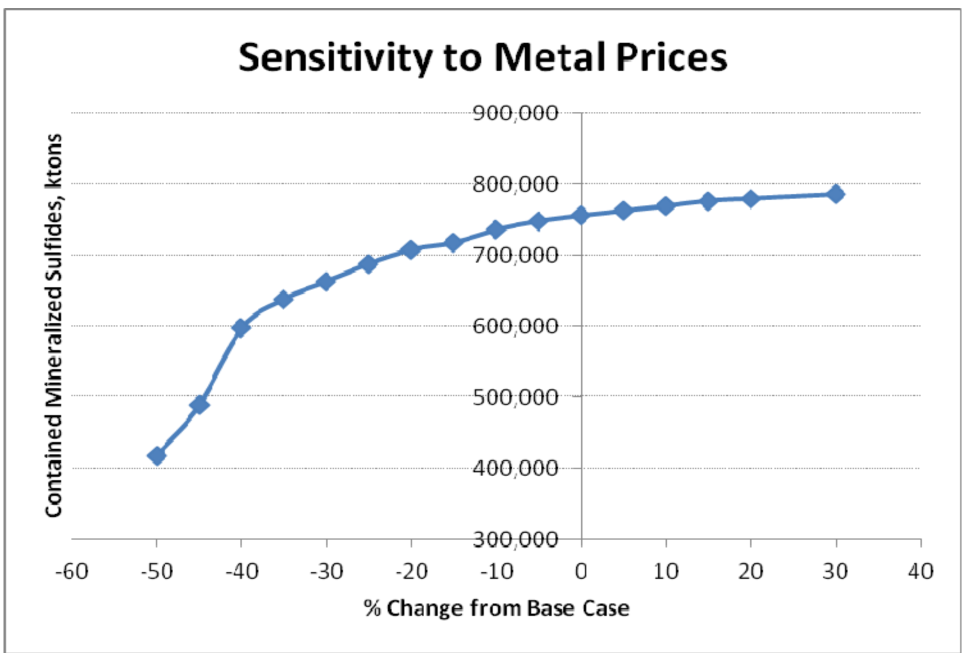

Figure 15-5: Sensitivity Analysis on Metal Prices

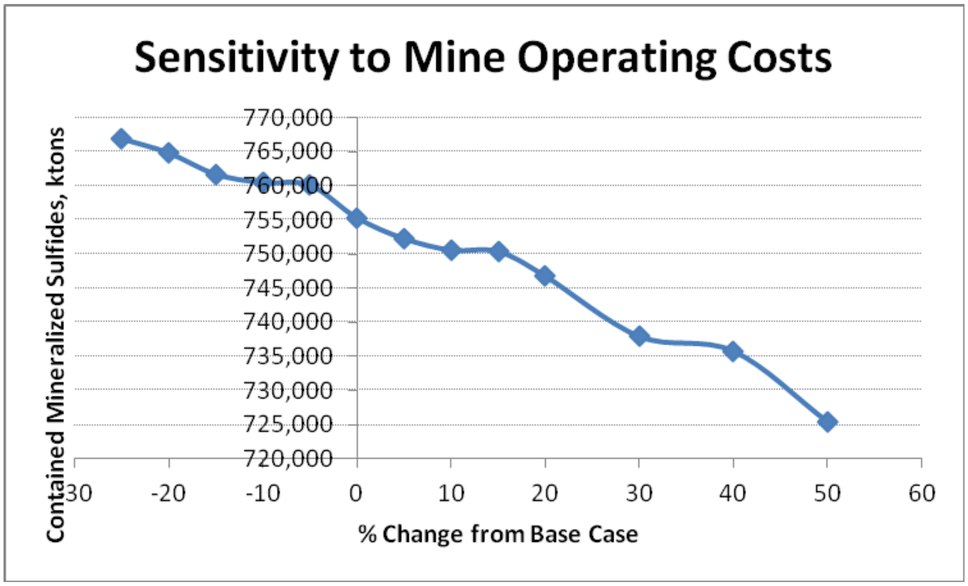

Figure 15-6: Sensitivity Analysis on Mine Operating Costs

15.4 MINING PHASE DESIGNS

The ultimate Rosemont pit is designed for large-scale mining equipment (specifically, 60-cu-yd class electric shovels and 260-ton haulage trucks) and is derived from the selected Lerchs-Grossman pit shell described in the previous section. The design process included smoothing pit walls, eliminating or rounding significant noses and notches that may affect slope stability, and providing access to working faces by developing internal ramps.

15.4.1 Pit Design Parameters

The slope angles used for the design of the Rosemont ultimate pit and internal mining phases were presented earlier in Table 15-1 (see Section 15.2). These slope angles allow for catch bench widths of 50-53 feet in the limestones/skarns and Bolsa Formation where the pit slopes are double-benched (i.e., vertical catch bench intervals of 100 feet). Slopes will be single-benched (i.e., on 50-foot intervals) in alluvium and arkose rock types, providing catch bench widths – toe to crest – of 25 to 48 feet. Interramp slopes and, hence, catch bench widths in alluvium and arkose vary according to the slope height and presence of groundwater.

The remaining parameters used in the designs of the ultimate pit and mining phases are presented in Table 15-7.

Table 15-7: Pit Design Parameters

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

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

15.4.2 Mining Phases and Ultimate Pit

Seven mining phases define the extraction sequence for the Rosemont Deposit. The phase development strategy consists of the extracting the highest metal grades along with minimum strip ratios during the initial years to maximize the economic benefits of the ore-body, while enabling smooth transitions in waste stripping throughout the life of mine to ensure availability of ore feed to the mill.

The starter pit, Phase 1, is fit approximately to the Lerch-Grossman pit shell defined by a $1.09/lb Cu price (the 43% of base metal price sensitivity case). The set of pit shells that were used to approximate this phase as well as the subsequent ones is in Appendix D. This pit is located about 3,200 feet west of the primary crusher and ranges in elevation from 5,650 to 4,350 feet. The phase is approximately 2,600 feet wide east-west and 3,300 feet north-south. The upper benches will be dozed down until haul road access can be developed to the 5,550 foot elevation. Phase 1 will develop approximately 61 million tons of sulfide ore at a stripping ratio of 2.3:1 (tons waste per ton of total ore). An illustration of the Phase 1 pit is shown in Figure 15-7.

Phase 1 material will be accessed via a haul road that will be constructed from the pit exit eastward to the primary crusher. This road will also branch off towards the waste rock storage (WRS) areas. These roads will be used for the life of the project, and will also be extended to access the dry stack tailings areas.

The pit entrance is at the 5,150 foot elevation, and a ramp from that location enters the pit in a counter clockwise direction. The ramp switches back at the 4,950, and 4,650 foot elevations before reversing to a clockwise direction to the bottom of the pit. The benches below 4,550 foot elevation are access by a single lane with haul road.

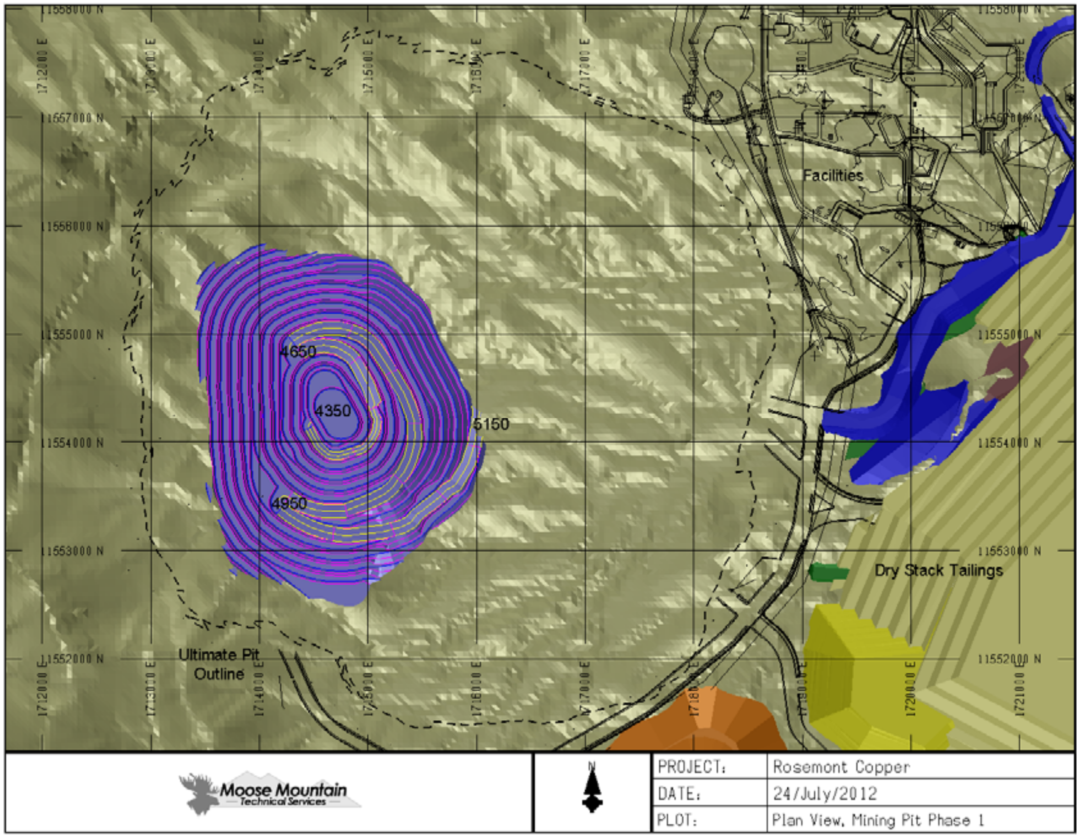

Figure 15-7: Plan View of Mining Pit Phase 1

Mining Phase 2 will expand the pit roughly 600 feet to the north, 400 feet to the east and 500 feet to the southeast. The eastern most limits of this pushback lie about 2,800 feet west of the primary crusher. Bench toe elevations will range from 5,750 to 4,300 feet. The phase is 2,900 feet wide east-west and 4,000 feet north-south. Phase 2 will supply over 27 million tons of sulfide ore. The average stripping ratio for this pushback is 3.1:1. An illustration of the Phase 2 pit is shown in Figure 15-8.

The pit entrance is at the 5,150 foot elevation, and a ramp from that location enters the pit in a clockwise direction. The ramp switches back at the 5,050, 4,750, and 4,550 foot-elevations before reversing to a counter clockwise direction to the bottom of the pit.

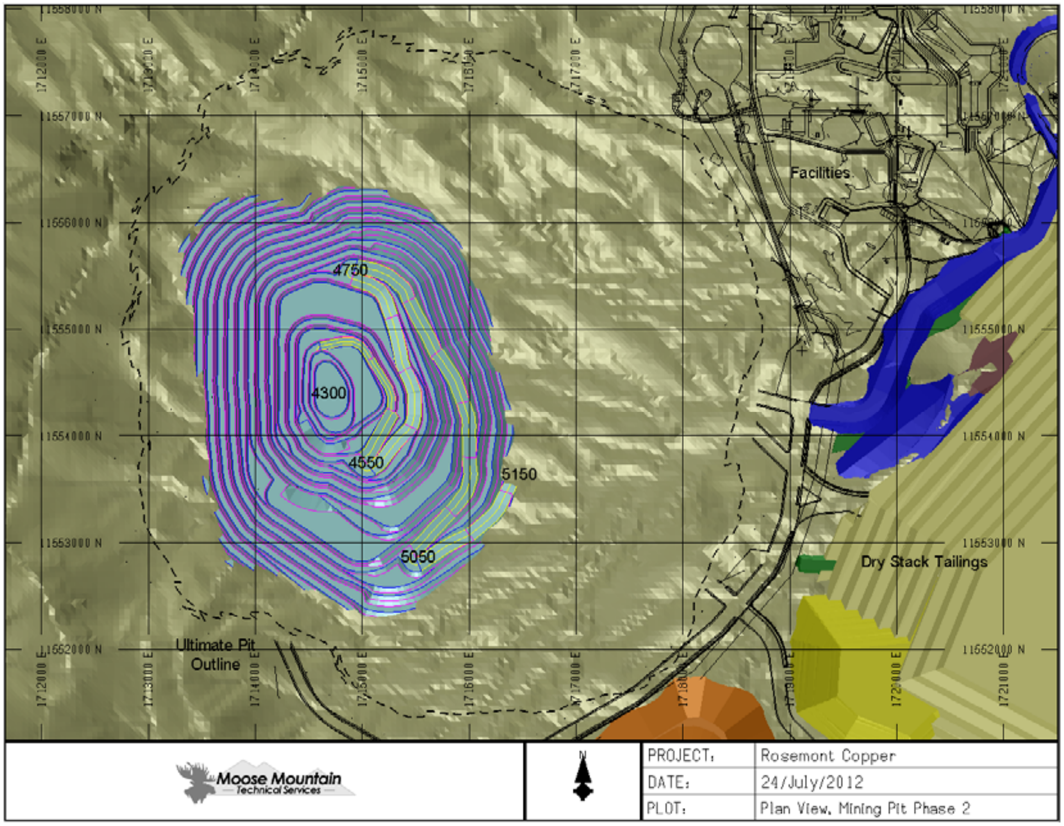

Figure 15-8: Plan View of Mining Pit Phase 2

The open pit is further expanded 300 to 400 feet to the east with the development of Phase 3. The eastern most limits of this pushback lie about 2,500 feet west of the primary crusher. Benches will range between 5,750 and 4,150 feet toe elevations. The phase is 3,300 feet wide east-west and 4,200 feet north-south. Over 42 million tons of sulfide ore will be generated by Phase 3 at an average stripping ratio of 1.4:1. Phases 2, and 3 fit approximately to the Lerchs-Grossman pit shell defined by a $1.13/lb Cu price (the 45% of base case metal price sensitivity).

This expansion from the Phase 1 pit is split into 2 separate pushbacks both in the same general direction. For each phase expansion the ramp on the east side of the pit is re-developed. An illustration of the Phase 3 pit is shown in Figure 15-9.

The pit entrance is at the 5,150 foot elevation, and a ramp from that location enters the pit in a counter clockwise direction. The ramp switches back at the 5,050, 4,750, 4,550 and 4,400 foot elevations before reversing to a counter clockwise direction to the bottom of the pit.

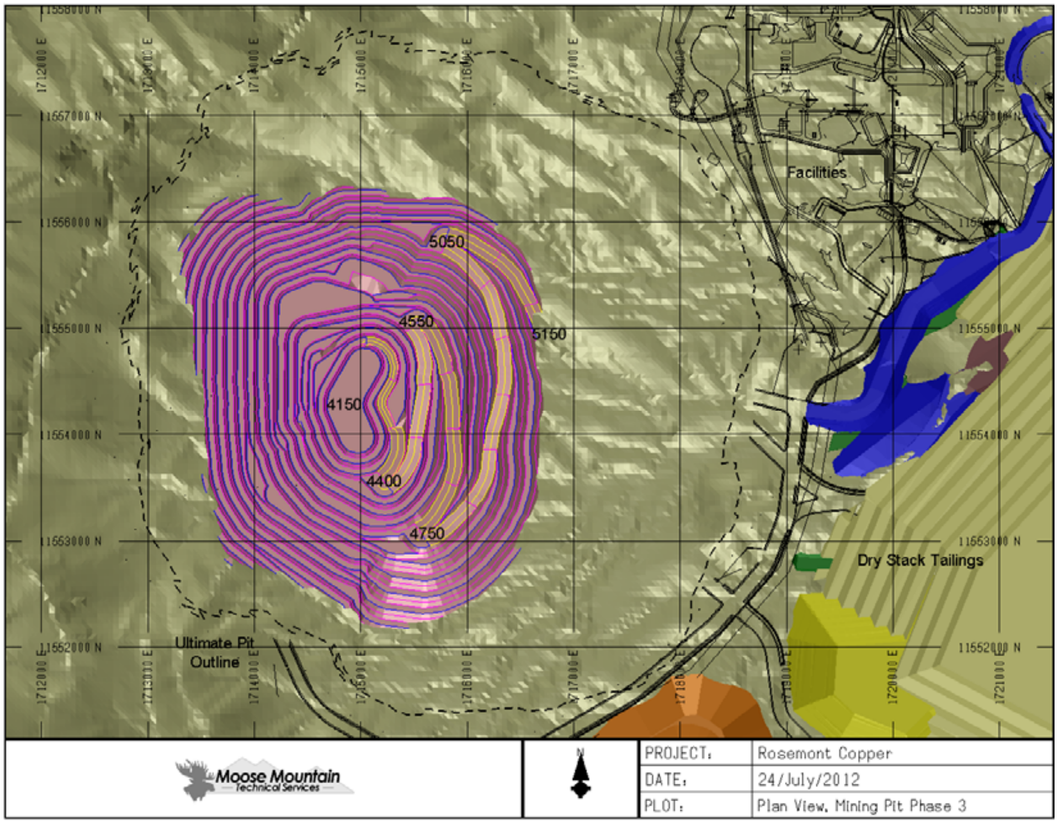

Figure 15-9: Plan View of Mining Pit Phase 3

Phase 4 will expand the open pit about 600 feet to the east and 400 feet to the north. The eastern most limits of this pushback lie about 1,700 feet west of the primary crusher. Phase 4 benches range between 5,450 and 3,950 feet. The phase is 3,800 feet wide east-west and 4,700 feet north-south. Phase 4 will produce nearly 43 million tons of sulfide ore at a stripping ratio of 2.4:1. Phase 4 is fit approximately to the Lerchs-Grossman pit shell defined by a $1.17/lb Cu price (the 47% of base case metal price value sensitivity). An illustration of the Phase 4 pit is shown in Figure 15-10.

The pit entrance is at the 5,100 foot elevation, and a ramp from that location enters the pit in a counter clockwise direction. The ramp switches back at the 4,950, 4,650, 4,450, 4,300 and 4,150 foot elevations before reversing to a clockwise direction to the bottom of the pit.

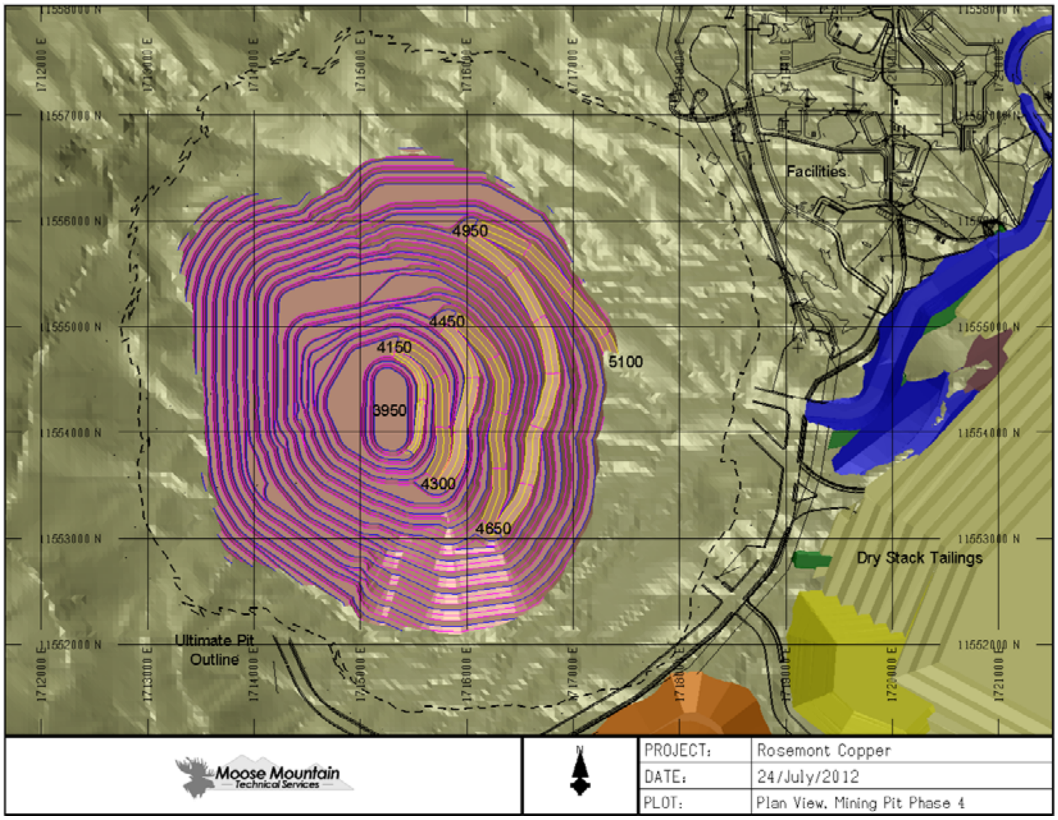

Figure 15-10: Plan View of Mining Pit Phase 4

Phase 5 is fit approximately to the Lerchs-Grossman pit shell defined by a $1.28/lb Cu price (the 51% of base case metal price value sensitivity). Mining Phase 5 expands the pit approximately 300 feet to the north and 600 feet to the east. The eastern most limits of this pushback lie about 1,200 feet west of the primary crusher. Phase 5 benches range between 5,650 and 3,750 feet. The phase is 4,500 feet wide east-west and 4,800 feet north-south. Phase 5 will produce nearly 80 million tons of sulfide ore at a stripping ratio of 2.0:1.The ramp on the east side of the pit is re-developed for this phase. An illustration of the Phase 5 pit is shown in Figure 15-11.

The pit entrance is at the 5,050 foot elevation, and a ramp from that location enters the pit in a counter clockwise direction. The ramp switches back at the 5,000, 4,800, 4,400, and 4,100 foot elevations before reversing to a counter clockwise direction to the bottom of the pit.

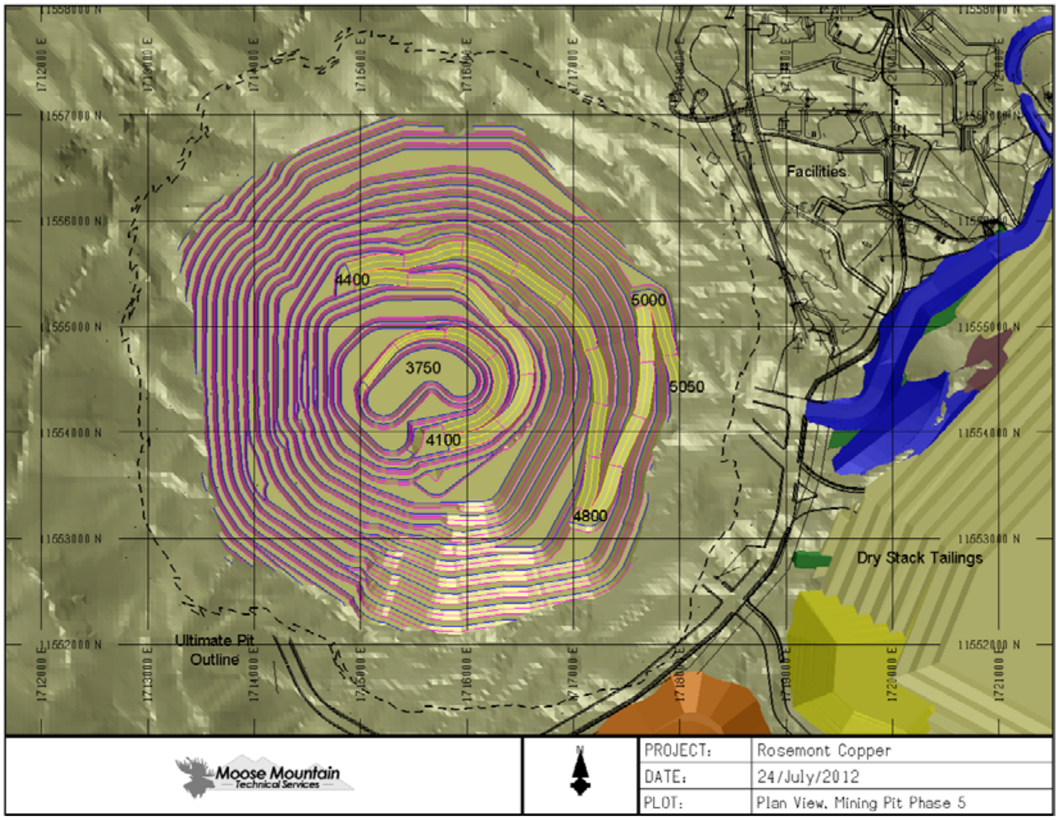

Figure 15-11: Plan View of Mining Pit Phase 5

Phase 6 is fit approximately to the Lerchs-Grossman pit shell defined by a $1.46/lb Cu price (the 58% of base case metal price value sensitivity). Mining Phase 6 expands the pit by various distances in all directions. This pushback put the pit at the ultimate design limits along the north, west and south sides. The eastern most limits of this pushback lie less than 1,000 feet west of the primary crusher. The top of Phase 6 is 6,000 feet, near the top of the west ridge. The waste material at the upper elevations will be dozed down to a bench at approximately 5,600 feet elevation that will be wide enough for loading and hauling. A haul road will be constructed on the north side of the pit to access the 5,600 bench in this phase. The bottom bench of Phase 6 is 3,350 feet. The phase is 5,500 feet wide east-west and 6,400 feet north-south. Phase 6 will produce over 241 million tons of sulfide ore at a stripping ratio of 1.7:1. The ramp on the east side of the pit is re-developed for this phase. An illustration of the Phase 6 pit is shown in Figure 15-12.

The pit entrance is at the 5,050 foot elevation, and a ramp from that location enters the pit in a clockwise direction. The ramp switches back at the 4,750 and 4,250 foot elevations before reversing to a clockwise direction to the bottom of the pit.

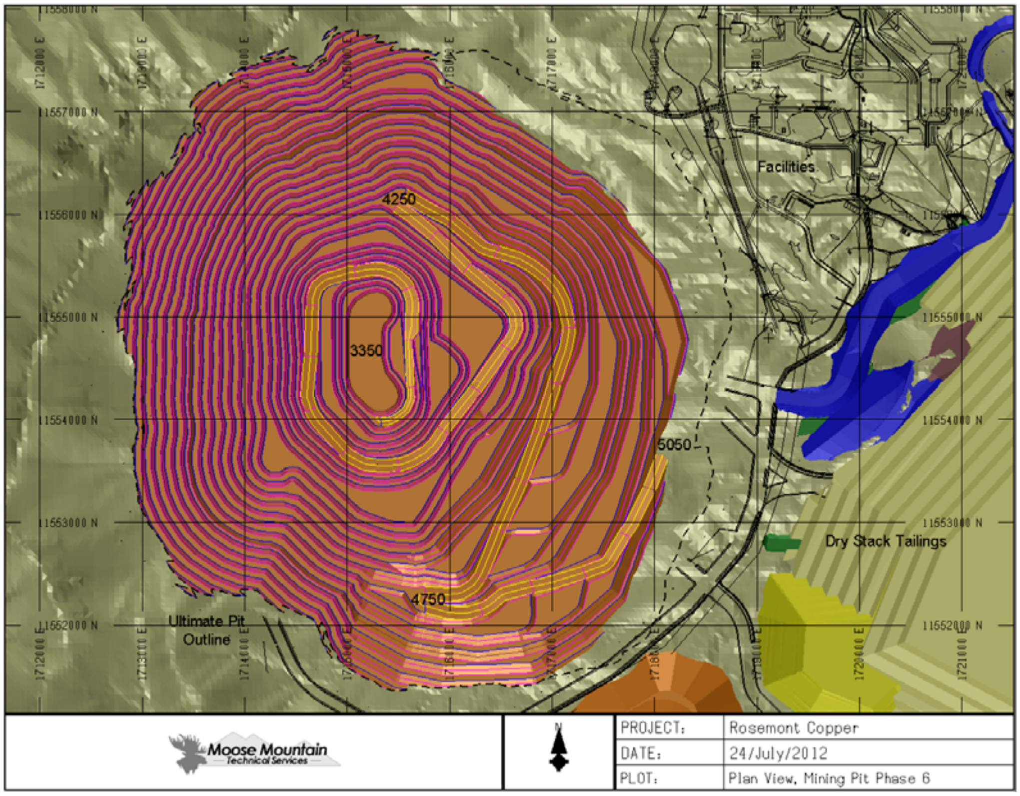

Figure 15-12: Plan View of Mining Pit Phase 6

The final pushback, Phase 7, extends the open pit from 300 to 600 feet along the east side to its ultimate limits and down to its maximum depth at the 2,900 foot elevation. The ultimate pit will be about 6,000 feet wide east-west and 6,500 feet wide north-south. The west wall will be over 3,000 feet high, while the east wall height will reach over 2,200 feet. Phase 7 is fit approximately to the Lerchs-Grossman pit shell defined by a $1.88/lb Cu price (the 75% of base case metal price value sensitivity). Phase 7 will generate nearly 172 million tons of sulfide ore at a stripping ratio of 1.7:1. An illustration of the Phase 7 pit, or final pit, is shown in Figure 15-13.

Total sulfide ore reserves in the final pit are estimated to be 667 million tons and 1.9 billion tons of waste material. Approximately 65 million tons of mineralized oxide material, indicated to be economic, are contained in this pit and included with the waste material instead of ore for the purpose of this study. Facilities to process this mineralized material is currently assessed and a portion of it may be included as ore in the future.

The ultimate pit is currently under-optimized due to the capacity limitations of the tailings storage facility. Potential expansion of this facility in the future will allow the pit to be designed to its optimum at the metal prices for $2.50 /lb Cu, $15.00 /lb Mo and $20.00 /oz Ag.

Throughout the mining phase and final pit designs, internal ramps are generally placed in arkose and alluvium pit walls in order to break up the interramp wall heights and limit the overall slope angles to the geotechnical recommendations. This allows the remaining walls to be steeper on an overall basis as fewer ramps were required in these design sectors. An additional benefit is to keep the main internal haulage ramps away from the high, western wall, which has the steepest inter-ramp slope angles. This allows additional sulfide ore to be developed at the base of the west wall.

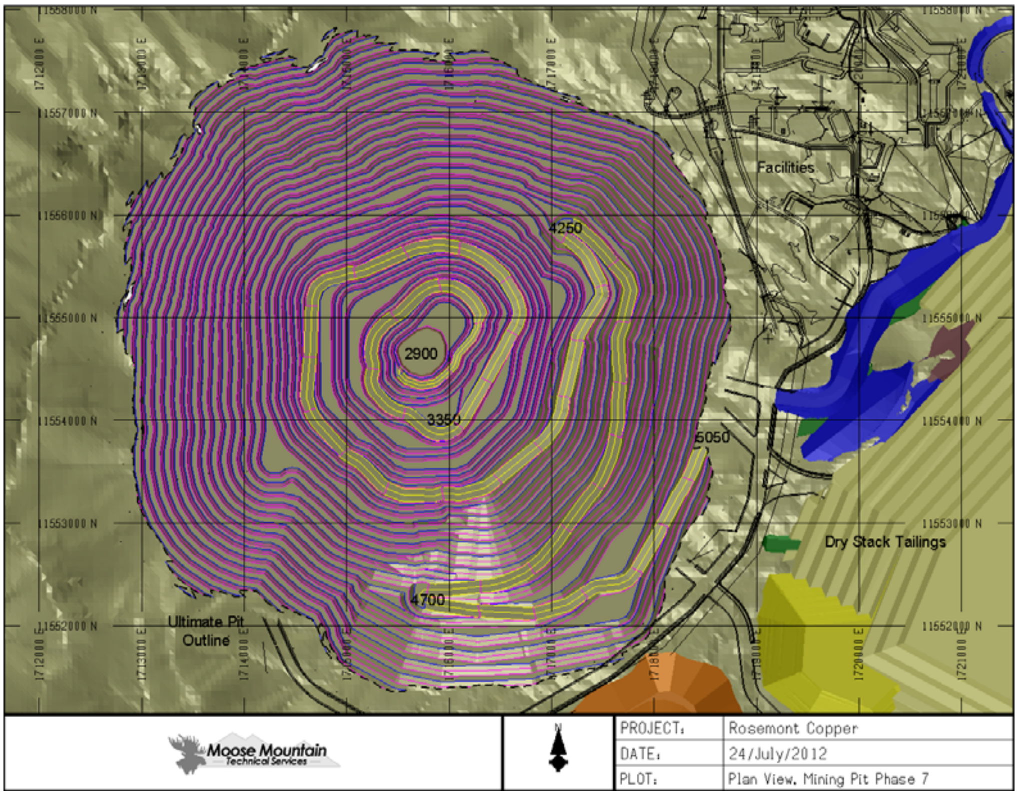

Figure 15-13: Plan View of Mining Pit Phase 7 (Ultimate Pit)

15.5 MINERAL RESERVES

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

The mining phase and ultimate pit designs were applied to the 3D block model of the deposit described in Section 14 to estimate contained tonnages and grades. All reserve estimates are reported in Imperial units.

15.5.1 Ore Definition Parameters

The base-case price and operating cost estimates presented in Table 15-3 are used as the economic envelope to define ore in the mineral reserve estimates. These parameters are restated in Table 15-8 below. All prices and costs are in US dollars.

Mineralized oxide materials that are indicated to be economic in the optimized pit analysis are not included in the pit ore reserves for this study. Designs for leaching facilities and recovery plans for the oxide materials are being undertaken, and they may be included as ore in future studies. All oxide materials are currently included with the waste materials.

Table 15-8: Ore Definition Parameters

Metal Prices:	
Copper (Cu)	$ 2.50 / lb Cu
Molybdenum (Mo)	$ 15.00 / lb Mo
Silver (Ag)	$ 20.00 / troy oz
Operating Costs (excl. oxide leaching):	
Base ore mining	$ 0.777 / ton
Base waste mining	$ 0.882 / ton
Incremental haulage (below pit rim at 5050 ft elevation)	$ 0.028 / ton / bench
Sulfide ore milling & flotation	$ 4.20 / ton ore
General/administration	$ 0.70 / ton ore
Copper Concentrate Processing:	
Cu grade in concentrate	30 %
Cu realization	96.5 %
Cu concentrate transportation	$ 75.00 / dry ton
Cu concentrate treatment	$ 55.00 / dry ton
Cu refining	$ 0.055 / lb Cu
Ag realization	90.0 %
Ag refining	$ 0.40 / troy oz Ag
Molybdenum Concentrate Processing:	
Mo grade in concentrate	50 %
Mo realization	90.0 %
Mo concentrate transportation	$ 0.00 / dry ton
Mo treatment & refining	$ 0.00 / lb Mo
NSR royalty	3 %

15.5.2 Material Densities

Bulk material densities, which vary by rock type, were read from values stored in the block model. These assignments are described in more detail in Section 14. Generally, rock tonnage factors range between 11.18 ft^3/ton to 13.72 ft^3/ton and average about 11.85 ft^3/ton for the rock contained within the ultimate pit.

15.5.3 Dilution

The Rosemont Deposit is a well-disseminated polymetallic deposit that has large ore zones above the anticipated internal cutoff grade. With the planned bulk mining method, external ore dilution along the ore - waste contact edges is generally assessed to determine whether the feed grade from the run of mine production is adequately represented by those predicted from the resource block model.

The sample compositing and block grade interpolation process used to construct the deposit block model was determined to have incorporated sufficient dilution, and hence, no additional internal or external dilution factors are applied. This is previously explained in Section 14.4.2 that provides an analysis comparing the Cu grades from ordinary kriging to nearest neighbor composites.

The resource model block dimensions are 50 feet X 50 feet X 50 feet. The interpolated metal grade is averaged for the entire block. When this mine commences operations, ore feed will be delineated by implementing a blasthole sampling program. Blasthole spacings will be smaller, 25 feet to 35 feet, than the resource block dimensions, thereby provide better definition than from the resource block model. Therefore, there will likely be some selectivity within the dimensions of each resource model block allowing for separation of the ore from waste that is not evident on a whole block basis. The actual feed grade may be higher than model block value as a result. If an ore-waste contact exists within a model block, dilution has already been applied by averaging the metal grades on a whole block basis.

15.5.4 Mineral Reserve Estimates

The mineral reserve estimates presented in this report were prepared by Mr. Robert Fong, P.Eng., Principal Mining Engineer for Moose Mountain Technical Services. Mr. Fong meets the requirements of an independent qualified person under NI 43-101 standards. The mineral reserve estimates are effective as of July 17, 2012.

Proven mineral reserves for the Rosemont Deposit are summarized by mining phase in Table 15-9 and probable mineral reserves are presented in Table 15-10. Table 15-11 lists the combined proven and probable mineral reserve estimates and waste rock for the Rosemont Deposit.

As previous discussed, the pit designed for this study is under-optimized. It reflects an optimum pit at metal price of $1.88 /lb, Cu $11.07 /lb Mo, and $14.87 /oz Ag. Proven and probable sulfide mineral reserves within the designed final pit total nearly 667 million tons grading 0.44% Cu, 0.015% Mo and 0.12 oz Ag/ton. There are 1.24 billion tons of waste materials, resulting in a stripping ratio of 1.9:1 (tons waste per ton of ore). Total material in the pit is 1.9 billion tons. Contained metal in the sulfide (proven and probable) mineral reserves is estimated at 5.88 billion pounds of copper, 194 million pounds of molybdenum and 80 million ounces of silver. No mineralized oxide materials are in the ore reserves, they are included with the waste materials.

Nearly 46% of the sulfide mineral reserves in the Rosemont ultimate pit are classified as proven and the remainder (54%) is considered probable. The classifications are based on the exploration drilling in the Rosemont Deposit. *All of the mineral reserve estimates reported above are contained in the mineral resource estimates presented in Section 14.*

For possible metallurgical considerations, the combined proven and probable mineral reserves are broken out by principal rock types and/or geologic formations in Table 15-12.

The Rosemont ultimate pit contains approximately 24 million tons of inferred sulfide mineral resources that are above the $4.90/ton NSR cutoff value for sulfides. These resources are included in the waste estimates presented in Table 15-11and Table 15-12. *Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Inferred mineral resources have a great amount of uncertainty as to their existence and as to whether they can be mined economically. It cannot be assumed that all or any part of inferred mineral resources will ever be upgraded.*

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

It should be noted that there is some local environmental and political opposition to the development of the Rosemont open pit copper mining project. However, the right to mine and extract the minerals is provided for under federal law and Rosemont has valid mines claims. Rosemont will need to acquire all applicable permits from primarily federal and state agencies. These permits fully address all environmental media and provide for public and stake holder input. (See Section 20)

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

Detailed listings of combined proven and probable mineral reserves by bench, by phase are presented in Appendix E.

Table 15-9: Proven Mineral Reserves by Phase

Phase	Sulfides >= 4.90 $/ton NSR Cutoff				
	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t
1	36,335	22.31	0.50	0.016	0.14
2	16,334	17.27	0.40	0.011	0.09
3	23,075	19.99	0.43	0.018	0.14
4	22,947	22.12	0.50	0.013	0.15
5	46,471	22.18	0.51	0.013	0.13
6	107,605	19.32	0.45	0.014	0.13
7	55,308	19.54	0.43	0.016	0.11
Total	308,075	20.29	0.46	0.015	0.12

(NSR values are based on metal prices of $2.50/lb Cu, $15.00/lb Mo and $20.00/oz Ag.)

Table 15-10: Probable Mineral Reserves by Phase

Phase	Sulfides >= 4.90 $/ton NSR Cutoff				
	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t
1	25,211	22.49	0.50	0.016	0.14
2	10,835	17.20	0.40	0.011	0.09
3	19,343	18.64	0.37	0.023	0.12
4	19,752	20.88	0.48	0.013	0.13
5	33,374	20.90	0.48	0.012	0.12
6	133,872	16.92	0.40	0.013	0.11
7	116,744	18.98	0.42	0.015	0.12
Total	359,131	18.67	0.42	0.014	0.12

(NSR values are based on metal prices of $2.50/lb Cu, $15.00/lb Mo and $20.00/oz Ag.)

Table 15-11: Combined Proven and Probable Mineral Reserves by Phase

Phase	Sulfides >= 4.90 $/ton NSR Cutoff					Waste	Total Material	Strip
	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	Ktons	Ratio
1	61,546	22.38	0.50	0.016	0.14	142,729	204,275	2.32
2	27,169	17.24	0.40	0.011	0.09	84,526	111,695	3.11
3	42,418	19.37	0.40	0.020	0.13	59,553	101,971	1.40
4	42,699	21.54	0.49	0.013	0.14	100,709	143,408	2.36
5	79,845	21.64	0.50	0.013	0.13	156,603	236,448	1.96
6	241,477	17.99	0.42	0.014	0.12	411,973	653,450	1.71
7	172,052	19.16	0.42	0.015	0.11	287,362	459,414	1.67
Total	667,206	19.42	0.44	0.015	0.12	1,243,455	1,910,661	1.86

(NSR values are based on metal prices of $2.50/lb Cu, $15.00/lb Mo and $20.00/oz Ag.)

Table 15-12: Combined Proven and Probable Mineral Reserves by Rock Formation

Rock Type /	Sulfides >= 4.90 $/ton NSR Cutoff					Waste	Total	Strip
Formation	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	Ktons	Ratio
Oxide - QMP / QLP						24,280	24,280	
Oxide - Andesite						95,887	95,887	
Oxide - Arkose						521,426	521,426	
Oxide - Other						0	0	
Sulfide - QMP / QLP	17,612	14.96	0.31	0.018	0.06	0	17,612	0.0
Sulfide - Abrigo	16,554	13.13	0.31	0.005	0.10	100,512	117,066	6.1
Sulfide - Concha	2,159	8.71	0.17	0.006	0.13	31,204	33,363	14.5
Sulfide - Epitaph	72,730	19.42	0.44	0.013	0.10	29,610	102,340	0.4
Sulfide - Colina	98,542	25.77	0.59	0.020	0.12	12,194	110,736	0.1
Sulfide - Earp	111,704	13.52	0.29	0.014	0.07	22,629	134,333	0.2
Sulfide - Horquilla	281,334	21.41	0.49	0.016	0.15	68,251	349,585	0.2
Sulfide - Escabrosa	22,325	25.07	0.60	0.007	0.17	21,767	44,092	1.0
Sulfide - Other	44,246	9.29	0.23	0.003	0.05	152,095	196,341	3.4
Overburden						7,053	7,053	
Tertiary Gravels						156,547	156,547	
Total	**667,206**	**19.42**	**0.44**	**0.015**	**0.12**	**1,243,455**	**1,910,661**	**1.9**

16 MINING METHODS

Mining sequence plans are developed to depict mining progress at regular intervals and to serve as the basis for a mine production schedule. The sequence plans are developed from the phase designs described in Section 15.4 and target a sulfide (mill) ore production base rate of 75,000 tpd. This rate is reduced in Year 1 for mill ramp-up, and increased later on for expansions and increased plant operating availabilities.

16.1 PRODUCTION SCHEDULING CRITERIA

The operating and scheduling criteria used to develop the mining sequence plans are summarized in Table 16-1 below.

Table 16-1: Mine Production Schedule Criteria

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

Pit and mine maintenance operations will be scheduled around the clock. Allowances for down time and weather delays have been included in the mine equipment and manpower estimations presented in Sections 16.7 and 16.14.

The mill ramp-up schedule used for Year 1 production targets is presented in Table 16-2. Quarterly mill production in Year 2 will average about 6.8 million tons, equivalent to an annual rate of 27.4 million tons.

Table 16-2: Mill Ramp-Up Schedule (Year 1)

Month	% of Full Production	Monthly Ktons	Quarterly Ktons
1	40	930	3,755
2	53	1,120	
3	73	1,705	
4	87	1,950	6,525
5	100	2,325	
6	100	2,250	
7	100	2,325	6,900
8	100	2,325	
9	100	2,250	
10	100	2,325	6,900
11	100	2,250	
12	100	2,325	
Total Year 1	88	24,080	24,080

The annual mill throughput for the life of mine is shown in Table 16-3. An expansion is planned beginning in Year 5 when the daily throughput will gradually increase from 75,000 tpd to 88,000 tpd by Year 7. In Year 12, an increase in plant operating availability will boost the daily throughput rate to 90,000 tpd.

Table 16-3: Annual Mill Throughput

Year	Average Throughput, tpd
1	65,973
2	75,000
3	75,000
4	75,000
5	78,000
6	84,000
7	88,000
8	88,000
9	88,000
10	88,000
11	88,000
12	90,000
13	90,000
14	90,000
15	90,000
16	90,000
17	90,000
18	90,000
19	90,000
20	90,000
21	90,000

16.2 MILL FEED CUT-OFF GRADE STRATEGY

An elevated cut-off grade strategy is implemented to bring forward the higher grade ore from the pit to the early part of the ore production schedule. Delivering higher grade ore to the mill in the early years will improve the net present value economics of the project.

NSR values are calculated for each block in the resource model to represent the net Cu, Mo, and Ag metal values. The pit reserves are estimated on a cut-off with an NSR value of $4.90 /t. This is the minimum value of mineralized material that will cover the processing and G&A costs, and is therefore reserved for mill feed. Applying an elevated cut-off for mill feed in a given period will result in assigning the ore blocks with NSR values between $4.90 /t and the elevated cut-off NSR value to the ROM stockpile. The stockpiled ore will be reclaimed later in the life of mine. It is necessary for ore blocks destined to the stockpile to have an NSR value high enough to also cover handling costs. The additional cost is estimated to be $0.40 /t, and therefore ore blocks that are destined to the ROM stockpile must have a minimum NSR value of $5.30 /t. Otherwise this material will be sent to the waste storage areas instead.

The elevated cut-off value to determine the head grade to the mill is limited at the upper end. Typically, more ore will be mined as well as additional waste stripping to extract sufficient mill feed as the mill cut-off value is elevated. The required mine equipment capacity to move the additional materials will limit the mill cut-off value. The maximum NSR cut-off value applied in this plan is $12.00 /t.

16.3 OVERBURDEN STRIPPING REQUIREMENTS

Mineral reserve tabulations by bench, by phase and a mine production scheduling program are used to analyze long-term stripping requirements for the Rosemont project. Elevation and phase order dependencies and sinking rate controls are used in conjunction with mill ore production targets and an internal NSR cutoff of $4.90/ton to simulate open pit mining. The program, through successive iterations, allows the user to examine waste stripping rates over the life of the mine and their impact on ore exposure and mill head grades.

The stripping analysis determined a minimum preproduction stripping requirement of approximately 99 million tons of waste. Approximately 6 million tons of sulfide ore will also be mined and stockpiled during this period. The estimated Year 1 waste stripping total is 88 million tons, and 70 million tons for Year 2. The annual waste stripping from Year 3 through Year 12 will average about 76 million tons per year to maintain adequate ore exposure levels for uninterrupted ore supplies to the mill. Waste stripping rates will decline to an annual average of 42 million tons for the next 5 year period, and drop to an average of 3 million tons for the last 5 production years as the final mining phase reaches near the pit bottom.

Preproduction stripping will be conducted over a 21-month time period and will ramp up according to the delivery of mining equipment (particularly electric shovels) and the hiring and training of work crews. The long-term and peak mining rates suggest the use of at least three large (60-cy class) electric shovels, two large (36-cy and 25-cy) front-end loaders and a hydraulic shovel (46-cy). The preproduction stripping ramp-up is based on the delivery of the front-end loaders and hydraulic shovel 21 months prior to mill startup. Delivery of the first operating shovel is anticipated 16 months prior to mill startup, with successive deliveries on three-month intervals until the last shovel is placed into production ten months before startup.

Mining crews would typically be expanded every one to two months to allow time for hiring and training. Crew efficiencies would start off at reduced levels and increase with experience. Table 16-4 summarizes the mine's preproduction stripping capacity ramp-up schedule if all equipment were utilized.

Table 16-4: Mine Production Estimated Stripping Capacity Ramp-Up Schedule

Preproduction Month	Mine Capacity	Quarterly Capacity
	Ktons	Ktons
1	785	
2	1,623	4,765
3	2,356	
4	2,759	
5	3,084	10,247
6	4,404	
7	4,877	
8	5,161	16,588
9	6,550	
10	6,963	
11	7,185	22,943
12	8,794	
13	9,112	
14	10,172	28,980
15	9,697	
16	10,329	
17	10,672	31,286
18	10,286	
19	10,850	
20	10,580	32,424
21	10,994	
Total	147,233	147,233

16.4 WASTE ROCK AND TAILINGS STORAGE

Overburden and other waste rock encountered in the course of mining will be placed into a waste rock storage (WRS) area located to the south and southeast of the planned open pit and into the dry stack tailings area, where dewatered mill tailings will be placed behind waste rock containment buttresses. The dry stack tailings area is north of the WRS area and east-northeast of the pit. The WRS and dry stack tailings facilities are fully contained within the Barrel drainage basin. The general mine site layout is shown in Figure 16-1.

The dry stack tailings facility is divided into two components, Phase 1 to the north and Phase 2 to the south. The dry stack tailings and WRS facilities were designed by AMEC/Tetra Tech, and are described in more detail in Section 18.5 of this report. MMTS provided estimates for waste rock quantities contained in the open pit and generated plans showing its development through the life of mine

16.4.1 Waste Rock Storage Design Criteria

The design criteria for the WRS area and associated haul roads are summarized in Table 16-5 below.

Table 16-5: Waste Rock Storage Design Criteria

Angle of Repose	37°
Swell Factor on dumps Average Tonnage Factor (with swell)	30 % 15.4 ft^3/ton
Swell Factor on roads and buttresses Average Tonnage Factor (with swell)	25 % 14.8 ft^3/ton

Tetra Tech generated the estimates on waste rock quantities that are required for construction of the dry stack tailings buttresses and other structures requiring waste rock from the mine during the preproduction periods. Construction material will be supplied by stripping operations in the pit unless an alternate source is closer by. Any “surplus” waste rock will be directed to the starter buttresses and waste rock piles in the WRS area.

One of the objectives in the early years of operation (specifically, Years 1 to 5) is to construct a series of starter and perimeter buttresses (screen berms) around the eastern and southern perimeters of the WRS, and Phase 1 Dry Stack areas to provide a visual barrier to most of the mine’s operations from views along State Highway 83 and nearby private land owners located southeast of the project. These starter buttresses will also allow regrading and revegetation of the WRS side slopes at much earlier time periods than with traditional mine waste rock stockpile construction. To the extent possible, the long haulage profiles to the starter buttresses are balanced with shorter profiles to internal waste rock stockpiles.

The WRS buttresses and internal stockpiles are designed to facilitate subsequent regrading and reclamation. Side slopes in the WRS area will be regraded to a maximum of 3:1 (horizontal:vertical) slopes. Buttress construction will consist of haul trucks end-dumping waste rock in 100 foot lifts at a 37° angle of repose. Buttress crests will be set back to allow simple dozing of the crests down to meet the target regraded slope angles.

16.5 MINE PLAN

Mining sequence plans are developed on a quarterly basis from preproduction through to the end of Year 2, and on an annual basis through Year 10. The preproduction period consists of seven quarters, or 21 months. Additional plans include mining progress through the end of Year 10, Year 15, Year 19, and Year 21.3 (end of mining). The mine plan drawings for these periods are show in Figure 16-1 to Figure 16-27. The mine production schedule is summarized in Table 16-6. Tables showing the benches mined by pit phase for each period are in Appendix F.

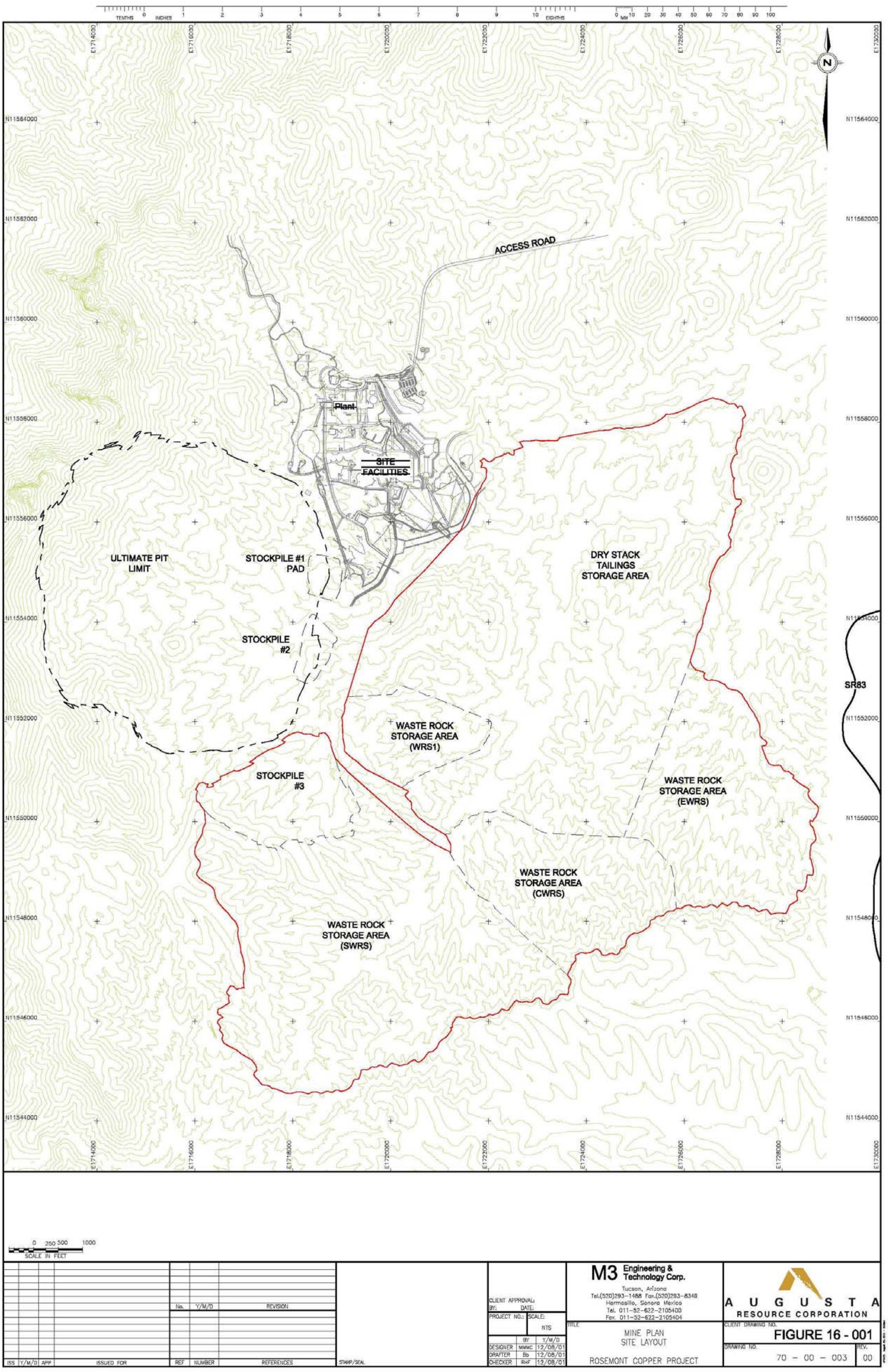

Figure 16-1: Mine Plan Site Layout

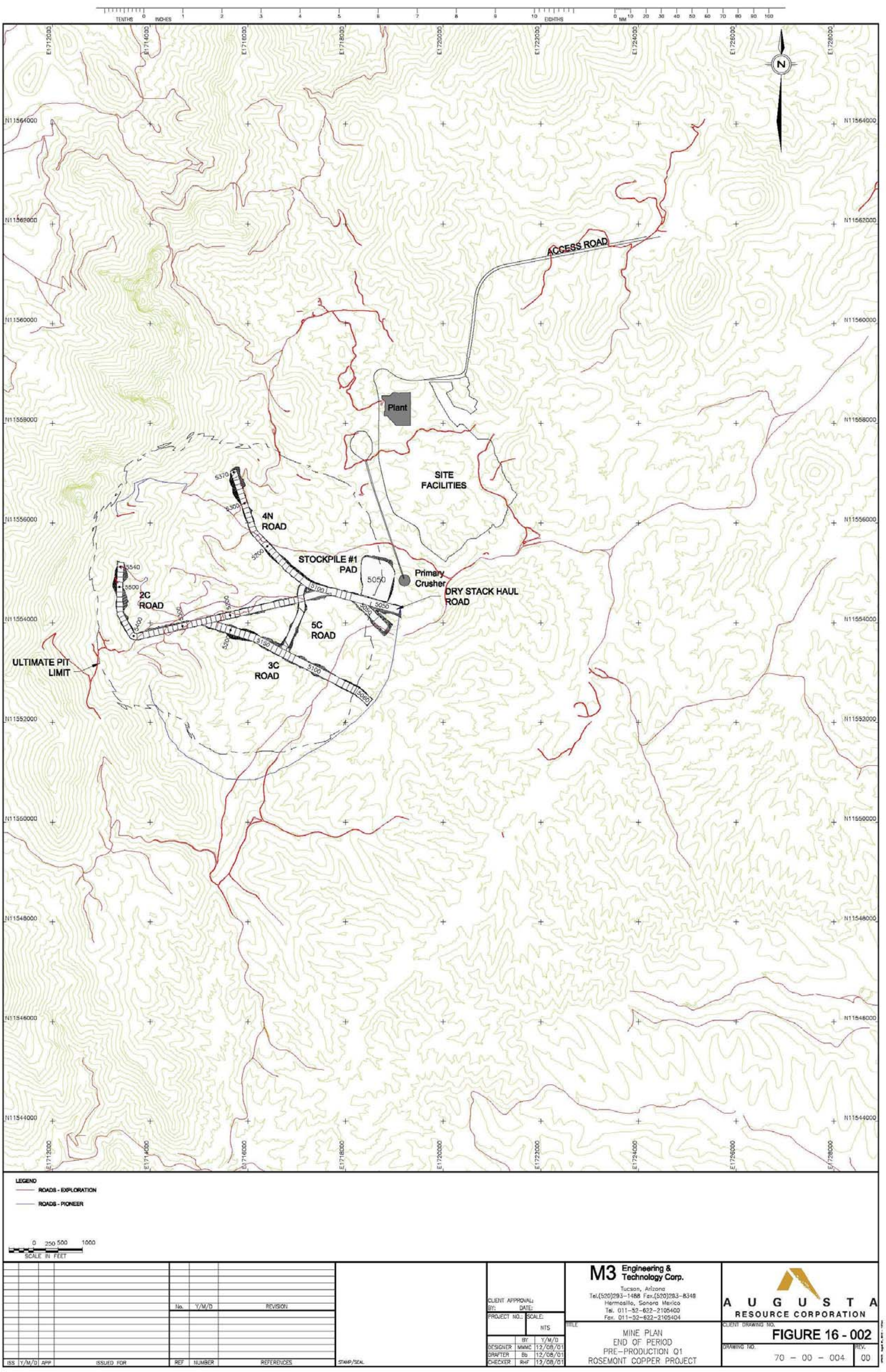

Figure 16-2: Mine Plan end of Period Pre-Production Q1

Figure 16-3: Mine Plan End of Period Pre-Production Q2

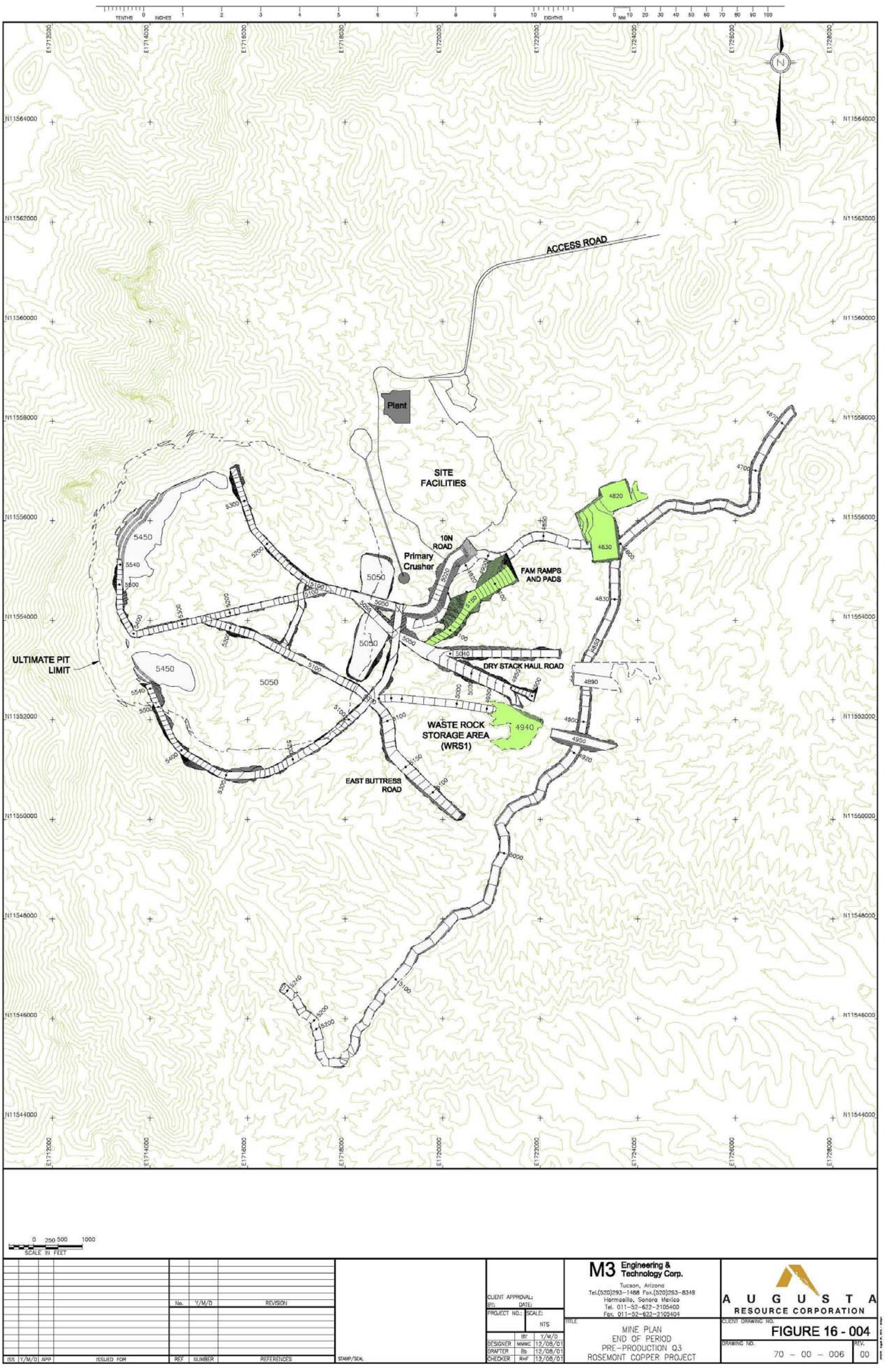

Figure 16-4: Mine Plan End of Period Pre-Production Q3

Figure 16-5: Mine Plan End of Period Pre-Production Q4

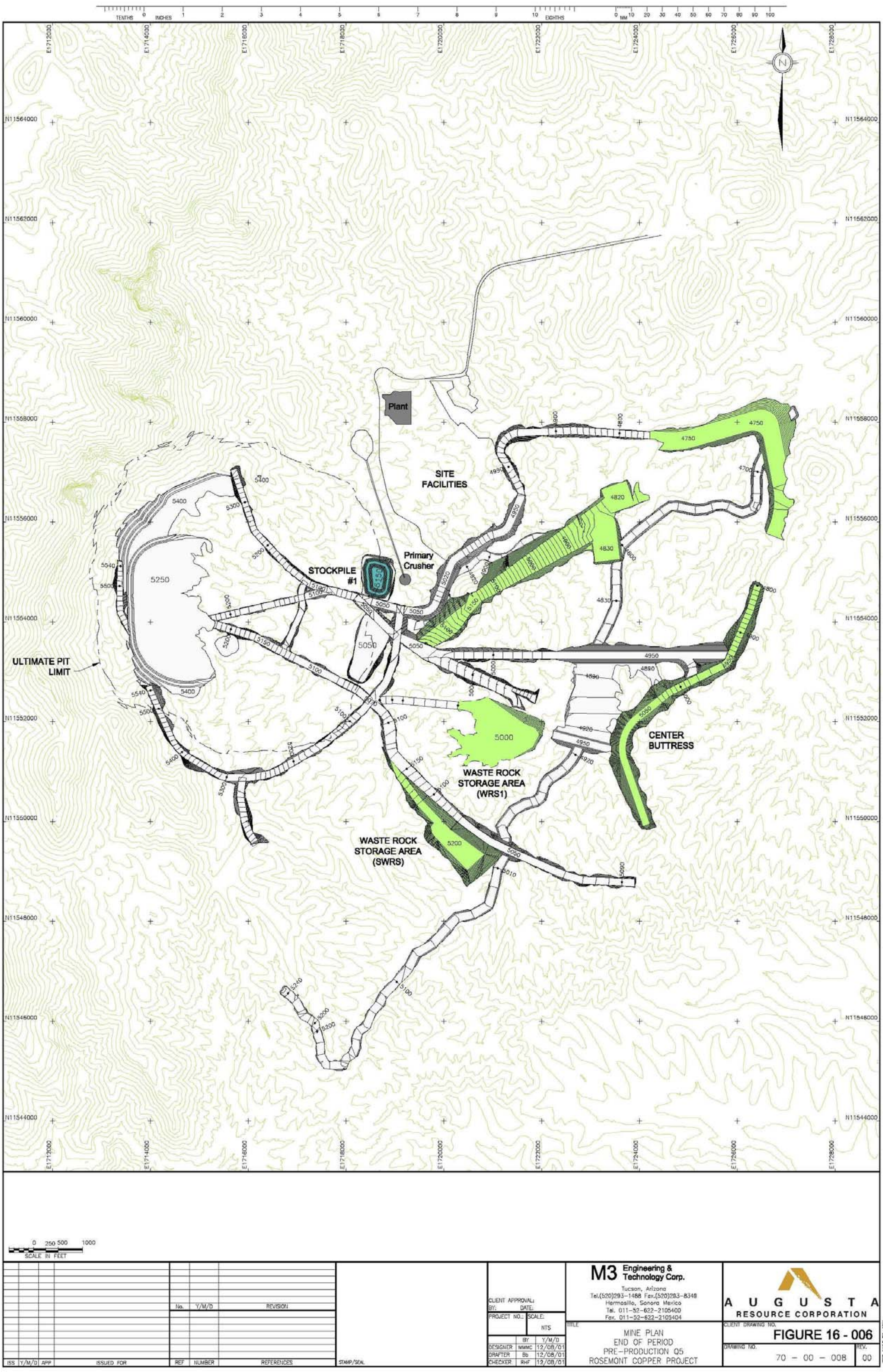

Figure 16-6: Mine Plan End of Period Pre-Production Q5

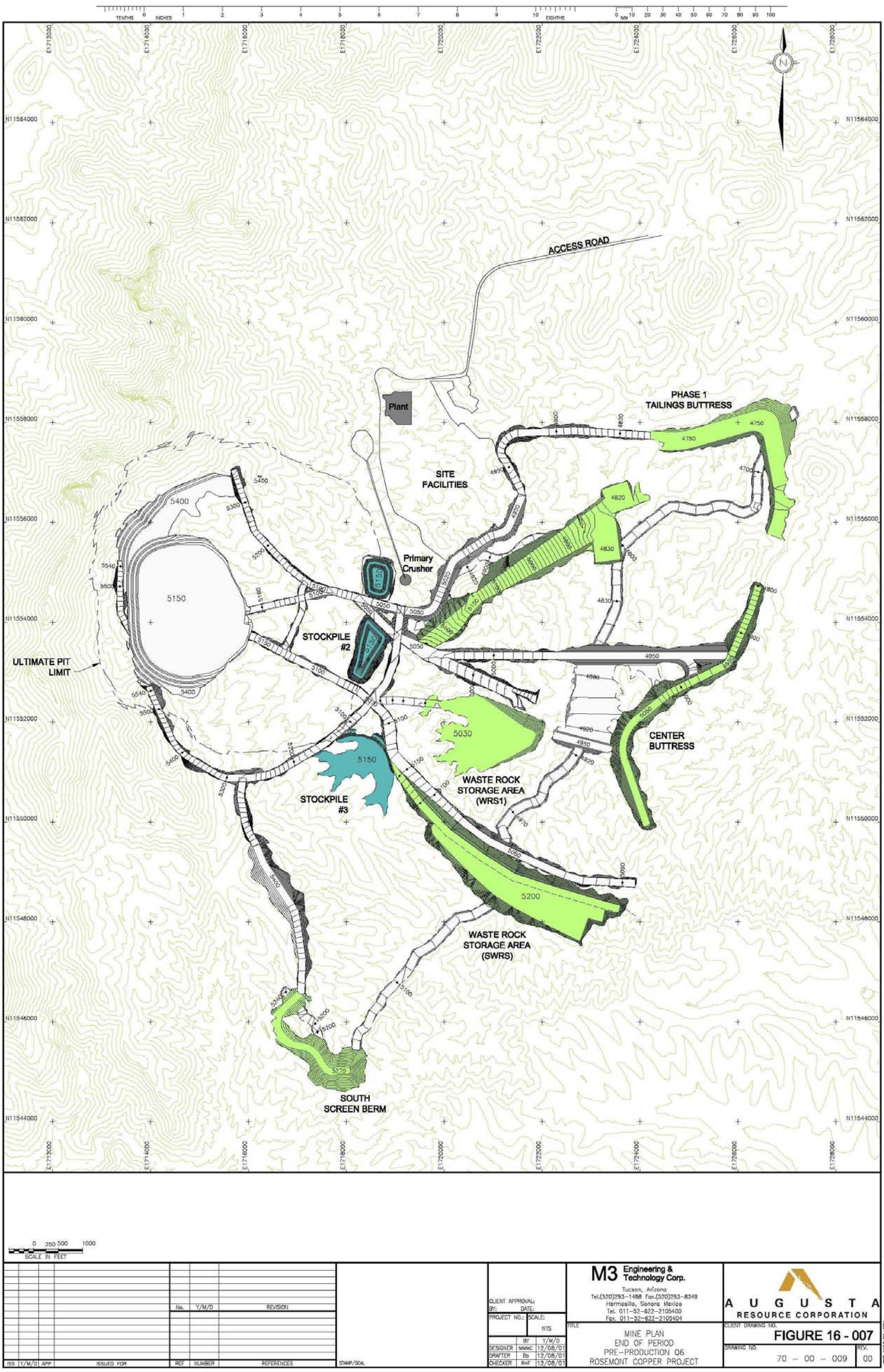

Figure 16-7: Mine Plan End of Period Pre-Production Q6

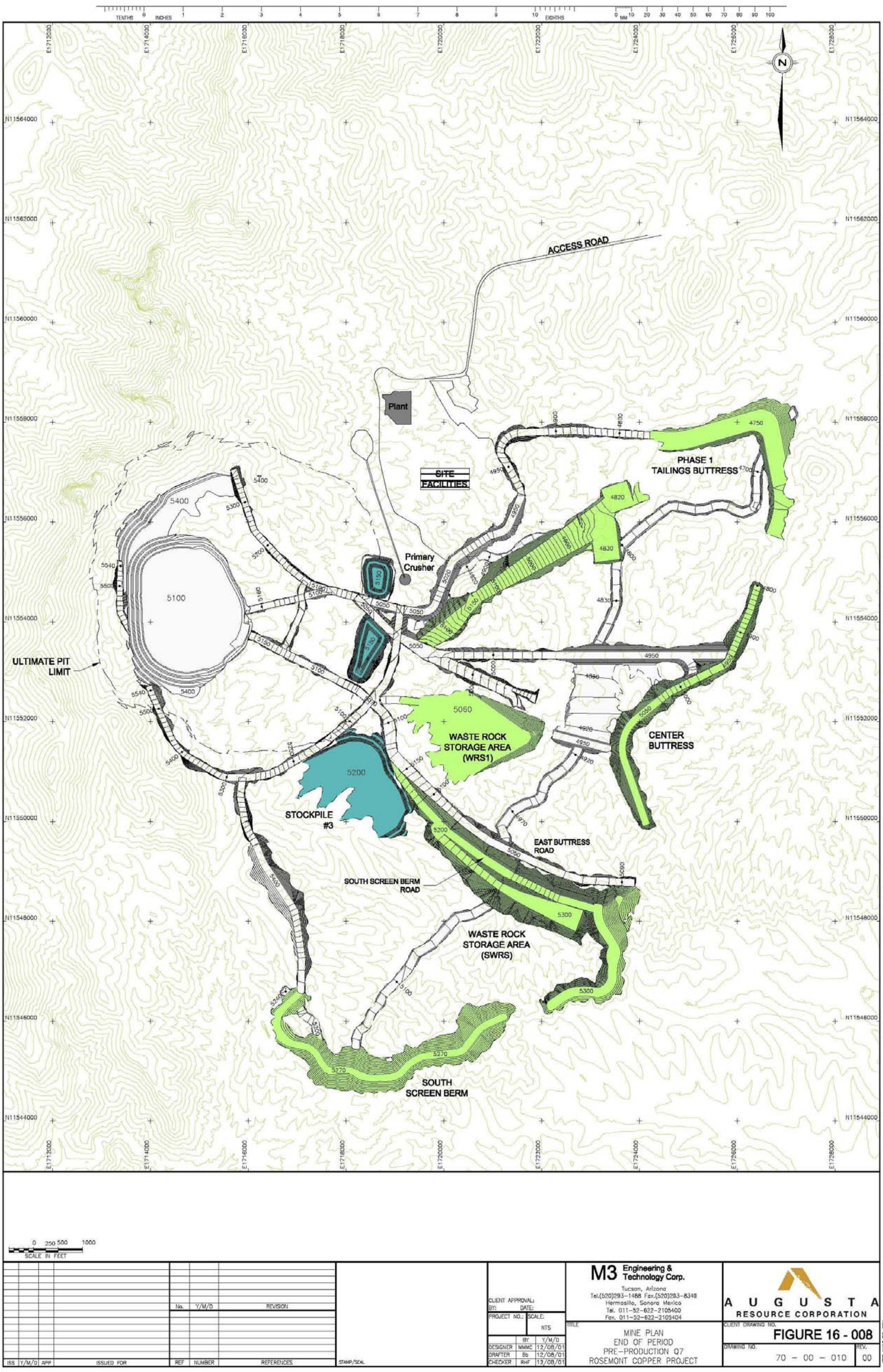

Figure 16-8: Mine Plan End of Period Pre-Production Q7

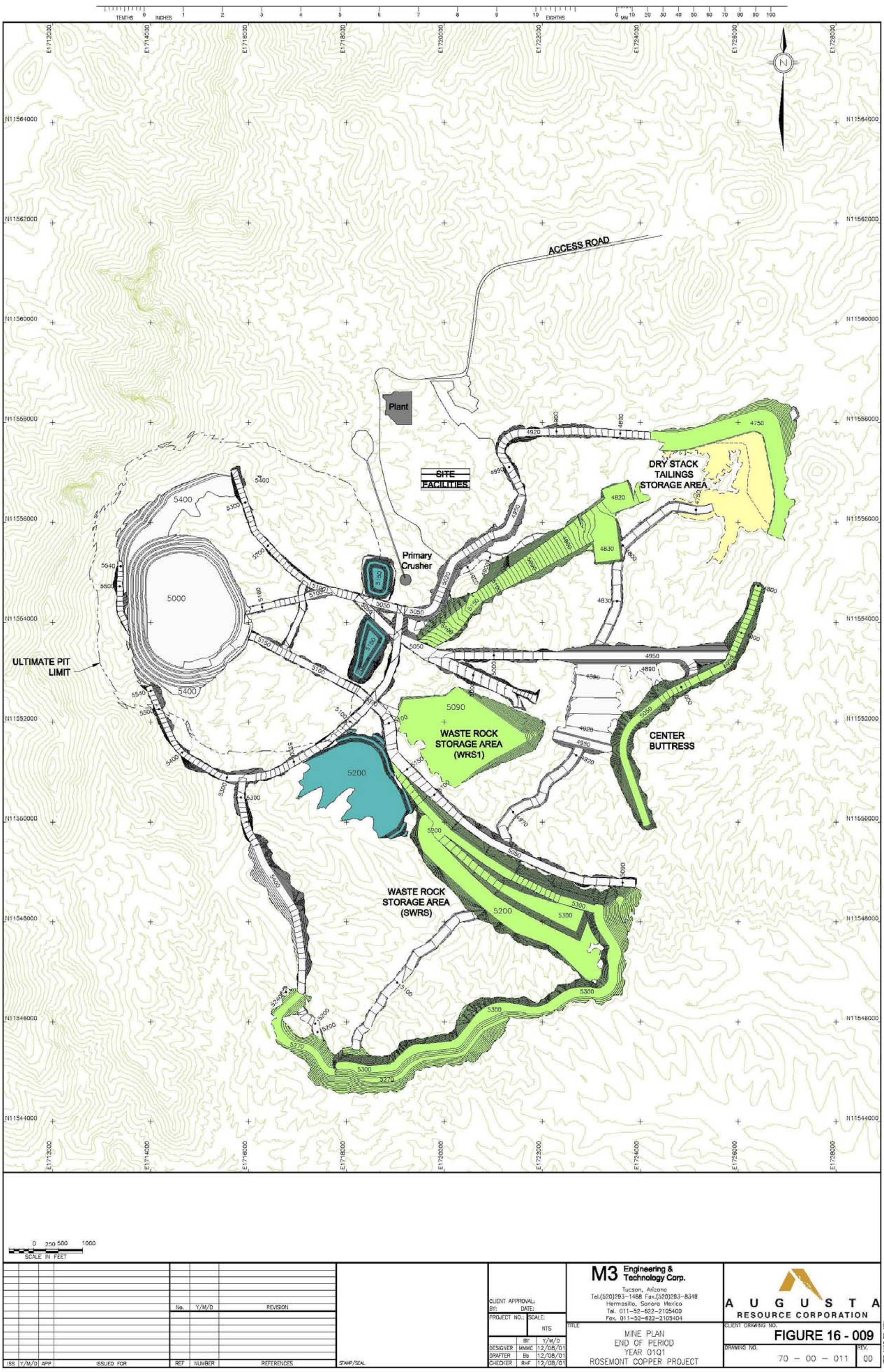

Figure 16-9: Mine Plan End of Period Year 01Q1

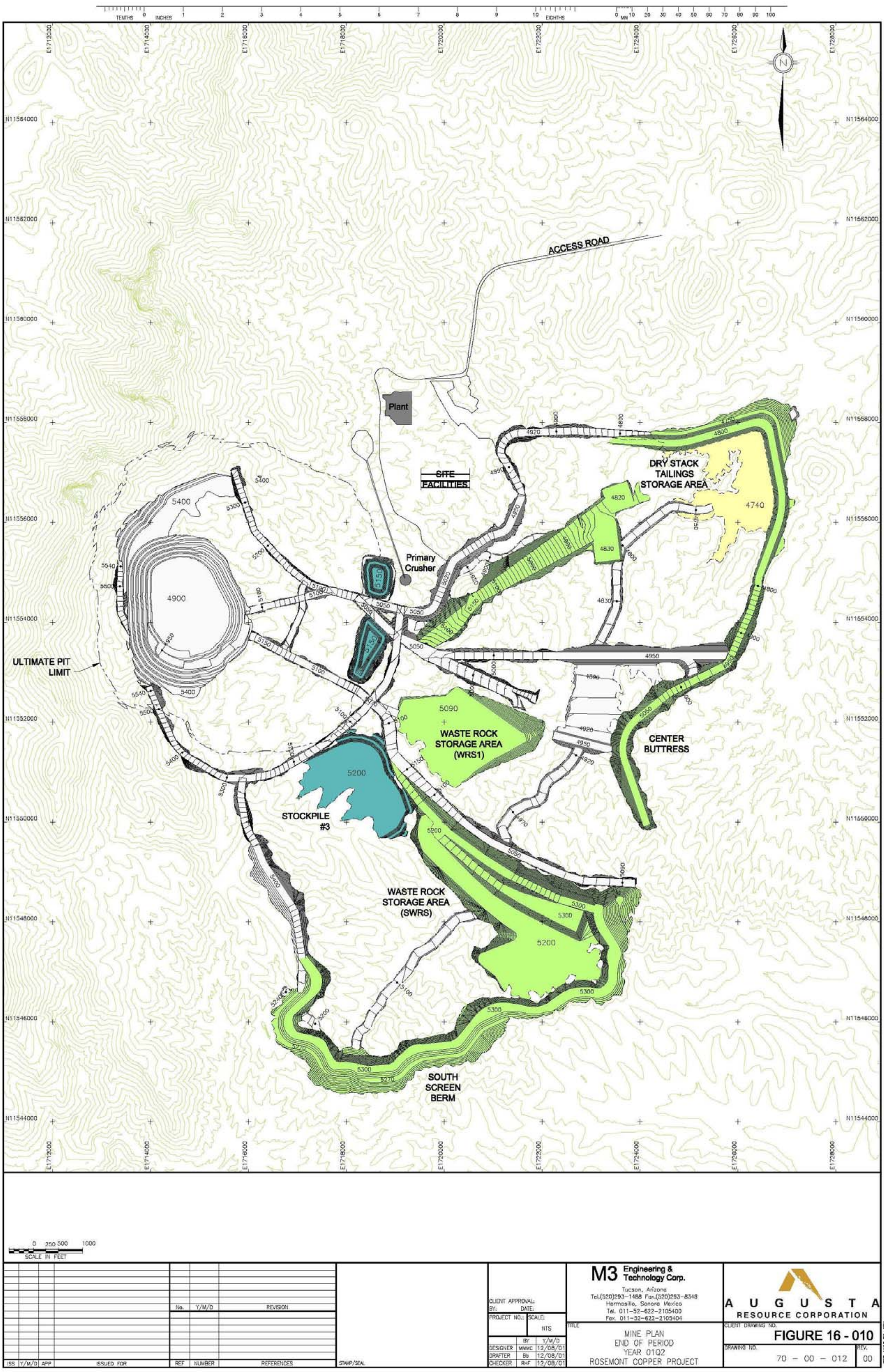

Figure 16-10: Mine Plan End of Period Year 01Q2

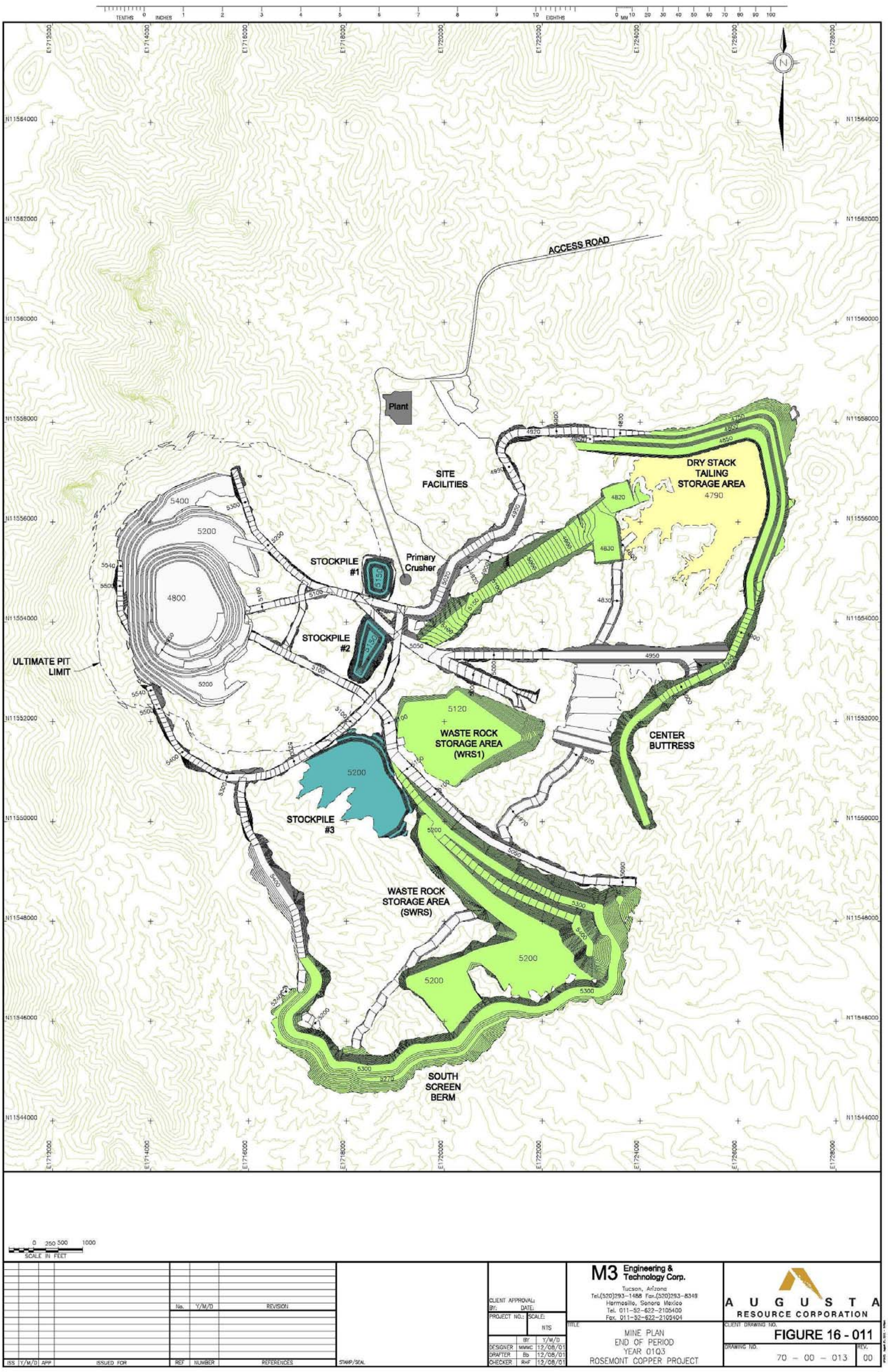

Figure 16-11: Mine Plan End of Period Year 01Q3

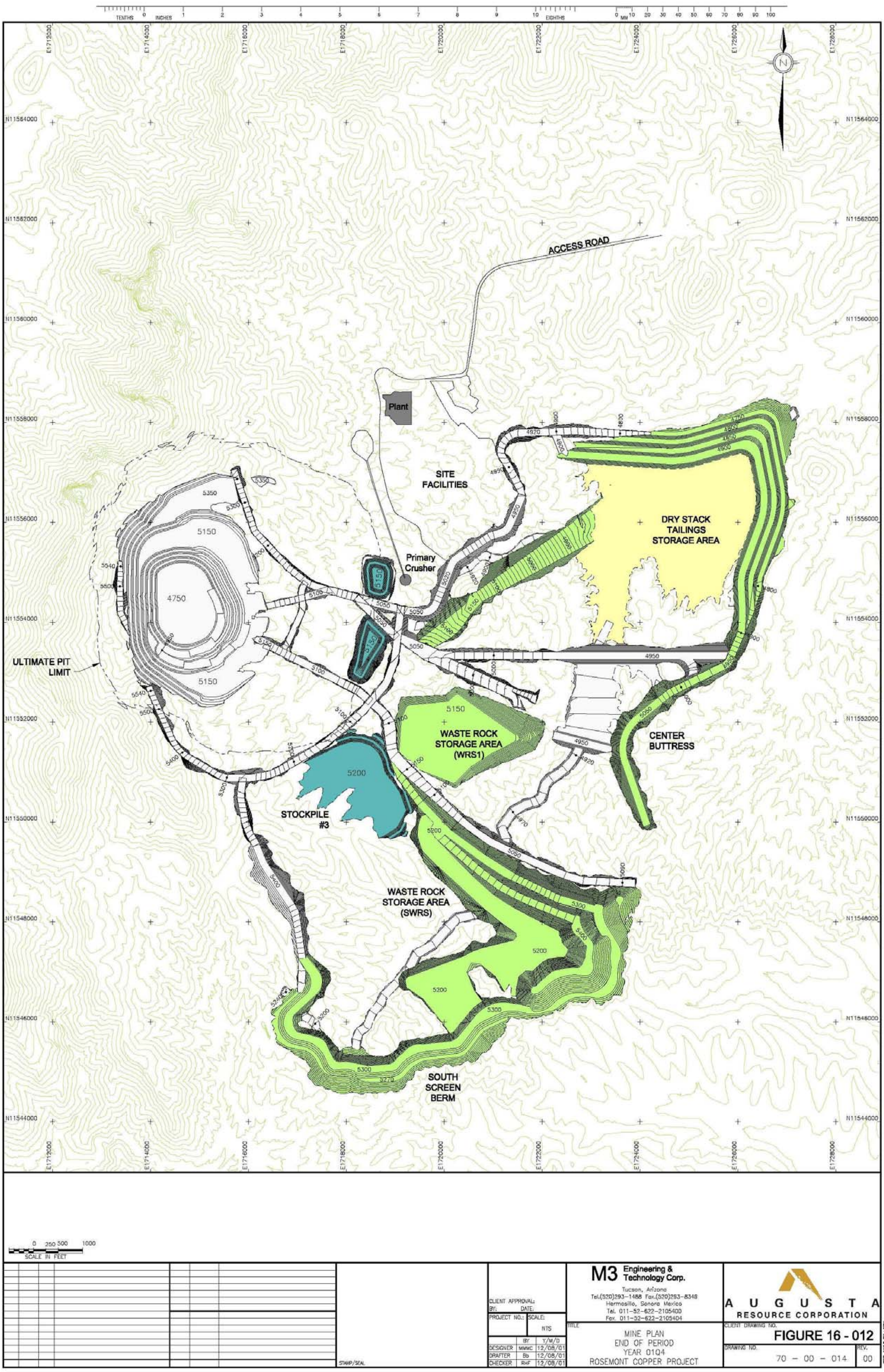

Figure 16-12: Mine Plan End of Period Year 01Q4

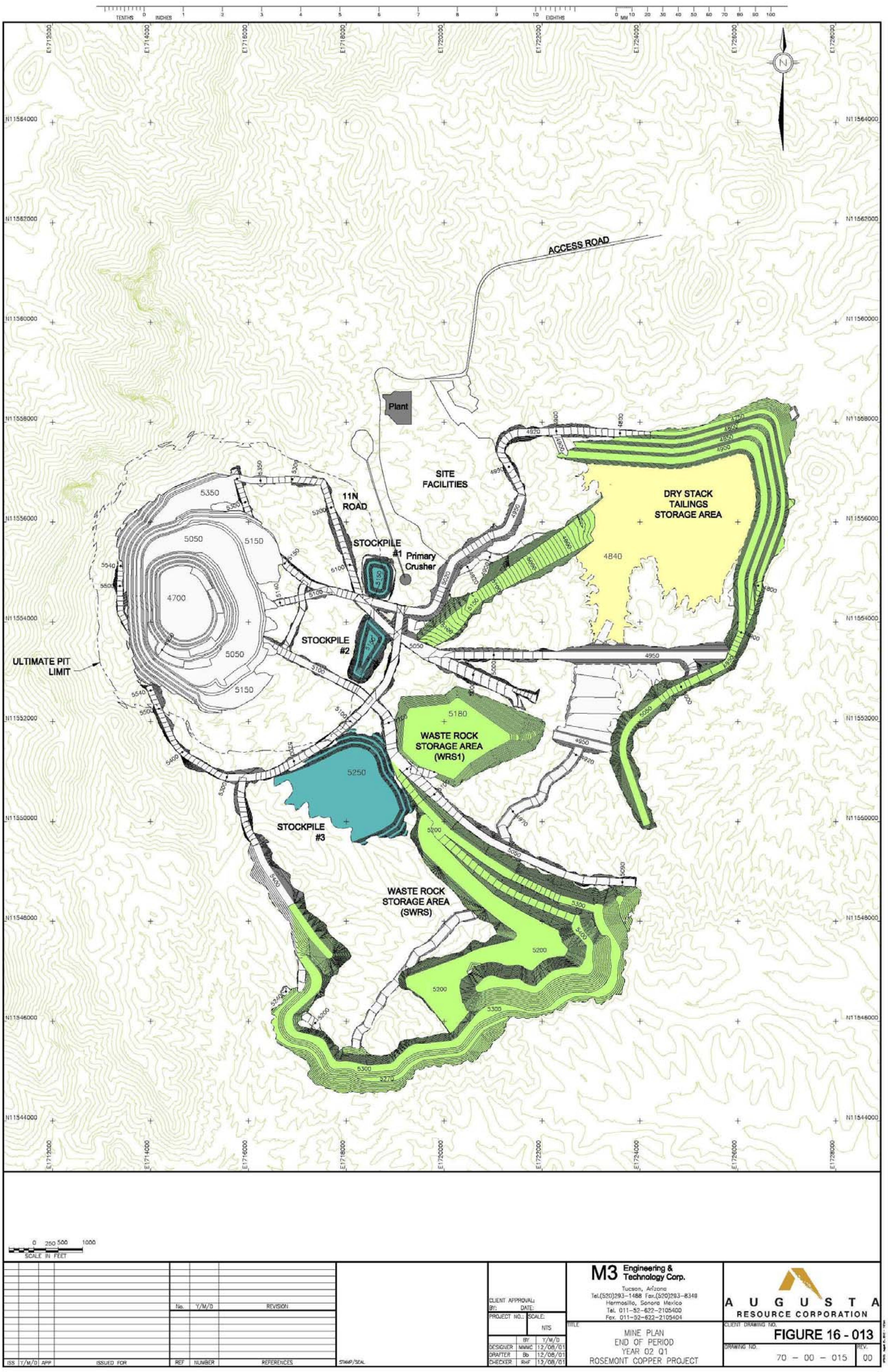

Figure 16-13: Mine Plan End of Period Year 02 Q1

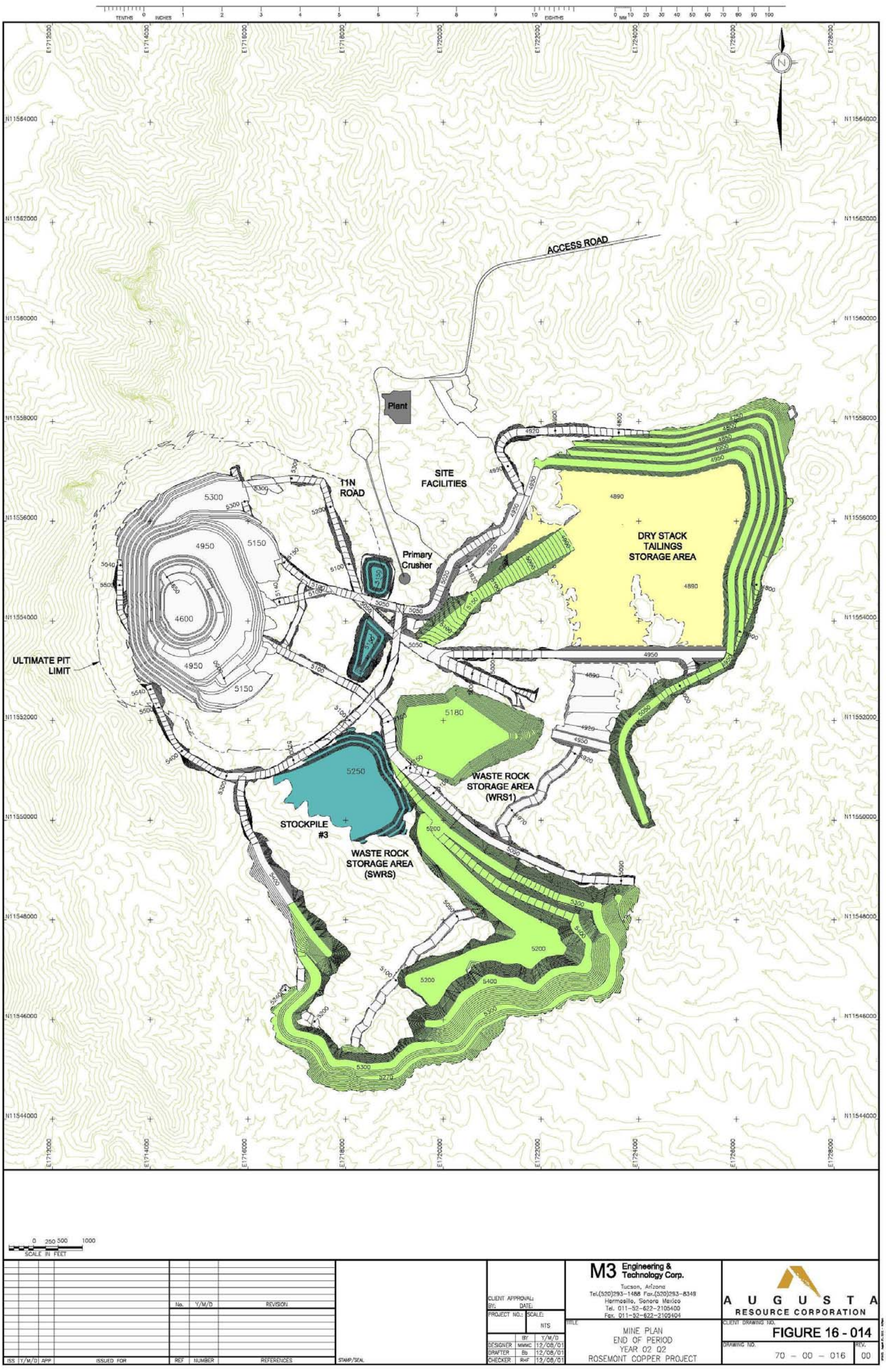

Figure 16-14: Mine Plan End of Period Year 02Q2

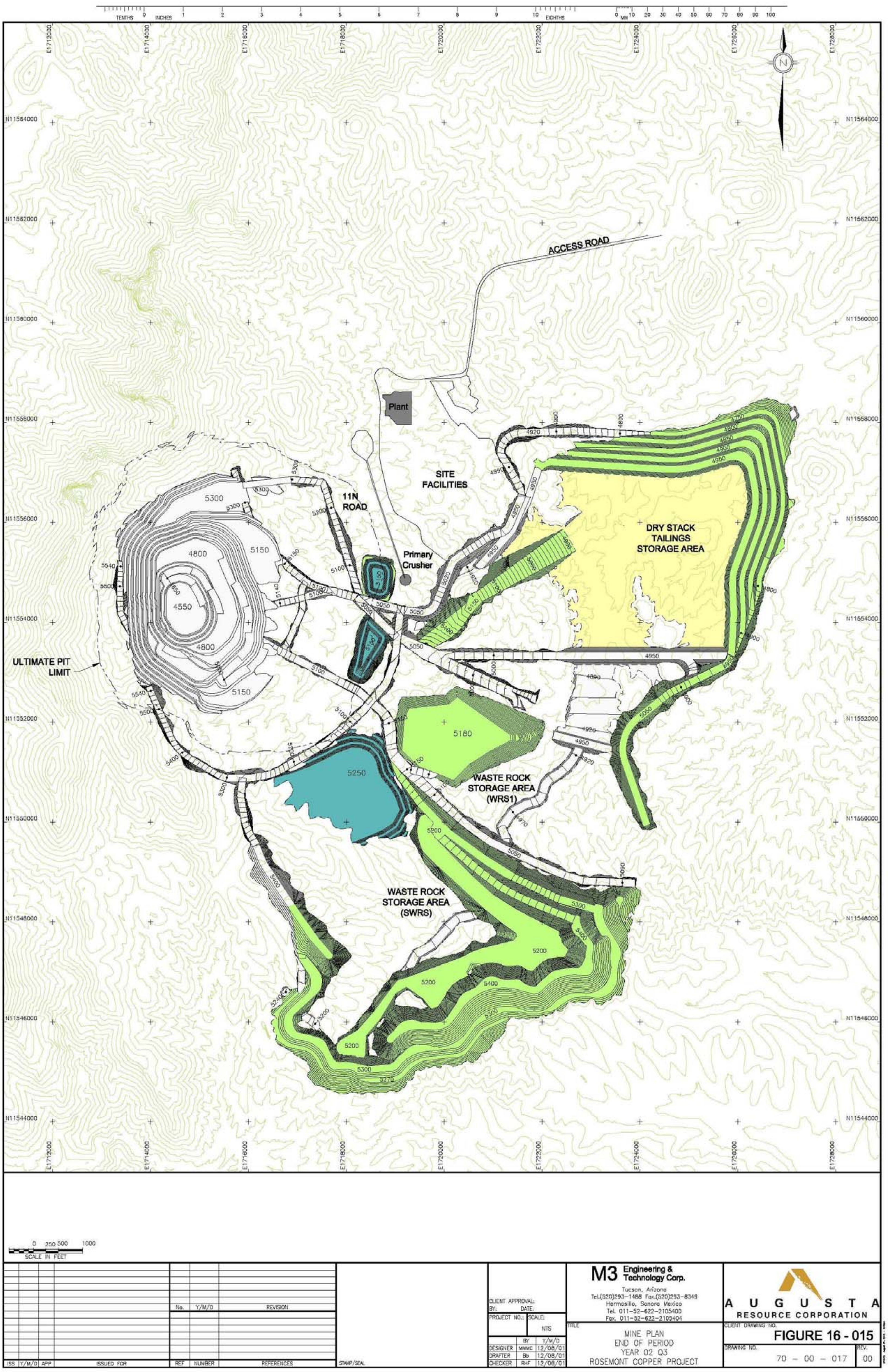

Figure 16-15: Mine Plan End of Period Year 02Q3

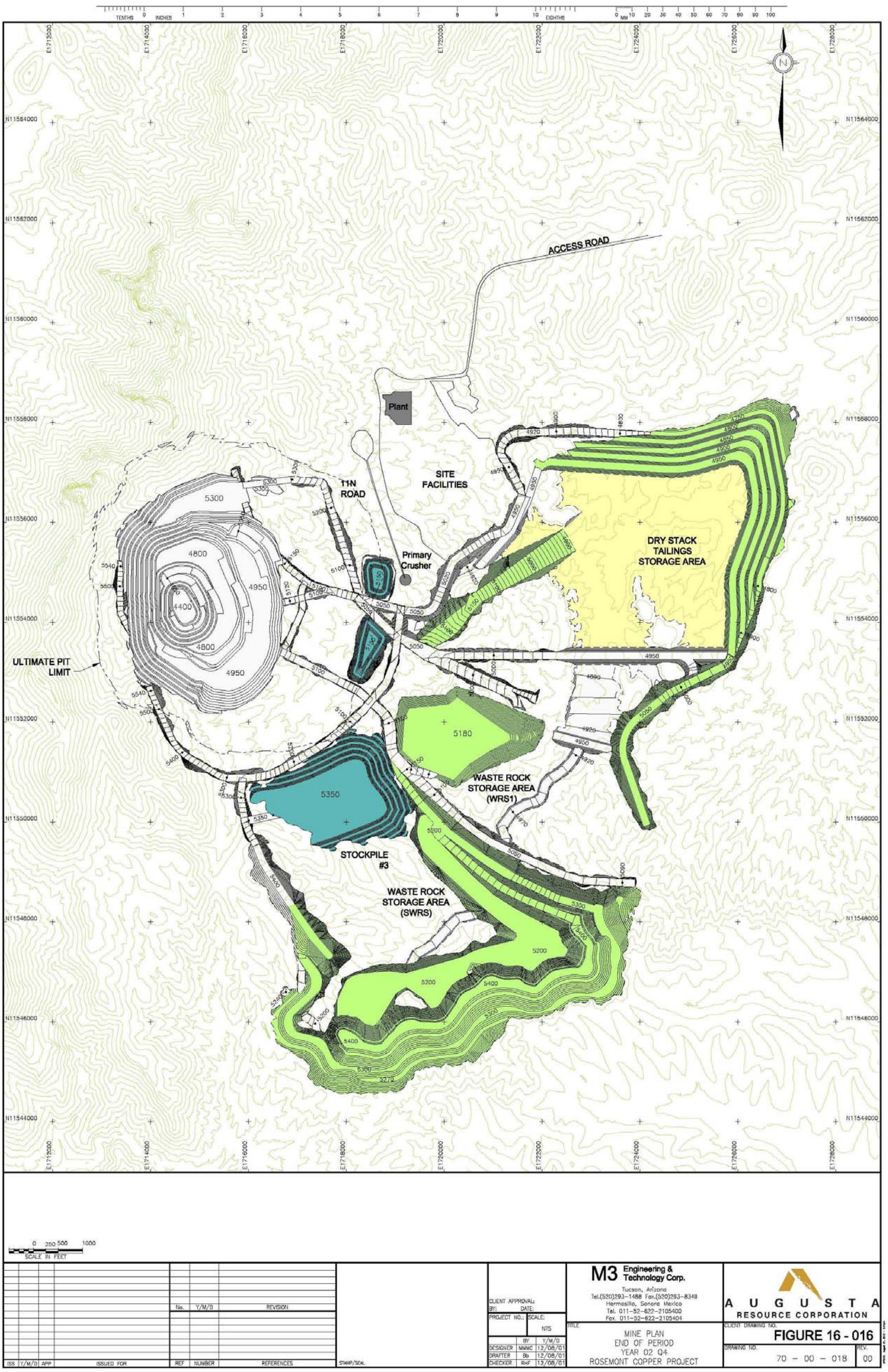

Figure 16-16: Mine Plan End of Period Year 02Q4

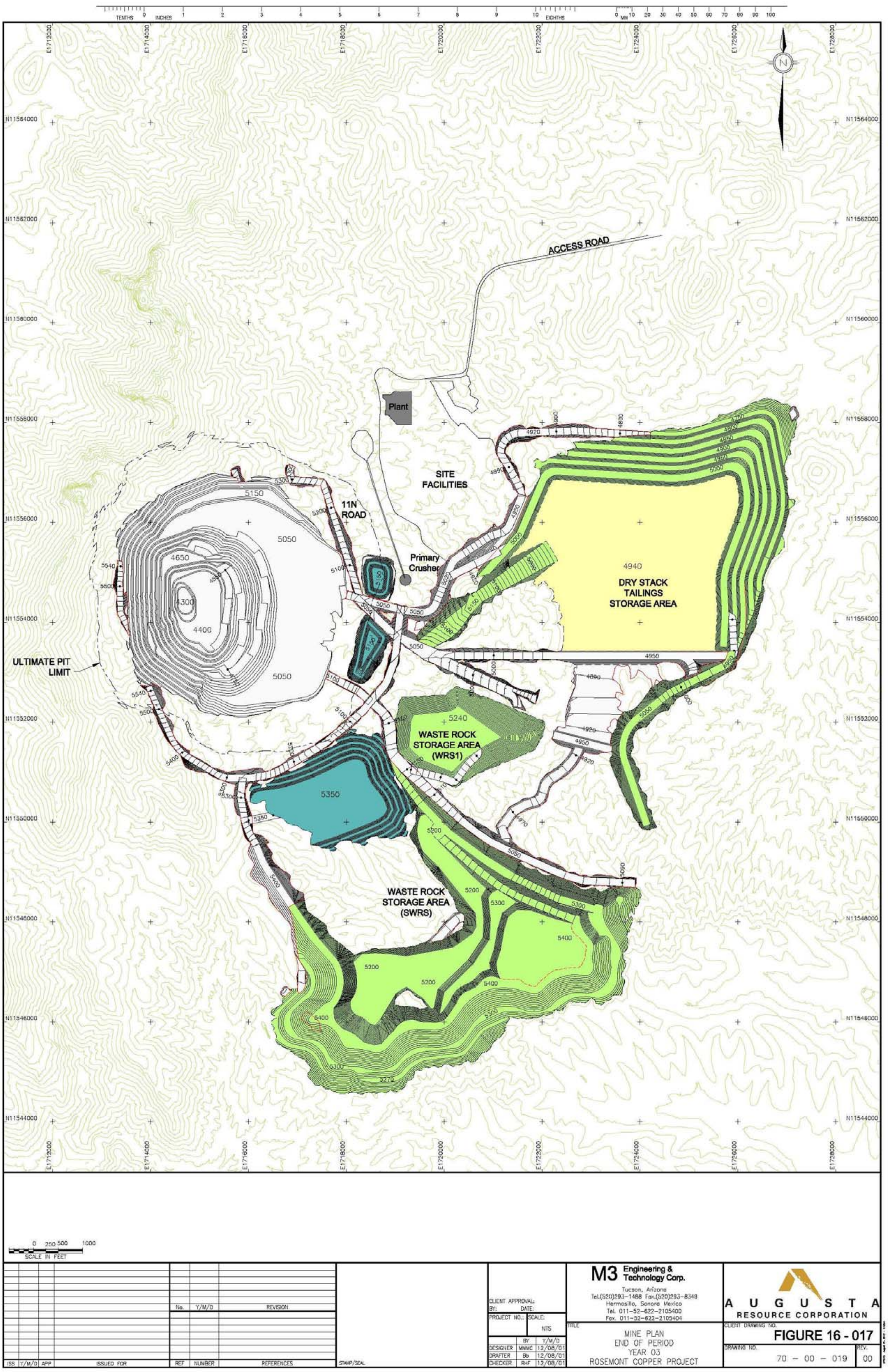

Figure 16-17: Mine Plan End of Period Year 03

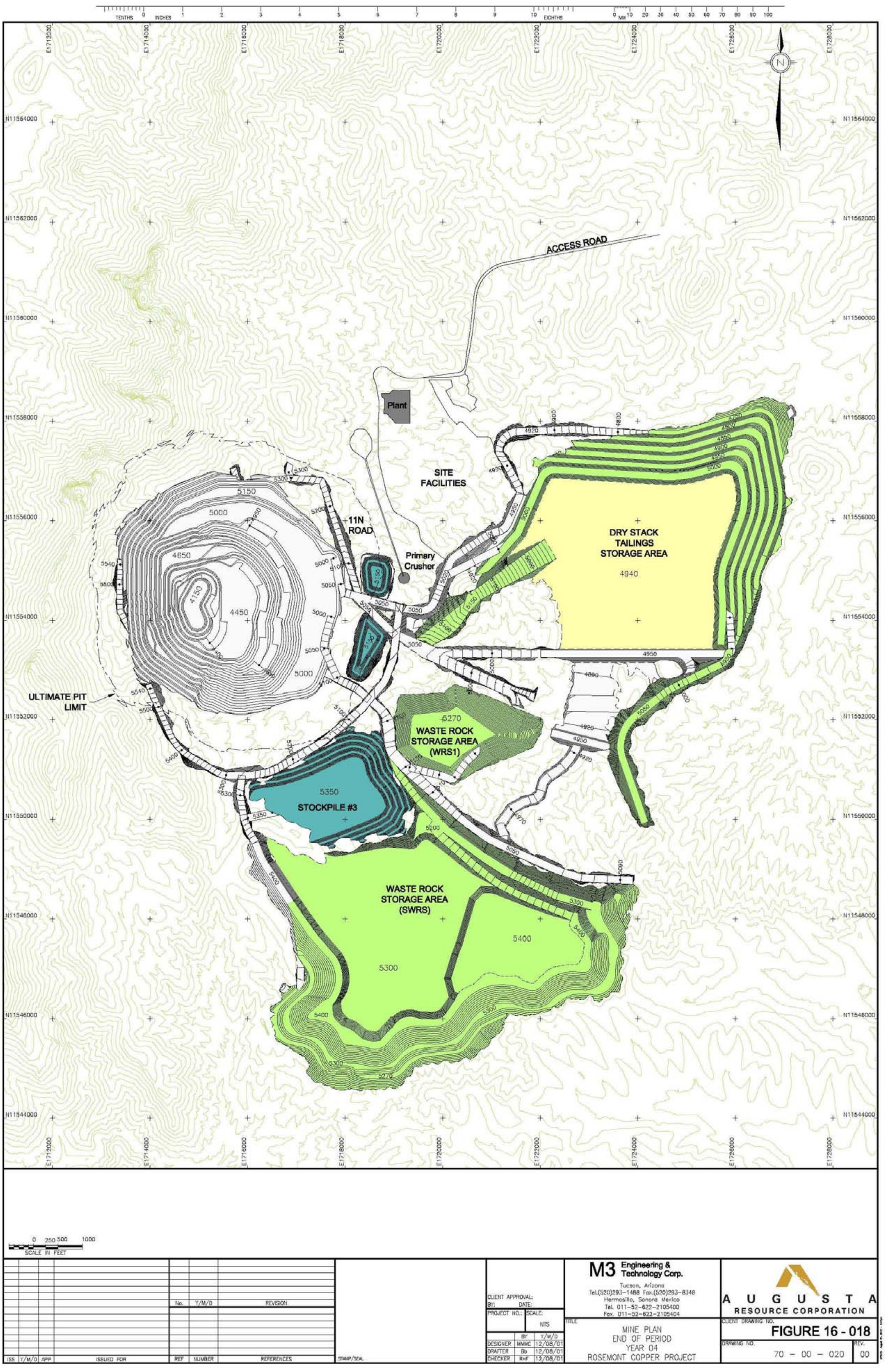

Figure 16-18: Mine Plan End of Period Year 04

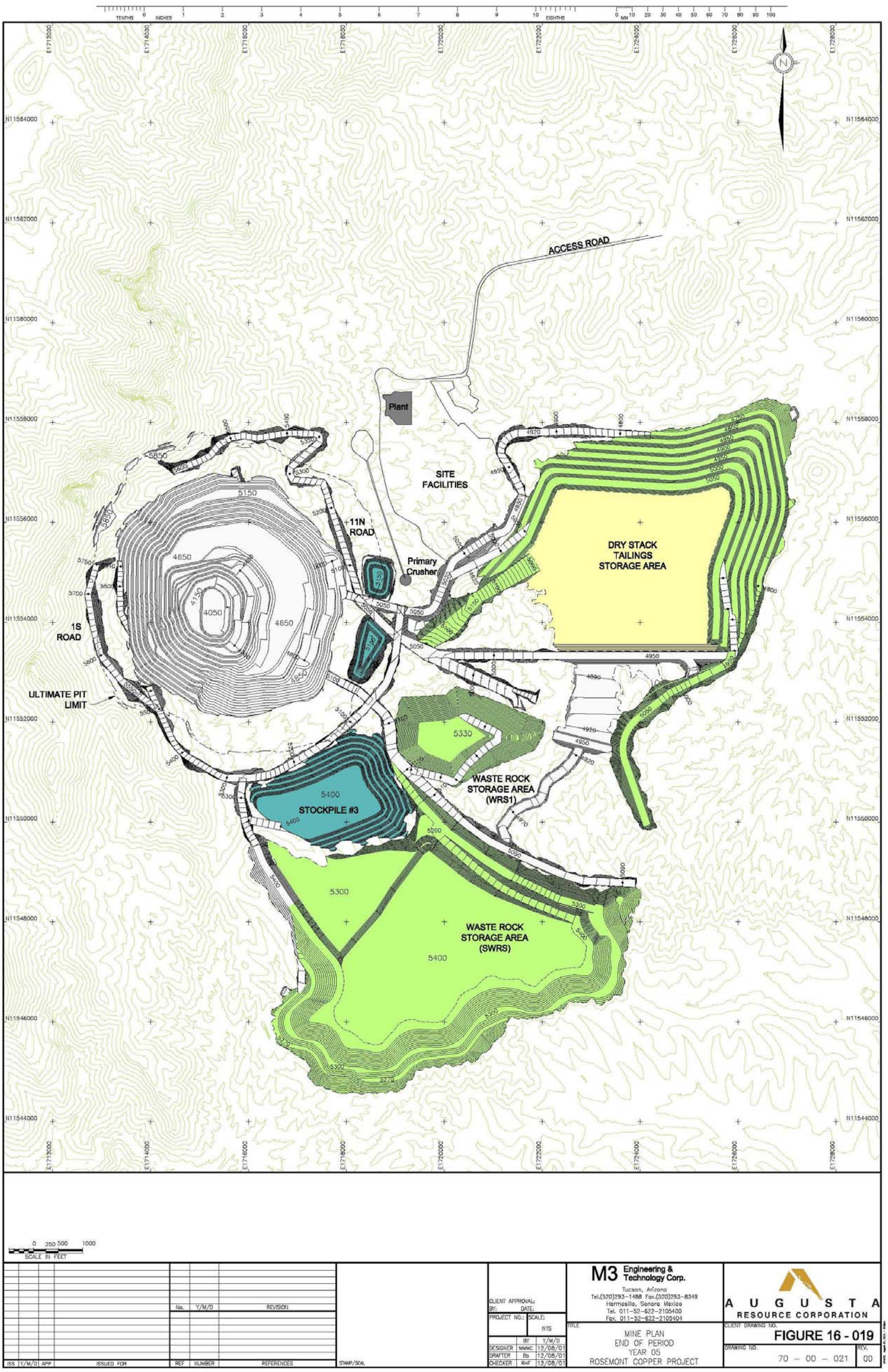

Figure 16-19: Mine Plan End Period Year 05

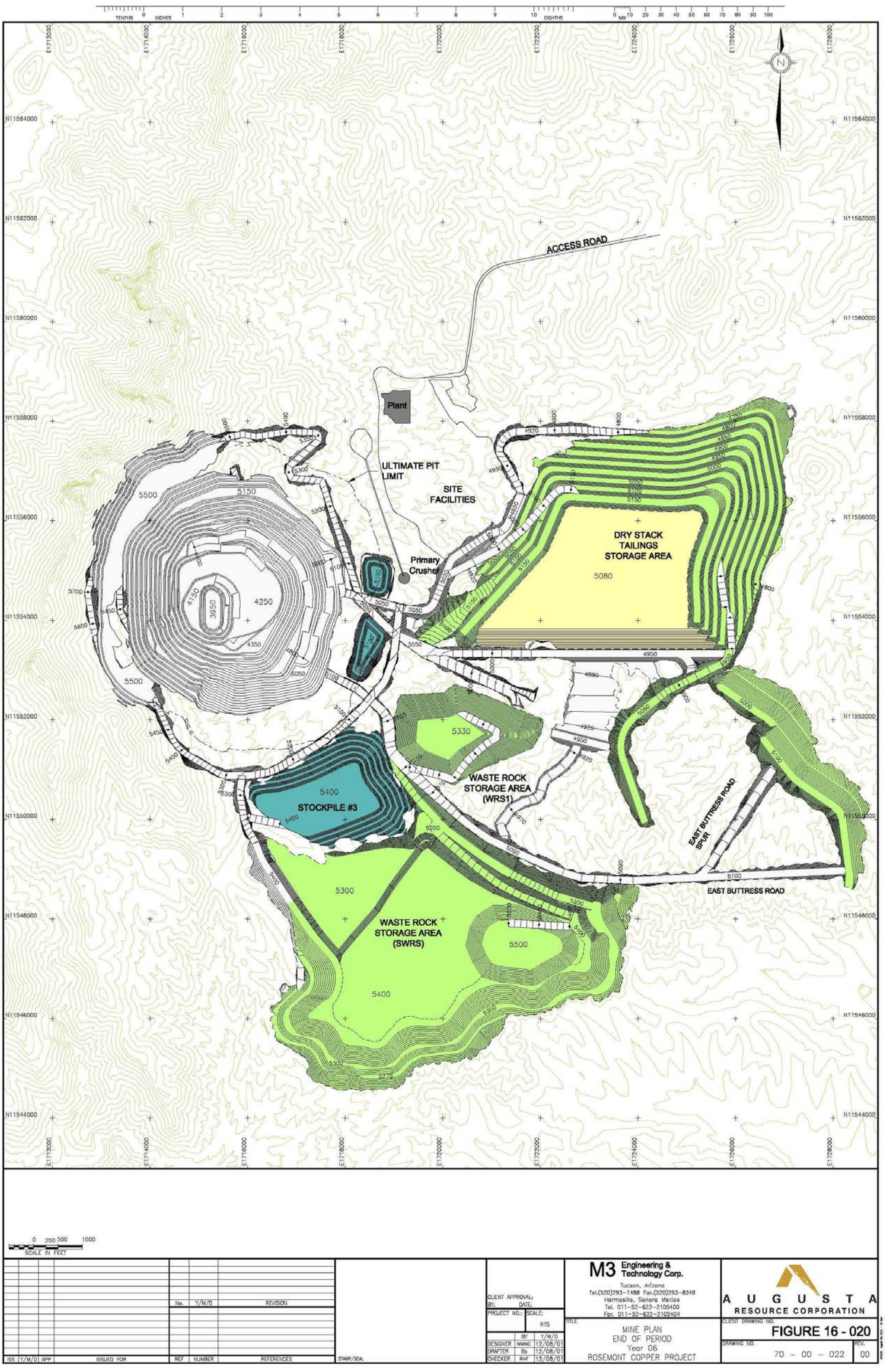

Figure 16-20: Mine Plan End of Period Year 06

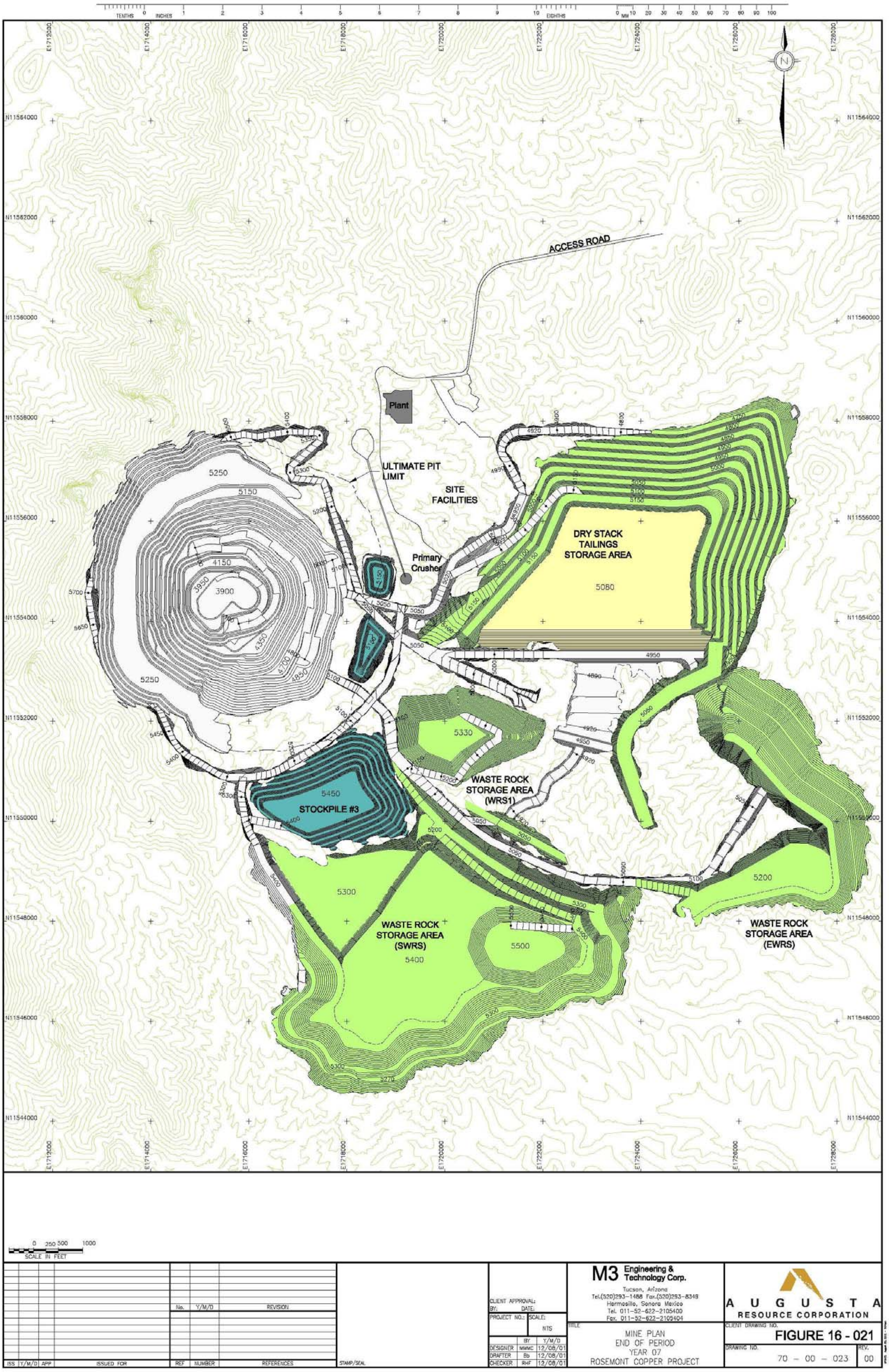

Figure 16-21: Mine Plan End of Period Year 07

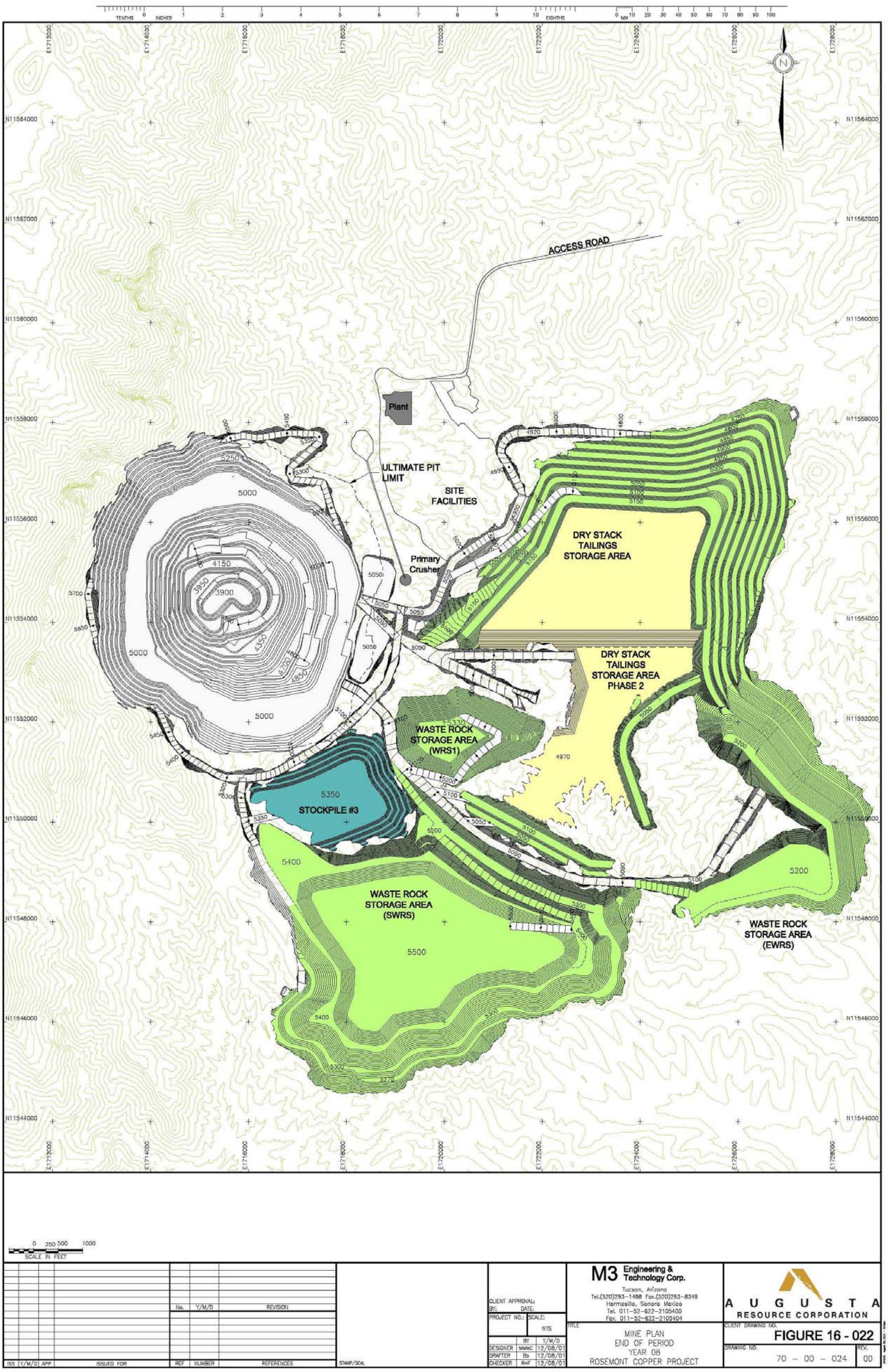

Figure 16-22: Mine Plan End of Period Year 08

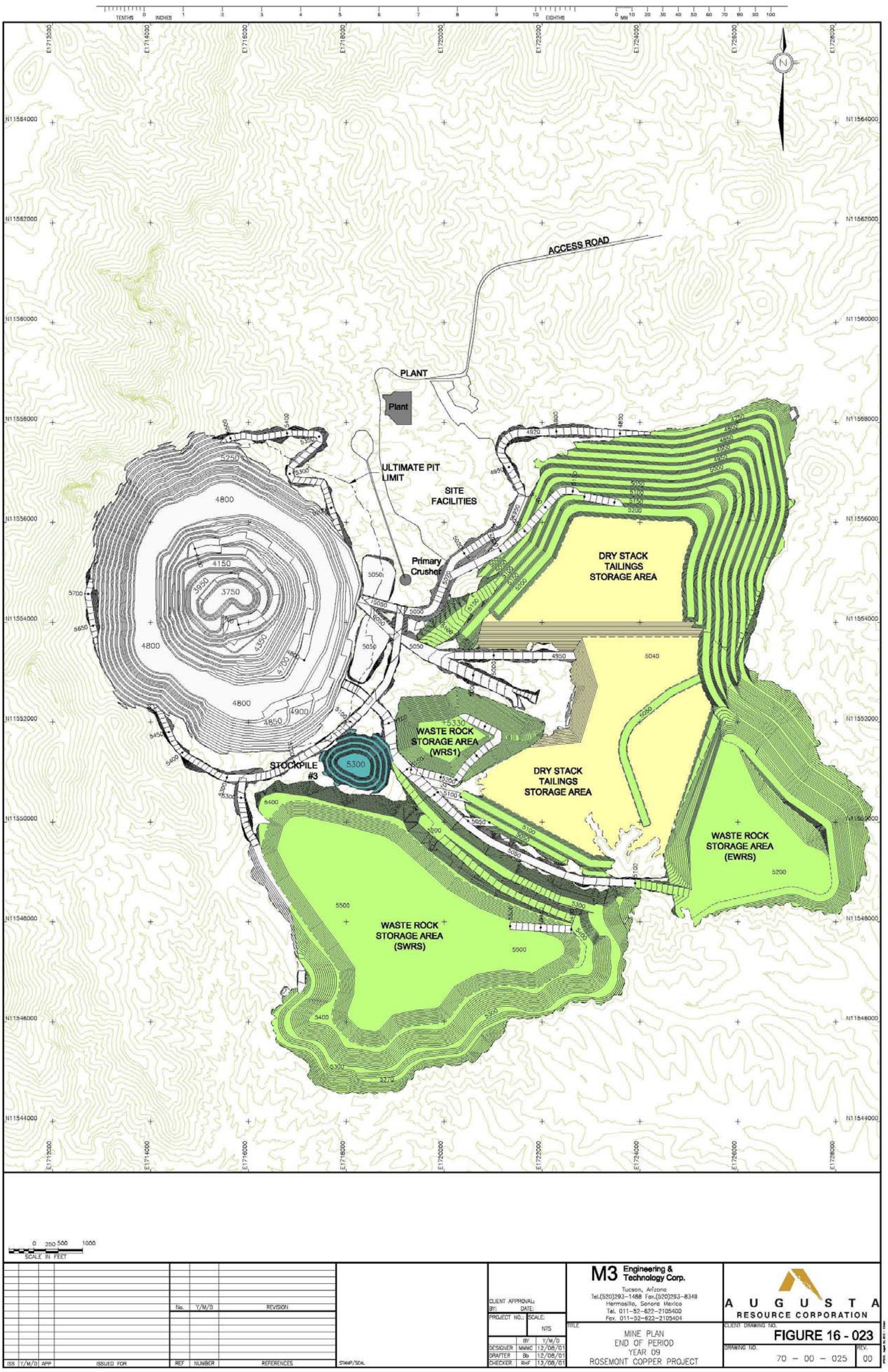

Figure 16-23: Mine Plan End of Period Year 09

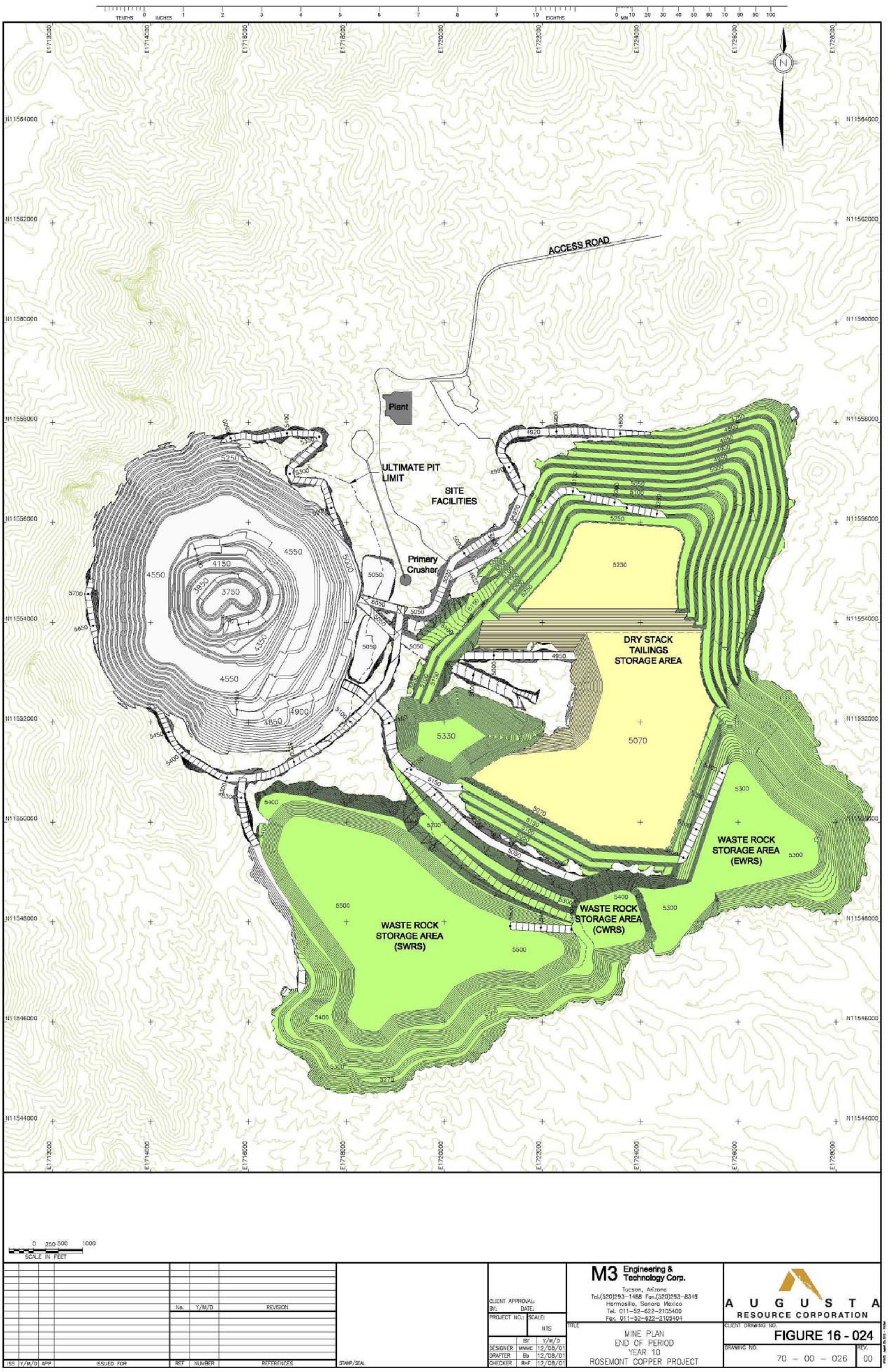

Figure 16-24: Mine Plan End of Period Year 10

Figure 16-25: Mine Plan End of Period Year 15

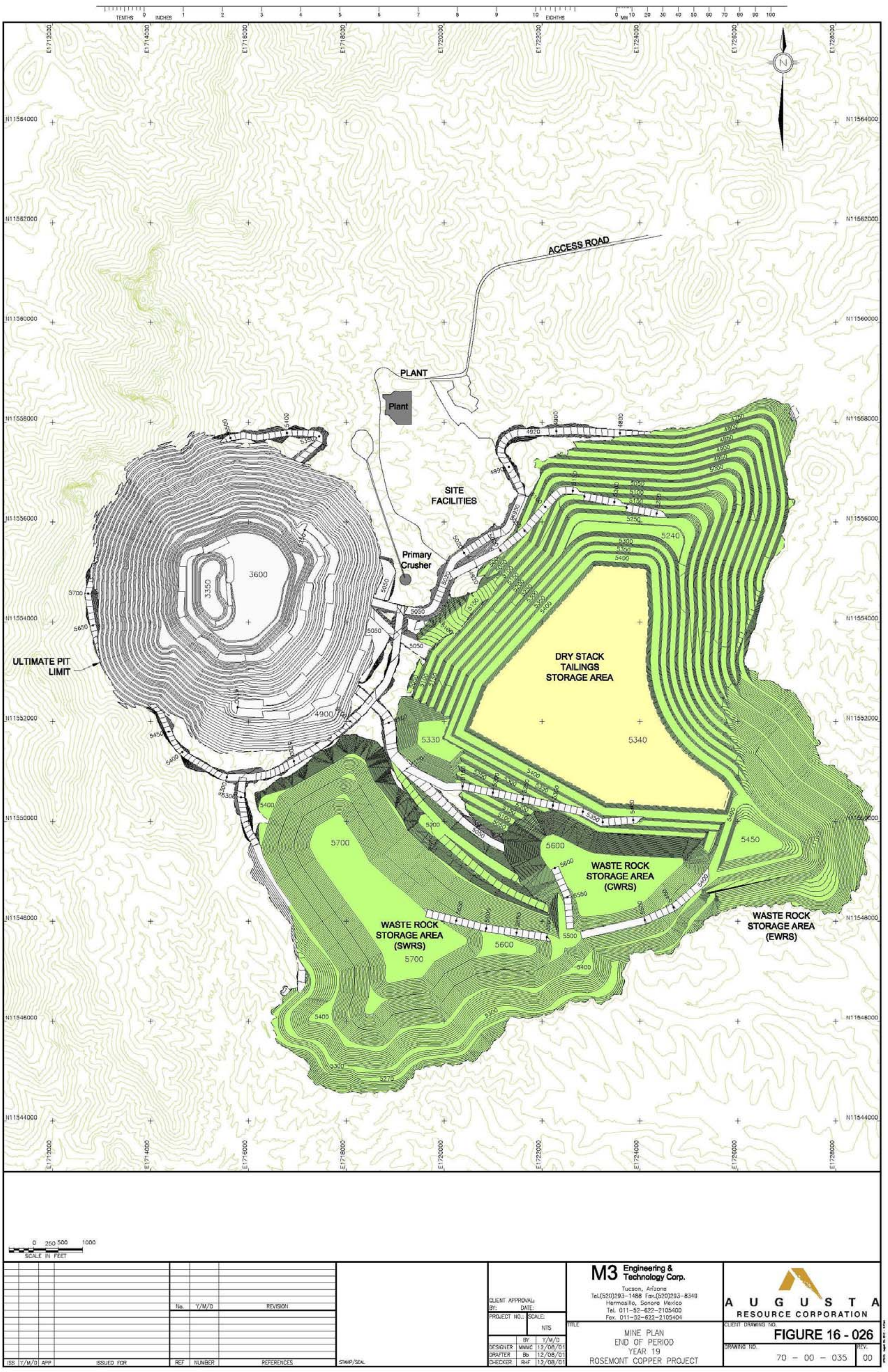

Figure 16-26: Mine Plan End of Period Year 19

Figure 16-27: Mine Plan End of Period Year 21.3

16.5.1 Mine Preproduction Summary

During the preproduction period before the first sulfide ore is delivered to the mill, the pit will be pre-stripped of waste to expose ore and develop the upper benches for subsequent push backs. Specifically, pre-stripping will occur in pit Phase 1 for ore exposure and in Phase 2 to 5 for development. By the end of preproduction the Phase 1 pit will be down to the 5,100 foot bench, and Phases 2 to 5 will be brought down together to the 5,400 foot bench. By the end of these seven quarters, sufficient ore will be exposed in Phase 1 pit to deliver uninterrupted ore to the mill. Phases 2 to 5 will be stripped sufficiently ahead to ensure supply of mill feed by Year 2.

16.5.1.1 Mine Road Development

Prior to the commencement of preproduction stripping operations, four primary haul roads will be constructed to access the top benches of the pit and connect to the crusher and waste storage areas. A 7,000-foot haul road (2C Road) will be constructed from the primary crusher to the high wall of Phase 1. The road will run west from the crusher, to the 5,400 foot elevation. From there it will swing north towards the pit highwall to 5,550 foot elevation. A second road (4N Road) will provide access for pit development at the north end. It will spur off the 2C Road at approximately 5,100 foot elevation running in a northwest direction to the 5,370 foot elevation. The third road will be constructed off the 2C Road at approximately 5,250 foot elevation and run southeast to approximately 5,060 foot elevation. This road will be used for hauling waste materials from the pit to various locations for waste rock placement or storage. A fourth road (1S Road) will be constructed at the south end of the upper pit benches starting from approximately 5,550 foot elevation. The road will roughly follow the final pit limit in a counter-clockwise direction and tie into the 4N Road at 5,050 foot elevation near the primary crusher. These roads will follow the topographic contours and will primarily be constructed by cut and fill. They will be constructed during the first and second quarters of the preproduction period.

Initial equipment access to the upper pit elevations to commence construction of these haul roads will be from existing exploration roads. Pioneer roads will be dozed following topographic contours where possible.

16.5.1.2 Mine Preproduction Periods –PP Q3 to PP Q7

By end of the third quarter of preproduction period (PP Q3) the upper benches on the west wall will be mined down to the 5,450 foot elevation. The main haul roads from the pit to the crusher, initial waste placement areas, and ore stockpiles will be completed. During this quarter, waste from the upper benches of Phases 2 to 5 along the west wall will also be dozed down. Material above the 5,600 foot elevation will be blasted as necessary and dozed down to the loading areas wide enough on benches below. Waste material totaling approximately 11.6 Mtons will be mined during the third preproduction quarter. Some of this material will be out of pit waste rock that has to be removed for development of the facilities within the Dry Stack Tailings footprint.

Waste rock will be hauled to extend roads, and commence building ramps and pads (FAM Pads and Ramps) for the dry stack conveyor installation. Over 7 Mtons of material will be required from the pit to complete this structure, and it is necessary that it is to be completed by end of PP Q4 to ensure that the facility will be constructed in a timely manner prior mill start-up.

Coarse and competent rock such as those from the Escabrosa and Glance formations will be hauled from the pit for pad or foundation materials. Excess waste rock not required for construction or development will be hauled to the initial Waste Rock Storage Area (WRS1).

The first cable shovel will be added to the excavating fleet in this quarter.

The second cable shovel will join the mining fleet in preproduction Quarter 4 (PP Q4). Mining in the Phase 1 pit during this quarter will be developed sufficiently and provide space for multi-bench mining. Stripping operations will advance Phase 1 to 5,350 foot elevation. Phases 2 to 5 will be mined down to the 5,400 foot elevation. 4N Road will be the access to the 5,400 foot elevation at the north end. Phase 1 access will be from 2C Road. 18.4 Mtons of material will be mined, 16.5 Mtons of that will be from the pit.

Waste rock will be hauled to extend the construction of the Dry Stack Road as well as 9N and 10N Roads. The Dry Stack Road will be used to access and construct the Centre Buttress, while 9N and 10N Roads will be used to haul waste rock to the Phase 1 Tailings Buttress. Nearly 1.0 Mtons of material will be hauled to start construction of the Tailings Buttress in this quarter. Construction of the 6S road will be started. Construction of the FAM Ramps and Pads will be completed in this quarter, and installation of the conveyor facilities will commence thereafter.

The third cable shovel will be added and begin stripping operations in preproduction Quarter 5 (PP Q5). Mining will mainly occur in Phase 1 and will be developed to the 5,250 foot elevation. Material mined in this quarter will increase to 23.4 Mtons, which includes about 1.5 Mtons of sulfide ore that will be placed in the run-of-mine (ROM) ore stockpile.

Waste rock will be hauled to continue construction of the Tailings Buttress at the 4,750 foot elevation. Construction of the Centre Buttress will occur during this quarter. This Buttress will be part of the containment structure for Phase 2 Tailings in the future. It is constructed early for the purpose of providing a screen to the mine operation from the public highway to the east. Waste rock will also be hauled to construct the South Buttress Road that will eventually be the access to the development of the South Screen Berm as well as support for containment of Phase 2 tailings. Approximately 1.9 Mtons of material will be placed in the WRS1.

Phase 1 pit continues to be the main area of mining activity in the sixth quarter of preproduction (PP Q6). The 5,150 foot elevation in Phase 1 pit will be reached. A total of 26.3 Mtons of material will be mined in this quarter, including 2.3 Mtons of sulfide ore hauled to the ROM stockpile. Waste rock will be hauled to the South Buttress at 5,200 foot and 5,270 foot elevations. Some waste rock will be used to extend the 6S Road, and subsequently used to start construction of the South Screen Berm at the 5,270 foot elevation. Approximately 3.1 Mtons of material will be hauled to the WRS1.

Phase 1 pit continues to be the main area of mining activity in the final quarter of preproduction (PP Q7) and is advanced to the 5,100 foot elevation. A total of 25.7 Mtons of material will be mined in this quarter, including 2.4 Mtons of sulfide ore hauled to the ROM stockpile. Waste rock will be hauled to the South Buttress at 5,200 foot, 5,270 foot and 5,300 foot elevations. Approximately 3.6 Mtons of waste material will be hauled to the WRS1.

16.5.2 Mine Production

Year 1

For the first year of mill operations, most of the mining activity will be in Phase 1 pit. Phases 2 and 3 development will occur during the latter half of the year. Required mill feed in all months will be from the Phase 1. 27.9 Mtons of ore will be mined in the first year, of which 3.8 Mtons will be stockpiled. The average mill feed grade will be 0.50% Cu for this period. 116.1 Mtons of total material will be mined Year 1.

During Quarter 1, Phase 1 benches mined will be 5,100, 5,150, and 5,000 benches. The Phase 1 in-pit ramp will be established on the south side of the pit, sinking in a counter-clockwise direction. Direct sulfide ore shipments from the pit will total about 3.8 Mtons, averaging 0.27% Cu.

Waste rock from the pit will be used to extend the South Screen Berm at the 5,300 foot elevation. 12.2 Mtons of waste material will be destined to the 5,200 foot lift of the South Waste Rock Storage Area (SWRS). Where possible, the South Screen Berm should be a minimum 50 foot higher than the SWRS to screen the mine operating activities from the public lands to the south. Approximately 4.8 Mtons of waste rock will be hauled to the initial waste storage area, WRS1.

In Quarter 2, Phase 1 will progress to the 4,900 bench and will supply all of the 6.5 Mtons of sulfide ore. An elevated cut-off at $8.00 /t NSR is implemented to produce an average head grade averaging 0.43% Cu. 7.3 Mtons of sulfide ore will be produced by the pit, of which 0.8 Mtons will be sent to the ROM stockpile.

7.8 Mtons of waste rock will be hauled and used for raising the Phase 1 Tailings Buttress to 4,800 foot elevation. Approximately 9.2 Mtons of waste rock will be destined to the South Screen Berm and SWRS. The remaining 2.6 Mtons of waste material from the pit will be placed in the WRS1.

In Quarter 3, Phase 1 will reach the 4,800 bench, and continue to be the main source for sulfide ore delivery to the mill. The NSR mill feed cut-off value is elevated to $12.00 /t, resulting in 6.9 Mtons of ore at an average head grade of 0.59% Cu delivered to the mill. 1.1 Mtons of ore below the mill feed cut-off value will be stockpiled. Phase 2 and 3 will advance to the 5,200 bench, accessed from the 4N Road.

The Phase 1 Tailings Buttress will be raised to 4,850 foot elevation with 6.2 Mtons of waste rock hauled from the pit. 11.5 Mtons of waste rock will be hauled to the 5,400 foot lift of the South Screen Berm and the 5,200 foot lift of the SWRS. The remaining 3.9 Mtons of waste material will be hauled to the WSR1.

In Quarter 4, Phase 1 will reach the 4,750 bench. Benches 5,200 and 5,150 will be mined from Phase 2 and 3, and there will also be some activity in Phases 4 and 5 on the 5,350 Bench. Phase

1 will continue to be the main source for sulfide ore delivery to the mill. The NSR mill feed cut-off value is maintained at $12.00 /t during this period, resulting in 6.9 Mtons of ore at an average head grade of 0.59% Cu delivered to the mill. 1.9 Mtons of ore below the mill feed cut-off value will stockpiled. The 4N Road will be cut off by Phases 2 and 3 development in the next quarter, and the 11 N road will need to be constructed. This road will also be used to access development of Phase 6 later on.

In the final quarter of Year 1, the Phase 1 Tailings Buttress will be raised another 50 feet to 4,900 foot elevation with 6.1 Mtons of waste rock hauled from the pit. The 5,400 foot lift of the South Screen Berm and the 5,200 foot lift of the SWRS continue to be the main destination for the waste rock, where 10.0 Mtons will place. 5.4 Mtons of waste material will be hauled to WRS1.

Year 2

In Year 2, the annual pit production will be 105.5 Mtons. Mill requirements for sulfide ore will be provided from Phase 1 early in the year, transitioning to a Phase 2 supply later on. An elevated mill cut-off of $12.00 /t NSR value is applied to provide an average head grade of 0.62% Cu for the year. Approximately 8.2 Mtons will be stockpiled. Phase 3 will be further pre-stripped in preparation to supply sulfide ore in Year 3. Some development of Phases 4 and 5 will also occur this year.

During the first quarter of Year 2, Phase 1 will be mined down to the 4,700 foot bench and Phase 2 will reach the 5,050 foot bench. The access into Phase 2 is easily available along the east side from the 3C Road. Sulfide ore production from the pit will be at the rated mill throughput of 6.8 Mtons at an average head grade of 0.65% Cu. 2.1 Mtons of ore will be stockpiled during this quarter. The majority of the ore will be from Phase 1.

Total waste material mined in this quarter will be approximately 19.0 Mtons. The majority of the waste rock will be used to raise the South Screen Berm to the 5,400 foot elevation. The remaining waste material will be destined to the 5,200 foot lift on the SWRS, and the WRS1.

Mining in Phases 1 and 2 will reach the 4,650 and 4,950 levels, respectively, during Quarter 2 of Year 2. Phases 4 and 5 will be mine to the 5,300 foot benches, accessed from the 11N Road. 6.8 Mtons of sulfide ore required at the mill, will be from Phase 1 and 2. The average Cu head grade is 0.62% for this quarter. 2.4 Mtons of ore below the elevated NSR cut-off value will be sent to the stockpile.

Total waste material mined in this quarter will be approximately 16.8 Mtons. 4.7 Mtons of waste rock will be used to commence raising the Phase 1 Tailings Buttress to the 4,950 foot elevation. 10.3 Mtons will be used to extend the South Screen Berm on the 5,400 foot elevation. The remaining waste material will be destined to the 5,200 foot lift on the SWRS, and the WRS1.

In Quarter 3, Mining in Phases 1 and 2 will reach the 4,550 and 4,800 foot elevations, respectively. The average mill feed grade from these pit phases is 0.61% Cu for this quarter. To maintain the high head grade, approximately 2.4 Mtons of ore will be sent to the stockpile.

A total of 15.3 Mtons of waste material will be mined in the third quarter of Year 2. 4.9 Mtons of waste rock will be used to complete the 4,950 foot lift on the Phase 1 Tailings Buttress. 10.1 Mtons of waste rock will be placed on the South Screen Berm at the 5,400 foot elevation, as well as the 5,200 foot elevation of the SWRS. The remaining waste material will be place in the WSR1.

During the last quarter of Year 2, mining in Phase 1 will descend to the 4,400 foot elevation, one bench from the bottom of this pit phase. Phase 3 will be developed for primary sourcing of ore. It will be mined from 5,100 to 4,950 foot elevations during this period. The 4N Road will be the main pit access for ore and waste material to this phase. The average mill feed grade is maintained at 0.61% Cu for 6.8 Mtons. Approximately 1.3 Mtons of ore is stockpiled.

All of the 18.8 Mtons of waste material will be from Phase 3. Almost all - 18.5 Mtons, will be destined to extend the South Screen Berm and SWRS.

Year 3

Mining in Year 3 will complete Phases 1, and 2. Phase 3 will be lowered to the 4,400 foot elevation. Phases 4 and 5 will be developed to the 5,050 foot elevation. The 3C and 4N Roads will be the primary haul roads for ore and waste material out of the pit through Phase 4 and 5. Internal pit ramps provide haul roads out of Phases 1 and 2 onto the Phase 3 mining benches.. The NSR mill feed cut-off is for this year is $12.00 /t. Total ore mined from the pit during this year will be 42.7 Mtons, of which 27.4 Mtons will be directed to the mill at an average grade of 0.50 % Cu. The balance of the ore mined will be stockpiled.

Total waste material mined for the year will be 82.2 Mtons. 9.8 Mtons will be hauled to the Phase 1 Tailings Buttress to raise the containment elevation to the 5,000 foot level. Approximately 7.1 Mtons of waste material will be hauled to WRS1, while the remainder will be hauled to the complete the 5,400 foot lift of the South Screening Berm, and build on the SWRS behind it. Part of the SWRS will be up to 5,400 foot elevation, while the lifts established on 5,200 and 5,300 foot elevations will continue to be extended.

Year 4

Phase 3 will be completed in Year 4. Mining activities will primarily be in Phase 4 where it will advance from 5,050 to the 4,450 foot elevation. The push back from Phase 3 to Phase 4 is sufficiently large and will allow for three cable shovels to mine in that phase for the majority of the year. Two benches are also mined in Phase 5, down to 5,000 foot elevation. Total ore mined from the pit will be 27.4 Mtons, all directed to the mill. The mill NSR cut-off is not elevated for this year as the pit will be over-stressed if more ore is excavated in this period. There will not be any ore hauled to the stockpile, and the average head grade is lowered to 0.45 % Cu in Year 4.

Total material mined in Year 4 will be 123.4 Mtons, of which 96.0 Mtons will be waste rock. Approximately 2.8 Mtons will be destined to the WRS1, while the remainder will be hauled to the SWRS. By end of Year 4, the SWRS will be completed to the 5,300 foot elevation, and the 5,400 foot lift will continue to be advanced.

Year 5

In Year 5, Phase 4 is mined to the 4,050 foot elevation, near the bottom of this pit phase. Phase 5 is advanced to 4,650 foot bench, and along with Phase 4 will be the primary source for ore feed to the mill. Commencement of expansion activities will allow the mill to increase its throughput rate to an average of 78,000 tph for the year. An elevated NSR mill cut-off value is re-established for this period at $8.00 /t. This cut-off will allow the mill feed grade to average 0.55 % Cu for the year. 28.5 Mtons of ore will be fed directly to the mill, and 3.5 Mtons will be stockpiled.

Phase 6 development will commence during this period. Pioneer roads will be re-established for equipment access to develop the benches above Phase 5. Track dozers will sequentially prepare drill ramps and pads for the track drill that will drill holes for blasting. Some waste material will be cast over into the pit from the blasts. Dozers will push the remaining material to benches below until approximately the 5,650 foot bench is attained. Pit operations below will be intermittently disrupted while this activity above takes place. The 11 N Road will be extended at the north end from 5,200 foot elevation to 5,600 foot elevation below the ridge top. From the south, the S1 Road will be extended from 5,500 foot elevation to above 5,650 foot elevation. These roads will provide haulage access for the upper benches of Phase 6.

Total waste material mined in Year 5 will be 74.6 Mtons. Approximately 9.0 Mtons will be hauled to the Phase 1 Tailings Buttress, raising it to the 5,050 foot elevation. The 5,300 foot lift of the SWRS will be filled, and the majority of the waste rock will be place on the 5,400 foot lift. Approximately 4.2 Mtons of waste material will be sent to the WRS1 location. At the end of this year period, the WRS1 will be completed, and the area will be allocated for Phase 2 Tailings.

Year 6 to Year 10

In Years 6 to 10, ore feed to the mill will be sourced from Phases 4, 5 and 6. Mill feed throughput will continue ramping up to 84,000 tph in Year 6 and 88,000 tph in Years 7 to 10. An elevated NSR mill feed cut-off is maintained for Years 6 to 7, and dropped to the minimum economic cut-off in Years 8 to 10, resulting in an average Cu mill feed grade of 0.43 % during this 5 year period. 8.9 Mtons of ore will be added to the stockpile in Years 6 and 7, bringing total to 46.0 Mtons. All of the stockpiled ore will be reclaimed in Years 8 to 10 to supplement the pit feed.

Phase 4 will be completed in Year 6, and Phase 5 will be completed by Year 8. Phase 6 will be the primary source for ore feed in years 9 and 10. Development at the top of Phase 6 will commence in Year 6, with stripping down to the 5,500 foot bench by the end of the year. As Phase 6 advances close to the 5,000 foot bench (Year 8), it will be a push-back completely around the previous phases in all directions. It is currently designed and scheduled as a single phase, but it may possibly be further sub-divided to enhance the mine plan and production schedule in later years.

Average ore mined and waste mined from the pit during this five year period will be 24.4 Mtons and 79.0 Mtons, respectively. Total material mined will average 103.4 Mtons. Ore mined from the pit will be reduced as the ROM stockpile will be reclaimed commencing in Year 8, supplementing the pit feed.

Waste material will continue to be hauled to, and placed in the SWRS during this period. Construction of the East Buttress will also commence during these years, with 73.6 Mtons placed in sequence from the 5,000 ft lift up to the 5,200 foot lift by Year 10. This buttress will provide a screen from public lands when the East Waste Rock Storage (EWRS) is built behind it. Access to the East Buttress will be by the Dry Stack Road to the North and a spur off the East Buttress Road to the south from 5,100 foot elevation. Waste rock will be placed in the EWRS beginning in Year 7 on the 5,200 foot lift. This lift will be completed by Year 10, and the next lift at the 5,300 ft elevation will be started in the same year. A total of 105.2 Mtons of waste rock will be placed in the EWRS. 30.4 Mtons of waste rock will be hauled to construct the Center Waste Rock Storage (CWRS) area up to the 5,400 foot elevation. This area is located between the SWRS and the EWRS, and can be accessed from the road to the SWRS.

31.5 Mtons of waste will be hauled to Phase 1 Tailings Buttress in these years, raising it to its final elevation of 5,250 foot by Year 10. Construction of the Phase 2 Tailings Buttress will occur in Year 7 up to the 5,050 foot elevation, to prepare for tailing placement in Year 8. In Year 8, this Buttress will be further raised to the 5,100 foot elevation to keep ahead of the tailings elevation. A total of 18.7 Mtons of waste rock will be placed in the Phase 2 Tailings Buttress during this five year period.

Year 11 to Year 15.

In Years 11 to 15, all mining activities will be in Phases 6 and 7. Phase 7 is the final push back on the east side, and development will commence in Year 11 with waste stripping of the 5,450 foot bench down to the 5,100 foot bench. Ore feed from Phase 7 will be available by Year 12. By the end of Year 15, Phase 7 will have advanced to the 4,400 foot elevation. Phase 6 will be mined from the 4,550 foot bench down to the 3,700 foot elevation during these years. By Year 12, the mill throughput will be increased to 90,000 tpd. The average mill feed grade will be 0.43 % Cu over the five year period. All pit ore mined will be hauled directly to the mill, and there will be no ROM stockpiles.

The average annual waste mined will be 52.1 Mtons, significantly reduced from the previous periods, as the lower benches of Phase 6 will predominantly consist of ore. Total material mine from the pit will average 84.8 Mtons per year.

23.4 Mtons of waste rock will be hauled to raise the Phase 2 Tailings Buttress from 5,150 foot elevation to 5,250 foot elevation over this period. The remaining waste will be destined to the waste rock storage areas – SWRS, CWRS, and EWRS. By Year 15, the SWRS will be completed to 5,700 foot elevation, the CWRS will be at 5,600 foot elevation and the EWRS will be at 5,450 foot elevation. Both SWRS and EWRS are at their final elevations.

Year 16 to End of Mine

The last few benches in Phase 6 will be mined in Year 16, and all ore feed and mining activities will be from Phase 7 thereafter. The mill feed grade will average 0.41 % Cu. The average strip ratio for this period will be 0.4:1 (ratio of waste mined:ore mined) as the benches will consist predominantly of ore.

Waste rock will continue to be hauled to the Phase 2 Tailings Buttress until the final elevation of 5,450 foot is attained. It will be necessary to haul waste rock and build the final lifts early when there is still rock available from the pit. The CWRS will be completed by Year 17 to 5,600 foot elevation as all waste rock after this year will have to be available for the Phase 2 Buttress construction.

16.6 MINE PRODUCTION SCHEDULE

The estimated mine production schedule is presented in Table 16-16. The proven and probable mineral reserves summarized in this schedule are based on an internal NSR cutoff of $4.90 /ton for sulfide ore. All inferred mineral resources are treated as waste.

The total ore report in production schedule in Table 16-16 is 661.4 Mtons, compared to the pit reserves reported in Section 15, Table 15-11 of 667.2 Mtons. The difference of 5.8 Mtons (less than 1%) is the ore contained in the pit that has an NSR value between $4.90 /t and $5.30 /t when an elevated cut-off strategy is applied. If the pit ore has an NSR value below the elevated mill cut-off, it is destined to the ROM stockpile provided that the NSR value is above $5.30 /t to cover the stockpile handling costs. If the value is below, it will be sent to the waste rock storage locations. If all pit ore is directed to the mill and stockpiling is not planned (as in Year 8 and after), all ore with an NSR value above $4.90 will be processed.

A mine life of 21.3 years is projected by this development plan. Peak mining rates of 342,000 tpd of total material will be realized in Year 3. Average mining rates during Years 5-10 will be 285,000 tpd of total material, and be reduced to an average of 232,000 tpd from Years 11 – 15 as the strip ratio drops.

Table 16-6: Mine Production Schedule – Combined Proven & Probable Mineral Reserves

(NSR values are based on metal prices of $2.50/lb Cu, $15.00/lb Mo and $20.00/oz Ag. All inferred mineral resources are treated as waste.)

Time	Mined Sulfides >= 4.90 $/ton NSR Cutoff					Sulfide Ore to Stockpile					Reclaimed Sulfide Ore Stockpile					Total Mill Feed					Waste	Total	Strip
Period	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	NSR $/t	TCu %	Mo %	Ag oz/t	Ktons	Ktons	Ratio
PP Q1	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0	0.00
PP Q2	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	4,091	4,091	0.00
PP Q3	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	9,026	9,026	0.00
PP Q4	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	16,543	16,543	0.00
PP Q5	0	0.00	0.00	0.000	0.00	1,516	13.86	0.29	0.012	0.12	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	21,925	23,441	14.46
PP Q6	0	0.00	0.00	0.000	0.00	2,327	11.08	0.23	0.012	0.08	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	24,011	26,339	10.32
PP Q7	0	0.00	0.00	0.000	0.00	2,416	13.60	0.29	0.011	0.11	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	23,263	25,678	9.63
Y1 Q1	3,755	13.28	0.27	0.012	0.11	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	3,755	13.28	0.27	0.012	0.11	25,639	29,393	6.83
Y1 Q2	6,525	19.91	0.43	0.016	0.13	828	6.74	0.14	0.005	0.06	0	0.00	0.00	0.000	0.00	6,525	19.91	0.43	0.016	0.13	19,552	26,905	2.66
Y1 Q3	6,900	26.54	0.59	0.019	0.16	1,137	9.08	0.20	0.007	0.07	0	0.00	0.00	0.000	0.00	6,900	26.54	0.59	0.019	0.16	21,571	29,608	2.68
Y1 Q4	6,900	26.04	0.59	0.018	0.14	1,875	8.58	0.19	0.006	0.05	0	0.00	0.00	0.000	0.00	6,900	26.04	0.59	0.018	0.14	21,407	30,182	2.44
Y2 Q1	6,843	28.19	0.65	0.019	0.14	2,092	8.46	0.19	0.008	0.04	0	0.00	0.00	0.000	0.00	6,843	28.19	0.65	0.019	0.14	18,950	27,884	2.12
Y2 Q2	6,843	26.63	0.62	0.017	0.12	2,363	8.76	0.20	0.007	0.04	0	0.00	0.00	0.000	0.00	6,843	26.63	0.62	0.017	0.12	16,833	26,039	1.83
Y2 Q3	6,843	26.45	0.61	0.015	0.15	2,400	8.00	0.18	0.005	0.04	0	0.00	0.00	0.000	0.00	6,843	26.45	0.61	0.015	0.15	15,324	24,568	1.66
Y2 Q4	6,843	26.90	0.61	0.016	0.19	1,349	8.78	0.22	0.002	0.05	0	0.00	0.00	0.000	0.00	6,843	26.90	0.61	0.016	0.19	18,837	27,030	2.30
Y3	27,375	23.79	0.50	0.026	0.12	15,253	8.95	0.19	0.009	0.05	0	0.00	0.00	0.000	0.00	27,375	23.79	0.50	0.026	0.12	82,165	124,793	1.93
Y4	27,375	20.19	0.45	0.012	0.15	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	27,375	20.19	0.45	0.012	0.15	95,980	123,355	3.51
Y5	28,470	24.15	0.55	0.015	0.15	3,545	6.70	0.14	0.006	0.07	0	0.00	0.00	0.000	0.00	28,470	24.15	0.55	0.015	0.15	74,569	106,584	2.33
Y6	30,660	22.71	0.53	0.013	0.12	3,688	6.67	0.15	0.004	0.05	0	0.00	0.00	0.000	0.00	30,660	22.71	0.53	0.013	0.12	63,412	97,761	1.85
Y7	32,120	26.30	0.61	0.014	0.15	5,253	9.45	0.19	0.009	0.07	0	0.00	0.00	0.000	0.00	32,120	26.30	0.61	0.014	0.15	62,094	99,467	1.66
Y8 to Y10	50,316	18.94	0.46	0.010	0.12	0	0.00	0.00	0.000	0.00	46,044	9.02	0.19	0.008	0.06	96,360	14.20	0.33	0.009	0.09	269,243	319,559	5.35
Y11 to Y15	163,520	17.82	0.43	0.013	0.11	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	163,520	17.82	0.43	0.013	0.11	260,736	424,256	1.59
Y16 to Y21.3	204,097	19.25	0.41	0.017	0.12	0	0.00	0.00	0.000	0.00	0	0.00	0.00	0.000	0.00	204,097	19.25	0.41	0.017	0.12	83,990	288,086	0.41
Total	615,384	20.33	0.46	0.015	0.12	46,044	9.02	0.19	0.008	0.06	46,044	9.02	0.19	0.008	0.06	661,428	19.54	0.44	0.015	0.12	1,249,160	1,910,588	1.89

16.6.1 Waste Rock Storage Destinations and Quantities

Table 16-7 summarizes waste rock quantities to the assigned destination for the life of mine. The totals include approximately 10 Mtons of out of pit waste rock that will be used for base material and construction.

Table 16-7: Waste Rock Storage Tonnages by Destination

Waste Destinations	k-tons
Haul Road 4N	179
Haul Road 2C	0
Haul Road 3C	12
Haul Road 7N	420
Haul Road 1S	437
Haul Road 6S	246
Haul Road 9N	801
Haul Road 10N	3,798
Dry Stack Haul Road	5,843
Leach Haul Road	3,019
FAM Ramps and Pads	7,250
ROM Stockpile Pad	510
Phase 1 Tails 4750 Buttress	8,191
Phase 1 Tails 4800 Buttress	7,765
Phase 1 Tails 4850 Buttress	6,198
Phase 1 Tails 4900 Buttress	6,089
Phase 1 Tails 4950 Buttress	9,617
Phase 1 Tails 5000 Buttress	9,780
Phase 1 Tails 5050 Buttress	8,980
Phase 1 Tails 5100 Buttress	8,449
Phase 1 Tails 5150 Buttress	9,798
Phase 1 Tails 5200 Buttress	7,171
Phase 1 Tails 5250 Buttress	6,043
Phase 2 Tails 5050 Buttress	5,030
Phase 2 Tails 5100 Buttress	10,541
Phase 2 Tails 5150 Buttress	6,016
Phase 2 Tails 5200 Buttress	9,630
Phase 2 Tails 5250 Buttress	10,845
Phase 2 Tails 5300 Buttress	15,730
Phase 2 Tails 5350 Buttress	16,876
Phase 2 Tails 5400 Buttress	16,410
Phase 2 Tails 5450 Buttress	12,597
Center Buttress 5050	8,992
Initial Waste Rock Storage	45,115
Total Center Waste Rock Storage 5300 - 5600	154,936
Total South Screen Berm (Buttress) 5200 - 5400	131,792
Total South Waste Rock Storage 5000 - 5200	503,914
Total East Buttress 5000 - 5200	73,594
Total East Waste Rock Storage 5200 - 5400	126,104

16.7 MINE EQUIPMENT SELECTION

Mine equipment requirements were developed based on the annual tonnage movements projected by the mine production schedule in Table 16-6, bench heights of 50 feet, two twelve hour shifts per day, 365 days per year operation, manufacture machine specifications and material characteristics specific to the deposit.

Specific manufacturer's models used in this study are only intended to represent the size and class of equipment selected. The final equipment manufacturer selection will be done as required to meet delivery dates and current need of the operation.

A summary of fleet requirements by time period for major mine equipment is shown in Table 16-8. This represents equipment necessary to perform the following mine tasks:

- Mine Site clearing and topsoil salvage and stockpiling
- Construction of the main haul road with the exception of some of the initial haul roads built by a contractor.
- Production drilling.
- Loading and hauling of sulfide ore to the primary crusher (located on the east side of the pit), and waste rock to waste rock storage (WRS) areas.
- Maintain mine haulage and access roads.
- Maintain waste rock storage (WRS) areas, dry stack storage buttresses and berms, and regrading of slopes and final surfaces.

Table 16-8: Major Fleet Requirements

	PP-2	PP-1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11-15	Year 16-20	Year 21-22
CAT 7495 Electric Shovel #1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
CAT 7495 Electric Shovel #2	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
CAT 7495 Electric Shovel #3	0	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Hydraulic Excavator 46 yd.	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
LeTourneau FEL L-1850 - 36 yd	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
CAT 994F FEL - 25cy	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Cat 834H RTD Dozer	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3
CAT 793F Haul Truck - 250 Ton	13	27	27	32	35	35	35	35	35	35	35	35	29	18	18
CAT D11T Track Dozer	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
CAT D10T Track Dozer	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Cat 16M Motor Grader	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Cat 785 Water Truck - 30000 Gal	3	4	4	4	4	4	4	4	4	4	4	4	3	2	2
Electric Blasthole Drill - 12.25 in	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Diesel Blasthole Drill - 12.25 in	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
DML Highwall Perimeter Drill	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cat 385 Hyd Excavator	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cat_980H_Front-end Loader - 5 yd	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cat CS76LT Compactor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Light Plants	6	6	16	16	16	16	16	16	16	16	16	16	16	16	16
Water Pumps	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3
Total Number of Units	**45**	**62**	**73**	**78**	**81**	**81**	**81**	**81**	**81**	**81**	**81**	**81**	**73**	**59**	**58**

16.8 EQUIPMENT OPERATING PARAMETER

Mine equipment was selected based on the production requirements shown in Table 16-6. During the first three quarters of preproduction 46 cu-yd hydraulic excavator, 36 cu-yd and 25 cu-yd loader matched with 260-ton-class haul trucks, supported with dozers, graders and water trucks will be used to develop the initial mine area. At the end of the third quarter of preproduction the first 60-cy class electric shovel will come on line followed by two more in preproduction quarters four and five.

The mine will operate two shifts per day, 12 hours per shift for 365 days a year. No significant weather delays are expected and the mine will not be shut down for holidays. Craft work schedule will consist of a standard four crew rotation.

The majority of mining equipment is new with the exception of hydraulic excavator, 994 loader, water trucks and equipment transport unit. Equipment mechanical availabilities (MA) vary depending on period as shown in Table 16-9.

Material characteristics used to determine productivity calculations are listed in Table 16-10. There are several different rock types at the Rosemont Mine but for production estimation the weighted average of all rock types was used. Major loading and haulage equipment will be equipped with electronic load monitors which will insure maximum loading. All equipment production is reported in dry short tons which is consistent with the reserve model. Moisture content is expected to range between 3 and 4 percent; for haulage calculations 3.5 percent was used.

Table 16-9: Equipment Mechanical Availability

	PP-2	PP-1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11-15	Year 16-20	Year 21-22
CAT 7495 Electric Shovel #1	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
CAT 7495 Electric Shovel #2	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	0%
CAT 7495 Electric Shovel #3	0%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	0%	0%	0%
Hydraulic Excavator- 46 yd	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
LeTourneau FEL L-1850 - 36 yd	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%
CAT 994F FEL - 25cy	82%	82%	0%	0%	82%	82%	82%	82%	82%	82%	82%	82%	0%	0%	0%
Cat 834H RTD Dozer	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
CAT 793F Haul Truck - 250 Ton	92%	92%	92%	91%	90%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%
CAT D11T Track Dozer	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
CAT D10T Track Dozer	85%	85%	85%	86%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Cat 16M Motor Grader	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%
Cat 785 Water Truck - 30000 Gal	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%
Cat 385 Hyd Excavator	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%
Cat_980H_Front-end Loader - 5 yd	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%
Cat CS76LT Compactor	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%
Electric Blasthole Drill - 12.25 in	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Diesel Blasthole Drill - 12.25 in	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
DML Highwall Perimeter Drill	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%

Table 16-10: Material Characteristics

In Situ Bulk Density	11.85 cubic feet per ton
Material Swell	30 Percent
Loose Density	15.41 cu ft per ton
Moisture Content	3.5 Percent

16.9 DRILLING AND BLASTING

Production drilling will be done using 12.25-in holes on a 32-foot by 32-foot pattern. Hole depth will be 50 feet with 5 feet of sub-drilling. Subgrade drilling in limestones and skarns may be increased if hard toe conditions are encountered.

Penetration rates vary between 80-91 feet per hour depending on the rock type. The penetration rates are consistent with rates being used by other mines in the area. A factor of 10 percent was used for re-drilling holes or drilling trim holes.

Powder factors varied between 0.27 – 0.34 pounds per ton depending on rock type. Ammonium nitrate and fuel oil (ANFO) blasting agents will be loaded in dry holes, while wet holes will be pumped dry and sleeved before loading with ANFO.

Drills will be outfitted with GPS and electronic sensing units to allow recording of penetration rates in drill holes to assist in determining decking requirements for individual holes. Drill productivities are expected to range between 6,200 and 7,200 tons per hour (tph), depending on rock type.

A blasting contractor will provide explosive emulsion storage, blending facilities and equipment for loading the holes.

16.10 LOADING

Major loading equipment consists of a 36-cy front-end loader, used 25-cy front-end loader, a 46 cy hydraulic excavator and three 60-cy class electric shovels. On average, 20% of material will be handled by FELs and 80% by electric shovels. All stockpile rehandle will be done with FELs.

The equipment was selected to work a 50-foot bench height and load 260- to 400-ton-class trucks. For this study, the 260-ton-class trucks were chosen based on economics, but the loading tools are sized for the larger trucks to give the operator flexibility in fleet selection at a later date.

Loading 260-ton trucks with a 60-cy class shovel requires three passes at 30 seconds per cycle, 42 second spot and queuing for a total load time of 2.20 minutes per truck. Loading the 260-ton trucks with 36-cy FEL requires five passes at 42 seconds per pass, a 42-second spot time and queuing time, for total load time of 4.5 minutes.

Loading equipment production rates vary during equipment start up and according to operator training and experience. After reaching a steady state, the 60-cy class shovel productivity will be 5,500 tph and the FEL productivity will be 2,800 tph.

16.11 HAULING

The 260-ton class truck was chosen based on an economic evaluation and support in the region. Main factors influencing the study were fuel burn, tire costs and repair costs. Truck fleet requirements vary from 11 units during start of preproduction to 35 by Year 3. The fleet remains constant after Year 3 until Year 13 when the waste volumes start to decrease 29 units are required. In year 18 the truck requirements decrease to 18 units due to hauling mainly ore to the primary crusher.

An average load factor of 255 tons was used for production calculations for haulage trucks. Table 16-11 gives an annual summary of cycle times and truck productivity, which include operator efficiency factors in early years.

Table 16-11: Haul Cycle Times and Truck Productivities

	Material Movement					Hauler Operating Hours					Average Hauler Cycle Times				Average Hauler Productivity			
Year	Sulphide Ore	Oxide Ore	Rehandle	Waste	Total	Sulphide Ore	Oxide Ore	Rehandle	Waste	Total	Sulphide Ore	Oxide Ore	Rehandle	Waste	Sulphide Ore	Oxide Ore	Rehandle	Waste
	ktons	ktons	ktons	ktons	ktons	hours	hours	hours	hours	hours	minutes	minutes	minutes	minutes	ktons/hour	ktons/hour	ktons/hour	ktons/hour
PP Q1	0	0	0	1,688	1,688	0	0	0	1,745	1,745	0	0	0	18.78	0	0	0	0.97
PP Q2	0	0	0	7,802	7,802	0	0	0	7,810	7,810	0	0	0	18.18	0	0	0	1.00
PP Q3	0	142	0	15,421	15,563	0	112	0	13,543	13,655	0	14.39	0	15.96	0	1.26	0	1.14
PP Q4	508	3,140	0	20,429	24,077	287	2,325	0	19,272	21,885	10.28	13.45	0	17.14	1.77	1.35	0	1.06
PP Q5	1,510	5,365	1	18,577	25,453	873	3,611	0	21,325	25,810	10.51	12.23	7.72	20.86	1.73	1.49	2.35	0.87
YR1 Q1	2,448	6,406	976	19,091	28,921	1,565	5,269	415	20,007	27,255	11.61	14.94	7.72	19.04	1.56	1.22	2.35	0.95
YR1 Q2	5,476	6,301	0	17,699	29,476	4,807	5,616	0	19,134	29,557	15.95	16.19	0	19.64	1.14	1.12	0	0.93
YR1 Q3	6,730	5,438	0	17,562	29,730	5,886	5,371	0	22,514	33,771	15.89	17.94	0	23.29	1.14	1.01	0	0.78
YR1 Q4	6,844	2,529	0	18,470	27,843	6,509	2,370	0	18,661	27,539	17.28	17.03	0	18.36	1.05	1.07	0	0.99
YR2 Q1	6,844	5,554	0	16,445	28,843	6,871	3,874	0	14,735	25,480	18.24	12.67	0	16.28	1	1.43	0	1.12
YR2 Q2	6,844	3,532	0	18,466	28,842	6,212	2,838	0	17,769	26,819	16.49	14.6	0	17.48	1.1	1.24	0	1.04
YR2 Q3	6,844	2,644	0	19,354	28,842	6,691	2,261	0	24,421	33,373	17.76	15.53	0	22.92	1.02	1.17	0	0.79
YR2 Q4	6,844	3,021	0	17,978	27,843	7,152	3,219	0	21,856	32,227	18.98	19.36	0	22.09	0.96	0.94	0	0.82
YR3	27,375	9,629	0	72,369	109,373	25,809	8,124	0	79,017	112,950	17.13	15.33	0	19.84	1.06	1.19	0	0.92
YR4	27,375	3,901	0	78,094	109,370	30,476	3,079	0	81,022	114,578	20.23	14.34	0	18.85	0.9	1.27	0	0.96
YR5	27,375	1,821	0	80,177	109,373	36,893	1,920	0	109,278	148,091	24.48	19.15	0	24.76	0.74	0.95	0	0.73
YR6	27,375	9,758	0	71,241	108,374	33,632	7,773	0	80,137	121,542	22.32	14.47	0	20.44	0.81	1.26	0	0.89
YR7	27,375	0	0	81,997	109,372	32,761	0	0	92,220	124,982	21.74	0	0	20.43	0.84	0	0	0.89
YR8	27,375	0	0	81,996	109,371	37,199	0	0	61,875	99,073	24.69	0	0	13.71	0.74	0	0	1.33
YR9	27,375	0	0	81,995	109,370	39,751	0	0	82,699	122,450	26.38	0	0	18.32	0.69	0	0	0.99
YR10	27,375	0	0	81,500	108,875	38,713	0	0	88,710	127,423	25.69	0	0	21.94	0.71	0	0	0.92
YR11	27,185	0	190	77,000	104,375	38,370	0	81	115,155	153,605	25.64	0	7.72	27.17	0.71	0	2.35	0.67
YR12	27,375	0	0	68,000	95,375	39,238	0	0	124,137	163,375	26.04	0	0	33.17	0.7	0	0	0.55
YR13	27,375	0	0	77,999	105,374	42,610	0	0	105,197	147,807	28.28	0	0	24.5	0.64	0	0	0.74
YR14	27,375	0	0	64,998	92,373	45,456	0	0	90,525	135,981	30.17	0	0	25.3	0.6	0	0	0.72
YR15	27,375	0	0	51,998	79,373	50,437	0	0	89,254	139,691	33.47	0	0	31.19	0.54	0	0	0.58
YR16	27,375	0	0	40,512	67,887	54,195	0	0	79,784	133,979	35.97	0	0	35.78	0.51	0	0	0.51
YR17	27,375	0	0	4,928	32,303	48,233	0	0	10,475	58,708	32.01	0	0	38.62	0.57	0	0	0.47
YR18	27,375	0	0	1,434	28,809	50,227	0	0	3,173	53,400	33.33	0	0	40.2	0.55	0	0	0.45
YR19	27,375	0	0	144	27,519	55,334	0	0	345	55,679	36.72	0	0	43.59	0.49	0	0	0.42
YR20	27,375	0	0	4,369	31,744	62,131	0	0	12,381	74,513	41.23	0	0	51.49	0.44	0	0	0.35
YR21	2,886	0	851	2,525	6,262	7,406	0	362	9,754	17,523	46.89	0	7.72	69.96	0.39	0	2.35	0.26
TOTALS	**546,338**	**69,181**	**2,018**	**1,232,258**	**1,849,795**	**815,724**	**57,762**	**858**	**1,537,930**	**2,412,276**	**27.13**	**15.17**	**7.72**	**22.82**	**0.67**	**1.20**	**2.35**	**0.80**

16.12 SUPPORT EQUIPMENT

Major support equipment includes mine equipment that is not directly responsible for production, but which is scheduled on a regular basis to maintain haul roads, pit benches, WRS areas and to perform miscellaneous construction work as needed. Equipment operating requirements were estimated for this equipment based on the major mine equipment support requirements and WRS slope regrading schedules. Equipment in the mine support fleet includes:

- Crawler dozers, D11 class
- Crawler dozers, D10 class
- Rubber-tired dozers, 834 class
- Motor graders, 16H class
- Water trucks, 30,000 gallon capacities

In general, the rubber-tired 834-class dozers will be used in the pit to clean up around the 60-cy class electric shovels, with the track dozers used for haul road construction, pit development, WRS area management, and final re-grading requirements. The graders and water trucks will be used to maintain roads and control dust.

16.13 EQUIPMENT OPERATING HOUR REQUIREMENTS

Table 16-12 is a summary of operating hour estimates for all major mining equipment.

Table 16-12: Equipment Operating Hours

	PP-2	PP-1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11-15	Year 16-20	Year 21-22
CAT 7495 Electric Shovel #1	1,731	5,830	5,830	5,846	5,830	5,830	5,830	5,846	5,830	5,830	5,830	5,846	29,167	27,993	5,593
CAT 7495 Electric Shovel #2	110	5,830	5,830	5,846	5,830	5,830	5,830	5,846	5,830	5,830	5,830	5,846	27,705	6,697	0
CAT 7495 Electric Shovel #3	0	3,430	5,013	4,570	5,492	5,581	4,502	1,330	3,368	5,594	5,643	4,750	0	0	0
Hydraulic Excavator - 46 yd	2,827	3,758	4,218	3,821	4,636	4,883	4,462	4,482	4,454	3,513	4,534	3,173	19,997	10,654	2,176
LeTourneau FEL L-1850 - 36 yd	2,186	1,198	3,530	651	5,283	4,387	342	3,902	0	5,867	5,950	4,299	16,962	5,310	1,690
CAT 994F FEL - 25cy	2,186	774	0	0	0	0	0	0	0	0	0	0	0	0	0
Cat 834H RTD Dozer	2,263	15,090	16,673	16,262	17,152	17,241	16,162	13,022	15,028	17,254	17,303	16,442	56,872	34,690	5,593
CAT 793F Haul Truck - 250 Ton	33,904	119,269	165,468	166,092	196,246	210,318	209,940	210,397	210,200	210,092	210,219	208,147	914,497	633,624	144,445
CAT D11T Track Dozer	6,016	7,129	8,555	8,983	8,494	8,491	8,349	8,279	8,256	8,256	8,378	8,536	33,176	26,849	5,485
CAT D10T Track Dozer	11,490	17,769	15,038	15,302	15,923	15,923	15,923	15,966	15,923	15,923	15,923	15,966	67,703	51,781	10,579
Cat 16M Motor Grader	5,519	14,778	17,037	17,124	17,031	17,031	17,016	17,054	17,007	17,007	17,019	17,079	68,077	55,308	11,299
Cat 785 Water Truck - 30000 Gal	5,519	15,644	16,112	16,156	16,112	16,112	16,112	16,156	16,112	16,112	16,112	16,156	64,481	52,397	10,705
Cat 385 Hyd Excavator	814	1,341	1,192	1,195	1,192	1,192	1,192	1,195	1,192	1,192	1,192	1,195	4,770	3,876	792
Cat_980H_Front-end Loader - 5 yd	543	1,286	1,277	1,280	1,277	1,277	1,277	1,280	1,277	1,277	1,277	1,280	5,109	4,151	848
Cat CS76LT Compactor	543	473	278	278	278	278	278	278	278	278	278	278	1,389	1,389	369
Electric Blasthole Drill - 12.25 in	25	7,989	12,284	11,162	13,236	12,770	11,201	10,293	10,512	11,476	10,726	10,425	44,905	26,939	5,082
Diesel Blasthole Drill - 12.25 in	12	3,911	6,013	5,464	6,479	6,251	5,483	5,038	5,146	5,618	5,250	5,103	21,981	13,187	2,488
DML Highwall Perimeter Drill	1,110	5,099	6,191	5,628	6,656	6,579	5,684	5,214	5,305	5,997	5,652	5,395	22,627	12,992	2,373
Mechanic Truck	5,284	10,500	22,000	22,060	22,000	22,000	22,000	22,060	22,000	22,000	22,000	22,060	74,836	52,836	8,770
Fuel and Lube Truck -50 ton	3,519	7,000	13,200	13,236	13,200	13,200	13,200	13,236	13,200	13,200	13,200	13,236	52,827	38,527	5,847
Cat 966 Cable Reeler	557	1,536	1,536	1,540	1,536	1,536	1,536	1,540	1,536	1,536	1,536	1,540	5,283	3,771	638
Total Fleet Operating Hours	**86,157**	**249,633**	**327,275**	**322,497**	**363,880**	**376,709**	**366,318**	**362,415**	**362,453**	**373,850**	**373,851**	**366,754**	**1,532,364**	**1,062,971**	**224,770**

16.14 MINE PERSONAL REQUIREMENTS

Mine supervision and technical staff requirements over the life of the mine are shown in Table 16-13. Staff requirements are 37 during the start of preproduction, building up to 45 at the end of preproduction and remaining constant throughout the mine life.

Mine hourly requirements are shown in Table 16-13. Hourly staffing requirements are 220 in Year -1 of preproduction. In year three we approach 300 hourly employees and remain at this level through year 10. After year 15 we average 180 hourly employees until the end of mine life.

Hourly mine operation personnel requirements are calculated based on equipment operating hour requirements plus 1.5 hours of non-operational time each 12-hour shift. Maintenance personnel are calculated based on estimated maintenance repair time for mining equipment. The percent of maintenance to total hourly personnel averages 35% throughout the mine life.

Table 16-13: Hourly and Salary Employee Count

Craft Workforce		PP-2	PP-1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11-16	Year 16-21	Year 21-23
Production																
Shovel Operator	no.	4	14	15	15	15	15	15	15	15	15	15	15	12	10	6
Loader Operator	no.	4	2	3	1	4	4	4	4	0	4	4	4	4	2	2
Equipment Oper Grader	no.	3	4	6	6	6	6	6	6	6	6	6	6	6	6	6
Equipment Oper RTD	no.	1	12	12	12	12	12	12	12	12	12	12	12	8	6	2
Equipment Oper Dozer	no.	10	15	14	14	14	14	14	14	14	14	14	14	12	9	7
Equipment Oper WaterTrk	no.	3	9	9	9	9	9	9	9	9	9	9	9	8	6	5
Equipment Oper Driller	no.	1	10	14	13	15	15	13	12	12	14	13	12	11	6	4
Equipment Small FEL Oper	no.	1	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Truck Driver	no.	20	70	97	97	114	122	122	122	122	122	122	120	107	74	64
Total Operators	no.	**47**	**138**	**172**	**170**	**191**	**199**	**197**	**196**	**193**	**198**	**196**	**194**	**168**	**120**	**97**
Maintenance																
Mechanic A	no.	7	26	31	29	32	32	31	30	27	31	30	30	29	20	12
Mechanic	no.	14	45	56	56	62	65	65	65	65	65	65	64	50	36	30
Servicer	no.	5	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Laborer	no.	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Total Maintenance	no.	**27**	**82**	**97**	**95**	**104**	**107**	**105**	**105**	**102**	**106**	**105**	**104**	**89**	**66**	**51**
Total Variable Craft Workforce	**no.**	**74**	**220**	**269**	**265**	**295**	**306**	**303**	**301**	**295**	**303**	**301**	**299**	**257**	**185**	**148**
Salaried Employees	**no.**	**37**	**45**	**45**	**45**	**45**	**45**	**45**	**45**	**45**	**45**	**45**	**45**	**45**	**45**	**45**
Total Mine Operations	**no.**	**111**	**265**	**314**	**310**	**340**	**351**	**348**	**346**	**340**	**348**	**346**	**344**	**302**	**230**	**193**

Note: The mechanic count includes required electricians and welders.

17 RECOVERY METHODS

Sulfide ore will be transported from the mine to the primary crusher by off-highway haulage trucks then conveyed to the concentrator facility. Copper concentrate produced at the concentrator facility will be loaded into highway haul trucks and transported to a concentrate smelter and metal refinery. Molybdenum concentrate produced at the concentrator facility will be bagged and loaded onto trucks for shipment to market.

The process selected for recovering the copper and molybdenite minerals can be classified as "conventional". The sulfide ore will be crushed and ground to a fine size and processed through mineral flotation circuits, refer to Figure 17-1. The following items summarize the process operations required for sulfide ore:

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

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

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

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

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

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

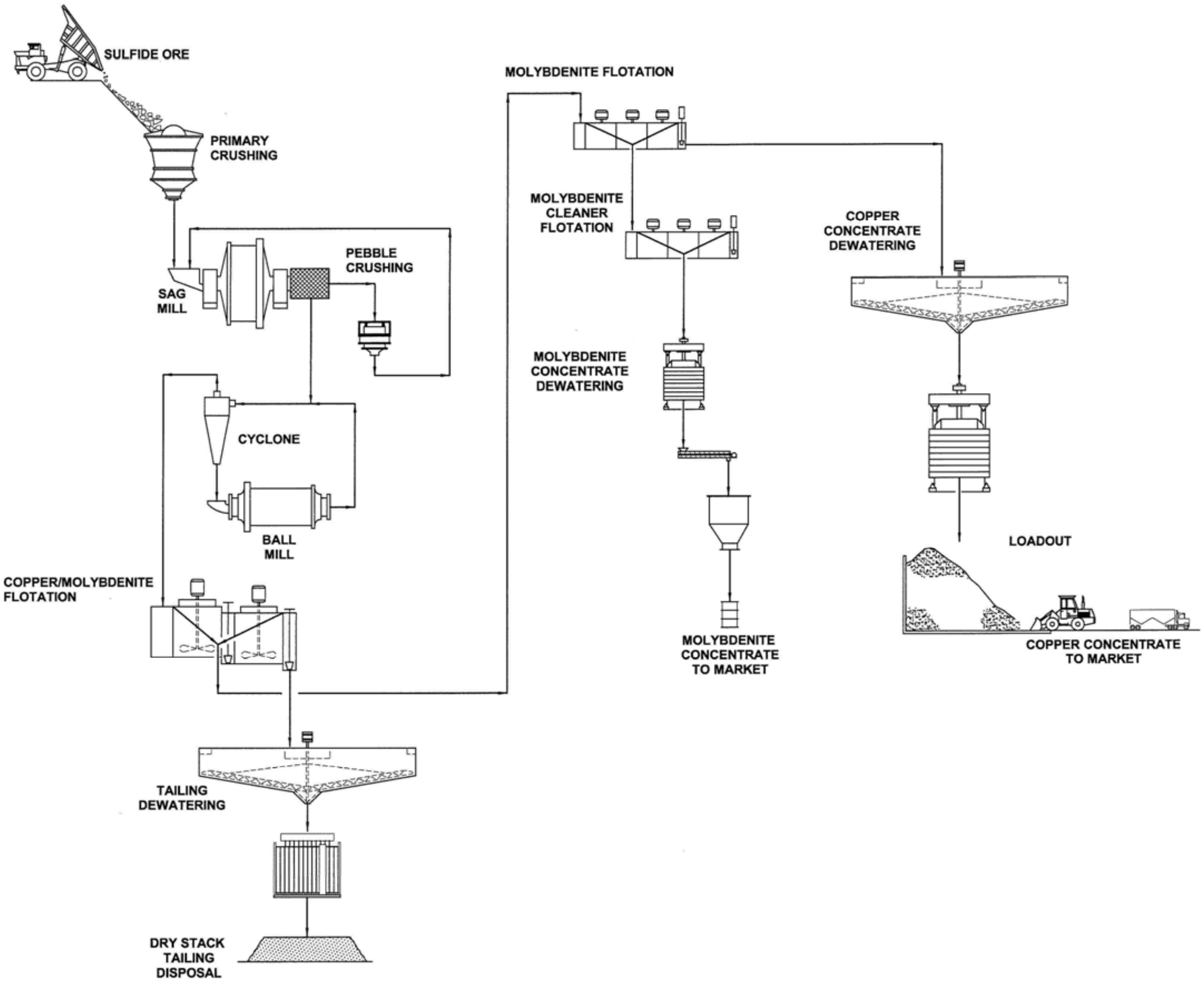

Figure 17-1: Overall Process Flowsheet

18 PROJECT INFRASTRUCTURE

This section of the Feasibility Study Report addresses the infrastructure facilities that will support the Rosemont mine and processing facilities. The infrastructure facilities include the access roads into the plant site, source of electrical power and power distribution, source of fresh water and water distribution, waste management, dry stack tailings storage facility, waste rock storage area, transportation and shipping, communications, and mobile equipment. Also included is a discussion of the geotechnical drilling and sampling program for foundation design.

18.1 ACCESS ROADS AND PLANT ROADS

Access and plant roads consist of a main east access road into the plant from State Highway 83, a secondary west access road over the Santa Rita Mountain ridge to Sahuarita, in-plant roads, haul roads and a perimeter road around the toe of the waste rock and dry stack storage areas. The plant and access roads are shown in Figure 18-1 and described below.

18.1.1 Main (East) Access Road

The primary access road to the property starts at State Highway 83 at a point between mile markers 46 and 47 and ends at the main guard house at the entrance to the plant. The main access road will be designed for 35 miles per hour traffic and consist of two lanes, one in each direction. Each lane will be 14 feet wide plus a four foot shoulder, providing a 36 foot wide road bed. A collection ditch will be provided on each side of the roadway, typically four feet deep with side slopes of 2:1, resulting in an eight foot wide channel on each side for collecting and directing rain water. The total road corridor with collection ditches is nominally 52 feet wide. The access road will be crowned in the center with the surface sloped 2% to each side. The road surface will be paved. The minimum easement for the access road on level ground will be 68 feet and wider where cut and fill toe lines extend beyond the minimum distance.

The intersection of the access road with State Highway 83 is at a location that provides stopping sight distance in excess of 2,000 feet in each direction along SR 83. State Highway 83 will be modified to provide safe ingress and egress from the access road in compliance with ADOT and AASHTO standards. Modifications will include an approximate 1,200-feet northbound acceleration lane, 12 feet wide, leaving the access road going north to safely accelerate to highway speed before merging with traffic. The improvements will also include an approximate 465-feet northbound left turn lane to facilitate access to the site from the south without impeding thru traffic. An approximate 475-feet southbound right turn lane, 12 feet wide, will also be provided on SR 83 approaching the intersection to allow traffic from the north to access the site without impacting thru traffic.

A total of five alternative alignments were evaluated for the original feasibility study completed in July 2007 before selecting the final alignment. The criteria for selecting the final alignment was to minimize sharp curves and switchbacks, maintain grades of 8% or less, minimize the distance to the plant, and balance cut and fill requirements. Two alternatives necessitated grades of 10% for some distances and two alternatives had sharp curves or switchbacks. Although the selected option was not the lowest cost, it did maintain grades of 8% or less and eliminated sharp turns to maximize safety. The length of the previously selected alignment was 3.7 miles from the State Highway intersection to the plant entrance. Additional road alignments were considered to further optimize the design and reduce costs where possible. A new alignment has been selected which meets original design objectives but results in a straighter, shorter road and fewer drainage crossings. Although earthwork volumes increase, elimination of several large drainage structures will offset these costs. This new alignment will be 3.3 miles long, resulting in less overall disturbance while also reducing disturbance of floodways and riparian areas.

18.1.2 Secondary (West) Access Road

A secondary access road is being provided to the west, over the ridge of the Santa Rita Mountains and then continuing along a service road that parallels Santa Rita Road. This west access road is considered a secondary access for plant maintenance employees to access the fresh water pump stations, pipeline, and transmission line. The design for the secondary access road is based on one 14 foot wide lane with guardrail or gabion as required. Existing Forest Service roads will be used as much as possible.

18.1.3 In Plant Roads

In plant roads are generally 24 feet wide with 5 foot drainage channels provided as required along both sides of the road. In plant roads extend from the plant entrance both through and around the perimeter of the process facilities and along the crushed ore conveyor to the mine truck shop. Secondary access roads leave this perimeter road to serve the crushed ore stockpile, the main substations, and water storage tanks. All traffic on plant roads will be right hand traffic until the mine truck shop. At this point, traffic becomes left hand drive to accommodate haul trucks in the area. Access to the powder magazines south of the mine and west of the waste rock stockpile will be from mine haul roads running from the primary crusher southwest between the open pit and the waste rock stockpile. This traffic is again left hand drive to accommodate the mine trucks in the area. In plant roads will be paved to reduce dust emissions.

18.1.4 Haul Roads

Haul roads are a minimum 125 feet wide (95 foot roadway surface and 30 foot wide safety berms) plus 10 foot drainage ditches as required. Mine trucks will have the right of way and all plant traffic crossing the haul roads must yield to the mine trucks. Haul roads used for access to and construction of perimeter waste rock buttresses shall be a minimum of 150 feet wide to allow trucks to turn around with the roadway surface.

18.1.5 Perimeter Road

A perimeter road will be provided around the toe of the mine Waste Rock Storage Area and the Dry Stack Tailings Facility along the security fence line. The road will start at the mine access road near the powder magazines at the south end of the pit and will follow the east perimeter of the waste rock and tailings facilities until it joins the main access road at the north entrance to the plant. The road follows the natural grade as much as possible. This road is for security to monitor the plant boundaries and provide maintenance access to the waste rock and tailings facilities.

18.2 POWER SUPPLY AND DISTRIBUTION

Pursuant to the Certificate of Environmental Compatibility (CEC) issued by the Arizona Corporation Commission (ACC) on June 12, 2012, Tucson Electric Power (TEP) will provide the electrical power supply for the Rosemont mine and process facilities. The total connected load for the Rosemont mine and process facilities is estimated to be approximately 126 MW and will require a transmission voltage of 138 kV. The estimated demand load is about 96.5 MW and the estimated operating load is about 92.8 MW.

18.2.1 Transmission Line Route

As part of the CEC application process, Rosemont evaluated five potential routes to connect the proposed Rosemont Substation to the proposed Toro Switchyard, located approximately 3 miles south of Sahuarita Road and 3.5 miles east of I-19 near the Country Club Road and Corto Road alignments. The proposed Toro Switchyard will tap into the existing 138kV transmission line that extends from the South Substation to the Green Valley Substation.

The ACC selected a preferred route that is approximately 13.2 miles long and originates at the proposed Toro Switchyard (Figure 18-2). The route travels east approximately 1 mile and then southeast paralleling Santa Rita Road and the Rosemont water pipeline alignment that is part of the Rosemont mine plan of operations (discussed further below). Near the intersection of Santa Rita and Helvetia roads, the Preferred Route turns northeast and generally follows the Rosemont water pipeline alignment over Lopez Pass on private property to the Rosemont Substation. The water pipeline right-of-way (ROW) would be 30 feet wide and include a 14 to 20-foot permanent access road for construction, operation, and maintenance. When co-located with the water pipeline, the transmission line ROW (100’) would be centered to include the entire water pipeline ROW so that the access road could be shared which would reduce construction disturbance.

Since the completion of the previous updated feasibility study, Rosemont has determined that a separate line for construction power would not be needed. That line, therefore, is no longer part of the proposed project.

18.2.2 Electrical Power Supply Description

The transmission line will be constructed using tubular steel monopole structures. The structures will range between 75 and 150 feet above ground, depending on the span length and terrain. In limited cases, structures could be as tall as 199 feet for site specific clearance issues. The span length between structures will be approximately 750 feet, according to existing conditions and engineering requirements, to achieve site-specific mitigation objectives. The tubular steel pole structures would have a self-weathering finish, and conductors would have a low-reflective (non-specular), dulled finish to reduce visibility.

The Toro switchyard consists of a ring-bus and equipment for voltage compensation. The switchyard provides power to the mine via the 138 kV transmission line. The switchyard also supplies power to the fresh water delivery system booster pump station substation. Distribution to the south well fields is provided via the substation at the booster pump station. TEP provides distribution power to the west and east well field

18.3 WATER SUPPLY AND DISTRIBUTION

18.3.1 Water Supply

The fresh water requirement for the Rosemont facilities is approximately 5,000 acre-feet per year (to as high as 6,000 acre-feet per year) with a peak delivery requirement of 5,000 gallons per minute (gpm) (Stantec 2009). The well fields and water supply pipeline are being designed for this peak demand, and are currently best described in a report by Stantec (2009) entitled *Rosemont Copper Water Supply Project Design Concept Report*. CDM Smith is currently finalizing designs.

Water supplies are limited in quantity and dependability throughout southern Arizona. The most viable source of water supply for the mine is from groundwater from various aquifers in the region. Potential sources of available groundwater identified for the Rosemont Project include: 1) bedrock and/or shallow alluvium aquifers on or near the Rosemont Project; 2) basin-fill deposit aquifers of Cienega Wash drainage basin and/or Davidson Canyon located east and north of the project; and 3) basin-fill deposit aquifers of the upper Santa Cruz basin west of the project.

The groundwater occurring in sedimentary rock units in the Rosemont Project area produce in the range of less than one to several tens of gpm and are considered to be insufficient as a primary source of water supply for the mine. The basin-fill deposits of the Cienega Creek and Davidson Canyon areas may have adequate pumping capacity; however, these areas are considered environmentally sensitive and Augusta has chosen to avoid groundwater pumping in these areas. Accordingly, groundwater resources in the Davidson Canyon and Cienega Creek areas were not considered as potential sources of water supply for the Rosemont Project.

The source of water supply identified for the project is groundwater in the basin-fill deposits of the upper Santa Cruz basin, which lies west of the Rosemont Project and the Santa Rita Mountains. To obtain sufficient quantities of groundwater from the Santa Cruz basin, it will be necessary to install well fields located a few to several miles west of the Santa Rita Mountains where there is a sufficient saturated aquifer depth to provide efficient pumping conditions.

Rosemont Copper has acquired a 53-acre land parcel near Santa Rita and Davis Roads (Sanrita West), a 20-acre land parcel near Alvernon and Dawson Roads (Sanrita East), and a 20-acre parcel near Santa Rita Road and Country Club Drive (Sanrita South, or Pump Station No. 1 site), for the purpose of constructing and operating a production well field for Rosemont water supply (Figure 18-3). Upon ultimate well field development, three wells are proposed for Sanrita West, two wells for Sanrita East and two wells for the Pump Station No. 1 site. One production well, RC-2 (at the Sanrita East site), has already been installed. Additionally, it is anticipated that the third well to be constructed at the Sanrita West property will provide a backup water supply in the event of well or pump failure.

Based on the results of pumping tests at wells E-1 and RC-2, sustainable long-term pumping rates for proposed production wells at the Sanrita West and Sanrita East properties are estimated to be approximately 1,500 GPM and 500 GPM, respectively. Summaries of results of drilling and testing of the E-1 and RC-2 wells are given in Errol L. Montgomery & Associates reports (2007 and 2009a).

Although wells have not yet been drilled or tested at the Pump Station No. 1 property, it is assumed for purposes of this document that wells constructed at this site will produce approximately 1,000 GPM each. Until well(s) are constructed and tested at the Pump Station No. 1 property, the total number and locations of wells should be considered tentative.

Based on the hydrogeologic investigations and well construction in the Sahuarita area, anticipated well production for each well site are summarized as follows:

Table 18-1: Conceptual Summary of Groundwater Well Production

Rosemont Property	Number of Wells	Anticipated Production Rate for Each Well (gpm)	Total Anticipated Production Rate (gpm)
Sanrita East	2	500	1,000
Pump Station No. 1	2	1,000	2,000
Sanrita West	2	1,500	3,000
Sanrita West (Back-up)	1	1,500	1,500
Total (w/Back-up)	**7**	**---**	**7,500**

*Includes existing Well RC-2

For purposes of estimating costs for production water wells, it was assumed that each well would be approximately 1,200 feet deep, would have 12- to 16-inch diameter well casing perforated from about 360 feet to the bottom of the well, and would be completed with a gravel pack in the well annulus.

18.3.2 Delivery System

Figure 18-3 depicts the proposed alignment of the water transmission pipeline. The pipeline begins at Pump Station No. 1 (at the Sanrita South property) and utilizes groundwater from the site storage reservoir. The pipeline alignment will extend east along the north section line of Sections 32, 33 and one-quarter mile of Section 34 (T17S, R14E), then turns southeasterly generally along the Santa Rita Road alignment.

The alignment will typically follow Santa Rita Road in a series of tangent lines between the north section line of Section 34 to section 35 (T17S, R14E). Figure 18-4 provides typical sections of the water supply and power line easements adjacent to Santa Rita Road. The alignment continues along Santa Rita Road between the northeast corner of Section 2 and across Sections 1 and 12 (T18S, R14E). The alignment then continues along Santa Rita Road between Section 7 and across Sections 18, 17, and 20 into section 21 (T18S, R15E). The proposed site for Pump Station No. 2 is located along the east side of Santa Rita Road near the section line between Sections 7 and 18.

The alignment then extends southeast through Section 21, around an unnamed hill, then east across Santa Rita Road, on the other side of the road from the Helvetia Cemetery. The alignment then goes to the east-west centerline of Section 22 (T18S, R15E), east to near the Section 23 section line, then sharply north-northeast to near the historic Helvetia site. The alignment from this point meanders extensively through sections 23 and 24, following the old Pima County pioneered road alignment for much of the length. The proposed alignment passes along the south side of the Helvetia Site Ruins, the north side of the Old Dick Mine hill, and into Section 24 and National Forest Land toward the west side peak of the Santa Rita Mountains. The pipeline will be routed over Lopez Pass near the mid-section of Section 24. The pipeline will discharge to the Rosemont one-million gallon storage tank system, which serves to provide storage and reserve for the operations. The alignment for these last 3 miles of pipeline includes a significant amount of vertical elevation change and the requirement for installation of two pump stations with associated water storage reservoirs.

The water delivery system consists of a 20-inch carbon steel pipeline and four booster stations along the route. The initial booster station will be located at the initial well field (Pump Station No. 1) at an approximate elevation of 2,735 feet and additional pumping stations along the pipe alignment at approximate elevations of 3,430 feet, 4,125 feet, and 4,845 feet. Pumping stations at these elevations will maintain pipeline pressures at reasonable levels. Air/vacuum valve structures will be located at regular intervals (~2,000 feet) along the route.

Each pump station consists of a concrete sump, 4 vertical turbine pumps (three operating and one stand-by) each rated at 1,667 gpm and approximately 500 horsepower, and a 4,000 gallon hydro pneumatic tank to absorb pressure fluctuations in the event of a power outage or equipment failure. The pipeline will be buried between pumping stations and into the plant. The additional well fields will be located on private land along the alignment south and east of the initial site.

18.3.3 Plant Water Distribution

Fresh water will be pumped from the well fields to a one million gallon fresh water and fire water tank located west of the process facilities at an approximate elevation of 5,310 feet. The lower portion of the tank, with an approximate capacity of 300,000 gallons, will be reserve capacity for the fire water system. Fresh water usage for the plant will be taken from the tank above the fire water reserve. Flow of fire water and fresh water is provided by gravity since the tank is located approximately 200 to 300 feet above the process facilities.

Water systems provided include a potable water system, fresh water system, process water system, and fire water system.

The potable water system consists of a potable water treatment package, 10,000 gallon potable water tank and a distribution network delivering potable water by gravity to all ancillary buildings, process facilities, restrooms, and safety showers. Consumption is about 20,600 gallons per day.

The fresh water system consists of the gravity distribution network from the fresh water storage tank to the process facilities requiring fresh water. The fresh water usage is for gland water pump seals, fresh water make-up to the mills, flotation plant make-up, and reagent make-up. The fresh water consumption is about 4.7 million gallons per day, including the potable water.

The process water system consists of a process water pond located south of the process facilities that collects process water from the concentrate and tailings de-watering equipment for recycling to the process. The process water pond is sized to hold three days of process flows plus the 100-year, 24 hour storm event. Storm water runoff from the process facilities will also report to the process water pond. Process water from the pond will be returned to the process in the grinding and flotation circuits. Process water recycled to the grinding and flotation circuits is about 53 million gallons per day.

The fire water system consists of a gravity distribution network from the fresh water / fire water storage tank to a system of hydrants around the ancillary buildings and process facilities. Fire protection hose cabinets are provided in process buildings and sprinkler systems are provided for the administration building, change house, laboratory and warehouse. The fire hydrant system is rated at 1,500 gpm.

18.3.4 Recharge Plan

Although Rosemont is not bound by any law or regulation to do so, Rosemont has committed to recharging 105% of the groundwater used during operations. The recharge will be back into the Tucson Active Management Area, as close to the water well field as possible. As of the end of 2010, Rosemont has recharged 45,000 acre-feet of water from the Central Arizona Project (CAP), which represents approximately eight years' worth of water usage for the Rosemont mine. The intent of the recharge program is to maintain a surplus of inventory storage credits of CAP water prior to pumping groundwater for mineral extraction use. Again, this recharge effort is not a requirement and is solely a result of Rosemont's efforts.

In addition, Rosemont, in partnership with Community Water Company of Green Valley, is working to build infrastructure from the CAP's current terminus at Pima Mine Road to a site much nearer the Rosemont well field. It should be noted that Rosemont is the first groundwater user to replace all of its pumping by recharging the aquifer. Completion of the CWC pipeline and recharge project would not only offset the impacts of Rosemont's pumping, but would also make it possible for other groundwater users to offset the impacts of their groundwater pumping. Again, this action is voluntary and not required by any regulation.

18.4 WASTE MANAGEMENT

Solid waste management is regulated in Arizona by The Arizona Department of Environmental Quality (ADEQ), through delegated authority from the Environmental Protection Agency (EPA). EPA regulations covering solid waste management are in the Code of Federal Regulations (CFR) Title 40 Sections 239 through 282. These regulations are more commonly known as the Resource Conservation and Recovery Act (RCRA). ADEQ statutes covering solid waste management are in the Arizona Revised Statutes (ARS) Title 49 Chapters 4 and 5 in Sections 701 through 973.

Solid waste can be classified as either hazardous or non-hazardous waste. Arizona regulations for hazardous and solid wastes are found in Arizona Administrative Code (AAC) Title 18 Chapters 13 and 18, respectively. The Pima County Department of Environmental Quality (PCDEQ) further regulates hazardous waste through delegated authority from ADEQ and has adopted the federal and state regulations. All handlers of hazardous waste must register with the county and provides reports as required.

18.4.1 Hazardous Waste

Hazardous waste is defined as a solid waste, or combination of solid wastes, which pose a substantial present or potential hazard to human health or the environment when improperly treated, stored, transported, or disposed of, or otherwise managed.

Hazardous waste must be managed from its creation through its disposal, a concept more commonly known as "cradle-to-grave." This concept requires that once a waste is determined to be hazardous it must be managed to eliminate the potential for the waste to enter the environment. There are a number of rules designed to control the generation, transport, and disposal of hazardous wastes. Generators are regulated according to the amount of waste generated per month. Conditionally exempt small quantity generators (CESQGs) generate less than 100 kilograms of hazardous waste per month, small quantity generators (SQGs) generate between 100 and 1,000 kilograms, and large quantity generators (LQGs) generate 1,000 kilograms or more hazardous waste per month. A facility may change generator status depending upon the amount of waste generated per month.

As part of the permitting effort, Rosemont will file for a hazardous waste identification number from the EPA and register as a generator of hazardous waste with ADEQ and PCDEQ. Proper management of wastes should allow Rosemont to have a conditionally exempt small quantity generator (CESQG) of hazardous waste status. However, in the event that it becomes necessary to manage quantities of waste in excess of the CESQG threshold, Rosemont will be in compliance by following all rules associated with proper management of waste on a larger scale.

In general, materials classified as hazardous waste will be shipped off-site for destruction or disposal. Materials such as contaminated greases, unused chemicals, waste paint related materials, and reagent wastes that may be classified as hazardous waste will be shipped off-site. Rosemont plans to dispose of these materials using the most permanent and practicable disposal method available. When off-site disposal is necessary, Rosemont will manage materials as required by RCRA and Department of Transportation (DOT) regulations. All shipments will be properly marked and manifested (using manifests or bills of lading as necessary), and characterizations of waste materials shipped will be available.

18.4.2 Non-hazardous Waste

As part of the on-site permitting effort, Rosemont will file for a private solid waste facility permit for managing non-hazardous waste. This facility will not accept any off-site wastes and will be managed for on-site use only. The primary disposal activities on-site will include, but may not be limited to, demolition and construction debris, non-putrescible materials, and waste from maintenance and operations meeting the definition of inert or non-hazardous such as respirator filters, gloves, boxes, non-recyclable packaging material, air filters, hoses, piping, etc.

The location for this facility is on Rosemont Property and is planned to cover approximately 1.5 acres. The facility will be managed using trenching and cover techniques.

18.5 TAILINGS MANAGEMENT

18.5.1 General

The Rosemont Dry Stack Tailings Storage Facility (TSF) will receive dry tailings from the sulfide ore processing plant at a nominal rate of 75,000 dry tons per day. This material will be stacked behind large containment buttresses constructed from pit run waste rock; consequently, this storage area will be active from late pre-production through the end of the mine's life (presently estimated at approximately 21 years).

The deposition of dry tailings, waste rock and overburden will be initiated with a series of perimeter buttresses and berms that are designed to reduce visual impacts from State Highway 83 and surrounding areas. The staging of these buttresses will also allow reclamation to begin early in the operation. Topsoil will be salvaged from pit and waste rock/tailings storage areas for use as a vegetation growth medium. Waste rock and tailings will be deposited behind, i.e. to the west of the perimeter buttresses and berms during the life of the mine. The dry tailings deposition will incorporate staged waste rock buttresses for screening and to improve mechanical and erosional stability of the tailings.

18.5.2 Location and Design Criteria

A siting study was conducted by Tetra Tech during previous studies to evaluate alternative sites based on defined design and selection criteria, as well as estimated development costs to identify the preferred tailings storage location and disposal method.

Design criteria and objectives for the original dry tailings storage facility included:

- Provision of secure long-term storage of a minimum 500 million tons (Mt), which is sufficient for the ore to be mined and processed during about 19 years of project life at a projected rate of 73,600 tons per day (tpd);

- Location within the immediate general area of the mine (approximately five-mile radius from the proposed mine pit);

- Prevention of airborne release of tailings solids to the environment by provision of dust suppression measures;

- Compliance with all applicable regulations including Arizona Best Available Demonstrated Control Technology (BADCT) standards;

- Creation of a site-specific design that accounts for local factors including climate, geology, hydrogeology, seismicity and vegetation; and

- Establishment of an effective and efficient reclamation program, with a focus on concurrent reclamation.

The results of the study indicated the dry tailings option as the most favorable disposal method. Advantages of the dry tailings stack over conventional tailings is that it eliminates the need for an engineered embankment and seepage containment system, maximizes water conservation and minimizes water makeup requirements, results in a very compact site limiting disturbance to a single drainage, and allows opportunities for concurrent reclamation and provisions for dust control.

18.5.3 Site Conditions and Geology

As previously described by Tetra Tech, the selected dry tailings storage site is located just east of the proposed mill site in Barrel drainage. The tailings facility site is characterized by terrain sloping generally east from the plant area to the Barrel drainage which runs generally north south in this area.

A geotechnical investigation was performed under the direction of Tetra Tech to characterize the site soil and rock conditions and to provide engineering parameters for feasibility design. Site investigation work within the limits of the proposed tailings facility included two geotechnical borings, two test pits and four miles of geophysical survey.

Surface soils within the proposed tailings facility area are comprised primarily of alluvial deposits in the drainages and floodplains. The thickness of the alluvium can range from 20 to 80 feet. Topsoil depths vary from a few inches to three feet across the site based on test pit logs. Bedrock depth varies from 80 feet within the drainages to ground surface within the hills.

The major rock units found within the area are the Mt. Fagan Rhyolite, the Willow Canyon Formation and the Apache Canyon Formation. The Mt. Fagan Rhyolite ranges from a phenocryst-rich ash-flow tuff to a megabreccia and the Apache Canyon Formation is a medium- to thick-bedded sequence of shale, laminated siltstone and fine-grained sandstone. The Willow Canyon Formation is a succession of medium- to coarse-grained, locally granular to pebbly, feldspathic sandstone, argillaceous sandstone and siltstone. A few moderate to small andestitc mafic lava flows flank the northwest edge of the tailings facility.

Insitu permeability testing was conducted in the boreholes using the double packer and falling head method at depths ranging from 19 to 63 feet. The results indicate fairly low permeability surficial soils with values generally ranging from 10^{-4} to 10^{-6} cm/s and bedrock permeabilities for the Apache Canyon Formation in the 10^{-6} to 10^{-7} cm/s range.

18.5.4 Tailings Properties

Laboratory testing of the tailings was performed by Tetra Tech for the original feasibility study. Additional testing was completed by AMEC Earth and Environmental on a broader range of tailing types. The purpose of the laboratory testing was to provide parameters for design and operation of the dry stack TSF. The testing programs included index testing (gradation, moisture/density relationship and specific gravity), capillary moisture retention tests, shear strength, consolidation and permeability tests.

The objectives of the testing program were to provide input parameters for engineering analyses to include slope stability, liquefaction potential, seepage (unsaturated flow modeling), as well as operational parameters for handle-ability and trafficability during transport and placement of the dry tailings.

The proposed disposal method for the dry tailings involves transporting the dewatered tailings to a secure disposal area via conveyor and stacking in relatively small lifts behind previously constructed waste rock buttresses. The dewatered tailings must be handleable for transfer and movement by conveyor. The placed tailings must support the stacking system, with a dozer to be used for spreading the tailings where necessary. Therefore, it was necessary to understand the effective range of moisture contents that would be suitable for the proposed operation. Shear strength is the primary parameter for evaluation of bearing capacity and trafficability of a material. Based on laboratory testing results completed by Tetra Tech on an initial sample of the Rosemont tailings, Tetra Tech concluded that trafficability of the Rosemont tailings will be very poor and likely impossible with the proposed stacking and operational equipment at moisture contents above 16 percent by dry weight. Testing by AMEC showed comparable results to those by Tetra Tech. As a result, the filter plant will dewater the tailings to no greater than 18% moisture content with placed tailings expected to be less than 16%

18.5.5 Dry Stack Stability

As discussed by Tetra Tech in the original feasibility design, the Rosemont tailings dry stack is designed as a low hazard facility with fully drained waste rock placed as buttressing material and low moisture tailings placed in relatively thin controlled lifts. The ultimate tailings mass will exhibit only partially saturated conditions with no excess pore pressure throughout the life of the facility and at closure. The tailings lifts will densify under successive controlled conveyor lift placement operations, which will result in an increase in the lower lift fill strength over time. The tailings facility stability analyses considered the maximum ultimate height at the maximum section through the facility for downstream and upstream stability.

The tailings will be placed in relatively dry state for acceptable handleability during conveyance and trafficability of the tailing surface, which will limit susceptibility to liquefaction under dynamic loading. Based on a liquefaction assessment for fine grained soils, the Rosemont tailings are potentially liquefiable at a moisture content greater than 19 percent by dry weight.

Since the tailings will not be handleable or trafficable at moisture contents above 16 percent, placement of tailings in the storage facility at moisture contents approaching 19 percent is not expected. However, limited higher moisture zones within the tailings mass created by meteoric water may occur. This condition was considered in the stability modeling by applying reduced shear strength to thin layers within the tailings mass at various levels to simulate these higher moisture zones and to evaluate the subsequent earthquake resistance of the facility.

Adequate factors of safety at 1.5 static and 1.1 pseudo-static (OBE earthquake) were obtained from the stability analyses based on the selected parameters and proposed facility configuration.

The Slope stability analyses performed on the outer slope indicate the dry tailings stack operations can be constructed with stable 3H:1V inter-bench slopes and an overall stable slope of approximately 3.5H:1V to a total maximum height of approximately 600 feet.

The potential for discrete liquefied tailings layers encapsulated in the tailings mass does not result in an unacceptable reduction of the factor of safety against slope failure. However, as an added measure against potential deformation of the outer waste rock buttresses constructed on tailings using the upstream method, compaction of the tailings below the waste rock buttresses is recommended. This measure will result in a higher density material, thereby reducing the liquefaction susceptibility of the tailings that form the foundation for subsequent waste rock buttresses.

18.5.6 Hydrologic Modeling

The dry stack TSF is constructed from material that contains moisture in excess of its field capacity. As such, it will drain under the effects of gravity from its placed moisture content to a lower moisture content. Unsaturated flow modeling of the dry stack TSF indicates that the solids will drain from a maximum moisture content of about 18% (as modeled)to a field capacity of about 11%. Seepage modeling performed to date has indicated a maximum of 8.4 gallons per minute for the entire facility, and will gradually decrease, ceasing in about 500 years.

Modeling calculations indicate that infiltration of rain into the Dry Stack Tailings Facility will be negligible, although seepage rates can increase temporarily in response to storm events. The negligible infiltration is a function of the fine-grained nature of the crushed and ground tailings material and the compaction that will occur with placement and facility construction. In addition, the 500-year seepage estimate suggests that any rain that does in fact infiltrate will not appear as seepage for hundreds of years.

Most of the dry stack TSF will ultimately lie above the ultimate groundwater capture zone predicted by the groundwater models. Within this zone any seepage that may occur would ultimately flow via groundwater to the open pit. If the capture zone does not encompass the entire facility it is possible that the seepage will reach the groundwater and flow down gradient. Water entrained in the dry stack tailings that comprises drain-down has been chemically analyzed and modeled and is not expected to affect AWQS at compliance locations, located around the perimeter of the facilities.

Modeling and analytical results indicate that seepage constituent concentrations will be below the AWQS for regulated constituents. This makes sense since flotation separates the vast majority of the sulfide-containing minerals and removes them from the tailings, leaving, in the case of Rosemont, a tailings product that is relatively free of minerals that have the potential to generate acid rock drainage.

18.5.7 Dry Stack Operations

Based on design criteria developed by AMEC, Tetra Tech prepared tailings stacking conveyor sequencing to accommodate the new facility arrangement. Dry tailings will be delivered by conveyor and placed with a radial stacker. A dozer will be used to spread the dry tailings and a vibratory smooth drum compactor will be used to provide sufficient compaction for trafficability of the conveyor and stacker as needed. Additional compaction will be conducted in specified areas to limit the possibility of dynamic (earthquake) liquefaction of the tailings material which may cause instability of the stack.

An initial starter buttress will be constructed with waste rock to accommodate approximately three months of tailings storage. Concurrent tailings and waste rock placement will occur throughout the life of the tailings facility. Waste rock will be advanced ahead of the tailings level in successive lifts using the upstream construction method. The waste rock buttresses will have top widths of 150 feet to accommodate two-way haul traffic and outer slopes generally of 3H:1V with benches to achieve an overall sloped facility of 3.5H:1V.

Dry tailings will be delivered by conveyor from the filter plant and placed with a radial stacker against the starter buttress. A dozer will be used to spread the dry tailings and a vibratory smooth drum compactor will be used to provide sufficient compaction for trafficability of the conveyor and stacker, as necessary.

The tailings material will generally be placed in 25-foot lifts by conveyor and trafficked with a dozer and compacted with a vibratory smooth drum compactor as needed to provide a suitable surface for the conveyor and stacking system. The outer perimeter of each tailings lift will be placed in 5-foot layers and compacted with a vibratory smooth drum compactor as needed to achieve a higher density to provide a suitable foundation for subsequent waste rock buttress construction using the upstream construction method. This will also reduce the potential for liquefaction of the tailings under dynamic (earthquake) loading.

18.5.8 Surface Water Control

Stormwater run-on will be limited by ponding stormwater upstream of the dry stack areas. Stormwater runoff sediments from the waste rock buttresses will be captured in sediment basins located downstream of the tailings stack. During operations, the tailings surface will be sloped away from the waste rock buttresses to limit potential water impoundment against the buttresses. Perimeter ditches will be constructed at the outer edges of the tailings surface and will convey stormwater to evaporation ponds located towards the back of the dry stack TSF. Stormwater collected in the evaporation ponds will be pumped to the PWTS Pond as necessary to limit infiltration of surface water into the tailings mass.

18.6 TRANSPORTATION AND SHIPPING

The Rosemont mine is located about 30 miles southeast of Tucson, Arizona along State Highway 83. Access to the site is from Interstate Highway I-10 between Tucson, Arizona and Benson, Arizona at the intersection of State Highway 83 and then south on Highway 83 about 12 miles to the primary access road. There is no rail service into the plant and all materials arriving and leaving the plant will be by truck. East–west rail service is available at Benson, Arizona, about 30 miles to the east, and the port of Tucson at Kolb Road and I-10, about 24 miles from the site. North–south rail service is also available at Sahuarita, Arizona, about 35 miles to the west. Although a west (secondary) access road is provided from the plant over the Santa Rita Mountain ridge to Sahuarita, all deliveries to the plant are assumed to enter the plant from the east access road and State Highway 83.

Table 18-2 shows the major products and consumables entering and leaving the plant along with the expected quantities and number of trips. A trip is considered as a round trip for one truck entering the plant to pick up or leave a load and leaving the plant empty or with the load. The transportation plan considers the most sensitive times of the day around shift change and early week day mornings with school bus activity on State Highway 83. Van pools for employees and staggered work shifts will be used to reduce the number of trips during these sensitive times of the day.

Table 18-2: Trip Data

Material	Quantity Per Year	Trips/ Week	Trips/ Day	Capacity/ Hour
Copper Concentrate, tons	439,000	352	50	4
Pebble Lime, tons	37,200	33	5	2
SAG & Ball Mill Balls, Tons	19,000	17	4	2
Diesel Fuel, gallons	9,000,000	29	4	2
Ammonium Nitrate, tons	20,075	18	4	1
Miscellaneous Reagents, tons	3,750	5	1	1
Wear Parts & Explosives, tons	3,250	5	1	1
Moly Concentrates, tons	4,670	4	0.8	
Fuels & Oils, gallons	105,000	1		

18.6.1 Copper and Moly Concentrates

Copper Concentrates are by far the highest volume of traffic into or out of the plant, with the exception of employees arriving and departing at shift change. Copper concentrates will be transported by tractor trailers to local smelters in Arizona or to rail sidings for shipment to the west coast for export. The tractor trailers have a capacity of about 24 tons and will be covered by tarp to prevent losses while in route. At an annual production of about 439,000 tons, approximately 352 trips will be required per week or about 50 trips per day, seven days per week. The plant can load about 4 concentrate trucks per hour for shipment which will require 13 hours per day to load and ship the concentrates. Copper concentrate shipments will be scheduled

7 days per week, 24 hours per day. The shipments will be scheduled to avoid the high traffic hours on Highway 83 during early mornings, afternoons, and at shift change.

The facility is estimated to produce about 4,700 tons per year of moly concentrates as a by-product of the copper concentrates. Moly concentrates will be shipped in bags by truck at the rate of about one truck per day, four days per week.

18.6.2 Pebble Lime

Pebble lime is a reagent used for pH control in the grinding and flotation process. Pebble lime will be received from local sources in bulk by special bottom discharge tank trucks with a capacity of about 22 to 24 tons. The pebble lime is pneumatically conveyed from the truck to a storage silo. At an annual requirement of about 37,200 tons, approximately 33 trips will be required per week, or about 5 trips per day. The plant can receive and unload about 2 trucks of lime per hour, which will require about 3 hours per day to receive and unload the pebble lime. Pebble lime receipts can be scheduled 7 days per week during the day and evening and again scheduled to avoid high traffic periods on State Highway 83 and shift change.

18.6.3 SAG and Ball Mill Grinding Balls

SAG and ball mill grinding balls are a major consumable for the grinding area. Grinding balls are available from local sources in Arizona and received in bulk by bottom dump or end dump trucks with a capacity of about 22 to 24 tons. At an annual requirement of about 19,000 tons, approximately 17 trips will be required per week or 4 trips per day. The receipt of grinding balls will be scheduled during the day, five days per week. The plant can receive and unload about 2 trucks of grinding medium per hour, which will require about 2 hours per day for the receipts. Two trucks can be received mid-morning and again mid-afternoon to avoid shift change and high commute periods on Highway 83.

18.6.4 Diesel Fuel

Diesel fuel is a major consumable for the mine haul trucks. It is also used as boiler fuel in the change house. Diesel fuel is available from local suppliers and is received in tank trucks with a capacity of about 6,000 gallons. At a peak capacity of about 9 million gallons per year, approximately 29 trips will be required per week or 4 trips per day to receive the diesel. Diesel receipts can be scheduled seven days per week during the day between shift changes. The plant can receive and unload about 2 trucks per hour, which will require about 2 hours per day for receiving the diesel into storage.

18.6.5 Ammonium Nitrate

Ammonium nitrate is a component used for blasting in the mine. It is received from local sources in bulk by tank truck and pneumatically conveyed into storage silos near the mine. The truck capacity is about 22 to 24 tons. The consumption of ammonium nitrate is about 20,075 tons per year, which will require about 18 trips per week or 4 trips per day based on five days per week receipts. A truck of ammonium nitrate can be received and unloaded into storage in about one hour, which will require four hours per day.

18.6.6 Miscellaneous Consumables

Miscellaneous quantifiable consumables consist of reagents used in the process and wear parts used in the crushing and grinding line. Also included is explosive powder and caps used by the mine. Reagents used in the flotation circuit are Aero 242 collector, Xanthates (SIPX), Frother (MIBC), flocculants, sodium hydrosulfide, sodium silicate, burner oil, Dowfroth, and polyglycol. Wear parts used in the crusher and grinding line include primary crusher liners, pebble crusher liners, SAG and ball mill liners, and regrind mill liners.

The total pounds of reagents used for the flotation plant is estimated to be less than 7.5 million pounds per year (3,750 tons). The estimated quantity of crusher and grinding wear parts is estimated to be approximately 4.1 million pounds per year (2,050 tons). The total quantity of explosives powder and caps is estimated to be approximately 1,200 tons per year. Total miscellaneous reagents and consumables that can be quantified total about 7,000 tons per year or about 135 tons per week. All miscellaneous reagents and consumables are assumed to arrive at the plant site by smaller 10 to 15 ton trucks. This requires about 10 trips per week or two trips per day on a 5 day per week basis.

Consumables such as office supplies, safety equipment, small tools, etc. cannot be quantified; however, is not considered significant to the transportation study.

18.6.7 Miscellaneous Fuels and Lubricants

Miscellaneous fuels and lubricants include gasoline, motor oils, lubricants, and antifreeze. Waste oils and waste antifreeze is also transported out of the plant for recycling. Consumption of all miscellaneous fuels and lubricants is estimated to be about 105,000 gallons per year, including the return of the waste oils and antifreeze. Fuels and lubricants are assumed to arrive at the plant in bulk by tanker trucks of capacities of 2,000 to 6,000 gallons or in barrels by truck. All miscellaneous fuels and lubricants will average about 2,000 gallons per week or one trip per week.

18.6.8 Employees

The work force for the Rosemont mine averages approximately 448 employees over the life of mine. General and administrative employees total approximately 43, all salaried. The mine operations total approximately 45 salaried supervisors and 248 hourly employees. The mill operations total approximately 29 salaried supervisors and 83 hourly employees.

The peak work force for the Rosemont mine is estimated to be about 506 employees in year 4. Approximately 117 are salaried, technical, and administrative employees scheduled five days per week on day shift only. The shift schedule is dependent upon area of responsibility but in general will be between 6:00 am and 3:30 pm and 7:00 am and 4:30 pm Monday through Friday. In addition, approximately 31 maintenance employees work day shift only Monday through Friday. Their scheduled shift is 7:00am to 4:00pm. Another 358 employees will work rotating 12 hour shifts from 6:00am to 6:00pm. One hundred five (105) will work on day shift, 74 on night shift, and 179 will be scheduled off at any one time. The shift change hours will start before 6:00am with the arrival of a new shift. Maintenance employees will start arriving before 7:00am after the night shift employees have left. The administrative and technical employees will start arriving before 8:00am. At the end of the day, maintenance employees will leave at 4:00pm, administrative and technical employees at 5:00pm and the shift operators at 6:00pm.

For the transportation study, it is assumed that van pooling will be provided and average a minimum of 5 per car. This will result in 50 trips arriving and 15 trips leaving in the morning. At the evening shift change, 50 trips will be leaving and 15 arriving. The shift change hours will see the highest volume of traffic into and out of the plant. The remaining time, traffic will be under 10 trips per hour

18.6.9 Safety Evaluation

Access to the plant is by Interstate and State roadways up to the access road into the plant. The access road is 3.3 miles long with no sharp turns and grades of less than 8 percent. The access road is a two lane road with 14 foot lanes and four foot shoulders. Drainage is provided on both sides of the road to control storm water runoff. The road surface will be compacted A.D.O.T. aggregate, class 2, eight inches thick. The road will be designed for 35 miles per hour traffic.

The intersection of the access road and State Highway 83 will be upgraded to include turnout and acceleration / deceleration lanes on Highway 83 approaching from either direction and leaving in either direction. Shift change will be staggered to spread the traffic over a 3 hour period for arriving and departing traffic. Van pools will also be used to reduce the number of vehicles entering and leaving the plant at shift change. Major shipments, such as copper concentrates and sulfuric acid, will be scheduled to avoid shift changes and high volume traffic times for State Highway 83.

18.7 COMMUNICATIONS

Modern mining and industrial plants require a data networking and telecommunication system similar to that found in office buildings and commercial businesses. There are requirements for accounting, purchasing, maintenance, and general office business as well as specialized requirements for control systems. Remote access from other owner locations as well as access to and from the internet is essential. Integration of the data networking and telecommunication systems many times results in constructed cost reductions, lower long term cost of ownership, and the ability to track and sometimes recover costs of the telecommunication system.

The two most common options are to design separate data networking and telecommunication systems or to integrate the two into a common infrastructure. For this project, the proposed approach is to integrate them. Based upon previous historical experience, the anticipated bandwidth required is between 6 and 10 Mbps or approximately 30% of an E3 connection. This bandwidth will be allocated between Internet service and telecommunication services. The service demarcation point and physical media is dependant upon the provider and will typically be either a microwave radio link or fiber optic. The demarcation point will pass through a firewall to provide network security and then into redundant high bandwidth network switches. The switches will then feed a dedicated office system Ethernet network and a dedicated control system network. A single connection with a gateway between the office system and the control system will allow business accounting systems to retrieve production data from the control system.

A voice over I/P (VoIP) phone system will be a part of the office network and VoIP handsets will be used for voice communication. A dedicated server will be provided for setup and maintenance of the VoIP system and for accounting of all long distance phone calling. Handsets will plug into any network connection point. It is anticipated that between 70 and 100 handsets will be required for this facility.

The office Ethernet network will support accounting, payroll, maintenance, and other servers as well as individual user computers. High bandwidth routers and switches will be used to logically segment the system and to provide the ability to monitor and control traffic over the network.

The control system Ethernet network will support the screen, historian, and alarm servers and connect to the Control Room computers as well as the Programmable Logic Controllers and other control systems provided with Ethernet communication capabilities. This system will incorporate redundancy and be designed to minimize traffic and latencies. No phone or user computer will be connected to this system.

A security system has been incorporated into the plant network. Using a dedicated video server and monitors, I/P cameras utilizing Power over Ethernet connections will be plugged into dedicated switches. The security server and workstation will be used to configure the system. Configuration includes determining scan rates, archive rates, pan/tilt/zoom, recording characteristics, and alarming. Security cameras are typically located in store rooms, parking lots, visitor lobbies, warehouses, and areas where sensitive materials are kept.

Internal communications within the plant will utilize the same voice over I/P phone system, which will provide direct dial to other phones throughout the plant site. Mobile radios will also be used by the mine and plant operation personnel for daily control and communications while outside the offices.

18.8 GEOTECHNICAL STUDIES

18.8.1 Geotechnical Study

In August 2006, Rosemont authorized Tetra Tech (previously known as Vector Arizona) to complete a geotechnical investigation on patented claims and fee lands in support of feasibility-level designs of a heap leach pad and associated ponds, dry stack TSF, plant site facilities, a waste rock storage area, and various water management facilities. In March 2008, the Coronado National Forest (CNF) granted approval to complete an additional geotechnical investigation by Tetra Tech on Forest Service land.

The Geotechnical Study Report issued by Tetra Tech in 2007 summarized results from the 2006-2007 geotechnical investigation. The Geotechnical Addendum report issued by Tetra Tech in 2009 presented findings from both the 2006-2007 and 2008 geotechnical investigations. A third report issued in 2009 presented a summary of geotechnical data specific to the plant site facilities and primary access road from both investigations.

The geotechnical site investigations included borehole drilling, test pit excavating, geotechnical logging, surface geology mapping, field penetration and hydraulic testing, review of historic condemnation boreholes, geophysical surveys and laboratory testing of selected samples.

The main objectives of the site investigation work performed included:

- Determine bedrock material properties for foundations, including permeability to support hydrogeologic assessments;
- Determine material properties (including in-situ permeability) of the overburden soils for excavations and structure foundations;
- Provide engineering properties for design of structure foundations; and
- Estimate quantities and quality of proposed borrow materials for facilities construction, including granular soils and rock material.

18.8.1.1 Geotechnical Program

The fieldwork carried out by Tetra Tech included mapping of bedrock outcrops and the extent of superficial materials within the plant site and heap leach areas to supplement published mapping previously performed by others.

A total of 18 miles of seismic refraction profiles were carried out across the site using refraction tomography methods in order to estimate the depth to bedrock and bedrock rippibilty within the proposed facility areas. Six shear wave soundings where completed using refraction microtremeter methods in order to perform liquefaction analysis. Ten magnetic profiles totaling approximately 40,000 feet were conducted across the Plant Site in order to better define the geologic contacts occurring within the area.

A total of 1,704 feet from 29 boreholes and 33 test pits were completed within the proposed facility areas. In-situ permeability testing included double packer tests within the bedrock and falling head tests within the overburden soils. In-situ density testing of overburden soils was completed using Standard Penetration Tests.

Laboratory testing was conducted on soil and rock samples from select boreholes and test pits. Testing was completed at various laboratories and included grain size distribution, Atterberg limits, modified proctor, direct shear tests, point load and uniaxial compression.

More than 70 exploration holes drilled by Augusta and previous owners were entered into a geo-data management and reporting program. This data supplied additional information for use in geologic cross-sections.

18.8.1.2 Investigation Findings

General

Due to the size of the Rosemont site, the overall Project area was broken up into four facility areas. The surficial geology for each area has been mapped and geological and geotechnical cross sections have been generated through each area, generally along the seismic survey lines, with lithological designations of soil, cemented soil/soft rock, rippable bedrock and non-rippable bedrock estimated from seismic velocity zones correlated with borehole and test pit data.

Facility Area 1

This area includes the proposed plant facilities, ore conveyor, tailings filter plant, PWTS Dam, truck shops and administration buildings. The majority of the site is underlain by a series of arkosic to tuffaceous siltstones, sandstones and conglomerates within the Willow Canyon Formation. There is an andesitic mafic lava flow cutting through the southeast portion of the plant site and several deposits of alluvium are within the area.

Test pits completed to the west of the plant site show that there is a thin layer of soil, 1 to 2 feet thick, overlying bedrock. The soil ranges from silty sand with gravel to clay with sand. Rock mass calculations indicate the Willow Canyon Formation can be classified as “Fair Rock.”

Several geological/geotechnical cross sections were developed by Tetra Tech throughout the plant site area. These cross sections show the anticipated geologic units, measured material velocities and building pad elevations proposed by M3 Engineering & Technology (M3). From the 6 seismic lines that intersect facilities in the plant site area, only one facility pad was found to be within non-rippable rock.

Facility Area 2

This area encompasses the previously considered leaching facility and associated ponds, which will be comprised solely of waste rock under the current design plan. The northern portion of the previously considered leach pad and process ponds are underlain by the Willow Canyon and Apache Canyon Formations. Rock mass calculations indicate the rock in these areas can be classified as “Fair Rock.” Given that the heap leach and associated ponds have been removed from the design, rippability of the bedrock is not a consideration.

The alluvium in this area was found to be as thick as 50 feet. The material is predominantly a moderately to weakly cemented sand with silt and gravel. Field penetration testing indicates the overburden materials exhibit increased density with depth.

Facility Area 3

Facilities in this area include the primary crusher, dry stack TSF and north diversion channel. The tailings facilities are mainly underlain by the Willow Canyon, Apache Canyon and Mt. Fagan Rhyolite Formations. There are smaller occurrences of an andesitic porphyry, a mafic lava flow, an extrusive rhyolite and alluvium deposits within the footprint. Apache Canyon and

Willow Canyon rock mass values within the tailings area are believed to be similar those for the nearby heap leach pad and plant site area. Rock mass calculations indicate that the Mt. Fagan Rhyolite can be classified as “Fair Rock.”

The alluvium in this area was found to be as thick as 27 feet. The material is predominantly a moderately to weakly cemented sand with silt and gravel. Field penetration testing indicates the overburden materials exhibit increased density with depth.

Facility Area 4

This area comprises the waste rock storage facility area. The waste rock facility is mostly underlain by the Gila Conglomerate except for deposits of alluvium located within the drainages and associated floodplains. Test pits indicate that the Gila Conglomerate is well to moderately cemented clayey sand with gravel containing trace amounts cobbles and boulders.

The alluvium in this area was found to be as thick as 80 feet. The alluvium material is predominantly a moderately to weakly cemented sand with silt and gravel. Field penetration testing indicates the overburden materials exhibit increased density with depth.

Borrow Materials

The site investigation identified potential borrow materials for use as subgrade and drain rock in the construction of the heap leach pad. Samples of the Gila Conglomerate were collected from test pits near the heap leach site and tested for subgrade suitability. Although the Gila Conglomerate is believed to be suitable subgrade material, it will need some processing to remove large cobbles and boulders prior to use as subgrade fill under the leach facility liner system.

Tetra Tech investigated the Concha limestone and Bolsa Quartzite within the pit limits for potential rock quarrying to provide aggregate material. It is currently anticipated that borrow materials for concrete aggregate required for the initial construction will primarily be from the Concha limestone based on accessibility of the material.

Geotechnical Design Parameters and Analyses

Geotechnical parameters for surface soils and bedrock have been developed from the field investigation and laboratory testing program and include rock mass rating, shear strength and foundation and earth pressure parameters.

Geotechnical analyses have been performed for the various the facility foundations based on preliminary anticipated foundation types and loads for bearing capacity, settlement, slope stability and foundation liquefaction.

General design recommendations are given for cut and fill slopes, site preparation, structural and controlled fill, access roads, erosion and drainage, corrosive soils and seismic design parameters.

18.8.2 Geochemical Study

In 2007 Tetra Tech completed two reports detailing the geochemical testing associated with the Rosemont Project, the Baseline Geochemical Characterization and the Geochemical Characterization, Addendum 1. The scope of this ongoing study is to determine the geochemical characteristics of the waste rock and the tailings materials. The following tasks have been and continue to be performed to support the study:

- The creation and maintenance of a database containing static and kinetic geochemical tests of waste rock collected from core, and tailings collected from metallurgical tests;
- The assessment of geochemical behavior and geochemical risks of the waste rock and the tailings materials; and
- A preliminary mitigation plan for preventing and/or controlling drainage that shows the potential for environmental impacts.

This study is part of a phased approach to quantify and mitigate geochemical risk throughout the planning, construction and operational life-cycles of the Project. The current results of the geochemical analyses are summarized in Rosemont's Integrated Watershed Summary (2012).

18.8.2.1 Waste Rock

The waste rock samples are enriched in copper, antimony, molybdenum, selenium, cadmium, arsenic, zinc, gold, lead and silver when compared to average concentrations in the earth's crust. The waste rock characterization is based on coarse reject samples collected from site core. Where possible, samples were composited over a 50 foot interval to simulate the bench height of the pit wall and/or the waste rock storage area. In total, 226 samples have been analyzed to determine the potential geochemical behavior of the waste rock.

Acid-Base Accounting (ABA) testing performed on the waste rock materials showed that 73% of the samples characterized to date meets the "inert" definition set forth in the Arizona Department of Environmental Quality (ADEQ) draft policy titled "Policy for the Evaluation of Mining Rock Materials for the determination of Inertness" (ADEQ, 1998). However, metals such as zinc, arsenic, and selenium can be mobile at alkaline pH values so even material defined as inert may require additional testing.

Kinetic testing was focused on the 27% of the waste rock with uncertain and potentially acid generating materials. At the conclusion of the 20 week testing cycles, the majority of the leachate analyzed from the waste rock resulted in most metal concentrations being below the Aquifer Water Quality Standards.

A portion of the samples have been characterized as potentially acid generating or moderate/uncertain acid generation behavior (Price, 1997). These samples are isolated to the Andesite, Arkose, and Bolsa materials. A review of the Open Pit cross sections suggest that samples characterized as potentially acid generating are isolated to a relatively broad, single, steeply dipping rock unit (Andesite) within the Willow Canyon Formation. A placement strategy within the Waste Rock Storage Area is required to properly blend materials containing abundant acid-neutralizing capacity with the uncertain acid generating materials to prevent stormwater infiltration to the potentially acid generating materials and ensure long-term water quality protection.

18.8.2.2 Tailings

In order to analyze the potential for Acid Rock Drainage (ARD) from the tailings material, ABA testing was completed on tailings samples generated from metallurgical testing. ADEQ Best Available Demonstrated Control Technology (BADCT) Mining Guidance Document specifies the type of testing that should be performed to characterize tailings. The selected tests were specifically designed to meet the ADEQ BADCT manual Tier #1 testing and include ABA testing, leachate analysis, whole rock analysis, and on-going kinetic testing. The results of testing performed to date on the tailings material indicate that they will not generate ARD, metal solubility is low, and it has considerable neutralization potential.

18.8.3 Geologic Hazards Assessment

A geologic hazards assessment was performed to identify potential hazards within the bounds of the Project site, estimate the risk associated with these hazards and present possible mitigation techniques.

The primary hazards identified by this study are rockfall hazards and abandoned mine workings. A large rockfall zone, identified along the western wall of the proposed Rosemont open pit, presents a high rockfall risk to the proposed Project and will require engineered mitigation to prevent rockfalls into the pit. Additional small areas of localized rockfall, many insignificant on the map scale, exist particularly in steeply incised alluvial valleys. The source material of these rockfall areas is not generally bare rock as described above, but rather loosely consolidated deposits where rockfall results from differential weathering between cobbles or boulders and matrix material.

The Project site and the general Project area have been the subject of historic mining activities, although none of these previous activities has matched the scale of the proposed mining project. Prior mining activities ranged from simple, shallow prospector borrow pits no more than a few feet in diameter and several feet in depth to well-developed mine adits that extend tens of feet or more into the subsurface. Additionally, one slag deposit and a historic ore leaching tank exist on the Project site. In order to inventory these previous mine workings, Tetra Tech researched the Abandoned Mine Lands (AML) database available on the World Wide Web through the USGS and conducted a ground survey. Six historic mine workings, absent from the USGS database, were discovered during field investigations. A geologist was also sent to the property to photographically document each AML site and provide a general description of the workings.

Many of the AML sites present negligible risks of mine subsidence hazards to the Project facilities. Other AML sites will require further investigations and possibly active engineered mitigation to reduce risks associated with mine subsidence. Possible mitigation efforts include backfilling, bridging or complete avoidance of AML sites. Classifications of these mine workings, and the efforts necessary to reduce risks, should be handled during final design efforts.

The geologic hazards assessment also estimated the seismic hazard for the Rosemont site in accordance with the Arizona Department of Environmental Quality (ADEQ) published guidelines for mining project design criteria. Seismic hazards for the Project site are defined by the Maximum Probable Earthquake (MPE) and the Maximum Credible Earthquake (MCE) design events. These events are correlated to specific engineered structures based on the level of risk and life of a particular facility. The estimated ground acceleration for the MPE and MCE is 0.045g and 0.326g, respectively. These values were used for design of the various engineered structures.

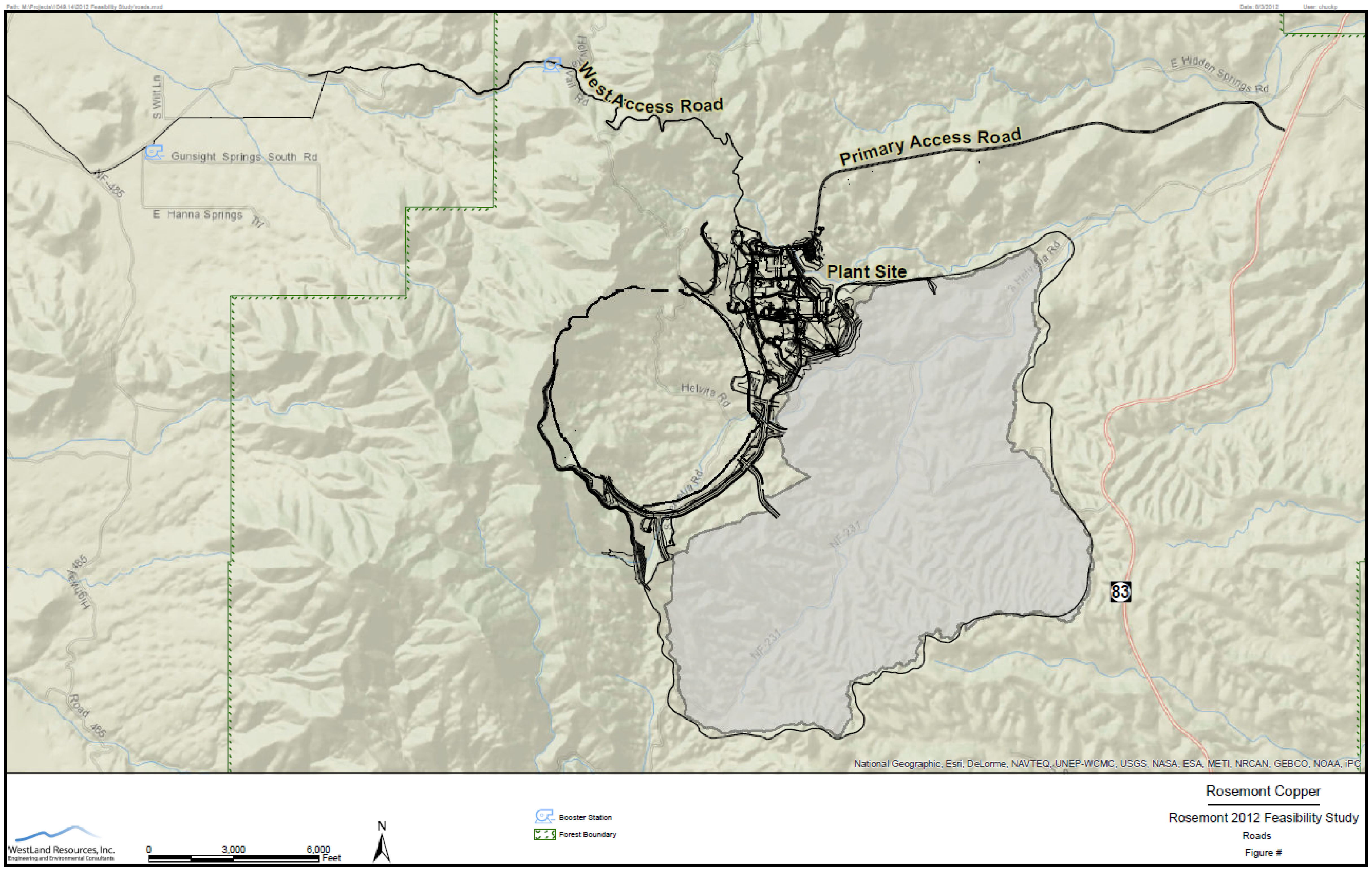

Figure 18-1: Roads

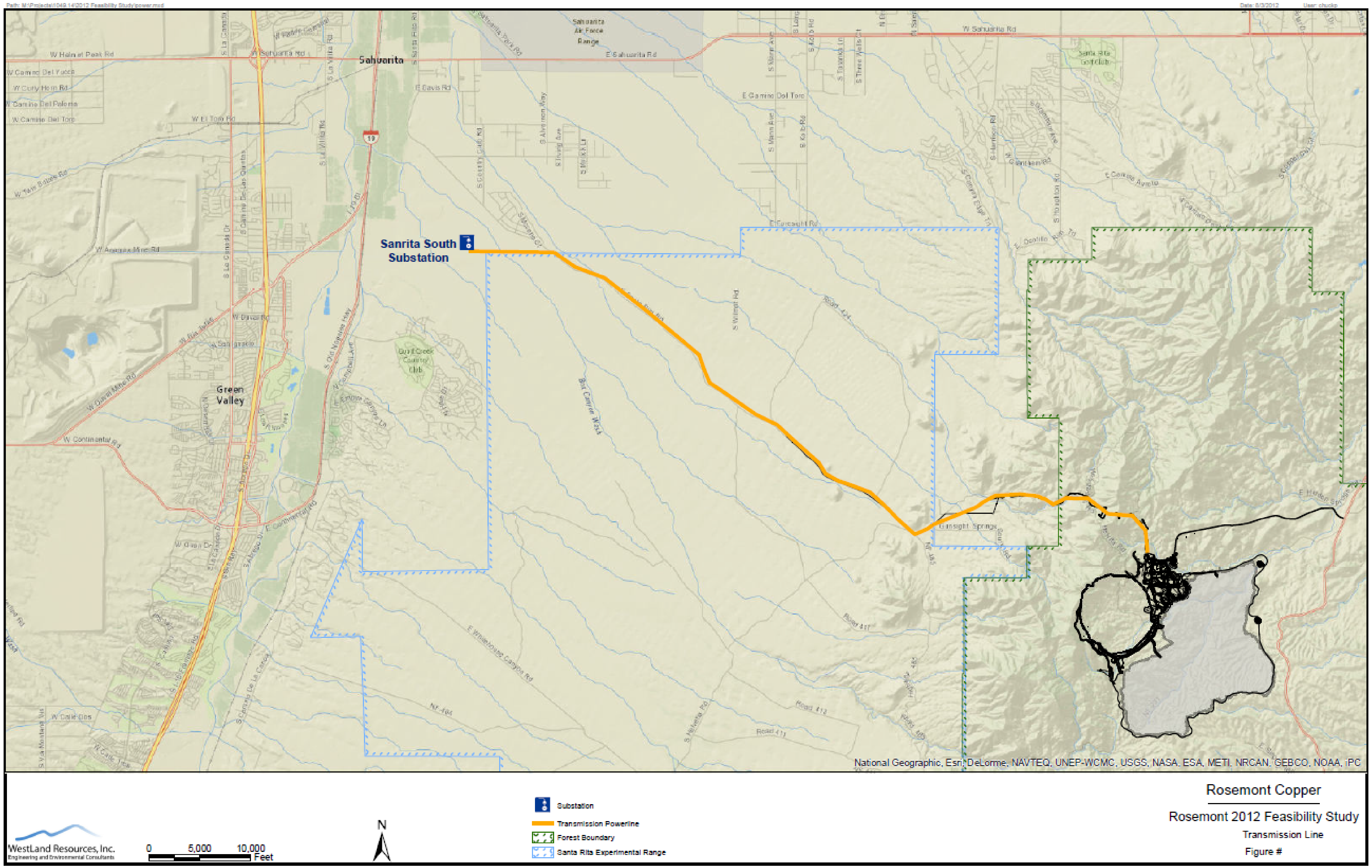

Figure 18-2: Transmission Line

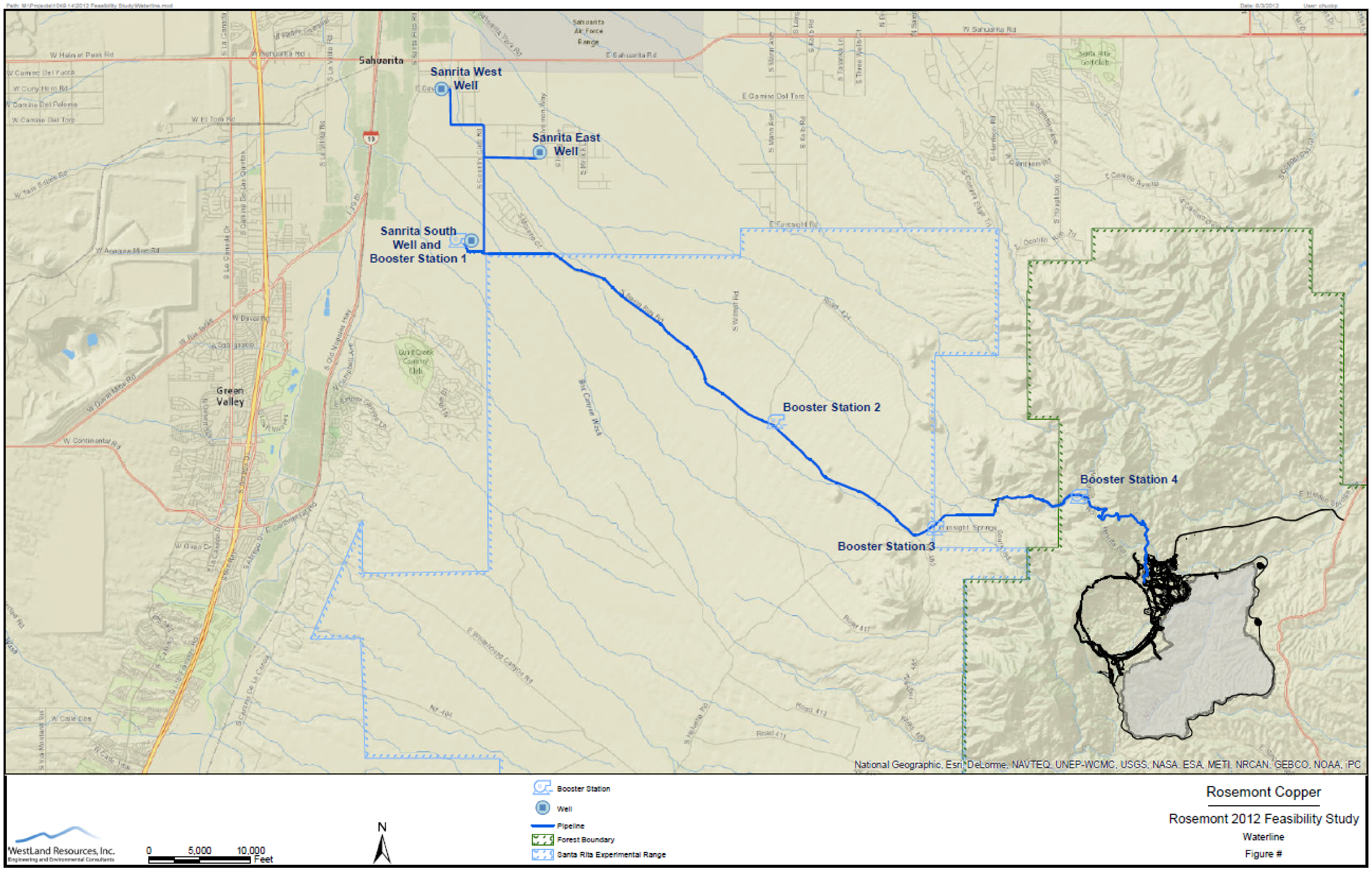

Figure 18-3: Waterline

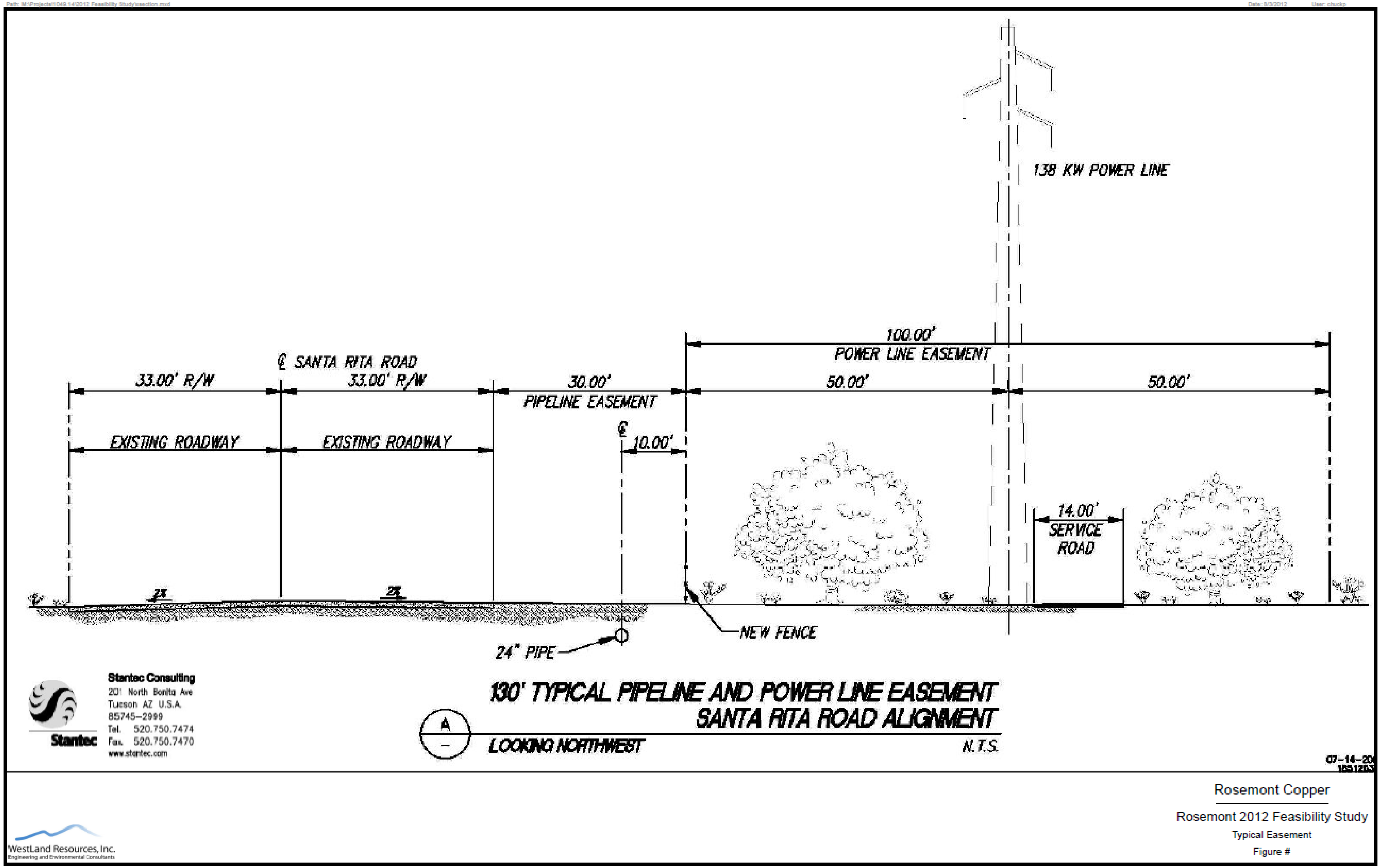

Figure 18-4: Typical Easement

19 MARKET STUDIES AND CONTRACTS

19.1 COPPER CONCENTRATES SALES

Rosemont's copper concentrates are very desirable in the international market because of their very high copper content and very low impurity levels. The average grade of copper concentrates currently traded internationally is about 26% Cu while Rosemont copper concentrates are expected to range between 28-32% Cu over the life of the project, and at the upper end of this range during the initial years of operation.

The following sales terms reflect Rosemont management's view that the global market balance for copper concentrates will remain tight during most of the Rosemont's life because of growing demand in China and the copper industry's challenges in keeping pace with timely new project developments and adequate production rates to meet that demand.

Delivery: CIF Major Asian smelter ports or parity.

Payable Metal Factors:

Copper: 96.5%

Gold: 92.5%

Silver: 92.5%

Treatment Charge: US$63.50 per dry metric ton (dmt) of concentrates.

Refining Charges:

Copper: US$0.0635/lb per payable lb.

Gold: US$5.00 per payable oz.

Silver: US$0.50 per payable oz.

Penalties:

Bismuth: US$2.00/dmt for each 100 parts per million (ppm) Bi above 250 ppm.

Fluorine: US$1.00/dmt for each 100 ppm F above 300 ppm.

Zinc: US$3.00/dmt for each 1% Zn above 3.5%.

Rosemont has placed 50% of its copper concentrates production under long-term sales contracts. 20% of production (up to a maximum of 1.5 million dmt over the life of the agreement) has been sold to Red Kite Explorer Trust. This off-take agreement includes market pricing, competitive payables for metals and international benchmark treatment and refining charges, and is linked to a pre-permitting bridge loan. Another 30% of production for the life of the mine (subject to a minimum of 100,000 dmt per year) is committed to LG International, a member of the Korean consortium that owns 20% of the Rosemont project. This contract is also at market-related sales terms. Rosemont is in active negotiations with several major international smelters in Asia and Europe for the sale of the remaining uncommitted production, and expects to conclude a sales contracts with one of these counterparties.

19.2 MOLYBDENUM CONCENTRATES

Rosemont's typical by-product quality molybdenum concentrates are expected to be sold to the trade at the following representative terms:

Delivery: FCA Rosemont mine.

Payable Metal Factor: 100% of the contained molybdenum.

Price: 89% of the Metals Week Dealer Oxide price (quoted on a $/lb contained Mo basis), subject to a minimum deduction of $1.50/lb contained Mo.

Penalties:

Copper – deduct $0.15/lb contained Mo for each 0.5% Cu >1.0% Cu.

Molybdenum – deduct $01.0/lb contained Mo for each 1% Mo <50% Mo.

Rosemont has sold 20% of its molybdenum concentrates production to its Korean partners as required under the joint venture agreement. Rosemont has been in active discussions with several major molybdenum traders and toll convertors for the balance of the production. Rosemont believes that there is ample conversion capacity in place and under development in the region to process its relatively modest amount of molybdenum concentrates.

19.3 COPPER CONCENTRATE TRANSPORTATION

Augusta Resource Corporation's Rosemont Copper Project is located in Pima County, Arizona about 30 miles southeast of Tucson. The mine access road will connect with Highway 83 about 15 miles south of its intersection with Interstate Highway 10 (I-10). Another road connects Highway 83 and Interstate Highway 19 (I-19), but this road is unsuitable for heavy truck traffic. On the west side of the Santa Rita Mountains are Freeport McMoran's Sierrita Mine near Green Valley and ASARCO's Mission Mine. Both mines use Union Pacific Railroad's Nogales Branch Line to connect the main line at Tucson. It is unlikely that an infrastructure sharing agreement can be reached with either company wherein Rosemont's production could be shipped to market. The facilities at both mines are fully utilized, antiquated and space constrained. It is therefore necessary to develop a transportation plan for the Rosemont Copper Project.

The Rosemont Copper Project expects to produce approximately 400,000 dry short tons of copper concentrate per year. This equates to approximately 435,000 tons per year of wet short (transportable) tons at 8% H_2O. For transportation purposes, concentrate is measured in wet short tons (wst) for domestic movements and in wet metric tons (wmt) for ocean, truck, rail for destinations beyond United States borders. There are four modes of transport amenable to concentrates; the modes are discussed below in order of transport economy from lowest to highest cost.

Pipeline

Concentrates are shipped by pipeline when an absence of infrastructure or rough terrain exists. Space or volume constraints or freight economics may also preclude shipping direct from the concentrator to an intermediate or final destination by more conventional means. It is always advantageous to have a declining route for a pipeline to avoid pumping costs. Pipelines, like electrical transmission lines, railroads, and highways require a right of way. Concentrates are shipped by pipeline in slurry form and dried through filtration at the terminus of the pipeline. Pipelines are characterized by initially high capital costs and low operating and maintenance costs. Capacity of a pipeline is governed by pressure and the size of the pipe as well as the distance to be transported. For the Rosemont Copper Mine, a pipeline is considered to be a potential alternative in the event that more conventional transport methods are unavailable.

Water

Water transport is accomplished by vessel or barge. Vessels are used to traverse long distances for example from the USA to N. China while barges are mostly used on inland waterways and lakes. Some ocean transport by barges is accomplished but generally, distances transported are short. Handysize or Handimax class vessels are the ships most used to transport copper concentrates. They are capable of carrying parcels ranging from 5,000 – 45,000 wet metric tons (wmt). Parcel size is generally limited by the shipper to 11,000 – 22,000 wmt due to the value of the product. Maritime insurance plays a large part in determining parcel size. As metal values decline, parcel sizes increase. Covered ground storage is generally required at the port to avert wind borne losses and to protect the material from rain and snow moisture accumulation. Moisture content in the concentrate must be closely managed to meet vessel flow moisture specifications.

Railroad

Rail transport of the Rosemont concentrates can be accomplished in either bottom discharge hopper cars or solid bottom gondola cars. Hopper cars are most preferred by North American smelters because they are less labor intensive to offload. Most smelters employ vibratory car shakers to assist in offloading the railcars. Moisture content must be closely managed to avoid concentrate sticking to the sidewalls and end plates of the rail cars. Ports can generally offload either type of railcar. Some ports employ machinery that inverts the railcars and dumps the concentrate into a hopper where it is conveyed to a storage facility to await the arrival of a vessel. Shipment by rail affords some convenience advantages over truck shipment in that the cars are offloaded as required. Railcars can be used as temporary storage, albeit at the risk of high demurrage costs while trucks must be offloaded as received. Most railcars in copper concentrate service today are of 90 wmt capacity.

Highway

Truck transport of copper concentrate is used when the destination is within a 200 mile or economic radius of the concentrator. End dump equipment is generally used; however, bottom dump equipment has been used when equipment shortages are present. Truck transport is also used when rail transport is not available or results in a circuitous route that is cost prohibitive. Trucks are often used during periods of railroad disability. Motive power or equipment shortages, derailments, or track outages can interrupt rail deliveries and temporarily shift concentrates to highway transport. This is usually accomplished at much higher cost than afforded by rail but is necessary to ensure production continuity at both concentrator and smelter operations. Truck transportation includes a human factor in that a driver must operate each truck. When delays in loading or offloading occur, drivers must have access to food, shelter, and sanitary facilities. It takes approximately 4 truckloads to equal the capacity of one railcar. Weighing, sampling and assaying and administrative transactions are increased by 300% (compared to rail) when trucks are employed to transport concentrates.

Concentrates produced at the Rosemont mine will initially be shipped by truck direct to smelters within a 200 mile radius or to a railhead in the vicinity of Tucson, AZ. There it will be transferred into railcars for furtherance to more distant North American smelters or to the ports in Mexico for ocean transport to offshore smelters.

Freight rate indications were obtained by the consultant from transportation service providers including railroads, truck lines, ports and port service companies, vessel owners and their agents. The rate indications were used to compile an average freight costs based upon assumptions provided to the anticipated concentrate markets. The rates per wet metric ton are as follows:

Case 1 – 80% N America, 20% Far East via Mexico	$54.63
Case 2 – 50% N America, 50% Far East via Mexico	$81.01
Case 3 – 30% N America, 70% Far East via Mexico	$98.59
Case 4 – 100% Far East via Mexico	$124.95

Current Economic Conditions

High fuel costs resulting in fuel surcharges (FSC) continue to be applied to basic freight rates for ocean, rail and truck shipments continue. The more fuel intensive the method of transport, the higher the fuel surcharge. Truck fuel surcharges are based upon regional fuel prices and fluctuate weekly. These charges are applied to the base freight rate and ranged from 38.5% to 29.6% in the past 18 months or through July 18, 2012. Fuel surcharges on rail shipments are presently assessed at $0.43 per mile transported. If (or when) oil prices stabilize fuel surcharges will be removed. Predictability and consistency of the fuel surcharge is nearly impossible since the surcharge is based upon fuel consumption in varying types of equipment and operating conditions.

For the past 20 years (or more), poor returns and sporadic need have curtailed investment by the railroads in new concentrate cars. There has been a tendency to modify unused coal hopper cars and place them into concentrate service. Therefore, the available car supply is not designed for the product. This results in inefficient material handling, in-transit losses and increased costs at the smelters and ports. Railcars in concentrate service are in short supply primarily due to their age and poor mechanical condition; railroads are insisting that concentrate producers provide the fleets necessary to transport their concentrate. Recent increased copper concentrate production in the southwest have strained the existing railcar supply and considerable volumes of concentrate are moving by truck.

A robust economy plays an important part in freight economics. For many years prior to 2003 the dry bulk cargo section of the ocean freight market experienced excess capacity resulting in depressed freight rates. Growth in coal and iron ore and port congestion, mainly in China, led to increased vessel utilization and increased freight rates. In 2003 and 2004 incentive returns for vessel owners were achieved and many additions were made to the dry bulk fleet. Charter rates are primarily a function of the balance between vessel supply and demand. Rates are also influenced by cargo size, commodity, port dues, and vessel specific factors such as age, speed and fuel consumption. Demand for larger dry bulk vessels is affected by the volume and trade in a small number of commodities. Rates tend to be more volatile in the larger vessels. Copper concentrate represents a very small segment of the bulk cargo market and moves in the smaller drybulk vessels. Consequently, rates for the Handy size and Handimax vessels that transport copper concentrate tend to be less volatile.

Freight buyers continue to enjoy lower ocean rates resulting from declined global economic performance. Vessel owners continue to scrap older vessels and new vessels are being constructed leaving the Handy size fleet at a fairly constant level.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 POLICY, LEGAL, AND REGULATORY FRAMEWORK

The primary purpose of this section is to describe the status of the permits needed to construct and begin to operate the mine. Certain permits, licenses and approvals are also needed during mine operations. These items (such as explosives permits, nuclear instrumentation licenses, hazardous waste identification and tracking numbers, spill control plans and other compliance plans) are not described in any detail in this section because they do not affect the timeline of project permitting and subsequent start up. A list of these items is provided in Table 20-1. Additional background detail regarding the nature of these regulations and permits are provided in Volume II of this Feasibility Study.

20.1.1 National Environmental Policy Act (NEPA)

When a mining project is a federal action, that is, if it requires some type of federal approval, then compliance with the National Environmental Policy Act (NEPA) is required. Under NEPA, a level of environmental analysis commensurate with expected impacts is required. There are three categories of documentation and approval under NEPA: 1) Categorical Exclusion, for small projects which do not exceed predefined impact thresholds; 2) Environmental Assessment (EA), for projects that will result in a finding of no significant environmental impacts; and 3) Environmental Impact Statement (EIS), for projects that will have significant environmental impacts.

For the Rosemont Copper Project, an EIS is being prepared. The steps that have either taken place, are in the process, or are planned for preparing the Rosemont EIS are summarized as follows:

- A Mine Plan of Operations (MPO) was submitted to the Forest Service (Coronado National Forest; CNF) on July 10, 2007;
- CNF determined that the MPO contained enough information to begin NEPA on March 2, 2008;
- CNF posted a notice of intent to prepare an EIS in the *Federal Register* (Vol. 73, No. 50) on March 13, 2008;
- The scoping period opened on March 18, 2008;
- During the scoping period, CNF hosted six public scoping meetings and opened a telephone line for comments in addition to accepting mail and email submissions;
- Scoping ended on July 14, 2008;
- CNF published a Draft EIS (DEIS) made available for public review and comment September 2011;
- The comment review period ended on January 31, 2012;

- During the comment period, CNF received over 25,000 comments submitted by members of the public, elected officials, organizations, and local, state and federal agencies;
- CNF has identified, coded, and organized all substantive comments and is currently developing responses for them;
- As a result of the public comments, CNF requested additional analyses of air quality, water quality and quantity, seeps and springs, riparian resources, dark skies, socioeconomics, and transportation. In addition, CNF requested revisions to a number of earlier plans including the reclamation and post-mine closure plan;
- CNF has posted December 2012 as the completion date for the Final EIS (FEIS); and
- CNF has same as previous for the final Record of Decision (ROD).

As previously described, NEPA has specific requirements with regard to Section 106 of the National Historic Preservation Act (NHPA) and Section 7 of the Endangered Species Act (ESA), which are discussed separately below.

20.1.2 National Historic Preservation Act (NHPA) / Cultural Resources Clearances

For projects such as Rosemont that require a federal permit (or permits), it is necessary to comply with the provisions of Section 106 of the NHPA. Section 106 addresses treatment of cultural resource sites that are eligible for listing on the National Register of Historic Places (Register). In addition, it sets forth requirements for tribal consultation regarding the potential presence of Traditional Cultural Properties (TCPs). The lead federal agency (in this case CNF) must consult with the State Historic Preservation Office (SHPO) and any interested tribal entities.

Register-eligible sites are known to be present in the Rosemont Project area. Section 106 consultation will be formally initiated in August 2012 and will be completed prior to CNF finalizing the ROD.

20.1.3 Endangered Species Act (ESA) and Other Biological Requirements

Plant and animal species that have been determined to be threatened or endangered are the focus of the federal *Endangered Species Act* (ESA), which is administered by the U.S. Fish & Wildlife Service (USFWS). Other federal agencies are required to consult with the USFWS regarding the effects of their actions on listed species, as required by Section 7 of the ESA. Further, the USFWS is required to designate “critical habitat” (specific areas essential for the conservation of the species and requiring special management) for listed species, if prudent and determinable. Once listed, an animal species may be “taken” only if authorized by the USFWS by an “incidental take” permit to do so. Compensation for impacts to species may be required through mitigation, such as protecting habitat for the species at an off-site location. Critical habitat may not be adversely modified by any federal agency action (direct, funded, or authorized) except as permitted by the USFWS.

CNF has formally initiated Section 7 consultation with the USFWS for the Rosemont Project. CNF submitted a biological assessment (BA) prepared in accordance with FS and USFWS requirements on June 7, 2012. The BA identifies the potential for listed or special status species to occur within the project area, and the potential adverse impacts by the project on those species. The USFWS has reviewed the BA and accepted the Section 7 consultation request. The USFWS established a 135-day response period, and anticipates providing a draft Biological Opinion by September 6, 2012 as required by the provisions of ESA Section 7. A final Biological Opinion is scheduled to be issued by October 21, 2012.

20.1.4 Water Permitting

20.1.4.1 Aquifer Protection Permit

Facilities potentially discharging pollutants that may adversely impact groundwater quality are required to secure an Aquifer Protection Permit (APP) from the Arizona Department of Environmental Quality (ADEQ). The APP for the Rosemont Project was issued by ADEQ on April 3, 2012 for discharging facilities under the Phased Tailings Alternative. ADEQ is aware that the APP will need to be modified once the final project design is determined in the CNF's Record of Decision. The permit requires routine quarterly groundwater monitoring at shallow points-of-compliance (POCs) throughout the operational, closure, and post-closure periods to ensure that state aquifer water quality standards (AWQS) are not being violated at these monitoring locations.

20.1.4.2 Clean Water Act Section 402 (AZPDES)

Discharges of process water and stormwater to waters of the US are regulated at the federal level by the EPA under the National Pollutant Discharge Elimination System (NPDES), as outlined in Section 402 of the Clean Water Act (CWA). In December 2002, the EPA delegated this program to ADEQ, which manages these discharges under the Arizona Pollutant Discharge Elimination System (AZPDES).

Currently, the Rosemont Project is operating under an AZPDES Construction General Permit (CGP); however, during mining operations the Rosemont Project will require authorization to discharge stormwater under the AZPDES Multi-Sector General Permit (MSGP-2010) for Mining. In general, stormwater will be managed by diverting unimpacted runoff from adjacent, undisturbed areas through diversion channels around the Rosemont Project facilities to the greatest extent practicable, while runoff from rainfall that falls within the Rosemont Project facilities will be retained and managed. During the development and operational periods, on-site stormwater runoff will be contained within the Project Site in a number of ponding areas that will develop and change along with the mine facilities and infrastructure (CDM Smith, 2012). Rosemont has developed the *Preliminary Site Water Management Plan for the Barrel Alternative* (Tetra Tech 2012) to describe designs for managing storm flows and sediment yield both during the active mine life and as part of the long-term reclamation plan.

In accordance with the requirements of ADEQ's MSGP-2010 program, once the final project design has been determined, a SWPPP will be developed for the Rosemont Project detailing components of stormwater management for the site including the installation and maintenance of practicable BMPS; discharge monitoring locations; applicable surface water quality standards; schedules for inspections, analytical monitoring and reporting; and corrective actions. In addition, Rosemont Copper will submit a complete and accurate NOI to ADEQ prior to any discharges.

20.1.4.3 Clean Water Act Section 404 (Dredge and Fill)

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

The Rosemont Project will result in the discharge of fill material to a network of ephemeral channels comprising the Barrel Canyon drainage. Rosemont Copper submitted a CWA Section 404 permit application for the proposed Rosemont Project on October 11, 2011 (Corps File No. SPL-2008-00846-MB). The Public Notice for the project was published on December 7, 201l, with the public comment period originally set to run through January 5, 2012. At the request of the EPA, the comment period was extended to January 18, 2012. A total of 6,637 comment letters were received during the comment period. Rosemont Copper has provided the Corps with a response to substantive issues raised in the public comments, and the Corps is currently conducting their review of the response. To mitigate the unavoidable loss of jurisdictional waters associated with the project, Rosemont Copper is developing a CWA Section 404 Habitat Mitigation and Monitoring Plan (HMMP) in coordination with the Corps. In addition, the Corps will issue their own NEPA decision document for the Rosemont Project.

20.1.5 Air Permitting

Air quality is regulated at the federal level by the EPA under the Clean Air Act (CAA). Authority for air quality permitting has been delegated by the EPA to the ADEQ.

In the arid southwest, fugitive emissions are a problem if not properly controlled. At the Rosemont Project, a combination of dust suppressants, water, and cover or hooding will be used to manage fugitive emissions from process areas. Capping, seeding, and land management techniques will be used on waste rock piles and storage areas. In addition, captured water from operations and stormwater will be used when and where appropriate to control dust to conserve groundwater resources. Management techniques for operations such as speed control, cleanup, and road maintenance will also be used to conserve resources and manage the potential to create fugitive emissions. Finally, the Plant Site roads will be paved to reduce dust generated by light vehicle traffic in these areas.

For the proposed operation of an open-pit copper mine and associated facilities, Rosemont Copper attempted to secure a Class II air permit from the Pima County Department of Environmental Quality (PDEQ); however, Rosemont Copper determined that it was more appropriate to seek authorization from the ADEQ. The ADEQ asserted complete air quality control authority over the Rosemont Project on August 3, 2012 and has proposed approval of the minor synthetic permit under their revised conditions. ADEQ has established a public comment period for the air quality permit from August 6 through October 9, 2012. In addition, one open house and one public hearing will occur during this period.

20.1.6 Arizona Native Plant Law

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

20.1.7 Pima County Conservation Lands System

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

Table 20-1: List of Agencies and Permits

Agency	Item	Description	Term	Conditions
Federal Permits				
Department of Transportation	Hazardous Materials Transportation Registration	Shipment of hazardous materials	Annual or 3 year renewal	Labeling, packaging, and shipping
U.S. Environmental Protection Agency	Hazardous Waste – RCRA, RCRA ID Number	Waste activities and disposal of hazardous waste	Life	Manifests, reporting, and inspections
U.S. Army Corps of Engineers	CWA Section 404 Permit	Discharge of fill material to onsite washes	5 years	Variety
Mine Safety and Health Administration	MSHA Number	Miner registration number	Life	Operate following MSHA rules
Forest Service	Plan of Operations	Plan for mining operations in the National Forest		Prepare a plan and manage according to the plan, update as required
Forest Service	Closure Plan	Bonding requirements for operations in the National Forest		Prepare a plan and manage according to the plan, updates as required
Forest Service	NEPA Review	Review of major federal action with CEQ oversight		Follow the Record of Decision
Bureau of Alcohol, Tobacco, and Firearms	Blasting Operator Registration	Registration of all personnel that may handle blasting materials	As needed	Background and fingerprint checks of all persons with access, update as required by Federal Agencies
Federal Communications Commission	Radio Licenses for Industrial/Business Pool Conventional Use	Communications equipment must be licensed	10 years	Follow license requirements
State Permits				
Arizona Department of Environmental Quality	Aquifer Protection Permit	Dumps, tailings, stormwater and process water ponds	Life	Inspections, monitoring, maintenance, and reporting
Arizona Department of Environmental Quality	CAA Title V Permit	Mobile and stationary emission sources	5 years	Inspections, monitoring, maintenance, and reporting
Arizona Department of Environmental Quality	AZPDES General Stormwater Permit	Discharge of stormwater	5 years	Delineated in stormwater management plan
Arizona Department of Environmental Quality	Solid Waste Management Inventory Number	Landfill and waste area requirements	Life	Monitoring, maintenance, and operations
Arizona Department of Environmental Quality	Hazardous Waste Management Number	Management of hazardous waste	Life	Monitoring, maintenance, and operations
Arizona Department of Environmental Quality	Waste Tire Cell Registration	Management of off-road tires greater than 3 feet in diameter	Life	Annual reporting, cover requirements
Arizona Department of Water Resources	Groundwater Withdrawal Permits	Groundwater withdrawal rights	20 years	Groundwater withdrawal, Annual reporting required
Arizona Department of Water Resources	Safety of Dams Permit	Requirements for dam construction	Life	Monitoring, maintenance
Arizona Department of Water Resources	Water Storage Permit	Underground storage of CAP water		Annual reporting, storage, and CAP purchase contracts
Arizona State Mine Inspector	Reclamation Plan	Post-mining land uses and plans for regrading	Life	Annual updates
Local Permits				
None	N/A	N/A	N/A	N/A

20.2 BASELINE DATA

Rosemont Copper and its contractors are continually collecting baseline data for a variety of resources in order to support an array of environmental impact analyses associated with project permitting and approvals. A summary of the data collection effort and the findings to date are provided in Volume II of this Feasibility Study.

20.3 ENVIRONMENTAL IMPACTS

As previously stated, a significant portion of the Rosemont Project occurs on the Coronado National Forest (CNF); therefore, the US Forest Service is required to complete a thorough review of the Project's environmental impacts under its NEPA obligations. The development of the Rosemont Project requires the completion of an EIS, because the Project is expected to have significant environmental impacts. The process of assessing impacts and identifying appropriate alternatives and mitigation measures have been ongoing under the direction of the CNF during the development of the EIS. An analysis of environmental impacts was provided in the DEIS published in October 2011. The purpose of the discussion in this section is to briefly identify anticipated environmental impacts that may result from the Project, while a more thorough and complete analysis of environmental impacts will be provided in the Final EIS.

20.3.1 Climate

Effects to climate from the burning of fossil fuels and the attendant release of carbon emissions into the atmosphere are the subject of much global concern. The Rosemont Project is designed to be energy efficient. In addition to conventional electrical power, the mine will generate some portion of its energy needs from solar installations. The project has a compact footprint which will reduce the traveled miles of haulage equipment.

20.3.2 Air Quality

Air impact analyses require specific information from detailed mine planning and final equipment selections. The Rosemont Project is expected to produce regulable emissions of dust and other EPA-regulated criteria pollutants from both point and fugitive sources. The Rosemont Project calls for reductions in base emissions levels through the application of appropriate pollution control technologies and management practices.

Assessments of impacts to air quality have been conducted for the Rosemont Project, and final approval of an air quality permit is pending. ADEQ has asserted complete air quality control authority over the Rosemont Project and has proposed approval of the air permit under revised conditions. A public comment period for the permit began August 6, 2012 and will end on October 9, 2012.

20.3.3 Soils

To the extent possible, soils will be salvaged and stockpiled for later use on reclaimed surfaces. Not all soil resources are salvageable due to topographic and equipment constraints. The total estimated volume of salvage soil in the prospective Barrel Alternative operational areas is approximately 2.8 million bank cubic yards. Additionally, underlying the salvageable soil throughout the site, and specifically underlying the above operational areas, is a substantial amount of unconsolidated and weathered bedrock which may be suitable growth media. The volume of these areas was estimated using a minimum depth of 4 feet. The estimated volume of unconsolidated and weathered bedrock is over 17 million bank cubic yards (CDM Smith, 2012).

Soil will be used as soon as possible in concurrent reclamation activities to avoid long storage times which may reduce seed source, micro-organism, and nutrient viability.

20.3.4 Surface Water

Surface water hydrology elements of concern include conservation of downstream surface water flows and the protection of surface water quality. In general, stormwater from above the mine pit would be diverted around disturbed areas to the extent practicable. Stormwater that falls within the mine pit and associated disturbed areas, will be contained onsite and used for mining and processing purposes. Downgradient sediment control structures will be porous rock-filled check dams located in Barrel Canyon Wash and Trail Creek. The check dams serve as the final point of compliance where stormwater can be monitored. As previously indicated, Rosemont Copper would operate under ADEQ's MSGP-2010 program for stormwater discharges. Details of site water management structures for the Barrel Alternative are provided in a report prepared by Tetra Tech (2012).

In addition, walk-away stormwater controls would be implemented at the closure of the mine. The following sections provide a brief discussion of these elements as presented in the current preliminary reclamation and closure plan for the Barrel Alternative (CDM Smith 2012).

20.3.4.1 Conservation of Downstream Surface Water Flows

At closure, stormwater wrap-a-round channels will route as much stormwater around the facility as practicable and will be located close as possible to the facility toe. At closure, stormwater flow from the reclaimed Plant Site area will report to the toe of the Barrel Alternative Landform ("Landform"; the earthen structure consisting of the Waste Rock Storage Area and the Dry Stack Tailings) once the liner system in the former Process Water Temporary Storage Pond (PWTS) is removed. As currently configured, a maximum ponding depth of 50 feet could occur in this area. In the event that ponding does occur, this maximum ponding depth could be reduced by grading or by partially filling the former PWTS Pond area at closure. It may also be possible to divert some of the former Plant Site area into McCleary Canyon via an intermediate diversion channel. This new diversion channel would be between Permanent Diversion Channel No. 1 and the former PWTS Pond area. An overflow channel from Perimeter Containment Area No. 2 (PCA2) will be constructed to route excess storm flows to Perimeter Containment Area No. 3 (PCA3). The perimeter containment areas are generally located along the south toe of the Landform and a natural ridgeline and function to control stormwater and sediments. Routing of the Pit Diversion Channel to the open pit can also be performed during the closure period to reduce inflow to PCA2.

20.3.4.2 Preservation of Pre-mining Surface Water Quality

The water quality issue of greatest concern at mining operations is the potential for acid rock drainage (ARD). ARD is an acidic (low pH) iron sulfate solution that is derived from the oxidation of sulfide minerals (e.g., pyrite) in the presence of water. If sufficient acid-neutralizing rock types (e.g., limestone) are present, the rate of sulfide mineral weathering or oxidation and leaching is minimized. A total of 226 samples of various rock types from the Rosemont mine area were collected and submitted for acid-base accounting (ABA) tests, 60 for metal leachability testing, and 16 for humidity cell tests (HCT). The mine rock characterization results indicate that a significant majority of the rock types are non-acid generating. Only 6% of all waste rock types were shown to have the potential to produce ARD. Simulated tailings samples were also submitted for ABA tests. The tailings samples consistently showed no potential for ARD formation. Therefore, it is anticipated that stormwater runoff from the waste rock and waste rock buttressing the tailings facility will meet all applicable surface water quality standards when ultimately discharged through the sediment control structures located down-gradient of the Landform toe in lower Barrel Canyon Wash and in Trail Creek.

As previously discussed, the Rosemont Project will be authorized to discharge stormwater under ADEQ’s MSGP-2010 program. The permit will require monitoring and sampling of stormwater at the sediment control structures to ensure that all stormwater leaving the site meets all applicable surface water quality standards. In addition, the Rosemont APP requires that shallow points-of-compliance (POCs) wells be constructed and monitored around the facility to ensure that AWQS are not being violated at these monitoring locations.

20.3.5 Groundwater

Implementation of the Project has the potential to impact both ground water quantity and quality. This section provides a brief analysis of both. More detailed evaluations will be provided in the Final EIS.

20.3.5.1 Groundwater Quantity

As indicated in the DEIS, Rosemont will extract approximately 5,000 acre-feet/year of groundwater for operational use from a well-field near Sahuarita, west of the Santa Rita Mountains. Although Rosemont is not bound by any law or regulation to do so, Rosemont has committed to recharging 105% of the groundwater used during operations. As of the end of 2010, Rosemont has recharged 45,000 acre-feet of water from the Central Arizona Project (CAP), which represents approximately eight years’ worth of water usage for the Rosemont mine. The intent of the recharge program is to maintain a surplus of inventory storage credits of CAP water prior to pumping groundwater for mineral extraction use. Again, this recharge effort is not a requirement and is solely a result of Rosemont’s efforts.

In addition, Rosemont, in partnership with Community Water Company of Green Valley, is working to build infrastructure from the CAP’s current terminus at Pima Mine Road to a site much nearer the Rosemont well field. It should be noted that Rosemont is the first groundwater user to replace all of its pumping by recharging the aquifer. Completion of the CWC pipeline and recharge project would not only offset the impacts of Rosemont's pumping, but would also make it possible for other groundwater users to offset the impacts of their groundwater pumping. Again, this action is voluntary and not required by any regulation.

20.3.5.2 Groundwater Quality

A thorough discussion of the anticipated Project effects to groundwater in the Davidson Canyon watershed is provided in the Integrated Watershed Summary (Rosemont 2012). A summary of that discussion is provided here.

The arid climate of the Project site combined with an abundance of limestone and general lack of sulfide minerals such as pyrite contribute to the protection of groundwater resources. The impact of the Project on groundwater quality is expected to be minimal if at all. Through the use of an industry standard, commonly accepted computer code (VADOSE/W) the Rosemont waste rock storage facility has been estimated to produce no seepage to groundwater. Indeed, with the development of a viable plant community on reclamation, this facility is estimated to lose water during typical climatic conditions. During short, intense storms, or multi-day precipitation events, some water is estimated to infiltrate the surface, but is expected to be removed through evapotranspiration when typical conditions return. The dry stack tailings facility is anticipated to slowly drain down from its placed moisture content to a value similar to its field capacity. The maximum rate of draindown is estimated at 8.4 gallons per minute, diminishing with time until drain down is complete in an estimated 500 years. Even though there is flow expected from the dry stack tailings facility during active mining, this expected flow rate is very small.

Laboratory testing of mine waste rock and tailings has documented material characteristics that, by professionally accepted standards, will not generate acid rock drainage (ARD). Although there is a small amount of mine rock that cannot be classified as inert with respect to generation of ARD, in aggregate the mine rock of the Rosemont project is overwhelmingly acid consuming. Owing to the abundance of acid neutralizing material, and its planned placement in the Project landform, any trace metals present in waste rock are largely insoluble, or attenuated by adsorption onto solid iron oxides. The overall result is water quality that is consistent with very low concentrations of regulated constituents of concern.

Groundwater quality in Arizona is regulated under ADEQ’s APP program which requires that potential discharges to groundwater meet state aquifer quality standards at the designated Point of Complain. Rosemont Copper has secured an APP from ADEQ, and a revised application will be submitted to ADEQ after the ROD is signed and the final project design determined. The Project design will meet or exceed prescriptive BADCT criteria, the accepted industry standards for the highest level of groundwater protection.

20.3.6 Vegetation and Wildlife

Activities associated with the Rosemont Project will result in the direct disturbance of approximately 7,016 acres (SWCA 2012). These impacts will be at least partially mitigated through the mitigation plan described in Section 20.4 of this document, which includes concurrent reclamation.

Impacts to wildlife will result primarily from the direct disturbance of natural habitat. Additional impacts may result from effects to wildlife travel corridors. As indicated previously, the CNF has submitted a final BA submitted to the USFWS initiating Section 7 consultation under the ESA. The BA concludes that the Rosemont Project will not jeopardize the continued existence of any federally listed threatened or endangered species, nor will it result in adverse modification of designated critical habitat. Rosemont has proposed a suite of conservation measures that will reduce the effects of the Project on federally listed species. These conservation measures are described fully in the BA, and the final list of approved conservation measures will be included in the Biological Opinion for the Project, a draft of which is currently scheduled to be provided by the USFWS by September 6, 2012.

20.3.7 Socio-Economics

It is expected that the Rosemont Project will have significant positive economic effects on the region. The results of an economic impacts assessment were recently published in May 2012 (et. al. Arizona State University 2012). The assessment measures the economic impact of the Rosemont Copper Project on employment, labor income, output, gross regional product and tax revenue in Pima County, Arizona, during the project's construction, production, and post-production phases which span a period of 27 years. Estimated impacts include both the direct effects of Rosemont Copper Project operations and multiplier effects that arise when income is recycled within the county's economy. However, as a result of public comments, CNF has requested additional analyses of non-market value socio-economic impacts by the Rosemont Project. A final impacts assessment will be provided in the published FEIS.

20.3.8 Visual Resources

The proposed Rosemont Project would alter the scenic character of the Barrel Canyon watershed and a portion of the viewshed along SR 83, as well as from other observation points. However; Rosemont Copper has proposed a number of mitigation measures for inclusion in the FEIS. These measures include but are not limited to:

- Concurrent reclamation of the outer waste rock and tailings buttress during operations, which will mitigate visual impacts starting in the first year of operations;
- Dust control measures (e.g. spraying water on roadways, water sprays at crushers, etc.) to reduce the visual impact of fugitive dust emissions;
- Evaluation of the potential to artificially oxidize, or weather, the upper benches of the open pit to reduce the contrast of color and tone between the pit wall and the surrounding landscape;
- Development of a limited color palette for the Plant Site area buildings, which will reduce the contrast of color and tone between the buildings and the surrounding landscape, while ensuring worker safety and meeting Mine Safety and Health Administration standards;
- Engaging the University of Arizona College of Architecture and Landscape Architecture to develop a design for the water supply booster stations on the west side of the Santa

Rita Mountains to reduce the visual impacts in the Santa Cruz River valley and on the Santa Rita Experimental Range.

20.3.9 Other Resources (Noise, Light, Recreation)

20.3.9.1 Noise

It is not expected that the Rosemont Project will create noise impacts in excess of background levels due to the operation of vehicles and equipment. Rosemont commissioned a study by Tetra Tech (2009) to assess the effects of noise and vibrations resulting from the Rosemont Project (as described in the DEIS). The Tetra Tech study monitored ambient noise at the Rosemont site and near vicinity. The study concluded that noise levels at an active copper mine were comparable to the ambient noise levels at the Rosemont area, with the exception of the open pit and within 100 feet of active haul roads. Blasting noise could not be detected more than 1.1 miles from the blast site (i.e. the open pit). Maximum construction noise levels generally would not be audible beyond the proposed Rosemont Project boundaries. Operational noise from the Plant Site area and noise form trucks, dozers, graders, and other motorized equipment would also not be generally audible beyond the proposed Project boundaries. When equipment is operating near the outer edges of the Waste Rock Storage Area or the Dry Stack Tailings Facility in areas that are close to or at the elevation of the perimeter buttress, the noise generally would not be audible at the nearest residential areas. In addition, noise from increased traffic would be reduced back to ambient levels as distances 1,500 feet from the highway. The study concluded that the effects of noise on neighboring properties would be minimal to negligible to nonexistent. Tetra Tech prepared a follow-up report in 2010 that showed that the differences in noise impacts for the Barrel Alternative were negligible when compared to the proposed action.

20.3.9.2 Light

It is expected that the Project activities will increase night time illumination in the area due to the 24-hours per day/7 days per week operation of the mine. Rosemont Copper commissioned a study by Monrad Engineering (2012) to develop mitigation opportunities to reduce the effect of lighting at the Project.

Although Pima County lighting codes do not apply to the Rosemont Project, Rosemont, as part of its commitment to best possible environmental practices, will voluntarily employ an advanced light pollution mitigation plan. The plan will include the use of state of the art lighting equipment and controls to minimize environmental impact to levels below the intent of the 2011 Pima County Outdoor Lighting Code (PCOLC), including other comparable modern light pollution control standards. Importantly, the plan must also comply with the project's operational safety requirements prescribed by MSHA. Proposed mitigation measures are as follows:

- Full cut off, solid state Light Emitting Diode (LED) lighting systems.
- High fitted target efficacy (FTE) lighting systems and optics.
- Specific purpose lighting systems with optics that match task requirements.

- Adaptive lighting controls to dim or extinguish lighting when not needed, and to provide immediate 'instant on' emergency or operational lighting.
- Where color rendering is needed, use of color tuned solid state light sources for superior energy efficiency and optical control with attenuated short wavelengths to minimize Rayleigh scattering.
- When color rendering light is not needed, use of narrow band solid state lighting to emulate low pressure sodium (LPS) but with superior optical and electrical control.
- Color adaptive lighting to shift from narrow band amber emissions to higher color rendering light when color rendering is needed.

20.3.9.3 Recreation

The project is expected to affect current recreational patterns in the area. Currently, all-terrain recreational vehicles traverse Barrel Canyon as well as adjacent areas, using both federal and private lands for their overland routes. Other users frequent Barrel Canyon and its surrounds as well. Barrel Canyon will be closed to the public in order to protect public safety and keep recreationists from entering the mine area.

Although impacts to recreation have previously been identified in the MPO and DEIS, CNF is currently conducting a final analysis of potential impacts to recreational opportunities by the Rosemont Project. The final assessment will be published in the FEIS.

20.4 MITIGATION PLAN

Activities at the site will culminate in a large landform which will be a consolidated and contoured earthen structure consisting of the Waste Rock Storage Area and the Dry Stack Tailings termed the Barrel Alternative Landform (Landform). The overall reclamation and closure plan proposed for the Rosemont Copper Project is based on several key components, or initiatives as initially established by the reclamation approach described in the *Reclamation and Closure Plan* (Tetra Tech 2007) and the *Reclamation Concept Update* (Tetra Tech 2010). These initiatives provide the physical and philosophical foundation for the CDM Smith (2012) reclamation and closure plan and will remain constant throughout the operation of the facility. These initiatives are described by the current reclamation plan for the preferred project alternative titled *Preliminary Reclamation and Closure Plan for the Barrel Alternative* (CDM Smith 2012).

In addition, Rosemont has secured seven parcels totaling approximately 4,570 acres to provide compensatory mitigation for the unavoidable loss of potential waters of the US resulting from the proposed Rosemont Project, as well as provide mitigation for other areas. A brief discussion of each of the specific parcels is provided below.

- *Sonoita Creek Ranch*: This parcel contains a total of approximately 1,200 acres of semi-desert grassland, Madrean evergreen forest, and riparian habitat along upper Sonoita Creek, and includes surface water rights that support two perennial ponds and associated riparian vegetation. With a 500+ acre-ft per year water right from the upgradient perennial Monkey Spring, this property provides a unique opportunity to establish riparian or wetland habitat on a portion of Sonoita Creek that has been heavily influenced by historical agricultural activities.

- *Fullerton Ranch*: This parcel contains approximately 1,780 acres of semi-desert grassland in the Sierrita Mountains. Because of the severely degraded, overgrazed condition of this parcel, significant mitigation for waters of the US impacts will be realized through rehabilitation of the xeroriparian vegetation.
- *Helvetia Ranch North*: This parcel contains approximately 940 acres of semi-desert grassland on the west side of the northern Santa Rita Mountains near the proposed Rosemont infrastructure corridor. This property connects lands managed by the BLM with the Santa Rita Experimental Range, and has been observed to support Pima pineapple cactus. This site provides an opportunity to preserve xeroriparian habitat associated with large, braided ephemeral drainages.
- *Rosemont Ranch Lands*: These properties consist of four distinct parcels on the east side of the Santa Rita Mountains totaling approximately 650 acres of semi-desert grassland. All of these parcels include some portion of one of the following significant drainages: Davidson Canyon, Barrel Canyon, and Mulberry Canyon.

All of these parcels are anticipated to provide mitigation for other effects of the Project, including loss of recreation and wildlife habitat.

Additional details of the mitigation plan are provided in Volume II of this Feasibility Study.

21 CAPITAL AND OPERATING COSTS

21.1 INITIAL CAPITAL COST

The capital cost for the Rosemont Copper Project is based on an open pit mine operation treating sulfide ore. Any run of mine oxide ore encountered will be treated as waste and stockpiled. The run of mine sulfide ore will be hauled to a primary crusher and crushed to a nominal size of six inches for further treatment in the grinding and flotation facility to produce a copper concentrate and a molybdenum concentrate product. The crushing, grinding and flotation circuits are based on an initial throughput rate of 75,000 tons per day of new sulfide ore with expectations of increasing production to 90,000 tons per day by the beginning of year 7 of the mine life with additional equipment and optimization of existing equipment.

The initial capital cost estimate is an update of the 2009 Updated Feasibility Study and reflects the status of engineering at the suspension of work in July 2011. It also reflects the deletion of the heap leach pad and solvent extraction facilities; including the tank farm, electrowinning tank house, sulfuric acid receiving and storage facilities, and the leach solution ponds. The dry stack tailing conveyor arrangement was modified to reflect the relocation of the dry stack impoundment area to a location south of Barrel Canyon in accordance with requirements from the US Forest Service.

The capital cost estimate is based on second quarter 2012 Dollars and is considered to be at a ±10% level of accuracy. Actual project costs could, therefore, range from 10% above the estimate amount to 10% below the estimate amount. The estimate accuracy is separate from contingency; which accounts for costs that are expected to be incurred, but which cannot be quantified with the level of information available. No allowance has been provided for escalation, interest, hedging, or financing during construction.

The Owner's costs were removed from the indirect cost in this estimate and carried as a line item in the economic model. The mine pre-production cost, considered an Owner's cost, was also moved from the estimate direct cost and included with the Owner's cost carried in the economic model.

The initial capital cost for the Rosemont Copper Project is summarized by area in Table 21-1 below. The mining capital cost estimate was developed based on the open pit mining equipment requirements. Haul cycles were developed from haulage profiles for the development of the open pit mining operations. The number of haulage and loading units were calculated by combining work schedules with the quantities mined each year. Support equipment such as track dozers, motor graders, and water trucks were estimated to maintain an efficient operation. Rosemont Copper Company (Rosemont) and URS Corporation specified the major mining equipment and ancillary mine equipment for the Rosemont Copper Project. The cost of the various mine equipment was provided by Rosemont Copper Company.

The following costs are based on this project being executed by experienced EPCM contractor(s) in the hard rock mining industry with a recent record of bringing projects on budget or under budget. In addition, it is assumed that all contracts and subcontracts are based on a lump-sum basis or a competitively bid unit cost basis, such as, per cubic yard of concrete placed. In particular, no time and material contracts are anticipated nor should they be allowed in order to ensure this budget is best maintained. In addition, it is assumed that at least two sufficiently sized self-performing local contractors are in place for all trade, such as civil, concrete, steel, architectural, mechanical, electrical, instrumentation and controls, and process piping. Certain contractors will have multiple trade capabilities.

Table 21-1: Capital Cost Summary by Area

Area	Description	Capital Cost
000	Site General	$20,901,243
010	Modifications to Highway 83	$2,051,500
050	Mine	$252,475,842
060	Mine Waste Rock Stockpile	$2,845,058
100	Primary Crushing	$27,233,116
150	Overland Conveyor & Crushed Ore Storage	$19,504,880
200	SAG Feed Conveyors	$14,354,630
300	Grinding & Classification	$143,324,365
400	Copper Flotation & Regrind	$38,335,262
410	Molybdenum Flotation & Regrind	$5,955,877
500	Copper Concentrate Thickening and Filtration	$18,854,379
510	Molybdenum Concentrate Thickeneing & Filtratio	$2,129,194
600	Tailings Dewatering & Stockpile	$175,346,385
650	Fresh Water System	$43,013,672
660	Process Water System	$11,582,261
670	Fire Water System	$2,501,752
700	Main Substation & Power Distribution	$12,667,938
750	Power Transmission Lines	$32,706,094
800	Reagents	$7,829,199
900	Ancillary Facilities	$37,015,115
	Sub-Total Direct Cost	**$870,627,762**
	Indirect Field Cost (Mobilization)	$3,065,779
	Arizona Transaction Privilege Tax	$12,056,808
	EPCM	$83,305,157
	Total Contracted Cost	**$969,055,506**
	Commissioning & Capital Spare Parts	$20,352,288
	Contingency	$51,526,777
	Power Line Gross-up Tax (TEP)	$19,480,369
	Total Evaluated Project Cost	**$1,060,414,940**

21.2 SUSTAINING CAPITAL COST

In addition to the initial capital required for the construction of the project, sustaining capital will be required over the life of the mine and will be funded by the cash flow from operations. Major cost items in sustaining capital include replacement of the mine mobile equipment, additional tailing dry stack tailing conveyors, replacement of mobile equipment for the process facilities, and Owner's sustaining capital. The sustaining capital also includes the additional equipment cost for the ramp up production schedule from 75,000 tpd to 90,000 tpd.

Rosemont Copper Company (Rosemont) and URS Civil Construction and Mining Group specified the major and ancillary mine equipment for the sustaining capital estimate and M3 provided the estimate of sustaining capital for the process facilities. Rosemont provided the Owner's sustaining capital.

A summary of the sustaining capital cost and the anticipated year to be incurred is shown in Table 21-2 below.

Table 21-2: Sustaining Capital

	Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
	Year	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
MINE SUSTAINING CAPITAL	**Totals**																					
Cat 834H RTD Dozer	$3,345,000											$2,230,000	$1,115,000									
CAT 793F Haul Truck - 250 Ton	$82,396,396		$17,912,260	$10,747,356												$25,077,164	$28,659,616					
CAT D11T Track Dozer	$3,850,000															$3,850,000						
CAT D10T Track Dozer	$3,975,000											$3,975,000										
CAT 24M Motor Grader	$0																					
Cat 16M Motor Grader	$2,550,000											$2,550,000										
Cat 785 Water Truck - 30000 Gal	$5,640,000																$5,640,000					
Electric Blasthole Drill - 12.25 in	$9,800,000																$9,800,000					
Diesel Blasthole Drill - 12.25 in	$4,700,000																$4,700,000					
DML Highwall Perimeter Drill	$1,250,000																$1,250,000					
Light Plants	$576,000	$180,000								$108,000		$180,000								$108,000		
Water Pumps	$525,000	$75,000					$75,000	$75,000	$75,000						$75,000	$75,000	$75,000					
5 Year Capital	$11,742,000		$305,000	$1,425,000	$1,425,000				$2,356,500	$1,425,000	$1,425,000					$530,500	$1,425,000	$1,425,000				
3 Year Capital	$7,895,000		$1,645,000	$60,000			$750,000	$600,000	$262,500		$750,000	$600,000			$1,045,000	$660,000			$750,000	$600,000	$172,500	
1 Year Capital	$2,840,000	$70,000	$50,000	$50,000	$570,000	$50,000	$100,000	$70,000	$50,000	$50,000	$70,000	$600,000	$50,000	$70,000	$50,000	$50,000	$120,000	$50,000	$550,000	$70,000	$50,000	$50,000
SUBTOTAL MINE SUSTAINING	**$141,084,396**	**$325,000**	**$19,912,260**	**$12,282,356**	**$1,995,000**	**$50,000**	**$925,000**	**$745,000**	**$2,744,000**	**$1,583,000**	**$2,245,000**	**$10,135,000**	**$1,165,000**	**$70,000**	**$1,170,000**	**$30,242,664**	**$51,669,616**	**$1,475,000**	**$1,300,000**	**$778,000**	**$222,500**	**$50,000**
PROCESS PLANT SUSTAINING CAPITAL																						
600 Area Tailing Conveyors																						
West Mobile Stacker (MSC)	$8,225,950			$8,225,950																		
West Elevating Conveyor	$447,500					$447,500																
West Elevating Conveyor	$362,250							$362,250														
East Mobile Stacker Conveyor	$2,700,000			$2,700,000																		
East Elevating Conveyor	$438,858										$438,858											
East Elevating Conveyor	$636,135												$636,135									
Filter unit	$3,300,000	$3,300,000											$0									
Subtotal Tailing Conveyors	**$16,110,693**	**$3,300,000**	**$0**	**$10,925,950**	**$0**	**$447,500**	**$0**	**$362,250**	**$0**	**$0**	**$438,858**	**$0**	**$636,135**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**
900 Plant Mobile Equipment																						
Boom Truck 10 ton- 45 ft.	$125,000								$125,000													
Boom Truck 15 ton- 60 ft.	$150,000								$150,000													
Front End Loader - 6 yard	$316,216										$316,216											
Front End Loader - 10 yard	$700,000										$700,000											
Front End Loader - 5 yard	$316,216										$316,216											
Bob Cat	$60,000										$60,000											
Bob Cat	$120,000							$120,000														
Fork Lift - 2000 lbs.	$26,000							$26,000														
Fork Lift - 5000 lbs.	$32,000							$32,000														
Fork Lift - 3000 lbs.	$26,000							$26,000														
Fork Lift - 3000 lbs.	$26,000							$26,000														
Track Dozer D9	$960,000											$960,000										
Flat Bed Truck - 10 ton	$75,000							$75,000														
Flat Bed Truck - 10 ton	$150,000							$75,000				$75,000										
Dump Truck - 10 ton	$300,000							$200,000					$100,000									
Pick-up Trucks, 1/2 ton, 4WD	$1,616,500					$185,500	$185,500	$185,500			$185,500	$185,500	$159,000			$185,500	$185,500	$159,000				
Mobile Crane - 60 tons	$475,000												$475,000									
Subtotal Plant Mobil Equipment	**$5,473,932**	**$0**	**$0**	**$0**	**$0**	**$185,500**	**$185,500**	**$765,500**	**$275,000**	**$0**	**$1,577,932**	**$1,220,500**	**$734,000**	**$0**	**$0**	**$185,500**	**$185,500**	**$159,000**	**$0**	**$0**	**$0**	**$0**
Freight	$794,786	$80,850		$432,012		$47,500	$18,356	$25,394	$10,554		$50,420	$6,700	$70,000			$18,550	$18,550	$15,900				
Construction Equipment	$200,490					$36,000					$164,490											
Subtotal Freight & Construction Eq.	$995,276	$80,850	$0	$432,012	$0	$83,500	$18,356	$25,394	$10,554	$0	$214,910	$6,700	$70,000	$0	$0	$18,550	$18,550	$15,900	$0	$0	$0	$0
SUBTOTAL PROCESS PLANT SUSTAINING	$22,579,901	$3,380,850	$0	$11,357,962	$0	$716,500	$203,856	$1,153,144	$285,554	$0	$2,231,700	$1,227,200	$1,440,135	$0	$0	$204,050	$204,050	$174,900	$0	$0	$0	$0
MILL EXPANSION CAPITAL	$96,214,192				$38,485,677	$57,728,515																
OWNER'S SUSTAINING CAPITAL	$16,212,500	$162,500	$60,000	$162,500	$560,000	$550,000	$672,500	$662,500	$560,000	$550,000	$672,500	$550,000	$560,000	$662,500	$672,500	$550,000	$7,272,500	$550,000	$560,000	$162,500	$60,000	
TOTAL SUSTAINING CAPITAL	**$276,090,989**	**$3,868,350**	**$19,972,260**	**$23,802,818**	**$41,040,677**	**$59,045,015**	**$1,801,356**	**$2,560,644**	**$3,589,554**	**$2,133,000**	**$5,149,200**	**$11,912,200**	**$3,165,135**	**$732,500**	**$1,842,500**	**$30,996,714**	**$59,146,166**	**$2,199,900**	**$1,860,000**	**$940,500**	**$282,500**	**$50,000**

21.3 OPERATING COSTS

The overall annual average life of mine operating cost for the Rosemont Copper Project is projected to be $8.05 per ton of ore treated and is summarized in Table 21-3 below. The operating cost includes the mine operations, mill operations, and support facilities. Total tons mined exclude pre-production.

Table 21-3: Life of Mine Operating cost Summary

Sulfide Ore Tons (Processed)	**661,428,205**		
Total Ore Tons	**661,428,205**		
Total Tons - Mined	**1,805,470,000**		
Mine Cost Area	**LOM Cost - $**	**LOM $/Total Tons Mined**	**$/Total Ore**
Mining Operations			
Clear and Grub	$4,834,666	$0.00	$0.01
Topsoil Stacking	$542,666	$0.00	$0.00
Drilling	$93,791,542	$0.05	$0.14
Blasting	$207,450,893	$0.11	$0.31
Loading	$363,100,984	$0.20	$0.55
Hauling	$1,019,798,618	$0.56	$1.54
Roads and Dumps	$257,682,540	$0.14	$0.39
Outside Services	$96,386,494	$0.05	$0.15
Mine Salary Personnel	$182,419,385	$0.10	$0.28
Subtotal Mining	**$2,226,007,788**	**$1.23**	**$3.37**
Processing Operations			
Mill Operations		**$/Sulfide Ore**	**$/Total Ore**
Crushing & Conveying	$113,985,032	$0.17	$0.17
Grinding & Classification	$1,416,938,790	$2.14	$2.14
Flotation and Regrind	$607,240,638	$0.92	$0.92
Concentrate Dewatering & Filtration	$63,336,487	$0.10	$0.10
Tailing Disposal	$527,910,456	$0.80	$0.80
Ancillary Services	$93,134,166	$0.14	$0.14
Total Mill Operations	**$2,822,545,570**	**$4.27**	**$4.27**
Supporting Facilities		**$/Sulfide Ore**	**$/Total Ore**
Laboratory	29,004,914	$0.04	$0.04
General and Administrative	247,703,926	$0.37	$0.37
Total Supporting Facilities	**$276,708,841**	**$0.42**	**$0.42**
Total Operating Cost	**$5,325,262,199**		**$8.05**

21.3.1 Mine Operating Cost

The mine operating costs were prepared under the direction of John I. Ajie – Vice President of Engineering Civil Construction & Mining Group of URS Energy and Construction. The life of mine average mining cost is $3.37 per ton of ore and $1.23 per ton of total material mined. The following factors were used to calculate the operating costs in the mine estimate:

- Costs are in second quarter 2012 Dollars
- A diesel fuel price varied from $2.53 per gallon during the early years to $1.81 per gallon during the latter years
- An electric power rate of $0.062 per kWh
- A delivered Ammonium Nitrate prill cost of $600 per ton
- Equipment operating costs from vendor supplied component replacement schedules and URS's data base for similar projects and equipment.
- Hourly labor and salary labor costs based on similarly sized mines in the Western US and local wage surveys.

21.3.2 Mill Operations

Mill operating costs were developed by M3 Engineering & Technology Corporation. The cost centers for the mill operation include crushing and conveying, grinding and classification, flotation and regrind, concentrate dewatering, tailing dewatering and disposal, and ancillary services. The costs were further distributed by cost elements or labor, electric power, reagents, maintenance parts and supplies and process supplies and services. The life of mine average mill operation cost is $4.27 per ton of ore processed.

Labor costs were developed from staffing plans and hourly labor rates were based on in-house data developed from industry surveys for this region. Salary rates for the mill operations were provided by Rosemont. The annual salaries include overtime and benefits for both salaried and hourly employees. Electric power costs were based on a power rate of $0.062 per kWh and the estimated power demand. Reagent consumption was determined from metallurgical test data or industry practice and recent budget quotes for reagents. Mill wear parts (liners) were based on a SAG mill reline schedule every 6 months and ball mill reline schedule every 18 months. Grinding ball consumption was based on the abrasion index of the ore and power used by the mills. Allowances for maintenance of the remaining facilities were factored from the equipment cost in each area of the facility. Allowances were also provided for process supplies and services based on historical in-house data.

21.3.3 Supporting Facilities

The operating cost for the support facilities was prepared by M3 Engineering & Technology Corporation and includes the analytical laboratory and the general administration departments. The life of mine average operating cost for the support facilities is $0.42 per ton of ore processed.

Labor cost for the analytical laboratory is based on a staff of 11 employees, including a chief chemist, chemists, assayers, and administrative personnel. Power cost was based on the connected load for the analytical laboratory, discounted for anticipated operating time. Annual allowances were provided for reagents, assay consumables, maintenance cost, and supplies and services.

General and administrative costs include labor and fringes for administrative employees, accounting, purchasing, human resources, safety and environmental departments. Also included are office supplies, power allocations, fuel, communications and outside services. Labor cost is based on a staffing plan of 43 employees for all the general and administrative departments. Salary levels for G&A staffing, including the analytical laboratory were provided by Rosemont. All other G&A costs are annual allowances for expenses to run the offices, legal fees, insurance cost, communications and other costs.

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

The financial evaluation presents the determination of the after tax Net Present Value (NPV), payback period (time in years after production commences to recapture the initial capital investment), and the after tax Internal Rate of Return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures, production costs and sales revenue. The sales revenue is based on the production of three commodities: copper, molybdenum and silver. Gold is also present in the copper concentrates in the form of a saleable byproduct credit. The estimates of capital expenditures and site production costs have been developed specifically for this project and are presented in Section 21 of this report.

22.2 MINE PRODUCTION STATISTICS

Mine production is reported as sulfide ore and waste material from the mining operation. The annual production figures were obtained from the Mine Plan as reported in Section 16 of this report.

The life of mine (including pre-production) ore and waste quantities and ore grade are presented in Table 22-1 below.

Table 22-1: Total Mine Production Statistics

	Tons (000)	Copper	Molybdenum	Silver
Sulfide ore	661,428	0.44%	0.015%	0.12 oz/ton
Waste	1,258,718			
Total	1,920,146			

The net smelter return (NSR) reflects the value of all payable metal contained in the concentrates produced less charges related to downstream smelting, refining and transportation charges. Detailed annual production statistics for the Combined Base Case (60/40 Pricing) can be found in the financial model (Table 22-12) at the end of this section.

22.3 PLANT PRODUCTION STATISTICS

In the pre-production time period, approximately 6.259 million tons of sulfide ore will be stockpiled and additional 39.785 million sulfide ore tons will be stockpiled from years 1-7. These tons will be processed in years 8 – 10.

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

The estimated LOM recovery for the copper is 87%, molybdenum recovery is estimated to be 58%, and the recovery for silver is 76%.

Average copper production is 255 million pounds for the first 3 years of production. Molybdenum production averages 6.9 million pounds per year and silver averages 2.8 million ounces per year for the first 3 years of production. Gold as a by-product averages 21 thousand ounces per year.

Life of mine saleable production is presented in Table 22-2 below.

Table 22-2: Life of Mine Metal Production

	Concentrate Tons (000)	Copper Tons (000)	Molybdenum Tons (000)	Silver Ozs (000)
Copper Concentrate	8,514	2,554		59,958
Molybdenum Concentrate	113		56	

22.4 SMELTER RETURN FACTORS

Copper and molybdenum concentrates will be shipped from the site to smelting and refining companies. The smelter and refining treatment charges will be subject to negotiation at the time of final agreement. A smelter may impose a penalty either expressed in higher treatment charges or in metal deductions to treat concentrates that contain higher than specified quantities of certain elements. It is expected that the concentrate will not pose any special restrictions on smelting and refining, and that the concentrates will be marketable to smelting and refining companies.

The smelting and refining charges calculated in the financial evaluation include charges for smelting copper and molybdenum concentrates. The off-site charges that will be incurred are presented in Table 22-3 below.

Table 22-3: Smelter Return Factors

Smelter Return Factors	
Copper Concentrate	
Payable copper	96.5%
Copper deduction	Nil
Treatment charge - $/ton	$57.62
Copper refining - $/lb	$0.064
Shipping charge - $/ton	$75.46
Payable gold	92.5%
Gold refining - $/oz	$5.00
Payable silver	92.5%
Silver refining – $/oz	$0.50
Silver deduction	Nil
Molybdenum Concentrate	
Payable molybdenum	100.0%
Molybdenum deduction	NA
Treatment charge - $/lb	$1.56
Shipping charge - $/ton	FCA site

22.5 CAPITAL EXPENDITURES

22.5.1 Initial Capital

The total capital of new construction (includes direct, indirect costs and mine pre-development) is estimated to be $1,226 million, including $15.6 million for spare parts moved to working capital.

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

22.5.2 Sustaining Capital

A schedule of capital cost expenditures during the production period has been estimated and included in the financial analysis under the category of sustaining capital. The total life of mine sustaining capital is estimated to be $276.1 million. This capital will be expended during a 21 year period, starting in Year 1 and ending in Year 21.

22.5.3 Working Capital

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

22.5.4 Salvage Value

An allowance of $53.7 million has been included in the cash flow analysis as a return of capital from the salvage and resale of equipment at the end of mine life. It was calculated using initial equipment capital cost at 10%.

22.6 REVENUE

Annual revenue is determined by applying estimated metal prices to the annual payable metal before treatment, refinery and transportation charges for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. Metal sales prices used in the evaluation are shown in Table 22-4 below and are taken from the Commodity Price Analysis dated June 30, 2012, shown in Table 22-6.

Table 22-4: Base Case and Historical Metals Prices

	60/40 Weighted Average *	3 Year Historical Average *
Copper	$ 3.50 / pound	$ 3.56 / pound
Molybdenum	$14.19 / pound	$15.06 / pound
Silver**	$3.90 / ounce	$ 3.90 / ounce
Gold**	$450.00 / ounce	$ 450.00 / ounce

** See Table 22-6 for definitions*
***Silver and gold prices from Silver Wheaton agreement.*

In addition to the above metal sales prices, cases with long term metal prices were also evaluated and are shown in Table 22-5.

Table 22-5: Long Term Metals Prices

	Year 1	Year 2	Year 3	Year 4	Year 5 forward
Copper	$ 3.50/lb	$ 3.25/lb	$ 3.00/lb	$ 2.75/lb	$ 2.50/lb
Molybdenum	$ 15.00/lb	$ 15.00/lb	$ 15.00/lb	$ 15.00/lb	$ 15.00/lb
Silver**	$ 3.90/oz	$ 3.90/oz	$ 3.90/oz	$ 3.90/oz	$ 3.90/oz
Gold**	$450.00/oz	$450.00/oz	$450.00/oz	$450.00/oz	$450.00/oz

***Silver and gold prices from Silver Wheaton agreement.*

The Silver Wheaton agreement specifies a set payment for gold and silver in consideration of a financial contribution to the project.

Table 22-6: Commodity Price Analysis

Summary of Historical and Future Commodity Prices
June 30, 2012

Commodity	Spot Price EOM June 2012	Prices used for US Sec filings — Historical Price (36-months)	Source of Data	Futures Price Forecast (24-Month) Projected thru June 2014		Prices used for NI 43-101 filings — Weighted Average (60-40)
Gold (USD per Tr Oz)	1,598.50	1,376.78	London PM Fix - Kitco	1,610.38	CME Group Futures	1,470.22
Silver (USD per Tr Oz)	27.08	26.20	London Fix - Kitco	27.22	CME Group Futures	26.61
Copper (USD per lb)	3.52	3.56	LME Monthly Ave	3.42	CME Group Futures & LME Futures	3.50
Lead (USD per lb)	0.84	1.00	LME Monthly Ave	0.89	LME Futures	0.96
Nickel (USD per lb)	7.57	9.48	LME Monthly Ave	7.68	LME Futures	8.76
Zinc (USD per lb)	0.86	0.96	LME Monthly Ave	0.87	LME Futures	0.92
Molybdenum (USD per lb Mo)	13.15	15.06	LME Monthly Ave	12.89	see Note 5	14.19

Notes:

1. Precious Metals updated through End-of-Month (EOM) June 2012
   Base metals updated through EOM June 2012
2. Sources: London Bullion Market Association price fixing for gold (PM) and silver daily historical and EOM futures pricing
   London Metals Exchange (LME) for copper, lead, nickel, and zinc monthly average settlement and futures pricing.
   Copper futures pricing is the weighted average of the LME (65%) and CME (35%) copper futures prices.
   Molybdenum prices from Infomine.com website.
3. M3 uses weighted average prices for NI-43-101 reporting purposes, 60 % historical prices; 40% futures forecast prices.
4. Spot prices are from London Bullion Market Association, Kitco Metals, & LME for precious metals, base metals, & molybdenum, respectively.
5. Historical Molybdenum prices are determined by interpolating values on reported by the LME. Moly futures are now a 100% based on the LME Moly 15-month futures price less the slope of the LME Mo prices for the last 30 months.

PROPRIETARY INFORMATION
M3 Engineering Technology
Tucson, Arizona
Monthly Posting of Metals Pricing

22.7 CASH COPPER UNIT COST NET OF BY-PRODUCT CREDITS

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

The three different cost comparison cases evaluated are summarized in Table 22-7 below:

Table 22-7: Cash Copper Unit Cost Net of By Product Credits

	Base Case (60/40)	Historical 36 month	Long Term Metal Prices
Mining	2,226,008	2,226,008	2,226,008
Processing - Mill	2,822,546	2,822,546	2,822,546
G & A [1]	388,169	388,169	388,169
Treatment & Shipping Charges	1,712,751	1,723,450	1,722,712
Severance Taxes [2]	145,589	150,511	92,577
Property Taxes [3]	66,500	66,500	66,500
Reclamation Expense [4]	34,657	34,657	34,657
Total Operating Cost	7,396,220	7,411,841	7,353,169
Moly - by-product credit	(1,598,773)	(1,696,795)	(1,690,035)
Silver - by-product credit	(216,299)	(216,299)	(216,299)
Gold - by-product credit	(147,357)	(147,357)	(147,357)
Silver Wheaton Contribution	(230,000)	(230,000)	(230,000)
Net Operating cost	5,203,791	5,121,390	5,069,478
Net Unit Cost per lb Cu	1.019	1.003	0.992

1 *G & A*
The G & A cost has a community endowment component which varies by metal prices and railcar lease and CAP water payments.

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

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

4 *Reclamation & Closure*
An allowance of approximately $34.7 million for the cost of the final reclamation bond has been included in the cash flow projection. Continual early reclamation is done throughout the life of the mine and costs have included for such, e.g. borrow pits.

Also included in the financial analysis are the following items:

22.7.1 Pre-production Mining Cost

A total of $116.1 million will be spent for pre-production mining. 70% of these costs are expensed and the remaining 30% is amortized over a 5 year period.

22.7.2 Fees and Royalties

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

22.7.3 Depreciation

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

Below are the percentages that were applied:

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

22.7.4 Depletion

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

The rates for depletion are as follows:

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

22.7.5 Income Taxes

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

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

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

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

22.8 PROJECT FINANCING

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

22.8.1 Net Income After Tax

Net Income after tax amounts for each of the cases evaluated is shown in Table 22-8 below:

Table 22-8: Net Income After Tax

$ Millions	Base Case (60/40)	Historical 36 month	Long Term Metal Prices
Net Income After Tax	$ 6,914.3	$ 7,155.2	$ 4,211.2

22.8.2 Net Present Value, Internal Rate of Return and Sensitivity Analysis

The base case (60/40 metal pricing) economic analysis (Table 22-9) indicates that the project has an after tax Internal Rate of Return (IRR) of 37.9% with a payback period of 2.3 years.

A sensitivity analysis was conducted on metal prices, capital expenditures, operating costs and metal production. The results are included in Table 22-9. The project IRR is most sensitive to variation in metals price followed by metal production, operating cost, and capital cost.

Table 22-9: After Tax Economic Analysis – Combined Base Case (60/40) ($ millions)

	NPV @ 0%	NPV @ 5%	NPV @ 8%	IRR %	Payback years
Base Case (60/40 weighted average)	7,257.5	3,645.8	2,507.6	37.9%	2.3
Metals Price +10%	8,449.1	4,295.0	2,983.5	42.4%	2.1
Metals Price -10%	6,050.5	2,987.6	2,024.9	33.0%	2.6
CAPEX +10%	7,181.8	3,567.1	2,428.4	34.1%	2.5
CAPEX -10%	7,333.3	3,724.5	2,586.8	42.8%	2.1
OPEX +10%	6,946.5	3,473.4	2,380.5	36.6%	2.4
OPEX -10%	7,562.5	3,814.5	2,631.9	39.1%	2.2
Metal Production +10%	8,352.5	4,241.4	2,943.8	42.0%	2.1
Metal Production -10%	6,155.5	3,045.7	2,067.9	33.5%	2.5

The historical case (36 month trailing price) economic analysis shown in Table 22-10 indicates that the project has an after tax Internal Rate of Return (IRR) of 38.8% with a payback period of 2.2 years.

Table 22-10: After Tax Economic Analysis – Combined Case – Historical 36 Month Prices ($ millions)

	NPV @ 0%	NPV @ 5%	NPV @ 8%	IRR (%)	Payback (years)
Historical Case (36 month)	7,498.4	3,776.4	2,603.1	38.8%	2.2
Metals Price +10%	8,719.2	4,441.9	3,091.0	43.4%	2.0
Metals Price -10%	6,279.1	3,111.7	2,115.7	34.0%	2.5
CAPEX +10%	7,422.7	3,697.7	2,523.8	34.9%	2.4
CAPEX -10%	7,574.2	3,855.1	2,682.3	43.9%	2.0
OPEX +10%	7,188.9	3,605.1	2,476.9	37.6%	2.3
OPEX -10%	7,804.6	3,945.8	2,727.9	40.0%	2.2
Metal Production +10%	8,619.6	4,386.9	3,050.5	43.0%	2.0
Metal Production -10%	6,374.5	3,164.3	2,154.5	34.4%	2.5

The long term price case economic analysis shown in Table 22-11 indicates that the project has an after tax Internal Rate of Return (IRR) of 30.9% with a payback period of 2.4 years.

Table 22-11: After Tax Economic Analysis- Combined Case- Long Term Prices* ($ millions)

	NPV @ 0%	NPV @ 5%	NPV @ 8%	IRR (%)	Payback (years)
Combined Long Term Prices	4,554.4	2,256.0	1,529.4	30.9%	2.4
Metals Price +10%	5,514.4	2,795.7	1,932.3	36.0%	2.2
Metals Price -10%	3,602.8	1,714.7	1,122.2	25.3%	2.8
CAPEX +10%	4,469.5	2,170.2	1,444.2	27.3%	2.7
CAPEX -10%	4,641.9	2,345.8	1,618.7	35.8%	2.2
OPEX +10%	4,228.2	2,070.5	1,390.9	29.2%	2.5
OPEX -10%	4,887.1	2,446.0	1,671.7	32.7%	2.3
Metal Production +10%	5,415.1	2,740.5	1,891.4	35.5%	2.2
Metal Production -10%	3,701.7	1,770.4	1,163.7	25.9%	2.8

- ** See Table 22-5 for the prices*

Table 22-12: Combined Base Case (60/40 split)

Augusta Resource Corporation
Rosemont Copper Project Feasibility Study

		Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10
Mining															
Sulfide Ore	Mined (kt)	661,428													
	Copper Grade -%	0.44%	0.00%	0.00%	0.26%	0.46%	0.52%	0.39%	0.45%	0.51%	0.49%	0.55%	0.51%	0.32%	0.50%
	Moly Grade -%	0.015%	0.000%	0.000%	0.012%	0.015%	0.014%	0.020%	0.012%	0.014%	0.012%	0.014%	0.012%	0.006%	0.011%
	Ag Grade - oz/ton	0.1201	0.0000	0.0000	0.1001	0.1262	0.1258	0.0982	0.1497	0.1405	0.1155	0.1430	0.1478	0.0920	0.1104
Waste	Mined (kt)	1,258,718	0	20,864	87,552	88,169	69,944	82,165	95,980	74,569	63,412	62,094	95,016	94,971	79,256
Total	Total Material Mined (t)	1,920,146	0	20,864	93,811	116,089	105,521	124,793	123,355	106,584	97,761	99,467	112,443	105,966	101,149
Process Plant	Cathode Production (klbs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Production (klbs)	5,108,580	-	-	-	216,060	305,860	243,800	207,540	264,040	273,260	330,040	219,460	135,640	214,880
	Gold Production (kozs)	354	-	-	-	18	25	20	14	18	18	22	15	9	14
	Silver Production (kozs)	59,958	-	-	-	2,556	3,210	2,626	2,977	3,094	2,758	3,608	2,884	1,652	2,200
	Molybdenum Production (klbs)	112,680	-	-	-	5,310	5,888	9,418	2,312	2,854	2,710	3,124	5,256	4,148	4,294
Dollars in Thousands															
Capital Cost															
Initial Capital - Equity $(000's)		$ 977,752	$ 160,750	$ 122,828	$ 595,533	$ 98,641	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Oxide $(000's)		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sulphide $(000's)		$ 953,302	$ 145,483	$ 238,326	$ 470,853	$ 98,641	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Mine Mobile Equipment $(000's)		$ 254,450	$ 15,267	$ 114,503	$ 124,681	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Silver Wheaton Streaming Contribution		$ (230,000)	$ -	$ (230,000)	$ -	$ -									
Sustaining Capital		$ 276,092	$ -	$ -	$ -	$ 3,869	$ 19,972	$ 23,803	$ 41,041	$ 59,046	$ 1,802	$ 2,561	$ 3,590	$ 2,133	$ 5,150
Oxide $(000's)		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sulphide $(000's)		$ 135,008	$ -	$ -	$ -	$ 3,544	$ 60	$ 11,521	$ 39,046	$ 58,996	$ 877	$ 1,816	$ 846	$ 550	$ 2,905
Mine Mobile Equipment $(000's)		$ 141,084	$ -	$ -	$ -	$ 325	$ 19,912	$ 12,282	$ 1,995	$ 50	$ 925	$ 745	$ 2,744	$ 1,583	$ 2,245
Total Capital (Initial + Sustaining) $(000's)		$ 1,253,844	$ 160,750	$ 122,828	$ 595,533	$ 102,509	$ 19,972	$ 23,803	$ 41,041	$ 59,046	$ 1,802	$ 2,561	$ 3,590	$ 2,133	$ 5,150
Working Capital															
Metal WIP and Finished Goods (1 1/2- month) $(000's)		$ -	$ -	$ -	$ -	$ 103,000	$ 39,000	$ (20,000)	$ (28,000)	$ 25,000	$ 3,000	$ 26,000	$ (44,000)	$ (38,000)	$ 34,000
Inventory - Parts, Supplies and Comodities $(000's)		$ 0	$ -	$ -	$ 6,231	$ 9,347	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Pre-production mining (amortized)		$ -	$ -	$ 10,704	$ 24,126	$ (2,141)	$ (6,966)	$ (6,966)	$ (6,966)	$ (6,966)	$ (4,825)	$ -	$ -	$ -	$ -
Total Working Capital $(000's)		$ -	$ -	$ 10,704	$ 30,357	$ 110,206	$ 32,034	$ (26,966)	$ (34,966)	$ 18,034	$ (1,825)	$ 26,000	$ (44,000)	$ (38,000)	$ 34,000
Revenue															
Cathode	Copper	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Concentrate	Copper & Moly	$ 19,216,579	$ -	$ -	$ -	$ 821,777	$ 1,138,536	$ 974,853	$ 750,307	$ 950,957	$ 978,829	$ 1,181,202	$ 832,416	$ 526,708	$ 800,419
		$ 19,216,579	$ -	$ -	$ -	$ 821,777	$ 1,138,536	$ 974,853	$ 750,307	$ 950,957	$ 978,829	$ 1,181,202	$ 832,416	$ 526,708	$ 800,419
Cash Operating Costs															
Mine	Operations	$ 2,226,008	$ -	$ -	$ -	$ 102,583	$ 122,331	$ 119,053	$ 122,532	$ 142,776	$ 122,799	$ 118,206	$ 127,023	$ 129,805	$ 119,611
Processing	Operations - Mill	$ 2,822,546	$ -	$ -	$ -	$ 111,056	$ 117,274	$ 119,623	$ 115,935	$ 119,356	$ 134,362	$ 137,612	$ 136,533	$ 135,860	$ 136,123
Processing	Operations - SXEW	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Other	G & A	$ 388,169	$ -	$ -	$ -	$ 25,428	$ 19,797	$ 20,052	$ 20,562	$ 20,586	$ 20,666	$ 20,551	$ 20,301	$ 20,291	$ 20,341
Shipping, Refining and Smelting		$ 1,712,751	$ -	$ -	$ -	$ 73,368	$ 101,128	$ 88,015	$ 66,348	$ 83,957	$ 86,299	$ 104,127	$ 74,426	$ 47,344	$ 71,251
Total Cash Operating Costs		$ 7,149,473	$ -	$ -	$ -	$ 312,435	$ 360,531	$ 346,743	$ 325,378	$ 366,675	$ 364,127	$ 380,496	$ 358,283	$ 333,300	$ 347,325
Cash Costs															
Royalty		$ 526,000	$ -	$ -	$ -	$ 22,000	$ 31,000	$ 27,000	$ 21,000	$ 26,000	$ 27,000	$ 32,000	$ 23,000	$ 14,000	$ 22,000
Severance Tax		$ 145,589	$ -	$ -	$ -	$ 5,322	$ 7,572	$ 6,212	$ 3,814	$ 5,827	$ 6,186	$ 8,677	$ 5,348	$ 2,582	$ 6,019
Property Tax		$ 66,500	$ -	$ -	$ -	$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ 3,500
Total Cash Costs		$ 7,887,562	$ -	$ -	$ -	$ 343,256	$ 402,603	$ 383,455	$ 353,692	$ 402,002	$ 400,813	$ 424,672	$ 390,131	$ 353,382	$ 378,845
Production Costs															
Pre-production mining cost		$ 116,100	$ -	$ 24,976	$ 56,294	$ 2,141	$ 6,966	$ 6,966	$ 6,966	$ 6,966	$ 4,825	$ -	$ -	$ -	$ -
Reclamation bond fee		$ 11,043	$ -	$ -	$ 698	$ 773	$ 776	$ 789	$ 706	$ 686	$ 373	$ 373	$ 373	$ 373	$ 373
Reclamation expense		$ 34,657	$ -	$ -	$ -	$ 1,366	$ 979	$ 1,697	$ 204	$ 626	$ 747	$ 747	$ 747	$ 747	$ 747
Reimbursable expenses/salvage value		$ (73,190)	$ 3,033	$ 3,033	$ 3,033	$ (3,896)	$ (3,896)	$ (3,896)	$ (3,896)	$ (3,896)	$ -	$ -	$ -	$ -	$ -
Depreciation		$ 1,483,844	$ -	$ -	$ -	$ 129,765	$ 233,922	$ 188,477	$ 160,374	$ 168,623	$ 171,443	$ 168,307	$ 93,040	$ 17,921	$ 15,753
Total Production Cost		$ 9,460,016	$ 3,033	$ 28,010	$ 60,025	$ 473,404	$ 641,350	$ 577,487	$ 518,047	$ 575,007	$ 578,201	$ 594,099	$ 484,291	$ 372,423	$ 395,718
Income from Operations		$ 9,756,563	$ (3,033)	$ (28,010)	$ (60,025)	$ 348,373	$ 497,186	$ 397,366	$ 232,260	$ 375,950	$ 400,628	$ 587,102	$ 348,125	$ 154,285	$ 404,701
Taxes with loss carry forward applied		$ 2,842,233	$ -	$ -	$ -	$ 60,509	$ 125,063	$ 105,512	$ 53,670	$ 94,770	$ 108,710	$ 172,204	$ 94,457	$ 33,356	$ 114,002
Net Income After Taxes		6,914,330	(3,033)	(28,010)	(60,025)	287,864	372,122	291,854	178,591	281,181	291,918	414,898	253,668	120,928	290,699
Cash Flow from Property after Tax															
Net Income After Taxes		$ 6,914,330	$ (3,033)	$ (28,010)	$ (60,025)	$ 287,864	$ 372,122	$ 291,854	$ 178,591	$ 281,181	$ 291,918	$ 414,898	$ 253,668	$ 120,928	$ 290,699
Add Back - Depreciation		$ 1,483,844	$ -	$ -	$ -	$ 129,765	$ 233,922	$ 188,477	$ 160,374	$ 168,623	$ 171,443	$ 168,307	$ 93,040	$ 17,921	$ 15,753
Operating Cash Flow		$ 8,398,174	$ (3,033)	$ (28,010)	$ (60,025)	$ 417,628	$ 606,044	$ 480,330	$ 338,965	$ 449,803	$ 463,361	$ 583,205	$ 346,709	$ 138,850	$ 306,452
Working Capital		0	-	(10,704)	(30,357)	(110,206)	(32,034)	26,966	34,966	(18,034)	1,825	(26,000)	44,000	38,000	(34,000)
Capital Expenditures		(1,253,844)	(160,750)	(122,828)	(595,533)	(102,509)	(19,972)	(23,803)	(41,041)	(59,046)	(1,802)	(2,561)	(3,590)	(2,133)	(5,150)
Sunk Cost		113,187	113,187	-	-	-	-	-	-	-	-	-	-	-	-
Cash Flow		$ 7,257,517	$ (50,596)	$ (161,542)	$ (685,916)	$ 204,913	$ 554,038	$ 483,493	$ 332,890	$ 372,724	$ 463,385	$ 554,645	$ 387,119	$ 174,717	$ 267,303
Cumulative Cash Flow			$ (50,596)	$ (212,138)	$ (898,054)	$ (693,141)	$ (139,103)	$ 344,390	$ 677,281	$ 1,050,004	$ 1,513,389	$ 2,068,033	$ 2,455,152	$ 2,629,869	$ 2,897,172
Property Economics Indicators: Before Taxes															
Cash Flow before Taxes			$ (50,596)	$ (161,542)	$ (685,916)	$ 265,421	$ 679,101	$ 589,006	$ 386,560	$ 467,493	$ 572,095	$ 726,848	$ 481,575	$ 208,073	$ 381,305
NPV at 0% - ($000) after tax		$ 10,099,750													
NPV at 5% - ($000) after tax		$ 5,093,555													
NPV at 8% - ($000) after tax		$ 3,528,478													
IRR		45.6%													
Property Economics Indicators: After Taxes															
Cash Flow before Taxes															
NPV at 0% - ($000) after tax		$ 7,257,517													
NPV at 5% - ($000) after tax		$ 3,645,806													
NPV at 8% - ($000) after tax		$ 2,507,595													
IRR		37.9%													
Payback - Years from Startup		2.3													

		11	12	13	14	15	16	17	18	19	20	21	22	23	24
	Copper Grade -%	0.47%	0.41%	0.45%	0.42%	0.40%	0.38%	0.41%	0.46%	0.47%	0.37%	0.35%	0.62%	0.00%	0.00%
	Moly Grade -%	0.011%	0.012%	0.013%	0.013%	0.017%	0.024%	0.014%	0.017%	0.019%	0.017%	0.012%	0.011%	0.000%	0.000%
	Ag Grade - oz/ton	0.0937	0.0816	0.1073	0.1284	0.1449	0.1495	0.0746	0.1036	0.1061	0.1040	0.1352	0.3485	0.0000	0.0000
Waste	Mined (kt)	59,586	56,910	53,029	48,686	42,525	32,078	34,938	6,956	3,085	2,289	2,321	2,323	0	0
Total	Total Material Mined (t)	91,706	89,760	85,879	81,536	75,375	64,927	67,787	39,806	35,935	35,139	35,171	9,320	0	0
Process Plant	Cathode Production (klbs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Production (klbs)	245,680	225,340	248,760	236,580	233,320	222,460	239,100	271,820	274,240	218,320	205,660	76,720	-	-
	Gold Production (kozs)	16	15	17	16	16	15	16	18	18	15	14	5	-	-
	Silver Production (kozs)	2,162	1,970	2,577	3,133	3,662	3,778	1,887	2,619	2,682	2,629	3,417	1,877	-	-
	Molybdenum Production (klbs)	4,972	5,646	4,960	4,736	6,686	9,154	5,334	6,500	7,126	6,584	4,790	878	-	-
Initial Capital - Equity $(000's)		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Oxide $(000's)		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sulphide $(000's)		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Mine Mobile Equipment $(000's)		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital		$ 11,912	$ 3,165	$ 733	$ 1,843	$ 30,997	$ 59,146	$ 2,200	$ 1,860	$ 941	$ 283	$ 50	$ -	$ -	$ -
Oxide $(000's)		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sulphide $(000's)		$ 1,777	$ 2,000	$ 663	$ 673	$ 754	$ 7,477	$ 725	$ 560	$ 163	$ 60	$ -	$ -	$ -	$ -
Mine Mobile Equipment $(000's)		$ 10,135	$ 1,165	$ 70	$ 1,170	$ 30,243	$ 51,670	$ 1,475	$ 1,300	$ 778	$ 223	$ 50	$ -	$ -	$ -
Total Capital (Initial + Sustaining) $(000's)		$ 11,912	$ 3,165	$ 733	$ 1,843	$ 30,997	$ 59,146	$ 2,200	$ 1,860	$ 941	$ 283	$ 50	$ -	$ -	$ -
Metal WIP and Finished Goods (1 1/2- month) $(000's)		$ 14,000	$ (7,000)	$ 9,000	$ (5,000)	$ 2,000	$ -	$ (1,000)	$ 16,000	$ 3,000	$ (25,000)	$ (8,000)	$ (63,000)	$ (35,000)	$ -
Inventory - Parts, Supplies and Comodities $(000's)		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (15,578)	$ -	$ -
Pre-production mining (amortized)		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Working Capital $(000's)		$ 14,000	$ (7,000)	$ 9,000	$ (5,000)	$ 2,000	$ -	$ (1,000)	$ 16,000	$ 3,000	$ (25,000)	$ (8,000)	$ (78,578)	$ (35,000)	$ -
Cathode	Copper	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Concentrate	Copper & Moly	$ 914,825	$ 854,561	$ 926,909	$ 884,227	$ 902,757	$ 901,126	$ 896,728	$ 1,027,251	$ 1,044,561	$ 846,503	$ 780,727	$ 280,401	$ -	$ -
		$ 914,825	$ 854,561	$ 926,909	$ 884,227	$ 902,757	$ 901,126	$ 896,728	$ 1,027,251	$ 1,044,561	$ 846,503	$ 780,727	$ 280,401	$ -	$ -
Mine	Operations	$ 121,823	$ 116,661	$ 104,016	$ 105,960	$ 100,921	$ 82,283	$ 96,087	$ 68,588	$ 58,851	$ 64,622	$ 59,591	$ 19,884	$ -	$ -
Processing	Operations - Mill	$ 136,391	$ 137,478	$ 137,935	$ 137,727	$ 139,036	$ 140,711	$ 138,058	$ 138,887	$ 139,257	$ 138,919	$ 137,722	$ 36,689	$ -	$ -
Processing	Operations - SXEW	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Other	G & A	$ 17,116	$ 17,041	$ 17,106	$ 16,841	$ 15,263	$ 15,288	$ 15,338	$ 15,403	$ 15,263	$ 15,288	$ 11,197	$ 8,447	$ -	$ -
Shipping, Refining and Smelting		$ 81,401	$ 76,351	$ 82,485	$ 78,793	$ 81,117	$ 81,812	$ 79,895	$ 91,727	$ 93,449	$ 76,044	$ 69,866	$ 23,550	$ -	$ -
Total Cash Operating Costs		$ 356,731	$ 347,531	$ 341,542	$ 339,320	$ 336,338	$ 320,095	$ 329,378	$ 314,605	$ 306,820	$ 294,874	$ 278,376	$ 88,571	$ -	$ -
Royalty		$ 25,000	$ 23,000	$ 25,000	$ 24,000	$ 25,000	$ 25,000	$ 25,000	$ 28,000	$ 29,000	$ 23,000	$ 21,000	$ 8,000	$ -	$ -
Severance Tax		$ 7,437	$ 6,833	$ 7,900	$ 7,370	$ 7,619	$ 7,723	$ 7,456	$ 9,414	$ 9,775	$ 7,373	$ 6,704	$ 2,424	$ -	$ -
Property Tax		$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ 3,500	$ -	$ -	$ -	$ -	$ -
Total Cash Costs		$ 392,668	$ 380,864	$ 377,942	$ 374,191	$ 372,457	$ 356,318	$ 365,335	$ 355,520	$ 349,096	$ 325,247	$ 306,080	$ 98,994	$ -	$ -
Pre-production mining cost		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation bond fee		$ 362	$ 362	$ 362	$ 362	$ 362	$ 363	$ 363	$ 363	$ 363	$ 363	$ 363	$ 363	$ 399	$ -
Reclamation expense		$ 687	$ 687	$ 687	$ 687	$ 687	$ 2,832	$ 2,832	$ 2,832	$ 2,832	$ 2,832	$ 2,832	$ 2,832	$ 2,788	$ -
Reimbursable expenses/salvage value		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (53,710)	$ (9,100)	$ -
Depreciation		$ 13,304	$ 8,247	$ 4,230	$ 3,761	$ 6,741	$ 15,239	$ 18,687	$ 14,740	$ 12,339	$ 12,072	$ 11,854	$ 15,004	$ -	$ -
Total Production Cost		$ 407,022	$ 390,160	$ 383,222	$ 379,002	$ 380,248	$ 374,752	$ 387,217	$ 373,455	$ 364,630	$ 340,514	$ 321,128	$ 63,483	$ (5,912)	$ -
Income from Operations		$ 507,803	$ 464,401	$ 543,687	$ 505,225	$ 522,509	$ 526,374	$ 509,512	$ 653,796	$ 679,931	$ 505,989	$ 459,599	$ 216,917	$ 5,912	$ -
Taxes with loss carry forward applied		$ 153,592	$ 139,232	$ 167,200	$ 154,203	$ 159,534	$ 160,374	$ 155,114	$ 205,271	$ 214,577	$ 155,806	$ 141,301	$ 71,439	$ 2,338	$ -
Net Income After Taxes		354,211	325,168	376,488	351,022	362,975	366,000	354,398	448,525	465,355	350,183	318,298	145,479	3,574	0
Net Income After Taxes		$ 354,211	$ 325,168	$ 376,488	$ 351,022	$ 362,975	$ 366,000	$ 354,398	$ 448,525	$ 465,355	$ 350,183	$ 318,298	$ 145,479	$ 3,574	$ -
Add Back - Depreciation		$ 13,304	$ 8,247	$ 4,230	$ 3,761	$ 6,741	$ 15,239	$ 18,687	$ 14,740	$ 12,339	$ 12,072	$ 11,854	$ 15,004	$ -	$ -
Operating Cash Flow		$ 367,515	$ 333,415	$ 380,718	$ 354,784	$ 369,716	$ 381,239	$ 373,085	$ 463,266	$ 477,694	$ 362,255	$ 330,151	$ 160,483	$ 3,574	$ -
Working Capital		(14,000)	7,000	(9,000)	5,000	(2,000)	-	1,000	(16,000)	(3,000)	25,000	8,000	78,578	35,000	-
Capital Expenditures		(11,912)	(3,165)	(733)	(1,843)	(30,997)	(59,146)	(2,200)	(1,860)	(941)	(283)	(50)	-	-	-
Sunk Cost		-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cash Flow		$ 341,603	$ 337,250	$ 370,986	$ 357,941	$ 336,720	$ 322,093	$ 371,885	$ 445,406	$ 473,753	$ 386,973	$ 338,101	$ 239,061	$ 38,574	$ -
Cumulative Cash Flow		$ 3,238,775	$ 3,576,025	$ 3,947,010	$ 4,304,951	$ 4,641,671	$ 4,963,764	$ 5,335,649	$ 5,781,055	$ 6,254,808	$ 6,641,781	$ 6,979,882	$ 7,218,943	$ 7,257,517	$ 7,257,517
Cash Flow before Taxes		$ 495,195	$ 476,482	$ 538,185	$ 512,144	$ 496,254	$ 482,468	$ 526,999	$ 650,676	$ 688,330	$ 542,778	$ 479,402	$ 310,500	$ 40,912	$ -

23 ADJACENT PROPERTIES

There are no significant other mineral properties immediately adjacent to the Rosemont Project.

24 OTHER RELEVANT DATA AND INFORMATION

To the best of the Authors' knowledge, all relevant data and information has been addressed elsewhere in this Technical Report.

25 INTERPRETATION AND CONCLUSIONS

25.1 CONCLUSIONS

The intent of this technical report is to present updated mineral resource information and metallurgical testing information completed since the last feasibility study technical report update in January 2009. The major changes in this feasibility study update, since the previous 2009 update, are noted below. Augusta intends to continue with the permitting effort and initiate construction once the permitting effort has been completed and a Record of Decision has been issued by the US Forest Service.

1. Augusta's 2012 drilling campaign at the Rosemont Deposit has increased both the quantity and confidence level of the estimated mineral resources to 919.3 million tons of measured and indicated, sulfide and mixed resources.

2. Rosemont's proven and probable sulfide mineral reserves increased by 22%, or 121 million tons, to 667 million tons, when compared to the previous Updated Feasibility Study in January 2009.

3. As a result of the additional metallurgical test work and further optimizations, life of mine copper recoveries improved from 83% to 87%. Life of mine molybdenum and silver recoveries remained comparable at 58% and 76% respectively.

4. The heap leaching of oxide minerals and associated facilities, such as the heap leach pad, solvent extraction, electrowinning, and related facilities, have been eliminated.

5. The total initial capital cost has increased approximately 32% to $1.226 billion reflecting additional equipment and escalation cost of equipment, materials and labor. The capital cost includes Owner's cost and mine pre-development costs.

6. The price of copper in this study increased approximately 42% from the 2009 study update, based on a calculated three year historical price and 2 year future price. Molybdenum prices dropped about 38% from the 2009 feasibility study update.

7. The after-tax NPV at a discount rate of 5% increased approximately 50%, from $2.4 billion to $3.6 billion, based on 3 year historical metal prices and 2 year future metal prices. The after-tax IRR increased from 28.5% to 38%, and the payback period dropped from 3.1 years to 2.3 years.

8. Environmental permitting for construction and operations continues to advance with the Record of Decision expected from the US Forest Service at the end of 2012.

25.2 RISKS

The project risks identified at this time are noted below. Using a staged approach to advance the project to full production will allow Augusta Resource Corporation to adequately assess the risk and associated costs and develop mitigation strategies before progressing to the next stage.

There are no known fatal flaws; however, the risks noted below may have cost impacts if not addressed in a timely manner.

a) The risk of further delays in the permitting effort will continue to add costs to the project for storage of purchased equipment and escalation of equipment costs not released for manufacture.

b) Legal challenges to the Record of Decision can delay the resumption of design engineering and start of construction, incurring added costs noted above.

c) Holding the resumption of engineering until the Record of Decision may delay the construction schedule if contractors need to wait for design information. This can add to the cost of construction.

d) Further geotechnical testing at critical areas of the site is still necessary to identify bedrock. Engineering design can mitigate cost increases by re-designing to avoid the need for blasting.

e) Availability of construction water at site has not been confirmed. Alternatives to having construction water available at site when construction starts are costly.

f) Sand and aggregate availability on site has not been confirmed. A 404 permit is required to take sand from dry washes on site. If the permit is not available at start of construction, alternative sources for sand and gravel could be costly.

26 RECOMMENDATIONS

The Rosemont Copper Project continues to have metrics indicative of a stable and continuous hard rock mining operation. Based on this assessment, it is recommended that Augusta Resource Corporation continue with the project implementation plan to place the property in operation. This would largely consist of three main tasks outlined below. The third task would be contingent on completing the first two items.

1. Continue with the effort to secure the environmental permits required for construction and operations and the Record of Decision. This work is currently in progress.

2. Secure the appropriate financing to complete the engineering and construction of the facilities and start operations. This work is currently in progress.

3. Initiate the remainder of the design engineering work to advance engineering as much as possible prior to mobilization to the field for construction. Engineering is currently on hold.

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APPENDIX A: FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE of QUALIFIED PERSON

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

1. I am currently employed as Senior Vice President and Chairman of the Board by:
   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Suite 101
   Tucson, Arizona 85704
   U.S.A.

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

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

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

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

6. I have had prior involvement with the property that is the subject of the Technical Report. I was the principal author for the 2007 Feasibility Study Technical Report and the 2009 Updated Feasibility Study Technical Report.

7. I am responsible for the overall preparation of the "NI 43-101 Technical Report for the Rosemont Copper Project Updated Feasibility Study, Pima County, Arizona, USA" (the "Technical Report"), dated August 28, 2012, prepared for Augusta Resource Corporation; integrating the mineral resource estimate, mineral reserve estimate, mining methods, geological setting, mineralization, deposit type, exploration, drilling, sample preparation, data verification, and adjacent properties by Moose Mountain Technical Services and mining capital and operating costs by URS Corporation.

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

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

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

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

Dated this 28th day of August 2012.

Conrad E Huss

Signature of Qualified Person

Registered Professional Engineer (Civil)
CERTIFICATE NO. 9648
CONRAD E. HUSS
28 AUG 2012
ARIZONA U.S.A.

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

Consent of Expert

August 28, 2012

FILED VIA SEDAR

TO: **All Applicable Securities and Regulatory Authorities**

AND TO: **Augusta Resource Corporation**

Dear Sirs/Mesdames:

Re: Augusta Resource Corporation (the "Company") – Technical Report entitled "NI 43-101 Technical Report for the Rosemont Copper Project Updated Feasibility Study, Pima County, Arizona" dated August 28, 2012 (the "Report")

The undersigned hereby consents to the filing of the Report with the securities regulatory authorities referred to above. The undersigned confirms that the undersigned has read the news release of the Company dated July 24, 2012 (the "News Release") and the Material Change Report (the "MCR") dated July 30, 2012 filed with the securities regulatory authorities referred to above, and confirms that the News Release and the MCR fairly and accurately represents the information in the Report prepared by the undersigned that supports the disclosure in the News Release and the MCR.

Registered Professional Engineer (Civil)
CERTIFICATE NO.
9648
CONRAD E. HUSS
Date Signed 28 AUG 2012
ARIZONA, U.S.A.

Conrad E. Huss, P.E., Ph.D.
Senior Vice President and Chairman of the Board
M3 Engineering & Technology Corporation
2051 W. Sunset Road, Suite 101
Tucson, AZ, 85704

CERTIFICATE of QUALIFIED PERSON

I, Susan C. Bird, M.Sc., P.Eng., do hereby certify that:

1. I am a Sr. Associate Engineer of Moose Mountain Technical Services, residing at 32 Paddon Ave., Victoria, B.C. V8V 2M5.
2. I graduated with a B.Eng. from the Queen's University in 1989.
3. I graduated with a M.Sc. from Queen's University in 1993.
4. I am a member of the Association of Professional Engineers and Geoscientists of B.C. (#25007).
5. I have worked as an engineering geologist for a total of 16 years since my graduation from university.
6. My past experience with porphyry deposits includes exploration, resource/reserve reporting and engineering work on Gibraltar, Kerr-Sulphurets-Mitchell (KSM), Whistler, and Ilovitza among others.
7. I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person".
8. I am responsible for Sections 7 through 12, Section 14 and Section 23 of the report entitled "NI 43-101 Technical Report for the Rosemont Copper Project Updated Feasibility Study, Pima County, Arizona, USA" (Technical Report), with an effective date of 28 August 2012.
9. I am independent of Augusta Resource Corporation, and work as a geological and mining consultant to the mining industry.
10. To the best of my knowledge, information and belief at the effective date, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Date this 28th day of August 2012

Susan Bird

Signature of Qualified Person

PROFESSIONAL
PROVINCE OF
S.C. BIRD
25007
BRITISH COLUMBIA
ENGINEER

Susan C. Bird, M.Sc., P.Eng.
Print Name of Qualified Person

Consent of Expert

August 28, 2012

FILED VIA SEDAR

TO: **All Applicable Securities and Regulatory Authorities**

AND TO: **Augusta Resource Corporation**

Dear Sirs/Mesdames:

Re: Augusta Resource Corporation (the "Company") – Technical Report entitled "NI 43-101 Technical Report for the Rosemont Copper Project Updated Feasibility Study, Pima County, Arizona" dated August 28, 2012 (the "Report")

The undersigned hereby consents to the filing of the Report with the securities regulatory authorities referred to above. The undersigned confirms that the undersigned has read the news release of the Company dated July 24, 2012 (the "News Release") and the Material Change Report (the "MCR") dated July 30, 2012 filed with the securities regulatory authorities referred to above, and confirms that the News Release and the MCR fairly and accurately represents the information in the Report prepared by the undersigned that supports the disclosure in the News Release and the MCR.

Yours truly,

Susan Bird

PROFESSIONAL
PROVINCE OF
S.C. BIRD
25007
BRITISH COLUMBIA
ENGINEER

Susan Bird, P. Eng.
Senor Associate
Moose Mountain Technical Services
1975 1st Ave. S
Cranbrook BC V1X 6Y3

CERTIFICATE of QUALIFIED PERSON

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

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

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

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

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

4. I have practiced metallurgical and mineral processing engineering and project management for 42 years. I have worked for mining and exploration companies for 18 years and for M3 Engineering and Technology Corporation for 24 years.

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

6. I am responsible for the preparation of Section 1.8 "Mineral Processing And Metallurgical Testing", Section 1.9 "Recovery Methods", Section 13 "Mineral Processing And Metallurgical Testing", and Section 17 "Recovery Methods" of the technical report titled *NI 43-101 Technical Report Update Feasibility Study* dated August 28, 2012 (the "Technical Report").

7. I have had prior involvement with the property that is the subject of the Technical Report. I participated in the preparation of the technical report titled *NI 43-101 Technical Report for the Rosemont Copper Project Feasibility Study, Pima County, Arizona, USA* dated August 2007, relating to the Augusta Rosemont property. I participated in the preparation of the technical report titled *2008 Mineral Resource Update for the Rosemont Project, Pima County, Arizona, USA* and dated December 4, 2008, relating to the Augusta Rosemont property. I participated in the preparation of the technical report entitled *Rosemont Copper Project Updated Feasibility Study* dated January 14, 2009. I visited the subject property on August 21, 2007.

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

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

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

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

Dated this 28th day of August 2012.

Signature of Qualified Person

Thomas L. Drielick

Print Name of Qualified Person

Registered Professional Engineer (Mining)
CERTIFICATE NO
22958
THOMAS L
DRIELICK
Date Signed 8-28-12
ARIZONA U.S.A.
Expires 9/30/14

Consent of Expert

August 28, 2012

FILED VIA SEDAR

TO: **All Applicable Securities and Regulatory Authorities**

AND TO: **Augusta Resource Corporation**

Dear Sirs/Mesdames:

Re: Augusta Resource Corporation (the "Company") – Technical Report entitled "NI 43-101 Technical Report for the Rosemont Copper Project Updated Feasibility Study, Pima County, Arizona" dated August 28, 2012 (the "Report")

The undersigned hereby consents to the filing of the Report with the securities regulatory authorities referred to above. The undersigned confirms that the undersigned has read the news release of the Company dated July 24, 2012 (the "News Release") and the Material Change Report (the "MCR") dated July 30, 2012 filed with the securities regulatory authorities referred to above, and confirms that the News Release and the MCR fairly and accurately represents the information in the Report prepared by the undersigned that supports the disclosure in the News Release and the MCR.

Yours truly,

Thomas L. Drielick, P.E.
Senior Vice President
M3 Engineering & Technology Corporation
2440 W. Ruthrauff Rd.
Tucson, AZ, 85705

Registered Professional Engineer (Mining)
CERTIFICATE NO 22958
THOMAS L DRIELICK
Date Signed 8-28-12
ARIZONA U.S.A.

CERTIFICATE of QUALIFIED PERSON

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

1. I am a Principal Mining Engineer associated with:

   Moose Mountain Technical Services (MMTS)
   1975 1st Avenue, S
   Cranbrook, B.C. Canada
   V1C-6Y3

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

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

4. I have worked as a mining engineering since graduation from university, and have provided over 18 years of engineering consulting services to projects in Canada, United States, South America, Mexico, Africa and Asia.

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

6. I am responsible for the preparation of Section 15 (Mineral Reserve Estimates), and Sections 16-1 to 16-5 (Mining Methods) of the technical report titled "*NI 43-101 Technical Report for Rosemont Copper Project, Updated Feasibility Study, Pima County, Arizona, USA*", dated August 28, 2012 (the Technical Report) relating to the Rosemont property.

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

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

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

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

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

Dated this 28th day of August, 2012.

Signature of Qualified Person

Robert H. Fong

Print Name of Qualified Person

Consent of Expert

August 28, 2012

FILED VIA SEDAR

TO: **All Applicable Securities and Regulatory Authorities**

AND TO: **Augusta Resource Corporation**

Dear Sirs/Mesdames:

Re: Augusta Resource Corporation (the "Company") – Technical Report entitled "NI 43-101 Technical Report for the Rosemont Copper Project Updated Feasibility Study, Pima County, Arizona" dated August 28, 2012 (the "Report")

The undersigned hereby consents to the filing of the Report with the securities regulatory authorities referred to above. The undersigned confirms that the undersigned has read the news release of the Company dated July 24, 2012 (the "News Release") and the Material Change Report (the "MCR") dated July 30, 2012 filed with the securities regulatory authorities referred to above, and confirms that the News Release and the MCR fairly and accurately represents the information in the Report prepared by the undersigned that supports the disclosure in the News Release and the MCR.

Yours truly,

Robert H. Fong, P.Eng.
Principal Mining Engineer
Moose Mountain Technical Services
1975 1st Avenue S
Cranbrook, BC, Canada
V1C-6Y3

CERTIFICATE of QUALIFIED PERSON

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

1. I am currently employed as Vice President of Engineering by:
   URS Energy and Construction
   7800 East Union Avenue, Suite 100
   Denver, CO 80237
   USA

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

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

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

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

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

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

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

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

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

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

Dated this 28 day of August 2012.

Signature of Qualified Person

(sealed)

JOHN I. AJIE
NEW MEXICO
14276
REGISTERED PROFESSIONAL ENGINEER

John I. Ajie
Print Name of Qualified Person

Consent of Expert

August 28, 2012

FILED VIA SEDAR

TO: **All Applicable Securities and Regulatory Authorities**

AND TO: **Augusta Resource Corporation**

Dear Sirs/Mesdames:

Re: Augusta Resource Corporation (the "Company") – Technical Report entitled "Rosemont Copper Project – NI 43-101 Technical Report, Updated Feasibility Study, Pima County, Arizona, USA" dated August 28, 2012 (the "Report")

The undersigned hereby consents to the filing of the written disclosure regarding section 21 of the Report with the securities regulatory authorities referred to above. The undersigned confirms that the undersigned has read the news release of the Company dated July 24, 2012 (the "News Release") and the Material Change Report (the "MCR") dated July 30, 2012 filed with the securities regulatory authorities referred to above, and confirms that the News Release and the MCR fairly and accurately represents the information in the Report prepared by the undersigned that supports the disclosure in the News Release and the MCR.

Yours truly,

John I. Ajie
Vice President of Engineering
URS Energy and Construction
Denver, Colorado, USA

JOHN I. AJIE
NEW MEXICO
14276
REGISTERED PROFESSIONAL ENGINEER

APPENDIX B: UNPATENTED CLAIMS LIST

Rosemont Property Unpatented Lode Mining Claims		
Count	**Claim Name**	**BLM Serial No.**
1	York Fraction	2198
2	Travis #1	2199
3	Jim	2200
4	Isle Royal Fraction	2201
5	Indian Club Fraction	2202
6	Pilot Fraction	2203
7	A.O.T. Fraction	2204
8	Malachite Fraction	2211
9	MAX 121	13284
10	MAX 123	13286
11 - 42	MAX 125 - MAX 156	13288 - 13319
43	Rosalind	14972
44	Michael M.	14973
45	Lydia J.	14974
46	Ida D.	14975
47	D & D #1	14976
48	D & DII	14977
49	El Frijoli	14978
50	Frijoli II	14979
51	Frijoli III	14980
52	Frijoli IV	14981
53	Frijoli V	14982
54	Frijoli VII	14984
55	Frijoli VIII	14985
56	Frijoli IX	14986
57	Frijoli X	14987
58	Frijoli XI	14988
59	Frijoli XI Extension	14989
60	Deering Springs No. 2	15002
61	Deering Springs No. 4	15003
62	Deering Springs No. 6	15004
63	Deering Springs No. 8	15005
64	Deering Springs No. 10	15006
65	Deering Springs No. 12	15007
66 - 69	Deering Springs No. 14 - Deering Springs No. 17	15008 - 15011
70 - 88	Deering Springs No. 21 - Deering Springs No 39	15012 - 15030
89	Deering Springs No. 42	15031
90	Deering Springs No. 51	15032
91	Deering Springs No. 52	15033
92 - 120	Kid 1 - Kid 29	25210 - 25238
121 - 134	Kid 34 - Kid 47	25243 - 25256
135 - 170	Wasp 52 - Wasp 130	25257 - 25294
171 - 188	Wasp 201 - Wasp 218	25295 - 25312
189	Wasp 313	25349
190	Wasp 315	25351
191	Wasp 317	25353
192	Wasp 319	25355
193	Wasp 321	25357
194	Wasp 323	25359
195	Wasp 325	25361
196	Wasp 327	25363
197	Wasp 329	25365
198	Wasp 331	25367
199	Wasp 333	25369
200	Wasp 335	25371
201	Wasp 337	25373
202	Wasp 339	25375
203	Wasp 341	25377
204 - 215	Wasp 343 - Wasp 354	25379 - 25390
216	Max 41	25662

217	Max 43	25664
218	Max 45	25666
219	Max 47	25668
220	Max 49	25670
221 - 241	Max 71 - Max 91	25692 - 25712
242	Max 93	25714
243	Max 95	25716
244	Max 97	25718
245	Max 99	25720
246 - 265	Max 101 - Max 120	25722 - 25741
266 - 271	Elk 1 - Elk 6	27423 - 27428
272 - 274	Elk 35 - Elk 37	27451 - 27453
275	Elk 39	27455
276	Elk 41	27457
277	Elk 43	27459
278	Elk 45	27461
279 - 290	Elk 70 - Elk 81	27465 - 27476
291	Elk 83	27478
292	Elk 85	27480
293	Elk 87	27482
294 - 313	Alpine #5 - Alpine #24	27513 - 27532
314	Santa Rita Wedge	28871
315	Buzzard No. 5	36021
316	Shadow #4	36025
317	John 1	36026
318	John 2	36027
319 - 323	Flying Dutchman No. 2 - Flying Dutchman No. 6	36028 - 36032
324	Black Bess No. 2	36034
325	K.W.L.	36036
326	G.E.J.	36037
327	R.F.E.	36038
328	R.C.M.	36039
329 - 340	Sycamore #1 - Sycamore #12	36040 - 36051
341	Naragansett Extension #1	36052
342	Naragansett Ext. #2	36053
343 - 348	Naragansett Extension #3 - Naragansett Extension #8	36054 - 36059
349	Narragansett Ext. No. 9	36060
350	Schwab Extension #1 North West	36061
351	Rocky 1	36062
352	Amole No. 2	36063
353	Falls No. 3	36065
354	Falls No. 4	36066
355	Perry No. 1	36067
356 - 358	Perry #2 - Perry #4	36068 - 36070
359 - 364	Perry #7 - Perry #12	36073 - 36078
365 - 368	Perry #15 - Perry #18	36081 - 36084
369	Gunsite 1-A	36086
370 - 372	Gunsite No. 2 - Gunsite No. 4	36087 - 36089
373	Gunsite 5A	36090
374	Gunsite 6-B	36091
375	Gunsite No. 7	36092
376	Gunsite 7A	36093
377 - 402	Gunsite No. 8 - Gunsight No. 33	36094 - 36119
403	Gunsight No. 35 - Gunsight No. 43	36121 - 36129
412	Gunsight 44	36130
413 - 418	Gunsight #45 - Gunsight #50	36131 - 36136
419	Williams Folly	36137
420	Williams Folly #2	36138
421 - 423	Santa Rita #1 - Santa Rita #3	46740 - 46742
424	Santa Rita #7	46746
425 - 433	Santa Rita #17 - santa Rita #31	46756 - 46764
434 - 436	Santa Rita #29 - Santa Rita #31	46768 - 46770
437 - 440	Catalina #1 - Catalina #4	46771 - 46774

441	Catalina #5A	46775
442	Catalina #6A	46776
443	Catalina #7	46777
444	Catalina #8	46778
445	Fred Bennett	46779
446	Fred Bennett	46780
447	Rosemont #9	46781
448	Rosemont #11	46782
449	Rosemont #11-A	46783
450	Rosemont #12	46784
451	Rosemont #13	46785
452	Rosemont #15	46786
453 - 455	Rosemont #16 - Rosemont #18	46787 - 46789
456	Rosemont 21	46790
457	Fred Bennett Fraction	46791
458	Last Chance No. 3	46794
459	Cave	46796
460	Strip	46800
461	Cuba Fraction	46801
462	Patrick Henry Fraction	46802
463	R. G. Ingersoll Fraction	46803
464	Daylight Fraction	46804
465 - 469	Travis #2 - Travis #6	46805 - 46809
470	Art	46810
471	Al	46811
472	Sam	46812
473	Fred	46813
474	Bert	46814
475	Bob	46815
476 - 485	Canyon No. 34 - Canyon No. 43	47482 - 47491
486 - 501	Canyon No. 64 - Canyon No. 79	47512 - 47527
502	Telemeter Fraction	62785
503	West End Fraction	62786
504	Hattie Fraction	62787
505	Cactus	64123
506	Travis #7	64124
507	Fox #1	64125
508	Fox #2	64126
509	Fox #7	64131
510	Fox #13	64133
511	Cloud Rest	64134
512	Big Windy	64135
513	Big Windy Fraction	64136
514	Blue Wing	64137
515	Cloud Rest No. 1	64138
516	Kent #1 Long John	66835
517	Kent #2 Patricia C.	66836
518	Kent #3 Little Joe	66837
519	Belle of Rosemont	66838
520	John	74390
521	Joe	74391
522	Ben	74392
523	Pete	74393
524	Adolph Lewisohn	74394
525	Adolph Lewisohn	74395
526	Rosemont	74396
527	Rosemont	74397
528	Albert Steinfeld	74398
529	Albert Steinfeld	74399
530	Hugh Young	74400
531	Hugh Young	74401
532	Ethel	74402
533	Albert	74403

534 - 537	Rosemont #1 - Rosemont #4	74404 - 74407
538	Rosemont #7	74408
539	Rosemont #8	74409
540	Rosemont #14	74410
541	Rosemont #19	74411
542	Rosemont #20	74412
543	Rosemont #20	74413
544	Rosemont #22	74414
545 - 547	Rosemont #23 - Rosemont #25	74415 - 74417
548	RX	74418
549	Flying Dutchman #7A	75181
550	Blue Point No. 2A	75182
551 - 554	Alpine #1A - Alpine #4A	75183 - 75186
555	Frijole VI A	95315
556 - 577	Falcon 1A - Falcon 22A	99789 - 99819
578 - 583	Falcon 27A - Falcon 32A	99811 - 99816
584	Wasp 62A	99817
585	Wasp 63A	99818
586 - 589	Wasp 219A - Wasp 222A	99819 - 99822
590	Tecky	99823
591 - 599	MIA 1A - MIA 9A	117293 - 117301
600 - 602	MIA 12A - MIA 14A	117304 - 117306
603	BILLY C.	129394
604 - 613	Hope 1 - Hope 10	303950 - 303959
614	Hope 10-A	303960
615	Hope-11 - Hope-28	303961 - 303978
633	H29	303979
634 - 641	Hope-30 - Hope-37	303980 - 303987
642 - 775	H-38A - H-171A	313532-313665
776 - 794	H-177A - H-199A	313671 - 313689
795	Hope No. 201	330891
796	Hope 201A	330892
797 - 811	Hope No. 202 - Hope No. 216	330893 - 330907
812 - 815	Hope No. 222 - Hope No. 225	330910 - 330913
816 - 819	Hope 226A - Hope 229A	330914 - 330917
820 - 844	Hope No. 230 - Hope No. 257	330918 - 330942
845	Elk 47	330943
846 - 850	H-172B - H-176B	331308 - 331312
851	MMRE	367652
852	Tailor	367653
853 - 856	HV 1 - HV 4	380250 - 380253
857 - 865	ROSE 1 - Rose 9	385174 -385182
866	HV 6 - HV 13	387231 - 387238
874 - 876	HV 23 - HV 25	387241 - 387243
877 - 883	HV 16 - HV 22	390077 - 390083
884 - 915	WAIT-1 - WAIT-32	390084 - 390115
916	FALLS FRACTION	391154
917	H-69B	391155
918	NO CHANCE No. 3	391156
919	SCHWAB FRACTION	391157
920 - 927	H FRAC. 1 - H FRAC. 8	392445 - 392452
928	BILLY FRAC	393532
929 - 938	DSM 1 - DSM 10	393533 - 393542
939	HV5 A	393543
940	MIA FRAC 1	393544
941	MIA FRAC 2	393545
942	SON OF GUN 34	394006
943 - 946	RMT FRAC 1 - RMT FRAC4	394561 - 394564
947	NC-CF	396422
948	Thankful	404128
949 - 1048	RCC-1 - RCC-100	411964 - 412063
1049 - 1054	AGAVE-1 - AGAVE-6	412064 - 412069
1055 - 1060	CONTINENTAL-1 - CONTINENTAL-6	412070 - 412075

APPENDIX C: PATENTED CLAIMS & FEE LAND LIST

Rosemont Property Patented Claims		
Count	**Property Name**	**Pima County Parcel No.**
1	BLACK BESS	305540020
2	FLYING DUTCHMAN	305540030
3	WISCONSIN	305540040
4	EXCHANGE	305540050
5	EXCHANGE No. 2	305540060
6	COPPER WORLD	305540070
7	OWOSKO	305540080
8	BLACK HORSE	305540090
9	BRUNSWICK	305540100
10	ANTELOPE	305540110
11	NEWMAN	305550010
12	CHANCE	305550040
13	BLACK HAWK	305550050
14	TELEMETER	305550060
15	WEST END	305550070
16	HATTIE	305550080
17	SILVER SPUR	305550090
18	SLIDE	305550100
19	BACK BONE	305550110
20	BUZZARD	305550130
21	HEAVY WEIGHT	305550140
22	LIGHT WEIGHT	305550150
23	PEACH	305560040
24	SOUTH END	305560050
25	MONITOR	305560060
26	GAP	305560070
27	WATER WISH	305580080
28	NEW MEXICO	305580090
29	GRIZZLY	305580100
30	OLD DICK	305580110
31	AMERICAN	305580120
32	RECORDER	305580130
33	MOHAWK	305580140
34	WEDGE	305580150
35	DAN	305580160
36	GENERAL	305580170
37	ELGIN	305580180
38	SUNSETE	305580190
39	TELEPHONE	305580200
40	ELGIN MILLSITE	305580220
41	DAN MILLSITE	305580250
42	WEDGE MILLSITE	305580260

43	OLD DICK MILLSITE	305580270
44	ARCOLA	305590060
45	BONNIE BLUE	305590070
46	KING	305590080
47	EXILE	305590090
48	VULTURE	305590100
49	ISLE ROYAL	305590110
50	INDIAN CLUB	305590120
51	A.O.T.	305590130
52	BALTIMORE	305590140
53	PILOT	305590150
54	LITTLE DAVE	305590160
55	COPPER FEND	305590170
56	TALLY HO	305590180
57	LEADER	305590190
58	OMEGA	305590200
59	ECLIPSE COPPER	305590220
60	SCHWAB	305590230
61	NARRAGANSETT BAY	305590240
62	LANDOR	305590250
63	WARD	305590260
64	ALTA COPPER	305590270
65	BROAD TOP	305590280
66	MALACHITE	305590290
67	YORK	305600040
68	OLCOTT	305600050
69	HILO CONSOLIDATED	305600060
70	ELDON	305600070
71	RAINBOW	305600080
72	AJAX CONSOLIDATED	305600090
73	CUBA	305600100
74	FALLS	305600110
75	OLD PUT CON	305600130
76	FRANKLIN	305600140
77	CUSHING	305600150
78	CENTRAL	305600160
79	POTOMAC	305600170
80	MARION	305610010
81	EXCELSIOR	305610030
82	EMPIRE	305610040
83	ALTAMONT	305610050
84	ERIE	305610060
85	CHICAGO	305610080
86	COCONINO	305610090
87	OLUSTEE	305630020

88	AMOLE	305630040
89	CHICAGO MILLSITE	305640020
90	COCONINO MILLSITE	305640030
91	OLD PUT MILLSITE	305640040
92	OREGON MILLSITE	305640050
93	OLD PAP MILLSITE	305640060
94	AJAX CONSOLIDATED MILLSITE	305640070
95	R. G. INGERSOLL	305650020
96	PATRICK HENRY	305650040
97	MOHAWK SILVER	305660050
98	TREMONT	305660060
99	BLUE POINT	30554012A
100	HEAVY WEIGHT MILLSITE	30555012A
101	TELEPHONE MILLSITE	30558021A
102	RECORDER MILLSITE	30558023A
103	AMERICAN MILLSITE	30558024A
104	OMEGA FIRST EXTENSION SOUTH	30559021A
105	DAYLIGHT	30560003A
106	OLD PAP COPPER	30560012A
107	FALLS NO. 2	30560012D
108	WEDGE NO. 2	30560012F
109	WEDGE	30560012G
110	SANTA RITA FRACTION	30560012H
111	SANTA RITA #13	30560012J
112	OREGON COPPER	30561007A
113	SANTA RITA #15	30561007D
114	SANTA RITA #14	30561007E
115	SANTA RITA #12	30561007F
116	LAST CHANCE NO. 1	30561007G
117	LAST CHANCE NO. 2	30561007H
118	SANTA RITA #26	30561007J
119	SANTA RITA #27	30561007K
120	SANTA RITA #28	30561007L
121	SANTA RITA #16	30562034C
122	CUPRITE	30563003A
123	FRANKLIN MILLSITE	30564008A
124	LA FAYETTE	30565003A
125	SANTA RITA #4	30565003D
126	SANTA RITA #5	30565003E
127	SANTA RITA #6	30565003F
128	SANTA RITA #8A	30565003G
129	SANTA RITA #9	30565003H
130	SANTA RITA #10	30565003J
131	SANTA RITA #11	30565003K
132	DAN WEBSTER	30565005A

Rosemont Project Fee Lands		
Count	**Property Name**	**Pima County Parcel No.**
1	(SANRITA PROP./SAHAURITA 53	303601410
2	HELVETIA RANCH	305580280
3	HELVETIA RANCH ANNEX	305580330
4	HELVETIA RANCH ANNEX	305580360
5	DE LA OSSA	305580350
6	HELVETIA RANCH ANNEX	305580370
7	SINGING VALLEY	307200040
8	SANRITA SOUTH	30354005B
9	SANRITA EAST	30363013C
10	SANRITA EAST	30363013D
11	WILMOT JUNCTION	30365003C
12	WILMOT JUNCTION	30365003E
13	WILMOT JUNCTION	30365003F
14	WILMOT JUNCTION	30365004A
15	HELVETIA RANCH ANNEX	30553002H
16	HELVETIA RANCH	30553004D
17	HELVETIA RANCH ANNEX	30553004H
18	HELVETIA RANCH ANNEX	30556001B
19	HELVETIA RANCH ANNEX	30556001C
20	HELVETIA RANCH ANNEX	30557004B
21	HELVETIA RANCH ANNEX	30557004C
22	HELVETIA RANCH	30557004D
23	HELVETIA RANCH ANNEX	30557005B
24	HELVETIA RANCH ANNEX	30557013B
25	HELVETIA RANCH ANNEX	30557013C
26	HELVETIA RANCH ANNEX	30557013D
27	HELVETIA RANCH ANNEX	30557013E
28	PIPELINE TRIANGLE	30558034C
29	ROSEMONT RANCH	30562006B
30	ROSEMONT RANCH	30562007D
31	ROSEMONT RANCH	30562007F
32	ROSEMONT RANCH	30562007G
33	ROSEMONT RANCH	30562007H
34	ROSEMONT RANCH (HIDDEN VALLEY)	30562008C
35	ROSEMONT RANCH (HIDDEN VALLEY)	30562008F
36	ROSEMONT RANCH (HIDDEN VALLEY)	30562008G
37	ROSEMONT RANCH (HIDDEN VALLEY	30562008H
38	ROSEMONT RANCH (HIDDEN VALLEY)	30562008J
39	ROSEMONT RANCH	30562009A
40	ROSEMONT RANCH	30562011A
41	ROSEMONT RANCH	30562012A
42	ROSEMONT RANCH	30562012C
43	SINGING VALLEY	30717001G
44	SINGING VALLEY	30717001K
45	SINGING VALLEY	30717001N
46	SINGING VALLEY	30718003D
47	SINGING VALLEY	30720003D
48	SINGING VALLEY	30720003F

APPENDIX D: LG SET 1 CASES

ROSEMONT FEASIBILITY STUDY 2012 - INPUT METAL PRICES FOR LERCHS GROSSMAN SENSITIVITIES									
	Net Price at Mine				Price at Market (less Off Sites Costs)				Approx Pit Ph #
	Variance	Cu $/lb	Mo $/lb	Ag $/oz	Variance	Cu $/lb	Mo $/lb	Ag $/oz	
P04a	10%	0.208	1.309	1.711	25%	0.629	3.215	4.600	
P05a	20%	0.416	2.619	3.422	33%	0.837	4.524	6.311	
P06a	30%	0.624	3.928	5.133	42%	1.045	5.834	8.022	
P07a	32%	0.665	4.190	5.475	43%	1.087	6.096	8.365	1
P08a	34%	0.707	4.452	5.818	45%	1.128	6.358	8.707	2,3
P09a	36%	0.748	4.714	6.160	47%	1.170	6.619	9.049	4
P10a	38%	0.790	4.976	6.502	48%	1.211	6.881	9.391	
P11a	40%	0.831	5.238	6.844	50%	1.253	7.143	9.734	
P12a	41%	0.852	5.369	7.015	51%	1.274	7.274	9.905	5
P13a	42%	0.873	5.500	7.187	52%	1.294	7.405	10.076	
P14a	43%	0.894	5.631	7.358	53%	1.315	7.536	10.247	
P15a	44%	0.915	5.762	7.529	53%	1.336	7.667	10.418	
P16a	45%	0.935	5.893	7.700	54%	1.357	7.798	10.589	
P17a	46%	0.956	6.024	7.871	55%	1.378	7.929	10.760	
P18a	47%	0.977	6.154	8.042	56%	1.398	8.060	10.931	
P19a	48%	0.998	6.285	8.213	57%	1.419	8.191	11.102	
P20a	49%	1.018	6.416	8.384	58%	1.440	8.322	11.273	
P21a	50%	1.039	6.547	8.555	58%	1.461	8.453	11.445	6
P22a	51%	1.060	6.678	8.727	59%	1.482	8.584	11.616	
P23a	52%	1.081	6.809	8.898	60%	1.502	8.715	11.787	
P24a	53%	1.102	6.940	9.069	61%	1.523	8.846	11.958	
P25a	54%	1.122	7.071	9.240	62%	1.544	8.976	12.129	
P26a	56%	1.164	7.333	9.582	63%	1.585	9.238	12.471	
P27a	58%	1.206	7.595	9.924	65%	1.627	9.500	12.813	
P28a	60%	1.247	7.857	10.266	67%	1.669	9.762	13.156	
P29a	62%	1.289	8.119	10.609	68%	1.710	10.024	13.498	
P30a	64%	1.330	8.381	10.951	70%	1.752	10.286	13.840	
P31a	66%	1.372	8.642	11.293	72%	1.793	10.548	14.182	
P32a	68%	1.413	8.904	11.635	73%	1.835	10.810	14.525	
P33a	**70%**	**1.455**	**9.166**	**11.978**	**75%**	**1.876**	**11.072**	**14.867**	**7-Ultimate**
P34a	72%	1.497	9.428	12.320	77%	1.918	11.334	15.209	
P35a	74%	1.538	9.690	12.662	78%	1.960	11.595	15.551	
P36a	76%	1.580	9.952	13.004	80%	2.001	11.857	15.893	
P37a	78%	1.621	10.214	13.346	82%	2.043	12.119	16.236	
P38a	80%	1.663	10.476	13.689	83%	2.084	12.381	16.578	
P39a	82%	1.704	10.738	14.031	85%	2.126	12.643	16.920	
P40a	84%	1.746	10.999	14.373	87%	2.167	12.905	17.262	
P41a	86%	1.787	11.261	14.715	88%	2.209	13.167	17.604	
P42a	88%	1.829	11.523	15.058	90%	2.251	13.429	17.947	
P43a	90%	1.871	11.785	15.400	92%	2.292	13.691	18.289	
P44a	95%	1.975	12.440	16.255	96%	2.396	14.345	19.144	
P45a	**100%**	2.078	13.095	17.111	100%	2.500	15.000	20.000	
P46a	105%	2.182	13.749	17.966	104%	2.604	15.655	20.856	
P47a	110%	2.286	14.404	18.822	108%	2.708	16.309	21.711	
P48a	115%	2.390	15.059	19.677	112%	2.812	16.964	22.567	
P49a	120%	2.494	15.714	20.533	117%	2.916	17.619	23.422	
P50a	125%	2.598	16.368	21.389	121%	3.020	18.274	24.278	

LG1a.xlsx-Input Metal Prices 1 of 1 14/08/2012

ROSEMONT FEASIBILITY STUDY 2012 - RESULTS FOR LERCHS GROSSMAN SENSITIVITIES

Run on April 15 Model Version*				Oxide Ore at $3.03 /t NSR Cut-off					Mixed and Sulfide Ore at $4.90 /t NSR Cut-off						
Pit	Pitres Report	Variance from Base	Cu Price $/lb	Oxide k-tons	NSRM $/ton	Cu %	Mo %	Ag oz/ton	Mixed & Sulfides k-tons	NSRM $/ton	Cu %	Mo %	Ag oz/ton	Waste k-tons	SR
P04a	p04a.rpt	-90%	0.63	0	0.00	0.000	0.000	0.000	30	167.52	4.203	0.034	0.858	253	8.43
P05a	p05a.rpt	-80%	0.84	0	0.00	0.000	0.000	0.000	30	167.52	4.203	0.034	0.858	1,569	52.30
P06a	p06a.rpt	-70%	1.05	18,339	5.40	0.171	0.005	0.016	65,175	24.45	0.553	0.016	0.154	109,141	1.31
P07a	p07a.rpt	-68%	1.09	22,011	5.60	0.178	0.005	0.017	80,195	24.30	0.551	0.015	0.151	136,073	1.33
P08a	p08a.rpt	-66%	1.13	33,569	5.76	0.183	0.004	0.024	202,193	22.54	0.504	0.016	0.141	340,060	1.44
P09a	p09a.rpt	-64%	1.17	60,446	5.41	0.171	0.003	0.021	339,536	21.73	0.491	0.015	0.130	537,001	1.34
P10a	p10a.rpt	-62%	1.21	64,175	5.37	0.170	0.003	0.020	385,077	21.38	0.481	0.015	0.128	580,100	1.29
P11a	p11a.rpt	-60%	1.25	65,113	5.38	0.170	0.003	0.020	410,625	21.16	0.477	0.015	0.128	607,078	1.28
P12a	p12a.rpt	-59%	1.27	65,628	5.37	0.170	0.003	0.020	424,097	21.03	0.475	0.015	0.127	621,569	1.27
P13a	p13a.rpt	-58%	1.29	66,156	5.38	0.170	0.003	0.020	448,131	20.90	0.473	0.015	0.126	665,356	1.29
P14a	p14a.rpt	-57%	1.32	66,238	5.39	0.171	0.003	0.020	451,135	20.86	0.473	0.015	0.126	667,626	1.29
P15a	p15a.rpt	-56%	1.34	66,341	5.39	0.171	0.003	0.020	463,690	20.76	0.471	0.015	0.126	685,548	1.29
P16a	p16a.rpt	-55%	1.36	66,399	5.39	0.171	0.003	0.020	478,620	20.65	0.470	0.015	0.126	709,368	1.30
P17a	p17a.rpt	-54%	1.38	66,430	5.39	0.171	0.003	0.020	489,071	20.59	0.468	0.015	0.125	730,458	1.31
P18a	p18a.rpt	-53%	1.40	66,477	5.39	0.171	0.003	0.020	491,952	20.56	0.468	0.015	0.125	733,894	1.31
P19a	p19a.rpt	-52%	1.42	66,518	5.39	0.171	0.003	0.020	507,816	20.41	0.465	0.015	0.124	755,214	1.31
P20a	p20a.rpt	-51%	1.44	66,805	5.40	0.171	0.003	0.020	531,611	20.27	0.463	0.015	0.123	807,855	1.35
P21a	p21a.rpt	-50%	1.46	66,805	5.40	0.171	0.003	0.020	557,919	20.12	0.458	0.015	0.123	861,753	1.38
P22a	p22a.rpt	-49%	1.48	66,825	5.40	0.171	0.003	0.020	558,465	20.12	0.458	0.015	0.123	862,407	1.38
P23a	p23a.rpt	-48%	1.50	66,825	5.40	0.171	0.003	0.020	584,030	20.00	0.456	0.015	0.123	923,850	1.42
P24a	p24a.rpt	-47%	1.52	66,858	5.40	0.171	0.003	0.020	588,096	19.96	0.455	0.015	0.122	930,094	1.42
P25a	p25a.rpt	-46%	1.54	66,858	5.40	0.171	0.003	0.020	597,225	19.87	0.452	0.015	0.122	941,783	1.42
P26a	p26a.rpt	-44%	1.59	66,893	5.40	0.171	0.003	0.020	608,826	19.82	0.451	0.015	0.122	973,386	1.44
P27a	p27a.rpt	-42%	1.63	66,914	5.40	0.171	0.003	0.020	631,222	19.71	0.449	0.015	0.121	1,039,202	1.49
P28a	p28a.rpt	-40%	1.67	66,914	5.40	0.171	0.003	0.020	638,082	19.65	0.447	0.015	0.121	1,053,327	1.49
P29a	p29a.rpt	-38%	1.71	66,945	5.40	0.171	0.003	0.020	653,665	19.56	0.445	0.015	0.121	1,100,115	1.53
P30a	p30a.rpt	-36%	1.75	66,965	5.40	0.171	0.003	0.020	656,765	19.53	0.445	0.015	0.121	1,108,396	1.53
P31a	p31a.rpt	-34%	1.79	66,975	5.40	0.171	0.003	0.020	668,626	19.46	0.442	0.014	0.121	1,147,113	1.56
P32a	p32a.rpt	-32%	1.83	66,985	5.40	0.171	0.003	0.020	673,885	19.41	0.441	0.014	0.120	1,158,854	1.56
P33a	p33a.rpt	-30%	1.88	66,985	5.40	0.171	0.003	0.020	679,591	19.37	0.440	0.014	0.120	1,176,233	1.58
P34a	p34a.rpt	-28%	1.92	66,996	5.40	0.171	0.003	0.020	687,320	19.31	0.439	0.014	0.120	1,201,569	1.59
P35a	p35a.rpt	-26%	1.96	67,016	5.40	0.171	0.003	0.020	692,445	19.31	0.438	0.014	0.120	1,239,625	1.63
P36a	p36a.rpt	-24%	2.00	67,016	5.40	0.171	0.003	0.020	703,640	19.24	0.437	0.014	0.119	1,284,656	1.67
P37a	**p37a.rpt**	**-22%**	2.04	**67,026**	**5.40**	**0.171**	**0.003**	**0.020**	**706,824**	**19.21**	**0.436**	**0.014**	**0.119**	**1,291,218**	**1.67**
P38a	p38a.rpt	-20%	2.08	67,026	5.40	0.171	0.003	0.020	710,803	19.18	0.435	0.014	0.119	1,306,407	1.68
P39a	p39a.rpt	-18%	2.13	67,036	5.40	0.171	0.003	0.020	715,765	19.13	0.434	0.014	0.119	1,323,099	1.69
P40a	p40a.rpt	-16%	2.17	67,036	5.40	0.171	0.003	0.020	716,610	19.12	0.434	0.014	0.119	1,324,915	1.69
P41a	p41a.rpt	-14%	2.21	67,036	5.40	0.171	0.003	0.020	731,218	19.03	0.431	0.014	0.118	1,397,578	1.75
P42a	p42a.rpt	-12%	2.25	67,036	5.40	0.171	0.003	0.020	733,203	19.01	0.430	0.014	0.118	1,401,872	1.75
P43a	p43a.rpt	-10%	2.29	67,036	5.40	0.171	0.003	0.020	734,602	18.99	0.430	0.014	0.118	1,406,390	1.75
P44a	p44a.rpt	-5%	2.40	67,036	5.40	0.171	0.003	0.020	746,442	18.91	0.428	0.014	0.118	1,466,341	1.80
P45a	**p45a.rpt**	**0%**	2.50	**67,036**	**5.40**	**0.171**	**0.003**	**0.020**	**750,595**	**18.86**	**0.427**	**0.014**	**0.118**	**1,478,467**	**1.81**
P46a	p46a.rpt	5%	2.60	67,056	5.40	0.171	0.003	0.020	758,656	18.80	0.425	0.014	0.117	1,530,126	1.85
P47a	p47a.rpt	10%	2.71	67,067	5.40	0.171	0.003	0.020	762,111	18.76	0.424	0.014	0.117	1,543,550	1.86
P48a	p48a.rpt	15%	2.81	67,077	5.40	0.171	0.003	0.020	764,468	18.74	0.423	0.014	0.117	1,551,299	1.87
P49a	p49a.rpt	20%	2.92	67,087	5.40	0.171	0.003	0.020	769,178	18.69	0.422	0.014	0.117	1,576,192	1.88
P50a	p50a.rpt	25%	3.02	67,129	5.40	0.171	0.003	0.020	774,668	18.63	0.421	0.014	0.116	1,606,891	1.91

* This model version (312) was updated during the study, and subsequent LGs were run on the updated version (712).

LG1a.xlsx-LG1a1 1 of 1 14/08/2012

APPENDIX E: RESERVES REPORT

```
*** ------------------------------------*----------------------------*
***                                     *   Project RUN#  4705.       *
***    M708V1   - Revised on 06-AUG-09  *   Date started  05-28-2012  *
***                                     *   Time started  14:56:40    *
***                                     *   Project Acct  N/A         *
*** ------------------------------------*----------------------------*


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               MineSight Basis Release 2011.10.31

-----------------------------------------------------------------------
Run File = RUN708.TMP   Print File = P621.RPT

Line          R U N   F I L E   R E C O R D S
----
  1  MEDS-708V1  10=rose10.dat  15=rose15.dat  3=P621.rpt  28=P621.sum
  2  MEDS-708V1  29=zone1.zon  24=P621.scd  26=
  3  CONTENTS OF P621 PIT, BY CLZN
  4  USR =
  5  I-O = 0
  6
  7  COM Partials file: P621.out
  8
  9  IOP1 = 1  / Partials type: 1=Integer
 10  IOP3 =  2 / Call USR708 by row, @ end of level and end of run
 11  IOP4 =  0 / 1=Geologic reserve
 12  IOP5 =  1. 240. / X1-->X2
 13  IOP7 =  1. 200. / Y1-->Y2
 14  IOP9 =  0 / 1=Open output ASCII file on unit 19
 15  IOP11 = 0    / Number of grade items (set by program)
 16  IOP12 = 1 / Number of zones per block
 17  IOP13 = 0    / Number of grade cutoffs (set by program)
 18  IOP14 = 2 / Number of decimal places in cutoff value
 19  IOP15 = 0 / 1=Omit 1st grade item for scd file
 20  IOP16 = 1 / 0=Report waste volume; 1=Tonnage; 2=Long Tons; 3=Tonnes
 21  IOP17 = 0 / 1=Output MineSight Scheduling File using cutoffs, 2=using zone item
 22  IOP18 = 1 / 0=Report INSITU grades; 1=Diluted
 23  IOP19 = 0 / 0=Apply DILN to INSITU grades; 1=To recovered ore
 24  IOP20 = 0 / 0=Waste ore below cutoff; 1=Above
```

```
RUN# 4705.    Page    2    METL 708V1    Date 05-28-2012   Time 14:56:40
----------------------------------------------------------------------------
 ROSEMONT DEPOSIT - JANUARY 2007 MODEL

   CONTENTS OF P621 PIT, BY CLZN

 IOP21 = 1 / 0=Take ore*prt; 1=Ore 1st for TOPO<100; 2=Ore 1st for all blks
 IOP22 = 1 / 0=Take topo*prt; 1=Take min(TOPO,prt)
 IOP23 = 0. / 0=Use zone input file; 1=Don't(RECV=100,DILN=0 for all zones)
 IOP24 = 1 / 0=Use ore% item; 1=Don't
 IOP25 = 1 / 1=Don't use TOPO
 IOP26 = 0. / 0=Use zone item; 1=Don't
 IOP27 = 1 / Grade item# to use for ore/waste cutoff
 IOP28 = 1 / 1=Use waste density from model
 IOP30 = 1 / 1=Report all zones in totals section
 IOP31 = 0 / 1=Apply DIL'N to ore/waste contact blocks only
 IOP32 = 1 / 1=Use density from model
 IOP33 = 0 / -1,1=Use waste types, 2=waste pcts, 3=both
 IOP34 = 1    / 1=Report cutoffs in bench summary
 IOP35 = 1 / 1=Report summary sections only; 2=Summary + bench summary
 IOP36 = 0 / 1=Report cumulative >= each cutoff
 IOP37 = 1 / 1=Treat missing grades as 0
 IOP38 = 0 / 1=Read waste type input file
 IOP39 = 0 / Number of waste pcts items
 IOP40 = 0 / 1=Mine zone 1 ore first

 GET15=TOPO
 GET15=
 GET15=TF CLZN  TF
 GET15=NSRM CUOK MOOK AGOK


 PAR1 = 0     / 1=Output grade file on unit 25. Requires special version.
 PAR3 = 2500. /Toe elevation of bottom bench
 PAR6 = 6500. /Toe elevation of top bench
 PAR4 = 0 / Optional waste type to output to scd file
 PAR25 = 0 0 0 0 / Optional waste types to output to scd file
 PAR5 = 0 / Maximum number of lines per page
 PAR7 = 11.85   / Default density of ore
 PAR8 = 0.01    / Ore and waste items are pct (0.01) or fraction (1)
 PAR9 = 0.001   / Factor for reporting
 PAR10= 11.85    / Density of waste
 PAR11 = 0   / Grade cutoff on 1st grade item
 PAR17 = 0.  / Min ore% for applying DIL'N to contact block
 END
```

```
RUN# 4705.    Page    3    METL 708V1    Date 05-28-2012   Time 14:56:40
----------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    CONTENTS OF P621 PIT, BY CLZN

ITEM#  1  TOPO
ITEM#  2  TF
ITEM#  3  CLZN
ITEM#  4  TF
ITEM#  5  NSRM
ITEM#  6  CUOK
ITEM#  7  MOOK
ITEM#  8  AGOK
     COORDINATES  OF  MINE  MODEL  FILE   ROSE15.DAT
     XMIN, XMAX, DX, NX=  1710000.0  1722000.0  50.0  240
     YMIN, YMAX, DY, NY= 11550000.0 11560000.0  50.0  200
     ZMIN, ZMAX, DZ, NZ=     2500.0     6500.0  50.0   80


     Data file ZONE1.ZON  *  unit  size from program
     -------------------- *  ---  ---- ----  ----
     ZONE DATA       file *    29



     Data file P621.SUM   *  unit  size from program
     -------------------- *  ---  ---- ----  ----
     RESERVE SUMMARY file *    28



     Data file P621.OUT   *  unit  size from program
     -------------------- *  ---  ---- ----  ----
     USER INPUT DATA file *    30
```

RUN# 4705. Page 4 METL 708V1 Date 05-28-2012 Time 14:56:40

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

CONTENTS OF P621 PIT, BY CLZN

GRADE ITEMS FOR DEFAULTS: NSRM CUOK MOOK AGOK

NAME	ZONE#	DENS	RECV	DILN	DIL DENS	DILN GRADES, CUTOFF GRADES							
MOX	1	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
MMX	2	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
MSU	3	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
IOX	4	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
IMX	5	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
ISU	6	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
FOX	7	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	3.030	0.000	0.000	0.000
FMX	8	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
FSU	9	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
UNDEF	10	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000

RUN# 4705. Page 5 METL 708V1 Date 05-28-2012 Time 14:56:40

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

CONTENTS OF P621 PIT, BY CLZN

BENCH TOE	ZONE NAME	ZONE NO.	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
TOTALS	MOX	1	5741.	12768.	12768.	5.093	0.162	0.0043	0.0184
	MMX	2	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	MSU	3	15356.	36335.	36335.	22.314	0.500	0.0157	0.1360
	IOX	4	3881.	8600.	8600.	5.005	0.159	0.0049	0.0161
	IMX	5	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	ISU	6	10630.	25211.	25211.	22.486	0.501	0.0163	0.1387
	FOX	7	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FMX	8	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FSU	9	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	UNDEF	10	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
TOTALS	SUMMARY		35607.	82914.	82914.	17.919	0.413	0.0130	0.1063

WASTE 121361. (kTONS) ROM S/R= 1.46

RUN# 4705. Page 6 METL 708V1 Date 05-28-2012 Time 14:56:40

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

CONTENTS OF P621 PIT, BY CLZN

BENCH TOE	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	WASTE TOTAL (kTONS)	ROM S/R	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
5650.0	0.	0.	0.	34.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5600.0	0.	0.	0.	186.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5550.0	0.	0.	0.	730.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5500.0	0.	0.	0.	1781.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5450.0	0.	0.	0.	3537.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5400.0	9.	19.	19.	5346.	283.08	7.850	0.249	0.0016	0.0027
5350.0	106.	233.	233.	7731.	33.15	7.145	0.227	0.0044	0.0838
5300.0	492.	1110.	1110.	8707.	7.84	8.693	0.215	0.0073	0.0718
5250.0	1408.	3176.	3176.	10453.	3.29	7.621	0.189	0.0076	0.0444
5200.0	1956.	4416.	4416.	12827.	2.90	6.666	0.166	0.0078	0.0363
5150.0	2173.	4921.	4921.	15293.	3.11	7.198	0.183	0.0069	0.0377
5100.0	2343.	5292.	5292.	12906.	2.44	8.707	0.222	0.0070	0.0542
5050.0	2403.	5423.	5423.	11559.	2.13	8.378	0.210	0.0076	0.0521
5000.0	2284.	5190.	5190.	9465.	1.82	10.445	0.246	0.0089	0.0681
4950.0	2328.	5348.	5348.	7933.	1.48	12.834	0.295	0.0102	0.0796
4900.0	2233.	5201.	5201.	5878.	1.13	19.288	0.428	0.0154	0.1203
4850.0	2570.	6025.	6025.	4078.	0.68	22.024	0.498	0.0156	0.1343
4800.0	2574.	6093.	6093.	2150.	0.35	22.713	0.518	0.0153	0.1259
4750.0	2867.	6810.	6810.	695.	0.10	23.587	0.524	0.0190	0.1273
4700.0	2497.	5957.	5957.	49.	0.01	26.917	0.626	0.0165	0.1333
4650.0	2172.	5199.	5199.	13.	0.00	25.438	0.588	0.0168	0.1212
4600.0	1553.	3732.	3732.	0.	0.00	25.689	0.585	0.0165	0.1509
4550.0	1302.	3133.	3133.	0.	0.00	29.622	0.679	0.0169	0.1866
4500.0	883.	2132.	2132.	8.	0.00	31.191	0.690	0.0213	0.2219
4450.0	716.	1727.	1727.	0.	0.00	26.927	0.587	0.0175	0.2253
4400.0	466.	1125.	1125.	0.	0.00	26.123	0.581	0.0148	0.2120
4350.0	271.	654.	654.	0.	0.00	23.226	0.511	0.0131	0.2046
TOTAL	35607.	82914.	82914.	121361.	1.46	17.919	0.413	0.0130	0.1063

```
RUN# 4705.    Page    7    METL 708V1   Date 05-28-2012  Time 14:56:40
----------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    CONTENTS OF P621 PIT, BY CLZN


         *** M708V1   finished on 05-28-2012  14:56:41


*** Current program execution:  Elapsed time (sec)   Date        Time
             M708V1                       1        05-28-2012  14:56:41
```

```
*** ------------------------------------*-----------------------------*
***                                     *  Project RUN#  4712.        *
***    MTINC    - Revised on 23-AUG-10  *  Date started  05-28-2012   *
***                                     *  Time started  14:57:04     *
***                                     *  Project Acct  N/A          *
*** ------------------------------------*-----------------------------*


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-----------------------------------------------------------------------
Run File = RUNINC.TMP   Print File = P622I.RPT

Line          R U N   F I L E   R E C O R D S
----
  1  MEDS-MTINC 10=rose10.dat 15=rose15.dat 3=P622i.rpt 27=P622.sum;
  2  MEDS-MTINC 31=P622i.sum 29=zone1.zon 28=P621.sum  24=P622i.scd
  3  INCREMENTAL CONTENTS OF P622
  4  USR =
  5  I-O = 0
  6  IOP2 = 0 / 0=Subtract smaller from larger; 1=Add them; 2=Larger only
  7  IOP17 = 0 / 1=Output MineSight Scheduling File using cutoffs, 2=using zone item
  8  IOP14 = 2 / Number of decimal places in cutoff value
  9  IOP15 = 0 / 1=Omit 1st grade item for scheduling file
 10  IOP19 = 0 / 1=Summarize to larger benches
 11  IOP23 = 0. / 1=No zone input file
 12  IOP30 = 1 / 1=Report all zones in totals section
 13  IOP34 = 1    / 1=Report cutoffs in bench summary
 14  IOP35 = 1 / 1=Report summary sections only; 2=Summary + bench summary
 15  IOP36 = 0 / 1=Report cumulative >= each cutoff
 16  PAR1 = 0. /Top bench for summarizing if IOP19=1
 17  PAR2 = 0. /Bot bench for summarizing if IOP19=1
 18  PAR3 = 0. /Bench height for summarizing if IOP19=1
 19  PAR4 = 0 / Optional waste type to output to scd file (-1 means PAR25-26 are range of codes)
 20  PAR25 = 0 0 0 0 / Optional waste types to output to scd file.
 21  PAR5 = 0 / Maximum number of lines per page
 22  PAR9 = 0.001 / Factor for reporting
 23  END
```

```
RUN# 4712.    Page    2    METL MTINC    Date 05-28-2012  Time 14:57:04
---------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P622


     Data file ZONE1.ZON  *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     ZONE DATA       file *    29


     Data file P622.SUM   *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     LARGER SUMMARY  file *    27


     Data file P621.SUM   *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     SMALLER SUMMARY file *    28


     Data file P622I.SUM  *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     RESERVE SUMMARY file *    31
```

RUN# 4712. Page 3 METL MTINC Date 05-28-2012 Time 14:57:04

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P622

GRADE ITEMS FOR DEFAULTS: NSRM CUOK MOOK AGOK

NAME	ZONE#	DENS	RECV	DILN	DIL DENS	DILN GRADES, CUTOFF GRADES							
MOX	1	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
MMX	2	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
MSU	3	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
IOX	4	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
IMX	5	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
ISU	6	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
FOX	7	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	3.030	0.000	0.000	0.000
FMX	8	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
FSU	9	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
UNDEF	10	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000

RUN# 4712. Page 4 METL MTINC Date 05-28-2012 Time 14:57:04

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P622

BENCH TOE	ZONE NAME	ZONE NO.	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
TOTALS	MOX	1	2616.	6074.	6074.	6.540	0.207	0.0020	0.0434
	MMX	2	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	MSU	3	6929.	16334.	16334.	17.271	0.399	0.0111	0.0864
	IOX	4	2710.	6284.	6284.	5.711	0.181	0.0022	0.0445
	IMX	5	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	ISU	6	4592.	10835.	10835.	17.200	0.398	0.0109	0.0855
	FOX	7	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FMX	8	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FSU	9	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	UNDEF	10	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
TOTALS	SUMMARY		16846.	39527.	39527.	13.764	0.335	0.0082	0.0729

WASTE 72168. (kTONS) ROM S/R= 1.83

RUN# 4712. Page 5 METL MTINC Date 05-28-2012 Time 14:57:04

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P622

BENCH TOE	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	WASTE TOTAL (kTONS)	ROM S/R	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
5800.0	0.	0.	0.	2.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5750.0	0.	0.	0.	97.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5700.0	0.	0.	0.	210.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5650.0	0.	0.	0.	534.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5600.0	0.	0.	0.	668.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5550.0	0.	0.	0.	933.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5500.0	0.	0.	0.	982.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5450.0	1.	1.	1.	1127.	804.93	5.119	0.162	0.0000	0.0000
5400.0	37.	86.	86.	1168.	13.58	8.994	0.285	0.0000	0.0041
5350.0	246.	574.	574.	1896.	3.30	7.273	0.231	0.0001	0.0039
5300.0	488.	1139.	1139.	2300.	2.02	5.855	0.186	0.0004	0.0140
5250.0	553.	1285.	1285.	3520.	2.74	5.523	0.175	0.0013	0.0278
5200.0	1010.	2352.	2352.	3770.	1.60	6.162	0.195	0.0028	0.0686
5150.0	1072.	2495.	2495.	5760.	2.31	7.040	0.219	0.0022	0.0638
5100.0	889.	2072.	2072.	8098.	3.91	6.927	0.210	0.0029	0.0218
5050.0	802.	1867.	1867.	7818.	4.19	6.963	0.199	0.0037	0.0322
5000.0	718.	1671.	1671.	7203.	4.31	8.190	0.215	0.0043	0.0280
4950.0	770.	1799.	1799.	6805.	3.78	8.325	0.205	0.0052	0.0298
4900.0	680.	1592.	1592.	5536.	3.48	9.356	0.222	0.0060	0.0362
4850.0	611.	1426.	1426.	5023.	3.52	11.413	0.275	0.0056	0.0535
4800.0	715.	1658.	1658.	3676.	2.22	12.720	0.318	0.0047	0.0727
4750.0	699.	1632.	1632.	3245.	1.99	14.749	0.358	0.0071	0.0767
4700.0	1214.	2838.	2838.	1456.	0.51	15.362	0.359	0.0099	0.0694
4650.0	1598.	3749.	3749.	340.	0.09	17.987	0.403	0.0148	0.0832
4600.0	1200.	2815.	2815.	2.	0.00	16.281	0.351	0.0160	0.0803
4550.0	1072.	2535.	2535.	0.	0.00	19.596	0.436	0.0163	0.0965
4500.0	836.	1993.	1993.	0.	0.00	25.601	0.601	0.0135	0.1389
4450.0	667.	1604.	1604.	0.	0.00	30.003	0.711	0.0127	0.1812
4400.0	507.	1223.	1223.	0.	0.00	26.376	0.615	0.0119	0.1779
4350.0	203.	490.	490.	0.	0.00	20.176	0.469	0.0111	0.1151
4300.0	260.	628.	628.	0.	0.00	20.843	0.473	0.0108	0.1578
TOTAL	16846.	39527.	39527.	72168.	1.83	13.764	0.335	0.0082	0.0729

```
RUN# 4712.    Page    6    METL MTINC    Date 05-28-2012  Time 14:57:04
---------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P622


         *** MTINC    finished on 05-28-2012  14:57:05


*** Current program execution:  Elapsed time (sec)   Date        Time
             MTINC                        1        05-28-2012  14:57:05
```

```
*** -----------------------------------*-----------------------------*
***                                    *  Project RUN#  4713.         *
***    MTINC    - Revised on 23-AUG-10 *  Date started  05-28-2012    *
***                                    *  Time started  14:57:08      *
***                                    *  Project Acct  N/A           *
*** -----------------------------------*-----------------------------*


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-----------------------------------------------------------------------
Run File = RUNINC.TMP   Print File = P623I.RPT

Line          R U N   F I L E   R E C O R D S
----
  1  MEDS-MTINC 10=rose10.dat 15=rose15.dat 3=P623i.rpt 27=P623.sum;
  2  MEDS-MTINC 31=P623i.sum 29=zone1.zon 28=P622.sum  24=P623i.scd
  3  INCREMENTAL CONTENTS OF P623
  4  USR =
  5  I-O = 0
  6  IOP2 = 0 / 0=Subtract smaller from larger; 1=Add them; 2=Larger only
  7  IOP17 = 0 / 1=Output MineSight Scheduling File using cutoffs, 2=using zone item
  8  IOP14 = 2 / Number of decimal places in cutoff value
  9  IOP15 = 0 / 1=Omit 1st grade item for scheduling file
 10  IOP19 = 0 / 1=Summarize to larger benches
 11  IOP23 = 0. / 1=No zone input file
 12  IOP30 = 1 / 1=Report all zones in totals section
 13  IOP34 = 1    / 1=Report cutoffs in bench summary
 14  IOP35 = 1 / 1=Report summary sections only; 2=Summary + bench summary
 15  IOP36 = 0 / 1=Report cumulative >= each cutoff
 16  PAR1 = 0. /Top bench for summarizing if IOP19=1
 17  PAR2 = 0. /Bot bench for summarizing if IOP19=1
 18  PAR3 = 0. /Bench height for summarizing if IOP19=1
 19  PAR4 = 0 / Optional waste type to output to scd file (-1 means PAR25-26 are range of codes)
 20  PAR25 = 0 0 0 0 / Optional waste types to output to scd file
 21  PAR5 = 0 / Maximum number of lines per page
 22  PAR9 = 0.001 / Factor for reporting
 23  END
```

```
RUN# 4713.    Page    2    METL MTINC   Date 05-28-2012  Time 14:57:08
---------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P623


     Data file ZONE1.ZON  *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     ZONE DATA       file *    29



     Data file P623.SUM   *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     LARGER SUMMARY  file *    27



     Data file P622.SUM   *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     SMALLER SUMMARY file *    28



     Data file P623I.SUM  *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     RESERVE SUMMARY file *    31
```

```
RUN# 4713.     Page     3    METL MTINC    Date 05-28-2012  Time 14:57:08
---------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P623

GRADE ITEMS FOR DEFAULTS: NSRM  CUOK  MOOK  AGOK
```

NAME	ZONE#	DENS	RECV	DILN	DIL DENS	DILN GRADES, CUTOFF GRADES							
MOX	1	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
MMX	2	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
MSU	3	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
IOX	4	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
IMX	5	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
ISU	6	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
FOX	7	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	3.030	0.000	0.000	0.000
FMX	8	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
FSU	9	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
UNDEF	10	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000

RUN# 4713. Page 4 METL MTINC Date 05-28-2012 Time 14:57:08

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P623

BENCH TOE	ZONE NAME	ZONE NO.	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
TOTALS	MOX	1	211.	480.	480.	4.884	0.155	0.0029	0.0451
	MMX	2	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	MSU	3	9795.	23075.	23075.	19.989	0.425	0.0175	0.1388
	IOX	4	399.	919.	919.	4.789	0.152	0.0014	0.0211
	IMX	5	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	ISU	6	8229.	19343.	19343.	18.642	0.373	0.0225	0.1176
	FOX	7	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FMX	8	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FSU	9	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	UNDEF	10	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
TOTALS	SUMMARY		18634.	43816.	43816.	18.910	0.393	0.0192	0.1259

WASTE 58154. (kTONS) ROM S/R= 1.33

RUN# 4713. Page 5 METL MTINC Date 05-28-2012 Time 14:57:08

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P623

BENCH TOE	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	WASTE TOTAL (kTONS)	ROM S/R	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
5400.0	0.	0.	0.	2.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5350.0	0.	0.	0.	6.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5300.0	1.	2.	2.	62.	30.80	3.079	0.098	0.0029	0.0271
5250.0	21.	47.	47.	550.	11.67	3.520	0.112	0.0014	0.0166
5200.0	207.	482.	482.	1726.	3.58	5.331	0.162	0.0011	0.0173
5150.0	424.	987.	987.	3140.	3.18	7.610	0.205	0.0009	0.0465
5100.0	455.	1057.	1057.	5007.	4.74	10.016	0.278	0.0015	0.0268
5050.0	460.	1069.	1069.	5619.	5.26	8.950	0.230	0.0021	0.0502
5000.0	345.	806.	806.	5360.	6.65	13.521	0.345	0.0023	0.0628
4950.0	240.	563.	563.	5419.	9.63	11.598	0.291	0.0028	0.0570
4900.0	149.	350.	350.	5975.	17.05	10.405	0.252	0.0055	0.0378
4850.0	343.	805.	805.	5684.	7.06	12.042	0.186	0.0284	0.0523
4800.0	511.	1196.	1196.	5455.	4.56	14.643	0.205	0.0384	0.0707
4750.0	377.	878.	878.	5426.	6.18	15.836	0.177	0.0530	0.0545
4700.0	597.	1383.	1383.	4138.	2.99	14.620	0.198	0.0414	0.0528
4650.0	969.	2247.	2247.	2880.	1.28	14.612	0.235	0.0326	0.0624
4600.0	1482.	3443.	3443.	1384.	0.40	16.401	0.315	0.0240	0.0874
4550.0	1882.	4376.	4376.	319.	0.07	15.737	0.322	0.0192	0.0770
4500.0	1785.	4159.	4159.	2.	0.00	17.217	0.356	0.0193	0.0962
4450.0	1798.	4208.	4208.	0.	0.00	19.554	0.402	0.0215	0.1187
4400.0	1458.	3434.	3434.	0.	0.00	22.606	0.478	0.0196	0.1671
4350.0	1512.	3603.	3603.	0.	0.00	28.662	0.635	0.0160	0.2448
4300.0	1139.	2738.	2738.	0.	0.00	26.706	0.587	0.0157	0.2278
4250.0	1130.	2727.	2727.	0.	0.00	24.656	0.547	0.0140	0.2049
4200.0	725.	1752.	1752.	0.	0.00	26.571	0.592	0.0132	0.2346
4150.0	623.	1504.	1504.	0.	0.00	23.876	0.531	0.0125	0.2063
TOTAL	18634.	43817.	43817.	58154.	1.33	18.910	0.393	0.0192	0.1259

```
RUN# 4713.    Page    6    METL MTINC   Date 05-28-2012  Time 14:57:08
---------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P623


         *** MTINC    finished on 05-28-2012  14:57:08


*** Current program execution:  Elapsed time (sec)   Date        Time
             MTINC                        0        05-28-2012  14:57:08
```

```
*** -----------------------------------*----------------------------*
***                                    *  Project RUN#  4714.       *
***    MTINC   - Revised on 23-AUG-10  *  Date started  05-28-2012  *
***                                    *  Time started  14:57:11    *
***                                    *  Project Acct  N/A         *
*** -----------------------------------*----------------------------*


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-----------------------------------------------------------------------
Run File = RUNINC.TMP   Print File = P624I.RPT

Line          R U N   F I L E   R E C O R D S
----
  1  MEDS-MTINC 10=rose10.dat 15=rose15.dat 3=P624i.rpt 27=P624.sum;
  2  MEDS-MTINC 31=P624i.sum 29=zone1.zon 28=P623.sum  24=P624i.scd
  3  INCREMENTAL CONTENTS OF P624
  4  USR =
  5  I-O = 0
  6  IOP2 = 0 / 0=Subtract smaller from larger; 1=Add them; 2=Larger only
  7  IOP17 = 0 / 1=Output MineSight Scheduling File using cutoffs, 2=using zone item
  8  IOP14 = 2 / Number of decimal places in cutoff value
  9  IOP15 = 0 / 1=Omit 1st grade item for scheduling file
 10  IOP19 = 0 / 1=Summarize to larger benches
 11  IOP23 = 0. / 1=No zone input file
 12  IOP30 = 1 / 1=Report all zones in totals section
 13  IOP34 = 1    / 1=Report cutoffs in bench summary
 14  IOP35 = 1 / 1=Report summary sections only; 2=Summary + bench summary
 15  IOP36 = 0 / 1=Report cumulative >= each cutoff
 16  PAR1 = 0. /Top bench for summarizing if IOP19=1
 17  PAR2 = 0. /Bot bench for summarizing if IOP19=1
 18  PAR3 = 0. /Bench height for summarizing if IOP19=1
 19  PAR4 = 0 / Optional waste type to output to scd file (-1 means PAR25-26 are range of codes)
 20  PAR25 = 0 0 0 0 / Optional waste types to output to scd file
 21  PAR5 = 0 / Maximum number of lines per page
 22  PAR9 = 0.001 / Factor for reporting
 23  END
```

```
RUN# 4714.    Page    2    METL MTINC    Date 05-28-2012   Time 14:57:11
---------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P624


     Data file ZONE1.ZON  *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     ZONE DATA       file *    29


     Data file P624.SUM   *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     LARGER SUMMARY  file *    27


     Data file P623.SUM   *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     SMALLER SUMMARY file *    28


     Data file P624I.SUM  *  unit  size from program
     -------------------- *   ---  ---- ----  ----
     RESERVE SUMMARY file *    31
```

RUN# 4714. Page 3 METL MTINC Date 05-28-2012 Time 14:57:11

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P624

GRADE ITEMS FOR DEFAULTS: NSRM CUOK MOOK AGOK

NAME	ZONE#	DENS	RECV	DILN	DIL DENS	DILN GRADES, CUTOFF GRADES							
MOX	1	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
MMX	2	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
MSU	3	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
IOX	4	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
IMX	5	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
ISU	6	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
FOX	7	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	3.030	0.000	0.000	0.000
FMX	8	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
FSU	9	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
UNDEF	10	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000

RUN# 4714. Page 4 METL MTINC Date 05-28-2012 Time 14:57:11

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P624

BENCH TOE	ZONE NAME	ZONE NO.	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	DILUTED NSRM	GRADES CUOK	MOOK	AGOK
TOTALS	MOX	1	899.	2011.	2011.	4.590	0.146	0.0024	0.0253
	MMX	2	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	MSU	3	9882.	22947.	22947.	22.116	0.501	0.0134	0.1453
	IOX	4	1155.	2581.	2581.	4.750	0.151	0.0020	0.0266
	IMX	5	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	ISU	6	8512.	19752.	19752.	20.881	0.478	0.0126	0.1258
	FOX	7	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FMX	8	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FSU	9	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	UNDEF	10	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
TOTALS	SUMMARY		20447.	47290.	47290.	19.907	0.457	0.0120	0.1256

WASTE 96117. (kTONS) ROM S/R= 2.03

RUN# 4714. Page 5 METL MTINC Date 05-28-2012 Time 14:57:11

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P624

BENCH TOE	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	WASTE TOTAL (kTONS)	ROM S/R	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
5450.0	0.	1.	1.	3.	3.11	4.900	0.156	0.0000	0.0000
5400.0	13.	29.	29.	34.	1.16	4.207	0.133	0.0000	0.0000
5350.0	171.	382.	382.	138.	0.36	4.720	0.150	0.0001	0.0007
5300.0	166.	371.	371.	525.	1.42	4.460	0.141	0.0016	0.0010
5250.0	209.	468.	468.	825.	1.76	4.479	0.142	0.0009	0.0037
5200.0	186.	415.	415.	1564.	3.77	4.441	0.141	0.0016	0.0145
5150.0	285.	638.	638.	3171.	4.97	4.167	0.132	0.0021	0.0218
5100.0	156.	351.	351.	5653.	16.10	4.347	0.135	0.0039	0.0425
5050.0	319.	721.	721.	8903.	12.35	5.621	0.165	0.0031	0.0561
5000.0	442.	1001.	1001.	8786.	8.78	7.821	0.214	0.0030	0.0628
4950.0	618.	1403.	1403.	7955.	5.67	6.666	0.175	0.0024	0.0462
4900.0	498.	1141.	1141.	7566.	6.63	6.168	0.149	0.0031	0.0355
4850.0	273.	620.	620.	7797.	12.57	6.494	0.147	0.0039	0.0465
4800.0	260.	599.	599.	7251.	12.11	6.280	0.136	0.0045	0.0481
4750.0	96.	220.	220.	7496.	34.14	7.598	0.116	0.0181	0.0345
4700.0	108.	248.	248.	7453.	29.99	9.542	0.196	0.0110	0.0538
4650.0	204.	471.	471.	6849.	14.53	11.589	0.237	0.0141	0.0569
4600.0	326.	751.	751.	5778.	7.69	13.029	0.294	0.0114	0.0464
4550.0	589.	1361.	1361.	4500.	3.31	11.807	0.280	0.0071	0.0453
4500.0	1340.	3079.	3079.	2460.	0.80	15.518	0.369	0.0079	0.0739
4450.0	1672.	3830.	3830.	1194.	0.31	20.994	0.493	0.0124	0.0959
4400.0	1872.	4293.	4293.	184.	0.04	22.569	0.523	0.0147	0.1053
4350.0	1967.	4534.	4534.	11.	0.00	24.577	0.575	0.0146	0.1190
4300.0	1750.	4049.	4049.	21.	0.01	24.471	0.560	0.0159	0.1340
4250.0	1597.	3704.	3704.	0.	0.00	22.732	0.521	0.0130	0.1431
4200.0	1384.	3235.	3235.	0.	0.00	23.232	0.526	0.0121	0.1791
4150.0	1220.	2866.	2866.	0.	0.00	24.092	0.521	0.0181	0.1829
4100.0	958.	2269.	2269.	0.	0.00	24.385	0.534	0.0152	0.2039
4050.0	783.	1869.	1869.	0.	0.00	27.754	0.607	0.0163	0.2449
4000.0	534.	1286.	1286.	0.	0.00	30.145	0.661	0.0184	0.2537
3950.0	450.	1086.	1086.	0.	0.00	44.504	0.986	0.0194	0.4446
TOTAL	20447.	47290.	47290.	96117.	2.03	19.907	0.457	0.0120	0.1256

```
RUN# 4714.    Page    6    METL MTINC   Date 05-28-2012  Time 14:57:11
---------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P624


         *** MTINC    finished on 05-28-2012  14:57:11


*** Current program execution:  Elapsed time (sec)   Date        Time
             MTINC                          0       05-28-2012  14:57:11
```

```
*** ------------------------------------*----------------------------*
***                                     *  Project RUN#  4715.        *
***    MTINC    - Revised on 23-AUG-10  *  Date started  05-28-2012   *
***                                     *  Time started  14:57:14     *
***                                     *  Project Acct  N/A          *
*** ------------------------------------*----------------------------*


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               MineSight Basis Release 2011.10.31

-----------------------------------------------------------------------
Run File = RUNINC.TMP   Print File = P625I.RPT

Line          R U N   F I L E   R E C O R D S
----
  1  MEDS-MTINC 10=rose10.dat 15=rose15.dat 3=P625i.rpt 27=P625.sum;
  2  MEDS-MTINC 31=P625i.sum 29=zone1.zon 28=P624.sum  24=P625i.scd
  3  INCREMENTAL CONTENTS OF P625
  4  USR =
  5  I-O = 0
  6  IOP2 = 0 / 0=Subtract smaller from larger; 1=Add them; 2=Larger only
  7  IOP17 = 0 / 1=Output MineSight Scheduling File using cutoffs, 2=using zone item
  8  IOP14 = 2 / Number of decimal places in cutoff value
  9  IOP15 = 0 / 1=Omit 1st grade item for scheduling file
 10  IOP19 = 0 / 1=Summarize to larger benches
 11  IOP23 = 0. / 1=No zone input file
 12  IOP30 = 1 / 1=Report all zones in totals section
 13  IOP34 = 1    / 1=Report cutoffs in bench summary
 14  IOP35 = 1 / 1=Report summary sections only; 2=Summary + bench summary
 15  IOP36 = 0 / 1=Report cumulative >= each cutoff
 16  PAR1 = 0. /Top bench for summarizing if IOP19=1
 17  PAR2 = 0. /Bot bench for summarizing if IOP19=1
 18  PAR3 = 0. /Bench height for summarizing if IOP19=1
 19  PAR4 = 0 / Optional waste type to output to scd file (-1 means PAR25-26 are range of codes)
 20  PAR25 = 0 0 0 0 / Optional waste types to output to scd file
 21  PAR5 = 0 / Maximum number of lines per page
 22  PAR9 = 0.001 / Factor for reporting
 23  END
```

```
RUN# 4715.     Page     2     METL MTINC    Date 05-28-2012   Time 14:57:14
---------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P625


     Data file ZONE1.ZON  *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     ZONE DATA       file *    29


     Data file P625.SUM   *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     LARGER SUMMARY  file *    27


     Data file P624.SUM   *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     SMALLER SUMMARY file *    28


     Data file P625I.SUM  *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     RESERVE SUMMARY file *    31
```

RUN# 4715. Page 3 METL MTINC Date 05-28-2012 Time 14:57:14

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P625

GRADE ITEMS FOR DEFAULTS: NSRM CUOK MOOK AGOK

NAME	ZONE#	DENS	RECV	DILN	DIL DENS	DILN GRADES, CUTOFF GRADES							
MOX	1	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
MMX	2	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
MSU	3	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
IOX	4	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
IMX	5	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
ISU	6	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
FOX	7	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	3.030	0.000	0.000	0.000
FMX	8	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
FSU	9	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
UNDEF	10	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000

RUN# 4715. Page 4 METL MTINC Date 05-28-2012 Time 14:57:14

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P625

BENCH TOE	ZONE NAME	ZONE NO.	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
TOTALS	MOX	1	2781.	6244.	6244.	5.038	0.160	0.0008	0.0056
	MMX	2	1.	3.	3.	16.933	0.940	0.0086	0.0973
	MSU	3	19972.	46468.	46468.	22.181	0.511	0.0131	0.1270
	IOX	4	4024.	9022.	9022.	4.849	0.154	0.0014	0.0150
	IMX	5	60.	146.	146.	13.217	0.729	0.0078	0.0761
	ISU	6	14278.	33228.	33228.	20.931	0.482	0.0123	0.1225
	FOX	7	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FMX	8	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FSU	9	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	UNDEF	10	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
TOTALS	SUMMARY		41117.	95111.	95111.	18.961	0.445	0.0109	0.1067

WASTE 141337. (kTONS) ROM S/R= 1.49

4050.0	2252.	5253.	5253.	20.	0.00	24.228	0.554	0.0145	0.1477
4000.0	1971.	4613.	4613.	3.	0.00	24.972	0.553	0.0191	0.1499
3950.0	1759.	4138.	4138.	0.	0.00	24.803	0.552	0.0167	0.1676
3900.0	1315.	3104.	3104.	0.	0.00	31.398	0.716	0.0170	0.2192
3850.0	1122.	2651.	2651.	0.	0.00	33.372	0.770	0.0192	0.1958
3800.0	813.	1921.	1921.	0.	0.00	31.582	0.689	0.0211	0.2542
3750.0	701.	1658.	1658.	0.	0.00	20.738	0.444	0.0192	0.1253
TOTAL	41116.	95111.	95111.	141337.	1.49	18.961	0.445	0.0109	0.1067

```
RUN# 4715.    Page    6    METL MTINC   Date 05-28-2012  Time 14:57:14
----------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P625


         *** MTINC    finished on 05-28-2012  14:57:14


*** Current program execution:  Elapsed time (sec)   Date        Time
             MTINC                         0        05-28-2012  14:57:14
```

```
*** ------------------------------------*----------------------------*
***                                     *  Project RUN#  4716.       *
***   MTINC   - Revised on 23-AUG-10    *  Date started  05-28-2012  *
***                                     *  Time started  14:57:16    *
***                                     *  Project Acct  N/A         *
*** ------------------------------------*----------------------------*


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----------------------------------------------------------------------
Run File = RUNINC.TMP   Print File = P626I.RPT

Line          R U N   F I L E   R E C O R D S
----
  1  MEDS-MTINC 10=rose10.dat 15=rose15.dat 3=P626i.rpt 27=P626.sum;
  2  MEDS-MTINC 31=P626i.sum 29=zone1.zon 28=P625.sum  24=P626i.scd
  3  INCREMENTAL CONTENTS OF P626
  4  USR =
  5  I-O = 0
  6  IOP2 = 0 / 0=Subtract smaller from larger; 1=Add them; 2=Larger only
  7  IOP17 = 0 / 1=Output MineSight Scheduling File using cutoffs, 2=using zone item
  8  IOP14 = 2 / Number of decimal places in cutoff value
  9  IOP15 = 0 / 1=Omit 1st grade item for scheduling file
 10  IOP19 = 0 / 1=Summarize to larger benches
 11  IOP23 = 0. / 1=No zone input file
 12  IOP30 = 1 / 1=Report all zones in totals section
 13  IOP34 = 1    / 1=Report cutoffs in bench summary
 14  IOP35 = 1 / 1=Report summary sections only; 2=Summary + bench summary
 15  IOP36 = 0 / 1=Report cumulative >= each cutoff
 16  PAR1 = 0. /Top bench for summarizing if IOP19=1
 17  PAR2 = 0. /Bot bench for summarizing if IOP19=1
 18  PAR3 = 0. /Bench height for summarizing if IOP19=1
 19  PAR4 = 0 / Optional waste type to output to scd file (-1 means PAR25-26 are range of codes)
 20  PAR25 = 0 0 0 0 / Optional waste types to output to scd file
 21  PAR5 = 0 / Maximum number of lines per page
 22  PAR9 = 0.001 / Factor for reporting
 23  END
```

```
RUN# 4716.    Page    2    METL MTINC   Date 05-28-2012  Time 14:57:16
---------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P626


     Data file ZONE1.ZON  *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     ZONE DATA       file *    29


     Data file P626.SUM   *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     LARGER SUMMARY  file *    27


     Data file P625.SUM   *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     SMALLER SUMMARY file *    28


     Data file P626I.SUM  *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     RESERVE SUMMARY file *    31
```

RUN# 4716. Page 3 METL MTINC Date 05-28-2012 Time 14:57:16

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P626

GRADE ITEMS FOR DEFAULTS: NSRM CUOK MOOK AGOK

NAME	ZONE#	DENS	RECV	DILN	DIL DENS	DILN GRADES, CUTOFF GRADES							
MOX	1	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
MMX	2	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
MSU	3	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
IOX	4	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
IMX	5	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
ISU	6	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
FOX	7	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	3.030	0.000	0.000	0.000
FMX	8	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
FSU	9	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
UNDEF	10	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000

RUN# 4716. Page 4 METL MTINC Date 05-28-2012 Time 14:57:16

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P626

BENCH TOE	ZONE NAME	ZONE NO.	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
TOTALS	MOX	1	1439.	3209.	3209.	5.358	0.170	0.0021	0.0036
	MMX	2	2814.	6786.	6786.	11.719	0.644	0.0073	0.0696
	MSU	3	42654.	100819.	100819.	19.828	0.436	0.0149	0.1297
	IOX	4	2902.	6467.	6467.	5.384	0.171	0.0026	0.0096
	IMX	5	5255.	12667.	12667.	10.554	0.573	0.0075	0.0673
	ISU	6	51457.	121205.	121205.	17.590	0.385	0.0139	0.1119
	FOX	7	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FMX	8	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FSU	9	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	UNDEF	10	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
TOTALS	SUMMARY		106520.	251153.	251153.	17.504	0.414	0.0134	0.1116

WASTE 402297. (kTONS) ROM S/R= 1.60

RUN# 4716. Page 5 METL MTINC Date 05-28-2012 Time 14:57:16

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P626

BENCH TOE	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	WASTE TOTAL (kTONS)	ROM S/R	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
6050.0	0.	0.	0.	2.	-1.00	-1.000	-1.000	-1.0000	-1.0000
6000.0	0.	0.	0.	8.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5950.0	0.	0.	0.	83.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5900.0	0.	0.	0.	148.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5850.0	0.	0.	0.	574.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5800.0	0.	0.	0.	864.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5750.0	0.	0.	0.	1879.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5700.0	0.	0.	0.	2503.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5650.0	0.	0.	0.	4041.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5600.0	0.	0.	0.	5150.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5550.0	0.	0.	0.	7248.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5500.0	0.	0.	0.	8884.	-1.00	3.887	0.123	0.0007	0.0000
5450.0	3.	6.	6.	11056.	-1.00	3.924	0.124	0.0000	0.0000
5400.0	70.	161.	161.	11816.	73.26	8.421	0.201	0.0020	0.0789
5350.0	221.	514.	514.	12760.	24.84	16.820	0.421	0.0047	0.0812
5300.0	456.	1055.	1055.	13189.	12.51	18.928	0.476	0.0054	0.0874
5250.0	709.	1642.	1642.	14005.	8.53	20.019	0.492	0.0062	0.1096
5200.0	1012.	2336.	2336.	14296.	6.12	15.601	0.379	0.0068	0.0961
5150.0	1150.	2662.	2662.	16463.	6.19	16.720	0.399	0.0072	0.1119
5100.0	1530.	3525.	3525.	18857.	5.35	14.235	0.348	0.0066	0.0913
5050.0	1446.	3337.	3337.	21610.	6.48	12.204	0.306	0.0053	0.0617
5000.0	1549.	3570.	3570.	24428.	6.84	10.824	0.271	0.0047	0.0577
4950.0	1516.	3501.	3501.	23769.	6.79	9.883	0.241	0.0048	0.0594
4900.0	1553.	3577.	3577.	22585.	6.31	10.277	0.255	0.0047	0.0631
4850.0	1616.	3726.	3726.	21559.	5.79	11.258	0.276	0.0042	0.0739
4800.0	1773.	4101.	4101.	19158.	4.67	13.510	0.331	0.0069	0.0892
4750.0	1852.	4300.	4300.	18781.	4.37	15.769	0.393	0.0105	0.0930
4700.0	1972.	4602.	4602.	17288.	3.76	17.090	0.452	0.0111	0.0931
4650.0	2122.	4982.	4982.	16943.	3.40	17.516	0.464	0.0096	0.1060
4600.0	2392.	5661.	5661.	13742.	2.43	20.499	0.549	0.0121	0.1120
4550.0	2474.	5866.	5866.	12314.	2.10	21.877	0.597	0.0106	0.1306
4500.0	2746.	6518.	6518.	10307.	1.58	20.109	0.534	0.0122	0.1116
4450.0	2914.	6890.	6890.	9638.	1.40	16.493	0.452	0.0102	0.0846
4400.0	3690.	8685.	8685.	6898.	0.79	15.876	0.418	0.0101	0.0792

4350.0	4137.	9756.	9756.	5675.	0.58	16.056	0.418	0.0106	0.0848
4300.0	4676.	10980.	10980.	3645.	0.33	15.773	0.402	0.0109	0.0791
4250.0	4900.	11494.	11494.	3139.	0.27	15.695	0.393	0.0123	0.0780
4200.0	4980.	11671.	11671.	1482.	0.13	17.169	0.429	0.0122	0.0902
4150.0	4882.	11444.	11444.	1454.	0.13	18.423	0.441	0.0160	0.1031
4100.0	4568.	10732.	10732.	1104.	0.10	19.182	0.482	0.0120	0.1160
4050.0	4573.	10747.	10747.	1082.	0.10	18.038	0.431	0.0119	0.1134
4000.0	4549.	10708.	10708.	579.	0.05	18.067	0.427	0.0127	0.1243
3950.0	4483.	10561.	10561.	497.	0.05	18.608	0.424	0.0135	0.1432
3900.0	4085.	9706.	9706.	195.	0.02	17.451	0.402	0.0114	0.1216
3850.0	3893.	9258.	9258.	216.	0.02	18.967	0.419	0.0137	0.1453
3800.0	3588.	8549.	8549.	159.	0.02	20.383	0.426	0.0177	0.1653
3750.0	3364.	8030.	8030.	134.	0.02	18.800	0.384	0.0194	0.1368

RUN# 4716. Page 6 METL MTINC Date 05-28-2012 Time 14:57:16

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P626

BENCH TOE	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	WASTE TOTAL (kTONS)	ROM S/R	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
3700.0	3313.	7938.	7938.	73.	0.01	17.842	0.337	0.0242	0.1316
3650.0	3022.	7258.	7258.	7.	0.00	17.986	0.340	0.0246	0.1309
3600.0	2301.	5547.	5547.	0.	0.00	19.947	0.384	0.0258	0.1432
3550.0	1976.	4770.	4770.	0.	0.00	23.113	0.463	0.0252	0.1757
3500.0	1571.	3794.	3794.	0.	0.00	23.271	0.449	0.0274	0.1968
3450.0	1245.	3006.	3006.	0.	0.00	22.083	0.436	0.0243	0.1817
3400.0	885.	2137.	2137.	0.	0.00	18.142	0.364	0.0191	0.1452
3350.0	764.	1845.	1845.	9.	0.00	13.862	0.261	0.0173	0.1226
TOTAL	106520.	251153.	251153.	402297.	1.60	17.504	0.414	0.0134	0.1116

```
RUN# 4716.    Page    7    METL MTINC    Date 05-28-2012  Time 14:57:16
----------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P626


         *** MTINC    finished on 05-28-2012  14:57:16


*** Current program execution:  Elapsed time (sec)   Date        Time
             MTINC                         0        05-28-2012  14:57:16
```

```
*** -----------------------------------*----------------------------*
***                                    *   Project RUN#  4717.      *
***    MTINC    - Revised on 23-AUG-10 *   Date started  05-28-2012 *
***                                    *   Time started  14:57:18   *
***                                    *   Project Acct  N/A        *
*** -----------------------------------*----------------------------*


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               MineSight Basis Release 2011.10.31

----------------------------------------------------------------------
Run File = RUNINC.TMP   Print File = P627I.RPT

Line          R U N   F I L E   R E C O R D S
----
  1  MEDS-MTINC 10=rose10.dat 15=rose15.dat 3=P627i.rpt 27=P627.sum;
  2  MEDS-MTINC 31=P627i.sum 29=zone1.zon 28=P626.sum  24=P627i.scd
  3  INCREMENTAL CONTENTS OF P627
  4  USR =
  5  I-O = 0
  6  IOP2 = 0 / 0=Subtract smaller from larger; 1=Add them; 2=Larger only
  7  IOP17 = 0 / 1=Output MineSight Scheduling File using cutoffs, 2=using zone item
  8  IOP14 = 2 / Number of decimal places in cutoff value
  9  IOP15 = 0 / 1=Omit 1st grade item for scheduling file
 10  IOP19 = 0 / 1=Summarize to larger benches
 11  IOP23 = 0. / 1=No zone input file
 12  IOP30 = 1 / 1=Report all zones in totals section
 13  IOP34 = 1    / 1=Report cutoffs in bench summary
 14  IOP35 = 1 / 1=Report summary sections only; 2=Summary + bench summary
 15  IOP36 = 0 / 1=Report cumulative >= each cutoff
 16  PAR1 = 0. /Top bench for summarizing if IOP19=1
 17  PAR2 = 0. /Bot bench for summarizing if IOP19=1
 18  PAR3 = 0. /Bench height for summarizing if IOP19=1
 19  PAR4 = 0 / Optional waste type to output to scd file (-1 means PAR25-26 are range of codes)
 20  PAR25 = 0 0 0 0 / Optional waste types to output to scd file
 21  PAR5 = 0 / Maximum number of lines per page
 22  PAR9 = 0.001 / Factor for reporting
 23  END
```

```
RUN# 4717.    Page    2    METL MTINC    Date 05-28-2012  Time 14:57:18
---------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P627


     Data file ZONE1.ZON  *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     ZONE DATA       file *    29



     Data file P627.SUM   *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     LARGER SUMMARY  file *    27



     Data file P626.SUM   *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     SMALLER SUMMARY file *    28



     Data file P627I.SUM  *  unit  size from program
     -------------------- *  ---   ---- ----  ----
     RESERVE SUMMARY file *    31
```

```
RUN# 4717.    Page    3    METL MTINC    Date 05-28-2012  Time 14:57:18
---------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P627

GRADE ITEMS FOR DEFAULTS: NSRM  CUOK  MOOK  AGOK
```

NAME	ZONE#	DENS	RECV	DILN	DIL DENS	DILN GRADES, CUTOFF GRADES							
MOX	1	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
MMX	2	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
MSU	3	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
IOX	4	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	3.030	0.000	0.000	0.000
IMX	5	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
ISU	6	11.850	100.0	0.0	11.850	1.114	0.199	0.007	0.050	4.900	0.000	0.000	0.000
FOX	7	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	3.030	0.000	0.000	0.000
FMX	8	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
FSU	9	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000
UNDEF	10	11.850	0.0	0.0	11.850	0.000	0.000	0.000	0.000	4.900	0.000	0.000	0.000

RUN# 4717. Page 4 METL MTINC Date 05-28-2012 Time 14:57:18

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P627

BENCH TOE	ZONE NAME	ZONE NO.	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
TOTALS	MOX	1	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	MMX	2	1.	2.	2.	7.852	0.400	0.0119	0.0396
	MSU	3	24030.	55306.	55306.	19.539	0.429	0.0165	0.1090
	IOX	4	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	IMX	5	17.	41.	41.	7.311	0.413	0.0033	0.0284
	ISU	6	50661.	116703.	116703.	18.987	0.417	0.0150	0.1169
	FOX	7	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FMX	8	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	FSU	9	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
	UNDEF	10	0.	0.	0.	-1.000	-1.000	-1.0000	-1.0000
TOTALS	SUMMARY		74708.	172052.	172052.	19.162	0.421	0.0155	0.1144

WASTE 287362. (kTONS) ROM S/R= 1.67

RUN# 4717. Page 5 METL MTINC Date 05-28-2012 Time 14:57:18

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P627

BENCH TOE	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	WASTE TOTAL (kTONS)	ROM S/R	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
5450.0	0.	0.	0.	168.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5400.0	0.	0.	0.	432.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5350.0	0.	0.	0.	847.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5300.0	0.	0.	0.	1216.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5250.0	0.	0.	0.	1864.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5200.0	0.	0.	0.	3135.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5150.0	0.	0.	0.	4714.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5100.0	0.	0.	0.	6353.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5050.0	0.	0.	0.	9188.	-1.00	-1.000	-1.000	-1.0000	-1.0000
5000.0	0.	0.	0.	12024.	-1.00	-1.000	-1.000	-1.0000	-1.0000
4950.0	0.	0.	0.	13476.	-1.00	-1.000	-1.000	-1.0000	-1.0000
4900.0	0.	0.	0.	13478.	-1.00	-1.000	-1.000	-1.0000	-1.0000
4850.0	0.	0.	0.	13198.	-1.00	-1.000	-1.000	-1.0000	-1.0000
4800.0	0.	0.	0.	13611.	-1.00	-1.000	-1.000	-1.0000	-1.0000
4750.0	0.	0.	0.	13460.	-1.00	-1.000	-1.000	-1.0000	-1.0000
4700.0	6.	14.	14.	12983.	920.76	6.454	0.181	0.0000	0.0002
4650.0	2.	5.	5.	13184.	-1.00	7.601	0.215	-0.0001	0.0000
4600.0	0.	0.	0.	14943.	-1.00	-1.000	-1.000	-1.0000	-1.0000
4550.0	14.	31.	31.	15542.	507.92	5.235	0.129	0.0013	0.0291
4500.0	0.	0.	0.	15459.	-1.00	-1.000	-1.000	-1.0000	-1.0000
4450.0	5.	10.	10.	15137.	-1.00	4.991	0.111	0.0035	0.0365
4400.0	23.	52.	52.	14516.	277.55	6.204	0.141	0.0042	0.0350
4350.0	116.	260.	260.	14007.	53.89	6.811	0.161	0.0048	0.0187
4300.0	694.	1547.	1547.	12511.	8.09	11.845	0.280	0.0080	0.0419
4250.0	1355.	3019.	3019.	10336.	3.42	14.593	0.334	0.0121	0.0492
4200.0	2172.	4846.	4846.	7608.	1.57	15.586	0.365	0.0105	0.0608
4150.0	2437.	5448.	5448.	6422.	1.18	18.980	0.438	0.0145	0.0666
4100.0	2979.	6703.	6703.	4937.	0.74	20.929	0.473	0.0169	0.0916
4050.0	3213.	7242.	7242.	3964.	0.55	18.454	0.419	0.0137	0.0872
4000.0	3506.	7911.	7911.	2640.	0.33	16.837	0.375	0.0144	0.0778
3950.0	3683.	8329.	8329.	1883.	0.23	18.650	0.404	0.0186	0.0831
3900.0	4158.	9433.	9433.	1552.	0.16	22.663	0.521	0.0154	0.1097
3850.0	4224.	9618.	9618.	1205.	0.13	22.580	0.501	0.0180	0.1242
3800.0	4126.	9424.	9424.	778.	0.08	19.589	0.433	0.0165	0.1020

3750.0	4028.	9225.	9225.	817.	0.09	22.292	0.491	0.0194	0.1121
3700.0	3964.	9117.	9117.	975.	0.11	22.102	0.492	0.0186	0.1071
3650.0	3818.	8812.	8812.	1101.	0.12	20.141	0.439	0.0186	0.1008
3600.0	3557.	8228.	8228.	765.	0.09	17.959	0.382	0.0183	0.0937
3550.0	3513.	8161.	8161.	751.	0.09	16.433	0.345	0.0175	0.0888
3500.0	3202.	7475.	7475.	889.	0.12	18.274	0.386	0.0170	0.1226
3450.0	3279.	7691.	7691.	647.	0.08	17.425	0.373	0.0153	0.1141
3400.0	3024.	7112.	7112.	183.	0.03	16.533	0.350	0.0145	0.1189
3350.0	2825.	6656.	6656.	176.	0.03	14.610	0.313	0.0120	0.1057
3300.0	2509.	5960.	5960.	674.	0.11	15.272	0.320	0.0126	0.1284
3250.0	1865.	4429.	4429.	1149.	0.26	14.068	0.301	0.0105	0.1140
3200.0	1839.	4358.	4358.	140.	0.03	16.126	0.340	0.0125	0.1392
3150.0	1464.	3472.	3472.	394.	0.11	25.346	0.561	0.0111	0.2525

RUN# 4717. Page 6 METL MTINC Date 05-28-2012 Time 14:57:18

ROSEMONT DEPOSIT - JANUARY 2007 MODEL

INCREMENTAL CONTENTS OF P627

BENCH TOE	INSITU ORE (kYDS)	INSITU ORE (kTONS)	RUN OF MINE (kTONS)	WASTE TOTAL (kTONS)	ROM S/R	DILUTED GRADES NSRM	CUOK	MOOK	AGOK
3100.0	863.	2074.	2074.	858.	0.41	25.539	0.526	0.0190	0.2632
3050.0	896.	2143.	2143.	297.	0.14	25.268	0.545	0.0108	0.2922
3000.0	660.	1585.	1585.	210.	0.13	28.756	0.634	0.0082	0.3458
2950.0	457.	1100.	1100.	378.	0.34	36.099	0.785	0.0054	0.5210
2900.0	232.	560.	560.	190.	0.34	34.042	0.753	0.0035	0.4785
TOTAL	74708.	172052.	172052.	287362.	1.67	19.162	0.421	0.0155	0.1144

```
RUN# 4717.    Page    7    METL MTINC   Date 05-28-2012  Time 14:57:18
----------------------------------------------------------------------------
  ROSEMONT DEPOSIT - JANUARY 2007 MODEL

    INCREMENTAL CONTENTS OF P627


         *** MTINC    finished on 05-28-2012  14:57:19


*** Current program execution:  Elapsed time (sec)   Date        Time
             MTINC                         1        05-28-2012  14:57:19
```

APPENDIX F: BENCHES MINED BY PERIOD

Rosemont Feasibility Study, Ore Benches Mined in Mining Schedule scd4b

(Detail from spreadsheet: Rosemont - Mining Schedule - 4b - 120621.xlsx)

Ore Bench Mined from		Q-3	Q-2	Q-1	Y1Q1	Y1Q2	Y1Q3	Y1Q4	Y2Q1	Y2Q2	Y2Q3	Y2Q4	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22
P621	ft	5,300	5,200	5,150	5,100	5,000	4,900	4,800	4,750	4,700	4,600	4,550	4,400	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P622	ft	0	0	0	0	0	0	5,150	5,100	5,000	4,900	0	4,750	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P623	ft	0	0	0	0	0	5,200	5,200	0	0	0	5,100	4,950	4,400	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P624	ft	0	0	0	0	0	0	0	0	0	0	0	5,100	5,050	4,450	4,050	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P625	ft	0	0	0	0	0	0	5,350	0	5,350	0	0	5,300	5,050	5,000	4,650	4,250	3,900	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P626	ft	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5,400	5,250	5,000	4,800	4,550	4,350	4,200	4,050	3,900	3,700	0	0	0	0	0	0
P627	ft	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	4,700	4,550	4,400	4,300	4,000	3,800	3,650	3,450	3,150
Ore Bench Mined to		Q-3	Q-2	Q-1	Y1Q1	Y1Q2	Y1Q3	Y1Q4	Y2Q1	Y2Q2	Y2Q3	Y2Q4	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22
P621	ft	5,250	5,150	5,100	5,000	4,900	4,800	4,750	4,700	4,600	4,550	4,400	4,350	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P622	ft	0	0	0	0	0	0	5,150	5,050	4,950	4,800	0	4,300	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P623	ft	0	0	0	0	0	5,200	5,150	0	0	0	4,950	4,400	4,150	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P624	ft	0	0	0	0	0	0	0	0	0	0	0	5,050	4,450	4,050	3,950	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P625	ft	0	0	0	0	0	0	5,350	0	5,300	0	0	5,050	5,000	4,650	4,250	3,900	3,750	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P626	ft	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5,250	5,000	4,800	4,550	4,350	4,200	4,050	3,900	3,700	3,350	0	0	0	0	0	0
P627	ft	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	4,600	4,450	4,300	4,000	3,750	3,600	3,450	3,100	2,900

Rosemont - Mining Schedule - 4b, ore bences mined, 120816.xlsx-Ore 1 of 16/08/2012 16/08/2012

Rosemont Feasibility Study, Ore and Waste Benches Mined in Mining Schedule scd4b
(Detail from spreadsheet: Rosemont - Mining Schedule - 4b - 120621.xlsx)

Bench Mined from	Year	Q-6	Q-5	Q-4	Q-3	Q-2	Q-1	Y1Q1	Y1Q2	Y1Q3	Y1Q4	Y2Q1	Y2Q2	Y2Q3	Y2Q4	Y3	Y4
P621	ft	5,650	5,500	5,400	5,300	5,200	5,150	5,100	5,000	4,900	4,800	4,750	4,700	4,600	4,550	4,400	0
P622	ft	5,800	5,500	5,400	0	0	0	0	0	5,350	5,200	5,100	5,000	4,900	0	4,750	0
P623	ft	0	0	5,400	0	0	0	0	0	5,350	5,200	0	0	0	5,100	4,950	4,400
P624	ft	0	5,450	5,400	0	0	0	0	0	0	5,350	0	5,350	0	0	5,300	5,050
P625	ft	5,750	5,500	5,400	0	0	0	0	0	0	5,350	0	5,350	0	0	5,300	5,050
P626	ft	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P627	ft	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Bench Mined to	Year	Q-6	Q-5	Q-4	Q-3	Q-2	Q-1	Y1Q1	Y1Q2	Y1Q3	Y1Q4	Y2Q1	Y2Q2	Y2Q3	Y2Q4	Y3	Y4
P621	ft	5,550	5,450	5,350	5,250	5,150	5,100	5,000	4,900	4,800	4,750	4,700	4,600	4,550	4,400	4,350	0
P622	ft	5,550	5,450	5,400	0	0	0	0	0	5,200	5,150	5,050	4,950	4,800	0	4,300	0
P623	ft	0	0	5,400	0	0	0	0	0	5,200	5,150	0	0	0	4,950	4,400	4,150
P624	ft	0	5,450	5,400	0	0	0	0	0	0	5,350	0	5,300	0	0	5,050	4,450
P625	ft	5,550	5,450	5,400	0	0	0	0	0	0	5,350	0	5,300	0	0	5,050	5,000
P626	ft	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P627	ft	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0

Bench Mined from	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22
P621	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P622	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P623	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P624	4,450	4,050	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P625	5,000	4,650	4,250	3,900	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P626	6,050	5,800	5,500	5,250	5,000	4,800	4,550	4,350	4,200	4,050	3,900	3,700	0	0	0	0	0	0
P627	0	0	0	0	0	0	5,450	5,100	4,900	4,700	4,550	4,400	4,300	4,000	3,800	3,650	3,450	3,150
Bench Mined to	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22
P621	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P622	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P623	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P624	4,050	3,950	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P625	4,650	4,250	3,900	3,750	0	0	0	0	0	0	0	0	0	0	0	0	0	0
P626	5,850	5,500	5,250	5,000	4,800	4,550	4,350	4,200	4,050	3,900	3,700	3,350	0	0	0	0	0	0
P627	0	0	0	0	0	0	5,100	4,900	4,700	4,550	4,400	4,300	4,000	3,750	3,600	3,450	3,100	2,900

Rosemont - Mining Schedule - 4b, ore bences mined, 120816.xlsx-Total Material   1 of 1   16/08/2012