NI 43-101 PRE-FEASIBILITY STUDY ON THE CONTACT COPPER PROJECT

PREPARED FOR:

International Enexco, Ltd.

EFFECTIVE DATE:
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REPORT DATE:
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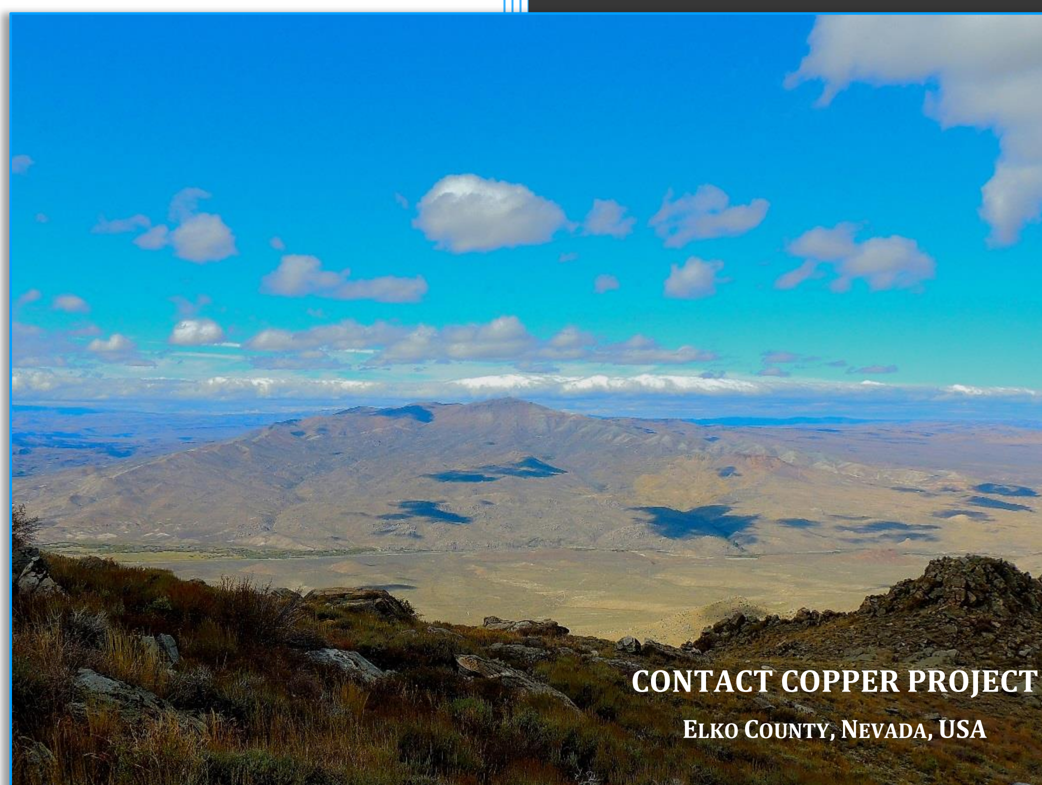

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Hard Rock Consulting, LLC

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1. SUMMARY

This updated Pre-feasibility Study on the Contact Copper Project Nevada, USA (the Report) was prepared by Hard Rock Consulting, LLC (HRC) for International Enexco, Ltd. (Enexco). The Report supports an updated Mineral Resource and Mineral Reserve estimate of the Contact Copper Project (the Project) and provides an up-to-date economic evaluation of the Project. The Report has been prepared in accordance with National Instrument 43-101 (NI 43-101) and Form 43-101F1 (43-101F1).

Enexco is a mineral resource company based in Vancouver, BC and publicly traded on the TSX Venture Exchange, OTC Markets and the Frankfurt Stock Exchange. Enexco is a junior minerals exploration company with a focus on North American exploration of copper, uranium and gold and the advancement of the company's 100% owned Contact Copper Project.

This report updates the Mineral Resource estimate from the *NI 43-101 Technical Report on the Contact Copper Project Nevada* (2012 RE), prepared by 3L Resources, Ltd (3L Resources), with an effective date of October 8, 2012. The updated NI 43-101 compliant Mineral Resource estimate contains 75 million tons at 0.21% Cu (314 million pounds of copper) in the Measured category, and 138 million tons at 0.19% Cu (518 million pounds of copper) in the Indicated category, for a total in Measured and Indicated Resources of 213 million tons at 0.20% copper (Cu) (831 million pounds of copper) at a 0.07% Cu cut-off. The estimate also contains 13 million tons of 0.20% Cu (52 million pounds of copper) in the category of Inferred Resource. The updated NI 43-101 compliant Mineral Resource estimate is constrained by a Lerchs-Grossman (LG) pit shell based on a copper price of $4.00 per pound.

The NI 43-101 compliant Mineral Reserve estimate determined within this report contains 58 million tons of 0.23% Cu (263 million pounds of copper) in the Proven category and 83 million tons of 0.21 % Cu in the Probable category (348 million pounds of copper) for a total in Proven and Probable Reserves of 141 million tons of 0.22% Cu (612 million pounds of copper) at a 0.07% Cu cut-off grade. The Mineral Reserves are extractable by conventional surface mining methods at an overall waste-to-ore ratio of 2.3:1. No Inferred Resources were included in the Proven or Probable Reserves, or in the economic analysis.

The Project as modeled will produce 49.2 million pounds of copper annually over a 9.4 year life. At a copper price of $3.20 per pound, the project cash flow generates a 25.9 % internal rate of return (IRR) and a $107 million net present value at an eight percent discount rate (NPV-8%) on an after-tax basis. The estimated capital costs are $188.9 million initially and $331 million over the life of the mine. The cash operating costs are estimated at $5.68 per ton of ore, or $1.73 per pound of copper produced.

1.1 PROJECT OVERVIEW

The Project is located west of the town of Contact, Nevada, one mile west of U.S. Highway 93, between the towns of Wells and Jackpot, Nevada. Enexco's property consists of approximately 2,650 acres in 156 patented claims and 4,320 acres in 288 unpatented claims. Previous exploration work was performed by four different companies from 1967 to 2004 on an intermittent basis. Enexco commissioned several independent resource estimates between 2006 and 2010. Based on these studies and acquisition of additional land in the Project area, Enexco conducted drilling and metallurgical programs from 2011 to 2013. This Report provides an updated economic model and Mineral Resource and Mineral Reserve for the Project.

1.2 EXPLORATION AND GEOLOGY

Copper mineralization occurs as an intrusive-related deposit within the Contact batholith and is observed in quartz veins within structural zones and in the surrounding granodiorite. The copper content is highest in the quartz veins, particularly where chalcocite is present, but grades outward into granodiorite where copper minerals occur in quartz veinlets, fracture coatings and disseminations. Mineralization is in the form of tenorite, chrysocolla and cuprite, and lesser chalcocite and covellite. Oxidation is observed to depths of 2,000 feet in drilling.

1.3 RESOURCE AND RESERVE

Mineral resources in this Report were estimated using a three dimensional block model and inverse distance squared weighting. The Mineral Resource is constrained within a Lerchs-Grossman pit shell based on a copper price of $4.00[1] per pound and operating cost and recovery parameters as described in Section 15. Table 1-1 shows the Mineral Resource at a 0.07% Cu cut-off grade.

Table 1-1 Mineral Resource Estimate Reported at 0.07% Cu Cut-off

Category	Cu %	Tons (000)	Pounds Cu (000)
Measured	0.21	75,473	313,968
Indicated	0.19	137,640	517,526
Total Measured + Indicated	**0.20**	**213,113**	**831,494**
Inferred	0.20	12,982	52,188

***Notes:**

(1) Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.
(2) Measured and Indicated Mineral Resources captured within the pit shell meet the test of reasonable prospect for economic extraction and can be declared a Mineral Resource.
(3) Inferred Mineral Resources are that part of the Mineral Resource for which the quantity and grade or quality are estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity.
(4) All resources are stated above a 0.05% Cu cut-off.
(5) Pit optimization is based on assumed copper price of US$4.00/lb.
(6) Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding

Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by social and economic factors; environmental, permitting, and legal aspects are discussed in Section 4.

The Mineral Resource and Mineral Reserve are reported at a 0.07% Cu cut-off, which reflects the estimations conducted in this Report on economics, marketing and other issues relevant to an open pit mining and heap leaching with solvent extraction/electrowinning (SX-EW) recovery of copper.

[1] The copper price of $4.00 per pound was selected for definition of mineral resources as the copper price under which the deposit has reasonable prospects for economic extraction. The copper price of $3.20 was selected for definition of mineral reserves and as the copper price over the duration of the project life, and represents 98% of the 5-year trailing price for copper as of the date of this Report.

The Mineral Reserve was estimated from the block model. At a 0.07% Cu cut-off grade, the Mineral Reserve classified as Proven and Probable totals 141 million tons at 0.22% Cu, shown in Table 1-2 below. The Mineral Resource and Mineral Reserve are NI 43-101 compliant. The Mineral Resource is inclusive of the Mineral Reserve. No Inferred Resources were used in either the production schedule or economic analysis.

Table 1-2 Mineral Reserve Estimate Reported at 0.07% Cu Cut-off

Category	Cu %	Tons (000)	Pounds Cu (000)
Proven	0.23	57,678	263,249
Probable	0.21	83,416	348,499
Total Proven + Probable	**0.22**	**141,094**	**611,748**

1.4 MINING, PROCESSING AND DEVELOPMENT

Mining will utilize open pit methods. Ore production is designed to increase from 29,000 to 52,000 tons per day, with an average rate of 41,000 tons per day at an overall waste-ore ratio of 2.3:1. Ore will be crushed in two-stage crushing and then heap leached with sulfuric acid solution. Copper will be recovered in a solvent extraction-electrowinning plant (SX/EW) to produce copper cathodes on site. The SX/EW plant is designed to produce 50 million pounds (25,000 tons) of copper annually. Metallurgical test work indicates the copper recovery will be 76% with an acid consumption of 17 pounds per ton of ore leached.

Preparation of a plan of operations leading to an environmental assessment is anticipated for the Project. Baseline studies and permitting are expected to extend into 2016.

1.5 CAPITAL COST

The capital costs were estimated from equipment quotes, factored estimates, and comparisons with other recently constructed projects and are $188.9 million including contingency.

Sustaining costs over the life of the Project are estimated at $126.0 million, and include $88.5 million in additional mining equipment and replacements, and $37.8 million for leach pad expansions and plant sustaining capital.

Table 1-3 Capital Costs

Description	Cost (000)
Direct Costs	
Site Preparation	$2,688
Mining Equipment	50,332
Crushing	11,533
Conveying	6,838
Pad & Ponds	26,146
SX-EW Plant	36,339
Infrastructure	11,050
Reagents & Initial Fills	2,532
Direct Costs Total	$147,459
Indirect Costs	
Construction Indirects	$2,838
Contingency (@ 20%)	19,425
Contingency Mine Equip. (@ 10%)	5,033
EPCM	7,095
Freight, Mobilization	2,365
Owners Costs	4,730
Indirect Costs Total	$41,486
Capital Costs Total	**$188,945**

1.6 OPERATING COST

The operating costs were estimated based on requirements in labor and supplies to support the designed production rates in mining, crushing and conveying, leaching and SX-EW. The operating costs are estimated at an average of $5.68 per ton of ore, or $1.73 per pound of copper over the life of the Project.

Table 1-4 Operating Costs

Operating Cost	Total Cost (000)	$/lb Cu	$/ton Ore
Mining	$424,936	0.92	3.01
Processing	325,359	0.70	2.31
G&A	30,001	0.06	0.21
Property Tax	16,913	0.04	0.12
Cash Operating Costs		*1.72*	*5.65*
Royalties		0.01	0.03
Total	$797,209	$1.73	$5.68

1.7 ECONOMIC ANALYSIS

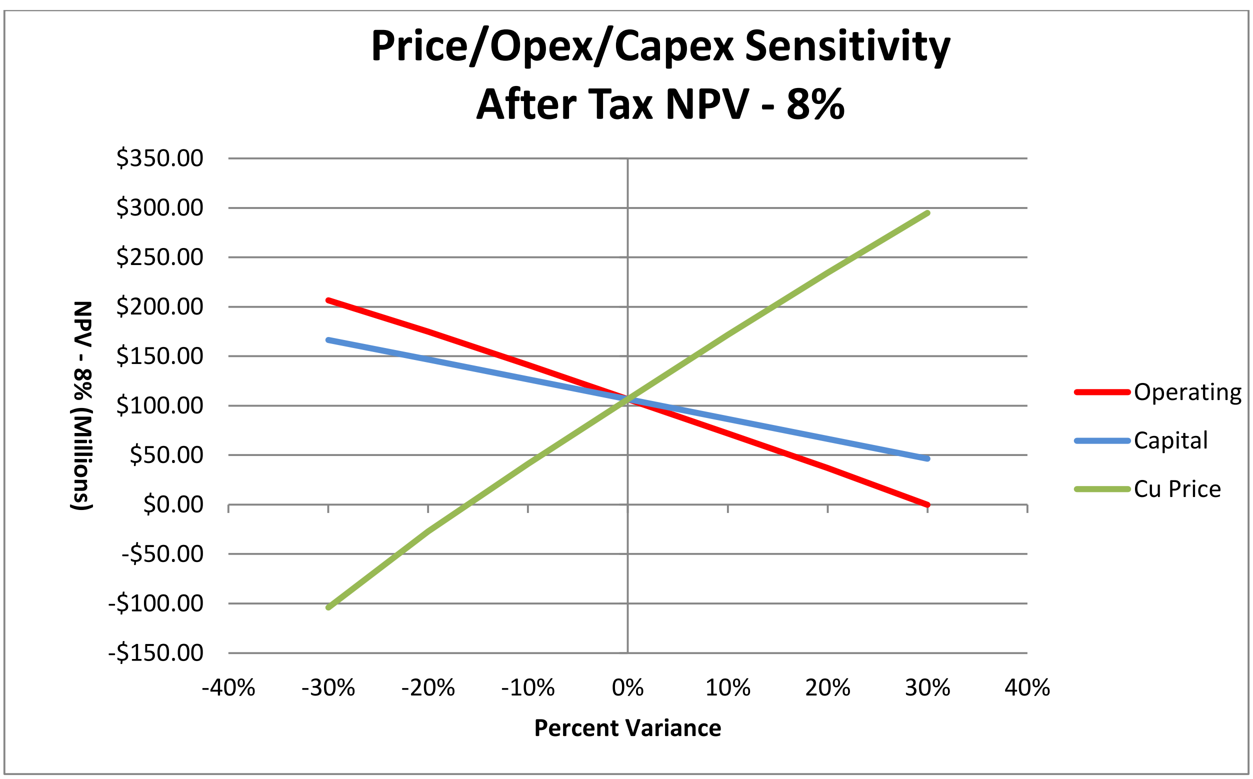
The economic analysis of the Project results in an internal rate of return (IRR) of 30.4% on a before-tax basis, and an IRR of 25.9% on an after-tax basis with a copper price of $3.20 per pound. Net present values (NPVs) at a discount rate of eight percent are $135 million and $107 million before- and after-tax, respectively.

Table 1-5 Project Economics with Proven and Probable Reserves

Project Valuation Overview	Before Tax Analysis	After Tax Analysis
Total Cash flow (millions)	$303.9	$255.6
NPV @ 5.0%; (millions)	$183.8	$149.1
NPV @ 8.0%; (millions)	$135.5	$106.7
NPV @ 10.0%; (millions)	$110.1	$84.5
Internal Rate of Return	30.4%	25.9%
Payback Period	3.0	3.4
Payback Multiple	3.8	3.4
Total Initial Capital (millions)	$188.9	$188.9
Max Neg. Cash flow (millions)	-$108.0	-$108.0

The economic results are most sensitive to changes in copper price. At a copper price of $2.90 per pound, the Project cash flow generates an after-tax IRR of 15.9% and NPV-8% of $45 million. At a copper price of $3.50 per pound, the after-tax IRR is 35.2% and the NPV-8% is $167 million.

Figure 1-1 Sensitivity of Copper Price



1.8 CONCLUSIONS AND RECOMMENDATIONS

HRC concludes the Contact Copper Project is potentially economic based on the development and operating cost estimates and price assumptions within this Report. HRC concludes the Mineral Resource and Mineral Reserve are sufficient to support the Project at the level of a feasibility study and recommends Enexco proceed with a feasibility study. HRC estimates the budget for a feasibility study at $1.25 million (Table 1-6). Activities for permitting are not included in the estimate.

Table 1-6 Estimated Costs for Contact Feasibility Study

		$ (x 1000)
A. Metallurgical Studies	Process optimization, design parameters	250
B. Geotechnical Studies	Pad & pond foundations, confirm water supply	250
C. Project Engineering & Report	Mine, Processing Plant, Infrastructure, Economics	750
	Total	**1,250**

2. INTRODUCTION

2.1 Purpose of Report

HRC was selected by Enexco to prepare the updated Pre-feasibility Study for the Project in Elko County, Nevada. The Mineral Resource and Mineral Reserve estimates described in this Report have been prepared in accordance with NI 43-101 and 43-101F1.

This Report supersedes existing pre-feasibility studies prepared for Enexco by Gustavson Associates in July, 2009 and October, 2010 and a resource estimate study prepared for Enexco by 3L Resources in October 2012. Ms. Terre Lane and Mr. Zachary J. Black, authors of the Report, were employees of Gustavson Associates and 3L Resources at the time and contributed to the previous resource estimates and pre-feasibility studies.

This Report is intended for use in a feasibility study on the Project. Since 2010, significant changes occurred in the Project with respect to land, resources, and economic factors. An updated pre-feasibility study was required to evaluate these changes and determine the scope of the Project prior to a feasibility study. This Report is prepared to support public disclosure of the updated Mineral Resource, Mineral Reserve, and economics of the Project.

This Report makes use of all relevant information provided by Enexco to HRC, and other information gathered by HRC. The purpose of this Report is to summarize and present the applicable information regarding Enexco's Project, and provide an independent estimate of the Mineral Resources and Mineral Reserves and outline the Project's economics.

The intended users of this Report are Enexco and its agents, as well as members of the general public accessing information about Enexco via their company website or the SEDAR information filing system. SEDAR is the Canadian Securities Administrator's (CSA) official site for public access to most public securities documents and information filed with the CSA by public companies and investment funds.

2.2 Units

American versions of Imperial English units of measure (U.S. Customary Units) are used throughout this Report, which are the commonly used units of reporting for base metal projects in the United States. Analytical results are reported in percent copper (% Cu) or parts per million (ppm) copper (10,000 ppm = 1.0%). Mining units are expressed in short tons (1 ton = 2000 pounds). All dollar amounts are in U.S. dollars.

The following conversions to Metric units are provided for the convenience of readers.

1 short ton (ton) = 2000 pounds (lb) = 0.9072 metric tons (tonnes (t))
1 foot (ft) = 0.3048 meters
1 yard (yd) = 3 feet = 0.9144 meters
1 mile (mi) = 5,280ft = 1.6093 kilometers
1 acre = 0.4047 hectares
1 square mile = 640 acres = 259 hectares
1 pound (lb) = 0.4536 kilograms (kg)

2.3 Basis of Report

This Report is based upon data, information obtained from external consultants, and the following information provided to HRC by Enexco:

- Core and reverse-circulation drill hole data – current and past drilling by Enexco and other companies
- Geological information and interpretations by Enexco and others
- Digital data provided to HRC by Enexco
- Site visit on June 7, 2012, by Zachary J. Black, EIT, SME-RM
- Site visit on August 1-2, 2013 by Jeff Choquette, PE, MMSA
- Site visit on August 1, 2013 by Dr. Deepak Malhotra, SME-RM
- Metallurgical tests
- Reports, listed in the references; and interpretations, opinions, assumptions, conditions, and qualifications set forth in these reports

2.4 Qualified Persons

The qualified persons, as defined by NI 43-101, responsible for this Report are:

- Jeff Choquette, PE, MMSA Qualified Person Member, Director, Mining Engineer, Hard Rock Consulting, LLC.
- Zachary J. Black, SME-RM, Director, Resource Geologist, Hard Rock Consulting, LLC.
- Terre Lane, MMSA Qualified Person Member, Consulting Mining Engineer, Hard Rock Consulting, LLC.
- Deepak Malhotra, Ph.D., SME-RM, President, Resource Development, Inc.

Mr. Jeff Choquette is responsible for Sections 15, 16 and 18 through 26. Mr. Zachary J. Black acted as project manager during preparation of this Report, and is specifically responsible for Sections, 1 through 12 and 14. Ms. Terre Lane is specifically responsible for Sections 15 through 22, and is responsible for the overall content and organization of the entire Report. Dr. Deepak Malhotra is responsible for Sections 13 and 17 of this Report.

2.5 Site Visit of Qualified Persons

Representatives from HRC, Zachary J. Black visited the Project on June 7, 2012 and Jeff Choquette on August 1-2, 2013. Resource Development, Inc. representative Dr. Deepak Malhotra visited the site on August 1, 2013. While on site, Mr. Black conducted general geologic field reconnaissance, inspected drill hole locations, and witnessed core drilling operations, sampling, and transportation. Mr. Choquette assessed site conditions, data and core storage. Dr. Malhotra inspected metallurgical sample locations, data and storage.

3. RELIANCE ON OTHER EXPERTS

The authors are not relying upon other experts for information except as follows:

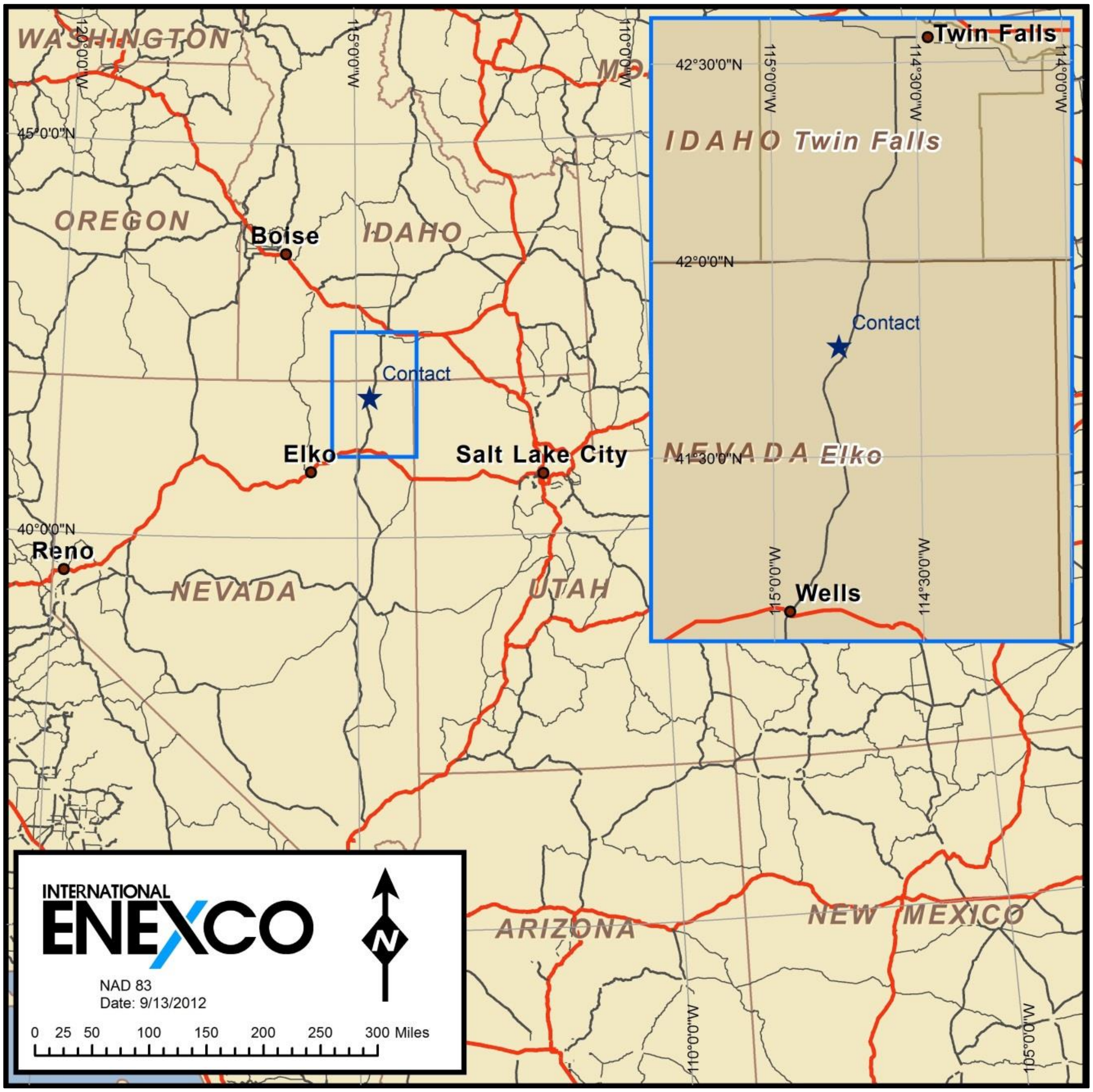
- The authors in the preparation of Section 4 have not independently conducted any title or other searches, but have relied upon Enexco for information on the status of claims, property title, agreements, permit status, and other pertinent conditions.

The authors have reviewed and incorporated reports and studies as described within this Report, and have adjusted information that required ammending.

4. PROPERTY DESCRIPTION AND LOCATION

The Project is located 62 miles south of the city of Twin Falls, Idaho, via U.S. Highway 93 in northeast Elko County, Nevada (see Figure 4-1 below). The Project's coordinates are 41° 47' North latitude and 114° 47' West longitude. The elevation at the Project is 6,000 feet above mean sea level. The mine local grid is in feet in NAD 83 Nevada State Plane 2701.

Figure 4-1 Contact Copper Project Location Map



4.1 Property and Mineral Tenure

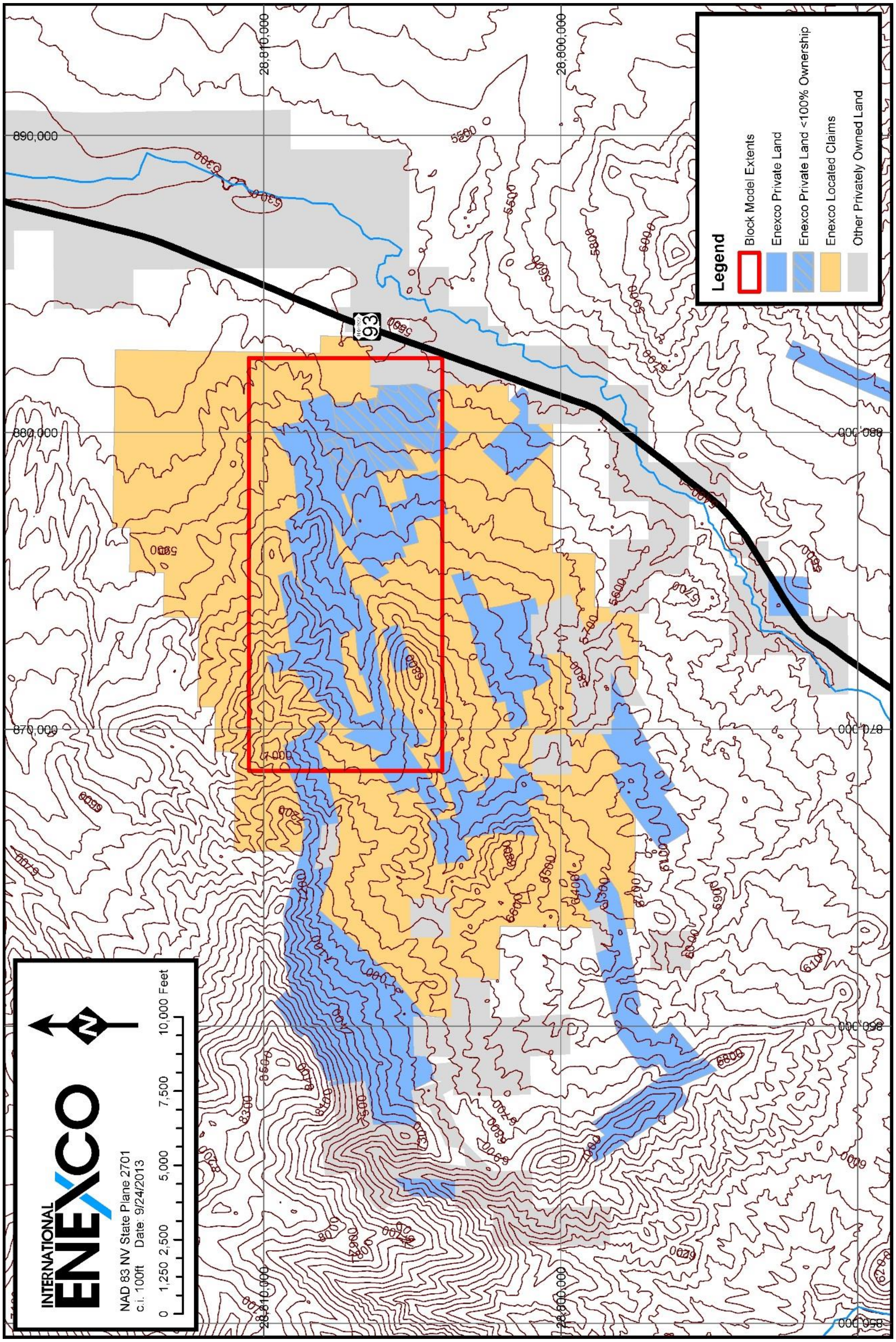
Enexco's land position at the Project comprises 2,650 acres in 156 patented claims and 4,320 acres in 288 unpatented lode claims (APPENDIX B LIST OF CLAIMS) as shown in Figure 4-2. All boundaries, whether patented or unpatented, were established by physical staking.

Enexco controls 156 patented mining claims of which, 154 are 100% owned by Enexco. Two claims, the Columbia and Columbia Fraction, are owned 87.5% by Enexco and are subject to an underlying 1.75% net smelter return royalty (NSR); the remaining 12.5% ownership is held by a private individual. An additional 44 patented mining claims are subject to a 0.25% NSR (APPENDIX B LIST OF CLAIMS). The holding cost for Enexco's patented mining claims is $2,895 per year in property tax payable to Elko County, Nevada, next due in August 2014. Enexco owns both the surface and mineral rights to the patented mining claims under the conditions described above. There are nine patented mining claims located within the Contact town site where Enexco holds all of the mineral rights, but only portions of the surface rights within the parceled lots of the town site on these claims (Figure 4-2 and APPENDIX B LIST OF CLAIMS). The surface rights may be acquired through purchase from the owner(s) at the current fair market value of the land, if needed for the Project.

Enexco owns a 100% interest in all 288 unpatented mining claims, which are on land administered by the U.S. Bureau of Land Management and cover areas in all directions from the exploration focus (Figure 4-2). The unpatented claims are subject to annual fees of $140 per claim with the Bureau of Land Management and $10.50 per claim with Elko County, Nevada, next due on September 1, 2014. Enexco holds the mineral rights and rights to surface use under the U.S. General Mining Law of May 10, 1872, as amended (30 U.S.C. §§ 22-54 and §§ 611-615) Title 43 of the Code of Federal Regulations (CFR) in Subparts 3700 and 3800 which is the major Federal law governing locatable minerals.

The Mineral Resource and Mineral Reserve defined and described in this Report fall entirely on Enexco's patented and unpatented claims. The location of the Mineral Resource and Mineral Reserve relative to the property boundaries is shown in Figure 4-2.

Figure 4-2 Map of Enexco's Patented and Unpatented Claims



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

The Project is west of the town site of Contact, Nevada, of which the year-round population is about 10. Fifteen miles north of the Project is the census-designated place of Jackpot, Nevada. Jackpot has a population of approximately 1,200 along with a post office, school, stores, and emergency services. The economic base of Jackpot includes hotel-casinos and ranching. The town is immediately south of the Idaho-Nevada border. The Jackpot labor force is housed at Jackpot or commute from Twin Falls, Idaho and the surrounding communities, 50 miles to the north.

The Project is located one mile west of U.S. Highway 93, a paved two-lane highway that connects Wells, Nevada and Twin Falls, Idaho. Access into the Project area is via an all-weather gravel road that runs west from the highway through Contact. Two high voltage power lines cross the highway on the northeast corner of the Project area. Enexco controls sufficient acreage to support a mining operation, including areas for mining, leaching, processing and waste storage.

The Project area lies on the eastern flank of Ellen D Mountain (elevation 8,631 feet). The resource area lies along a west-east drainage that extends from the divide on the south flank of Ellen D Mountain down to Contact (elevation 5,330 feet). The elevation at the divide is 7,054 feet. The north side of the drainage is formed by igneous (granitic) rocks and meta-sediments. The drainage and south slope are formed by weathered igneous rocks. At Contact, the drainage opens into a dry alluvial basin on the west side of U.S. Highway 93. The drainages in the Project area are ephemeral. With the exception of two small seeps, surface water in the Project area is absent. A small spring provides water to several residents of Contact. East of Contact, and east of U.S. Highway 93, is a perennial stream, Salmon Falls Creek.

The climate at the Project is semi-arid and typical of northeastern Nevada. Sagebrush, grass, and cactus grow on the property. The largest amount of precipitation occurs in the spring; summers are relatively warm with low rainfall; falls are cool and dry and winters are relatively cold with little snow. The total precipitation is 10.1 inches per year. Although the snowfall is light, high winds result in road closures on U.S. Highway 93 during some winter storms. Due to the high elevation and infrequent cloud cover, temperatures vary widely between day and night, but, overall, are generally moderate with an average annual maximum of 62ºF and an average annual minimum 30ºF. The relatively moderate climate and low snowfall have a minimal effect on exploration work and mining operations which are expected to be conducted year-round. During the summer and fall, rangeland fires occur within the region almost annually.

6. HISTORY

The first recorded discovery of copper in the Contact area was circa 1870. By 1908, the population of Contact reached 300 people. Production from the district according to sources such as Requa's *Laboratory Report on the Testing of Your Sample of Copper Ore* (1970), is reported as 300,000 tons of ore grading five percent Cu. However, only 34,404 tons of ore grading 4.8% Cu for the period from 1918-1949 is reported. The latter figure is probably closer to the actual production judging from the amount of underground workings in the area. Ores were mostly via rail to smelters near Salt Lake City, Utah. Reports show there were two attempts to operate small smelters at Contact, but the ore processed was minimal. There is no evidence of a mill at Contact, nor are there any reports of attempts to mill ores and produce flotation concentrates on a commercial scale. In the 1920s, an effort was made to in-situ leach copper from the adits on the Delano claim. The copper was leached with sulfuric acid brought in by rail cars from Salt Lake City, Utah and recovered in a scrap-iron launder located below town. Little physical evidence remains of the smelting or leaching operations.

From 1957-1967, the district was inactive. In 1967, Calta Resources, Ltd. acquired claims and started an exploration program for copper, which included geological mapping, IP surveys, trenching and drilling of 56 core holes totaling 47,417 feet. In 1970, the property was acquired by Coralta Resources Ltd., who optioned the property to Phelps Dodge Corporation. From 1973-1975, Phelps Dodge Corporation's work included IP-surveys, aeromagnetic surveys and drilling of 16 core holes totaling 30,594 feet, eight of which were drilled in excess of 2,000 feet with a maximum depth of 3,515 feet. The goal of this program was to test the property's porphyry copper potential; vein and disseminated copper mineralization were encountered in several holes, but the property was returned to Coralta.

In 1989, International Enexco, Ltd., the parent company of Enexco International Inc., acquired Coralta's patented claims for terms that included a 0.25 percent net smelter return (NSR) royalty. In 1998, International Enexco, Ltd. entered into a joint venture (JV) with Golden Phoenix Minerals, Inc. with the goal of defining a bulk tonnage copper deposit. From 1998-2004, Golden Phoenix conducted extensive rock-chip sampling, geophysical induced polarization and resistivity surveys, and drilling of 40 reverse circulation holes totaling 18,180 feet. Although results of the program were encouraging, Golden Phoenix discontinued the JV and the property reverted back to International Enexco, Ltd.

In 2006, International Enexco, Ltd. transferred ownership of the property to its subsidiary, Enexco International, Inc., a Nevada corporation. The Company commissioned a technical report in 2006 with Caracle Creek International Consulting Inc. (Jobin-Bevans and Kelso, 2006).

Based on recommendations in the Jobin-Bevans and Kelso report, in 2007, Enexco began a multi-phase drilling program with the objectives of upgrading the Mineral Resource base and defining an economic copper project. Using G & O Drilling of Alberta, Canada, 18 core holes totaling 15,354 feet were completed. Enexco then undertook subsequent drilling in-house. By December 2008, Enexco completed an additional 115 core holes totaling 103,821 feet. The drilling included infill, step-out, and metallurgical holes within the primary resource area. All assays for the 2007-2008 drilling were included in a pre-feasibility study in 2009.

Following the recommendations provided in the 2009 PFS, Enexco continued drilling in 2009-2010 and completed an additional 20 core holes totaling 19,120 feet. The assay results for these holes were included a 2010 PFS.

In 2010, Allied Nevada Gold Corporation (Allied) drilled four core holes east of Enexco's resource area totaling 2,670 feet in drill holes CON10-001 through CON10-004. This drilling was done by TonaTec Exploration, LLC of Mapleton, Utah. In September 2011, Enexco acquired all of Allied's land holdings within the Contact Mining District, making Allied's drill data part of the Project.

In 2010-2012, Enexco conducted two drilling programs. One program tested the northern extent of the resource area and consisted of three core holes totaling 3,664 feet in holes EN-154 through EN-156, which were drilled in-house by Enexco. The second program tested the east end of the resource area and in-filled areas as recommended in the 2010 PFS. This drilling comprised 24 core holes totaling 14,096 feet in holes EN-157 through EN-180, which were drilled by Rocky Peak Drilling of Twin Falls, Idaho; and 58 reverse circulation holes totaling 28,335 feet in holes ENR-1 through ENR-58 which were drilled by DeLong Construction, Inc. of Winnemucca, Nevada. The additional drilling by Enexco and Allied in 2010-2012 increased the sample density and overall size of the resource area. In October, 2012, 3L Resources prepared a technical report with an updated mineral resource estimate for the Project. No new drilling or sampling was done since the October, 2012 report.

7. GEOLOGICAL SETTING AND MINERALIZATION

All geologic ages used in this Report are from The Geological Society of America's *2009 Geologic Time Scale* (The Geological Society of America, 2009).

7.1 Regional Geology

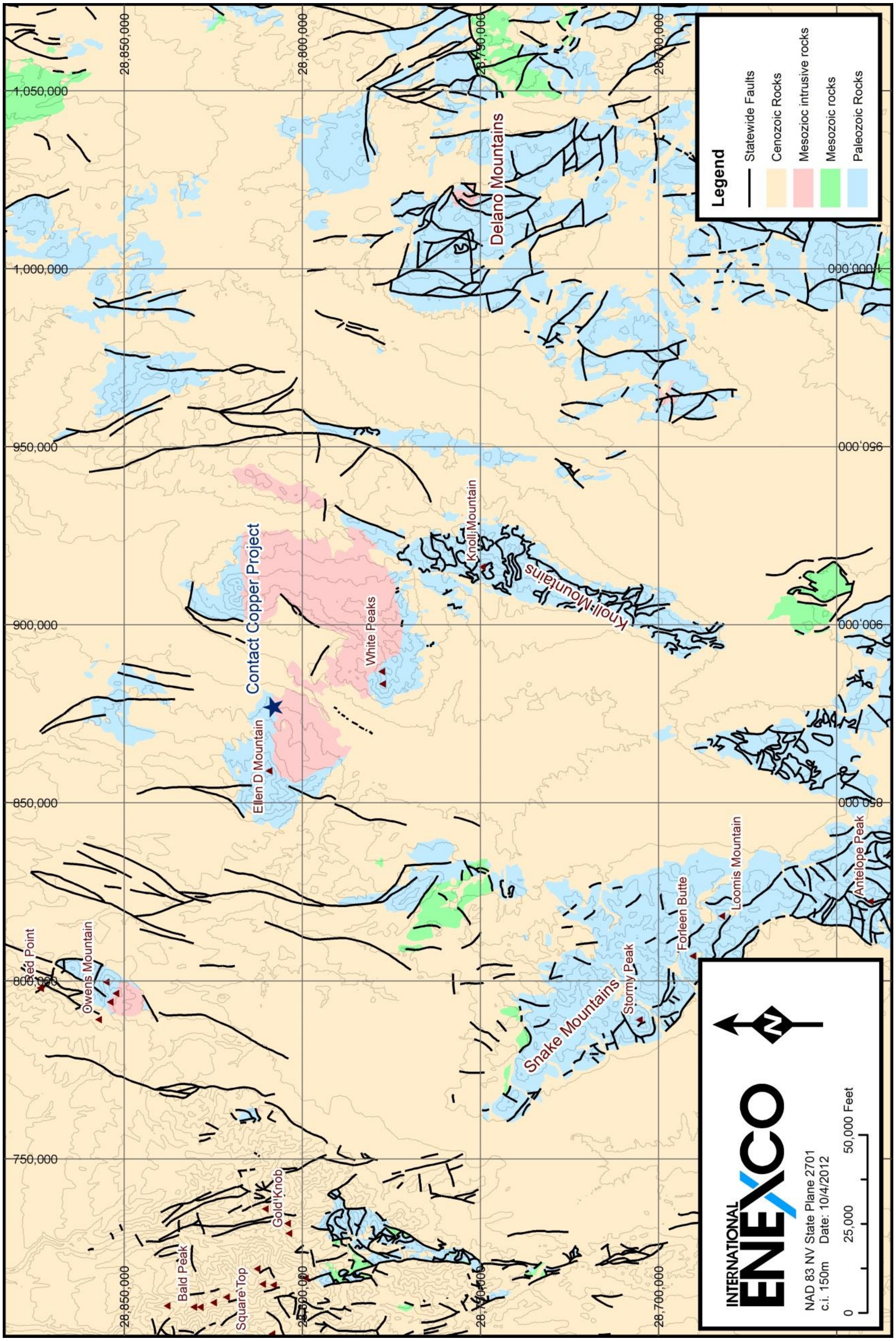
The exposed geologic record of northeast Elko County, Nevada begins with Precambrian sediments deposited on the continental shelf. During the Devonian period, the Antler Arc collided with North America, as seen in the Roberts Mountain Thrust, part of which is found in the Snake Mountains (Figure 7-1). The Roberts Mountain Thrust placed Ordovician through Devonian deep water sediments on top of shallow marine sediments as young as Mississippian in age. The Antler Mountains, unpreserved, were located in what is now western Nevada, creating a foreland basin over northeast Elko County.

These mountains fed sediments to the western side of the basin through the remaining of the Paleozoic. In the Earliest Triassic, the Sonoma Mountains were built in a similar location to the Antler Mountains from another collision. Sediments flowed into the foreland basin that comprised the region until the Late Triassic when inland seas regressed.

The region was then subjected to uplift from the Nevadan Orogeny in the late Jurassic around 150 Ma. At this time, the continental arc was 300 to 400 miles southwest of the region in the Sierra Nevada, at the edge of North America. From the Jurassic to the Miocene, the region was heavily eroded exposing the Paleozoic sediments.

During the Miocene, basin and range faulting extended the region east-west. Sediments of this region are terrestrial to lacustrine. Volcanic rocks in the region are related to this extension and the formation of the Jarbidge Caldera, which is part of the Yellowstone Hotspot track. The continued extension and erosion from basin and range faulting yields the landscape observed today (Blakey, 2011).

Figure 7-1 Regional Geologic Map of Northeast Elko County, Nevada



(Crafford, 2007)

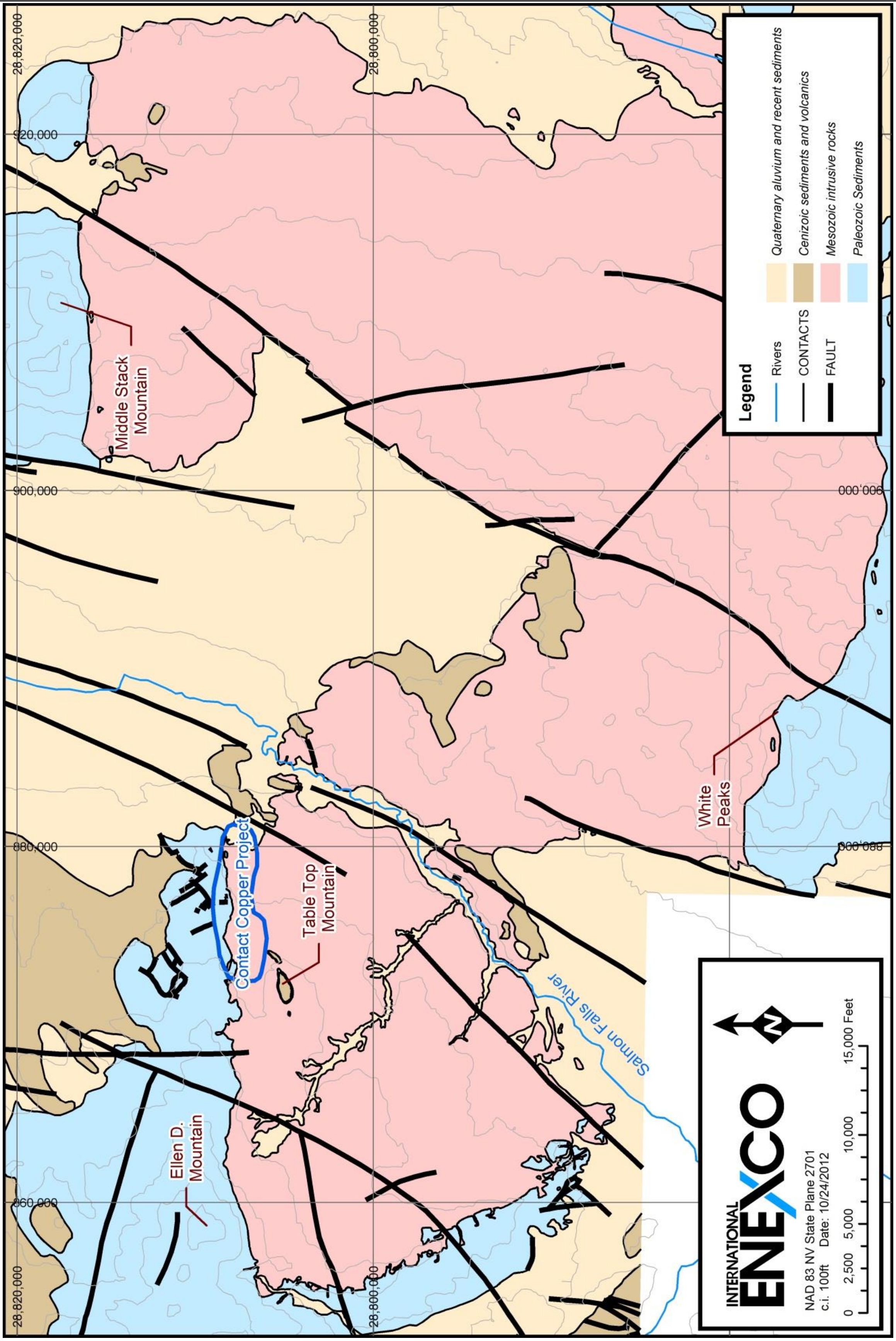
7.2 Local Geology

The Contact Mining District's earliest deposition, and earliest rocks began with sequences of marine sedimentary units of both siliciclastic and carbonate composition. The depositional time frame of these units occurred from the Carboniferous through the Permian (Larson & Scott, 1955) immediately following the Antler Orogeny. It is likely these siliciclastics were derived from the newly uplifted Antler Mountains of central Nevada. This deposition occurred in the foreland basin which covered eastern Nevada, western Utah, and eastern Idaho. Sediments in the district bear resemblance to the Permian Phosphoria Formation of Idaho and Utah (Gibbons, 1973). Deposition continued through to the Late Triassic. No evidence remains of Mesozoic sediments in the district. Structures unique to the Paleozoic rocks strike N 50° W (Gibbons, 1973). These steeply dipping lateral and normal faults are possibly associated with the Sonoma or the Nevadan orogenies.

The Contact batholith, which is exposed over most of the district, was likely the result of decompression melting during the Nevadan orogeny. The batholith, seen in Figure 7-2 below, consists of granitoid rocks, the most common being biotite-hornblende granodiorite. Mineralization in the district is associated with late stage differentiates of the cooling batholith. Emplacement of the batholith deformed the Paleozoic sediments in the area and tilted the sediments away from the batholith. The batholith has two major joint strikes in the Contact area: N 5° W and N 70° E (Gibbons, 1973). The latter joint set is coincident with the general trend of mineralization and alteration for the Contact Mining District.

The erosion which exposed the batholith and Paleozoic sediments began in Middle Triassic and continued into the Miocene when an increase in volcanism covered the Paleozoic sediments and Jurassic granites. Basin and range extension may have occurred penecontemporaneously with the volcanism and produced the Salmon Falls Creek valley which bisects the pluton. The Contact Mining District does not exhibit tilting typical of the basin and range province. Differential erosion over the last several million years has reduced relief of the batholith relative to the adjoining sediments.

Figure 7-2 Geologic Map of Part of the Contact Mining District



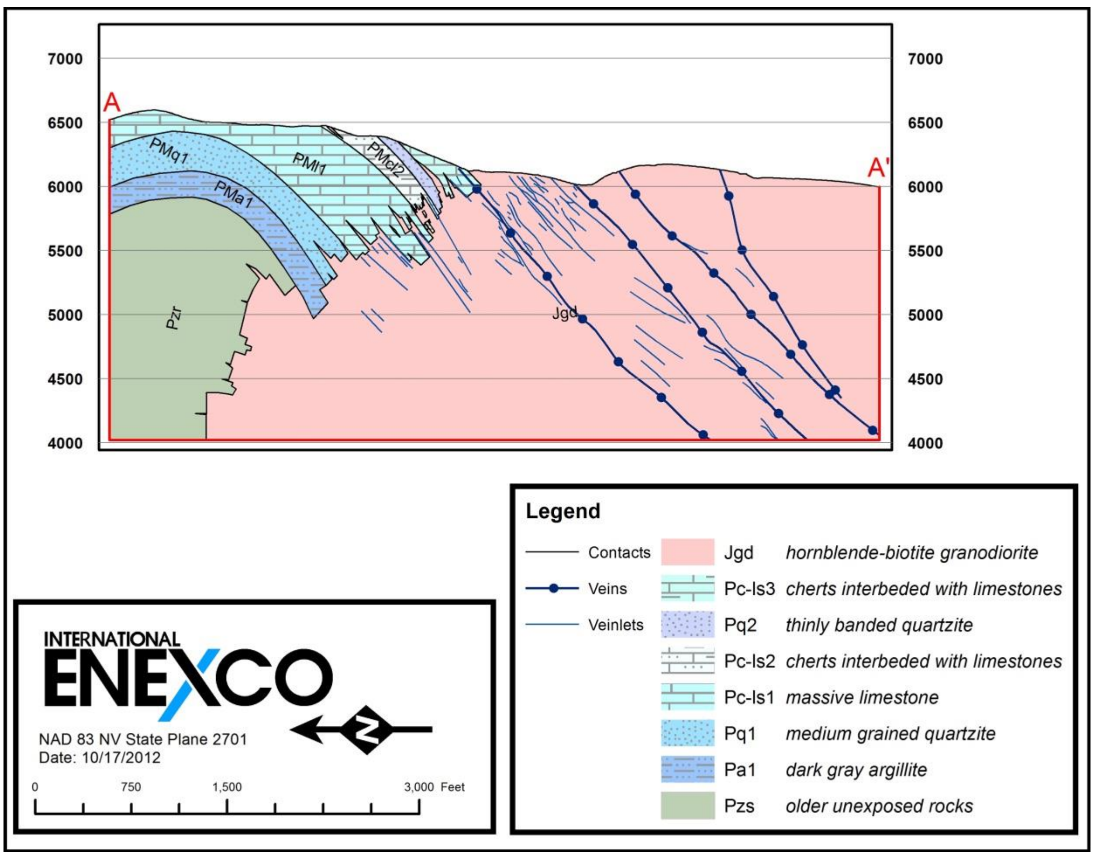
Adapted from Gibbons (1973), Plate 1

7.3 Property Geology

The Project lies one mile west of the town of Contact, Nevada and three and one-half miles east of Ellen D Mountain (Figure 7-2). It is located on the northern contact between the Jurassic batholith and the Paleozoic sediments. The Paleozoic sediments near this contact form an anticline with the south limb dipping steeply into the batholith and the other dipping gently away from the batholith (Figure 7-3). The contact between the intrusive and sediments dip between 45° and 60° North.

Paleozoic age sediments cap the ridges north of the intrusive and are of marine origin. The siliciclastic rocks range in grain size from clay to coarse-grained sand, and have carbonate contents ranging from zero to greater than 50% of the rock. Altogether, approximately 2,500 feet of Permian to Mississippian sediments are exposed on the property. Mineralization in the sediments consists most notably of skarn replacement deposits (see section 7.3.2.7). These deposits are generally hosted in silicified and garnetiferous limestone. The copper sulfide minerals, chalcopyrite and bornite, have partially replaced a percentage of the rock. In places, up to 20% of the rock has been replaced by copper sulfides; however, five percent is more commonly observed (Gibbons, 1973). Veins of quartz and metallic sulfides are also found in the sediments.

Figure 7-3 Cross-section A-A'



The granitoid rocks of the Contact batholith comprise a range of felsic compositions with the modal abundance being granodiorite. Nearly all mafics are either hornblende or biotite. The Contact batholith was emplaced as multiple phases of igneous activity. Feldspar porphyry dikes are thought to be related to early phases of the emplacement. These dikes are found cutting the Paleozoic sediments near the intrusion but not within the granodiorite of the main batholith (Gibbons, 1973); therefore, these dikes are considered precursor intrusions. Leucogranite and aplite dikes are found near the intrusive-sediment contact and throughout the batholith. These dikes are thought to be late stage differentiates of the main melt based on their composition and cross-cutting relationship with the granodiorite. These late stage intrusions show correlation to copper mineralization. South of Table Top Moutain, metasomatic zones with dikes and sills of aplite and leucogranite host copper mineralization in surface outcrops.

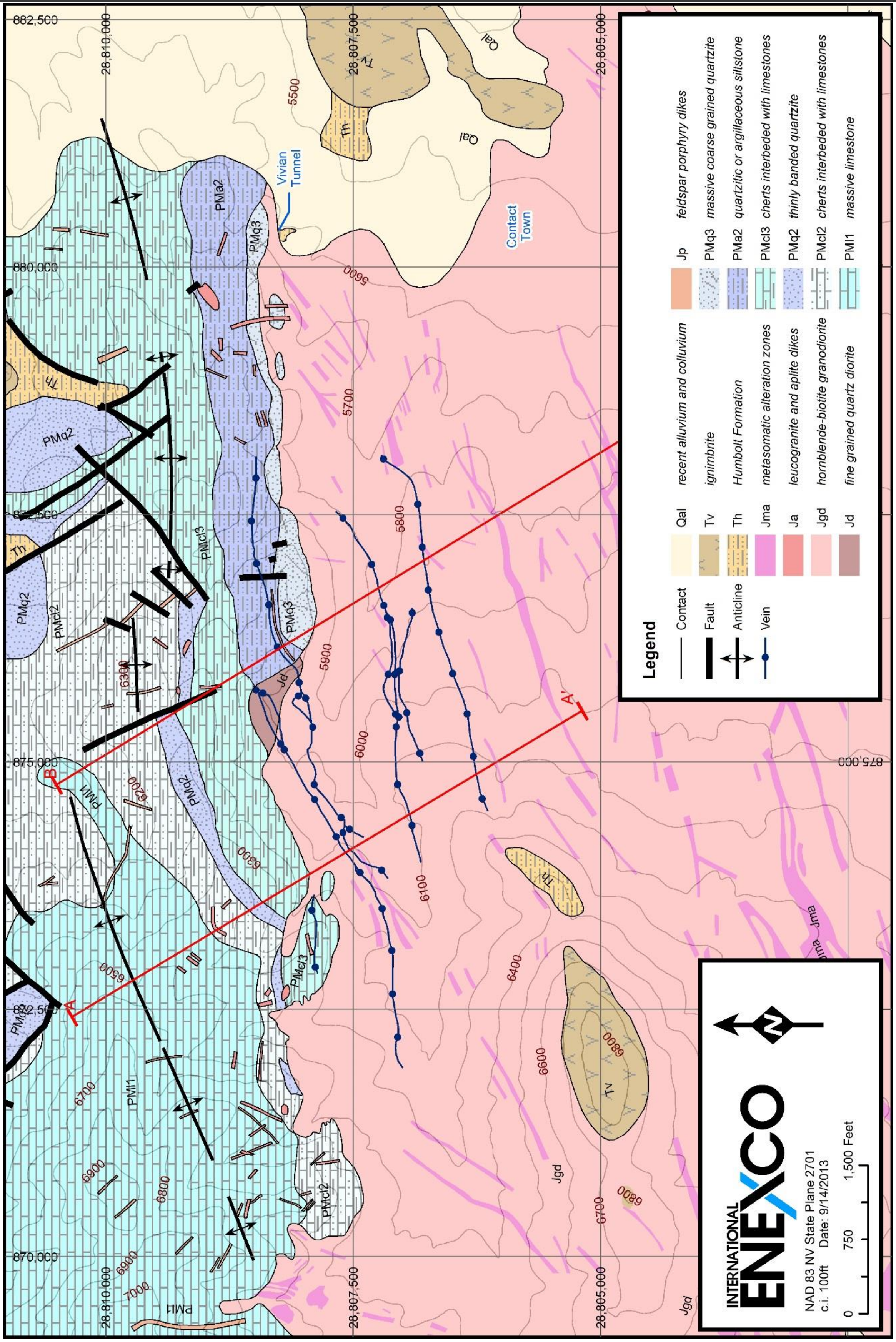
The metasomatic altered zones show an ENE to NE trend, as can be seen in Figure 7-4 below. These zones typically contain cores of leucogranite, aplite, and/or quartz veins. At the surface, these zones are leached and contain a matrix of felsic minerals and limonites (Figure 7-6, Photo d). Copper oxides can be found in these zones as fracture coatings, disseminations, and masses in gossans.

In the principal resource area, copper mineralization is defined by drilling for 7,500 feet along an east-west trend. The western end is open for further extension. The eastern end has been defined by drilling. Drilling has also outlined copper mineralization for 3,000 feet north-south along the trend with the northern and southern extents open. The copper mineralization occurs in and around quartz veins, and in smaller veins and veinlets accompanying the primary veins. Copper mineralization occurs in zones of disseminated copper oxides hundreds of feet wide around the major mapped veins. Examples in Figure 7-6 Photos a and e show densely fractured granodiorite with copper oxides and massive quartz veins with copper oxides. Coexistence of chalcopyrite and chalcocite, as seen in Figure 7-6 Photo c, in equilibrium indicates temperatures of 175° to 350°C (Gibbons, 1973, p. 126) in the veins. These temperatures place the deposit as a deep epithermal to mesothermal system (Gibbons, 1973).

The principal resource area is best described as a mesothermal sheeted quartz and copper vein system trending ENE and dipping 45-60°S. Supergene alteration has converted the primary sulfides to oxides in a zone about 1,000 feet from the surface. The oxides have disseminated into the wall rock away from the original fractures and veins to coat new fractures in the granodiorite.

Late Cenozoic sediments and volcanics overlie Miocene paleotopography. The Humboldt Formation is comprised of lacustrine sediments and water lain ash. Rhyolitic ignimbrites cap hills and give them flat tops. No primary copper mineralization is observed in any rocks younger than Jurassic age.

Figure 7-4 Contact Copper Project Geologic Map



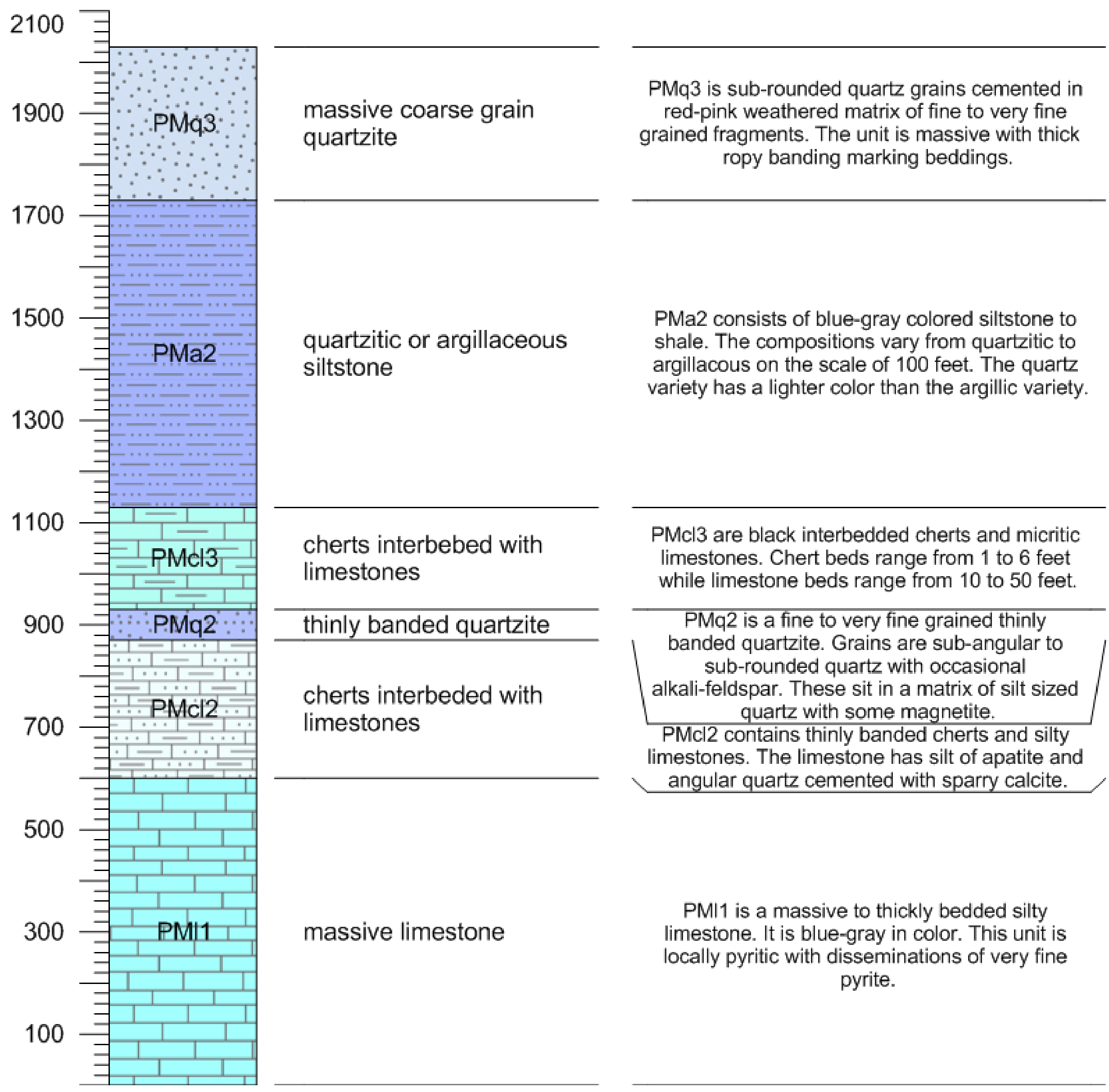
Adapted from Gibbons (1973), Plate 2

7.3.1 Lithological Descriptions

7.3.1.1 Marine Sediments of the Late Paleozoic

The sedimentary units at Contact were assigned Mississippian to Permian ages by Gibbons (1973) and others based on their similarity to rocks found in neighboring regions described by King in *Systematic Geology* (1878). The Mississippian to Permian sequence was deposited in the transition zone of a shallow inland sea. The majority of sediments exhibit planar bedding and were most likely deposited off shore. The siliciclastic sediments are generally clay to silt in size. Event beds with well-rounded very coarse sands are also present. The calcite in the limestones is sparry cement. Figure 7-5 below summarizes the sedimentary units observed at the Project.

Figure 7-5 Generalized Stratigraphic Column of Late Paleozoic Sediments of the Contact Deposit Area



Adapted from The Geology of Part of the Contact Mining District (Gibbons, 1973, p. 20).

7.3.1.2 Granitic Rocks of the Contact Batholith

The Contact batholith is composed of a range of felsic igneous intrusive rocks. The rocks have compositions from quartz diorite to leucogranite and have IUGS classifications of monzonite to syenite. Much of the metasomatic altered granodiorite mapped in Figure 7-4 (Jma) has the composition of tonalite from its sodic-calcic alteration. The bulk of the batholith is made up of granodiorite (Gibbons, 1973, pp. 35-56).

The biotite-hornblende granodiorite has an equigranular to weakly porphyritic texture. It is composed of euhedral plagioclase feldspar 5 to 15mm in length, and quartz, alkali-feldspars, biotite, and hornblende 2 to 10mm in length. Color is a medium gray and exhibits spherical weathering.

Porphyritic rhyolite dikes are observed cutting the sediments near Ellen D Mountain. These dikes contain plagioclase phenocrysts of 1-2mm in a gray aphanitic groundmass and are likely an early phase of intrusion due to absence of crosscutting relationships with the batholith.

Aplite and leucogranite dikes are observed crosscutting both the sediments and granodiorite. These dikes are highly felsic with less than five percent mafics; this and the crosscutting relationship with the granodiorite indicate their origin as late stage differentiates of the batholith. The difference between the leucogranite and the aplite is textural: the leucogranite has a medium grained granitic texture and the aplite has pegmatitic or aphanitic textures.

Zones of metasomatic altered granodiorite trend across the western batholith. In outcrop, these are white to orange to green. The orange color is derived from the oxidized and leached mafics and sulfides. The most commonly observed alteration on the surface is sodic-calcic, followed by potassic alteration. Potassic alteration is most commonly observed in drilling, and ranges from pink selvages inches in width to zones tens of feet wide containing 80% potassium feldspar. Propylitic alteration is also observed on surface and in drilling where mafic minerals are replaced by chlorite. Sericitic alteration is observed in drilling near the primary quartz veins.

7.3.1.3 Cenezoic Sedimentary and Volcanic Rocks

The Humboldt formation is a white thinly bedded unwelded ash. Interbedded in the ash are tan silts and clays. Thicknesses of this unit range from as little as 10 feet in the higher elevations to an estimated thickness of 400 feet or more in the basin to the east. This unit lies unconformably over the Miocene paleosurface of Jurassic and Paleozoic rocks. Abundant cross stratification and occasional slumping indicate a water lain origin for these sediments. A thin erosional lag surface of indurated gravels caps this formation.

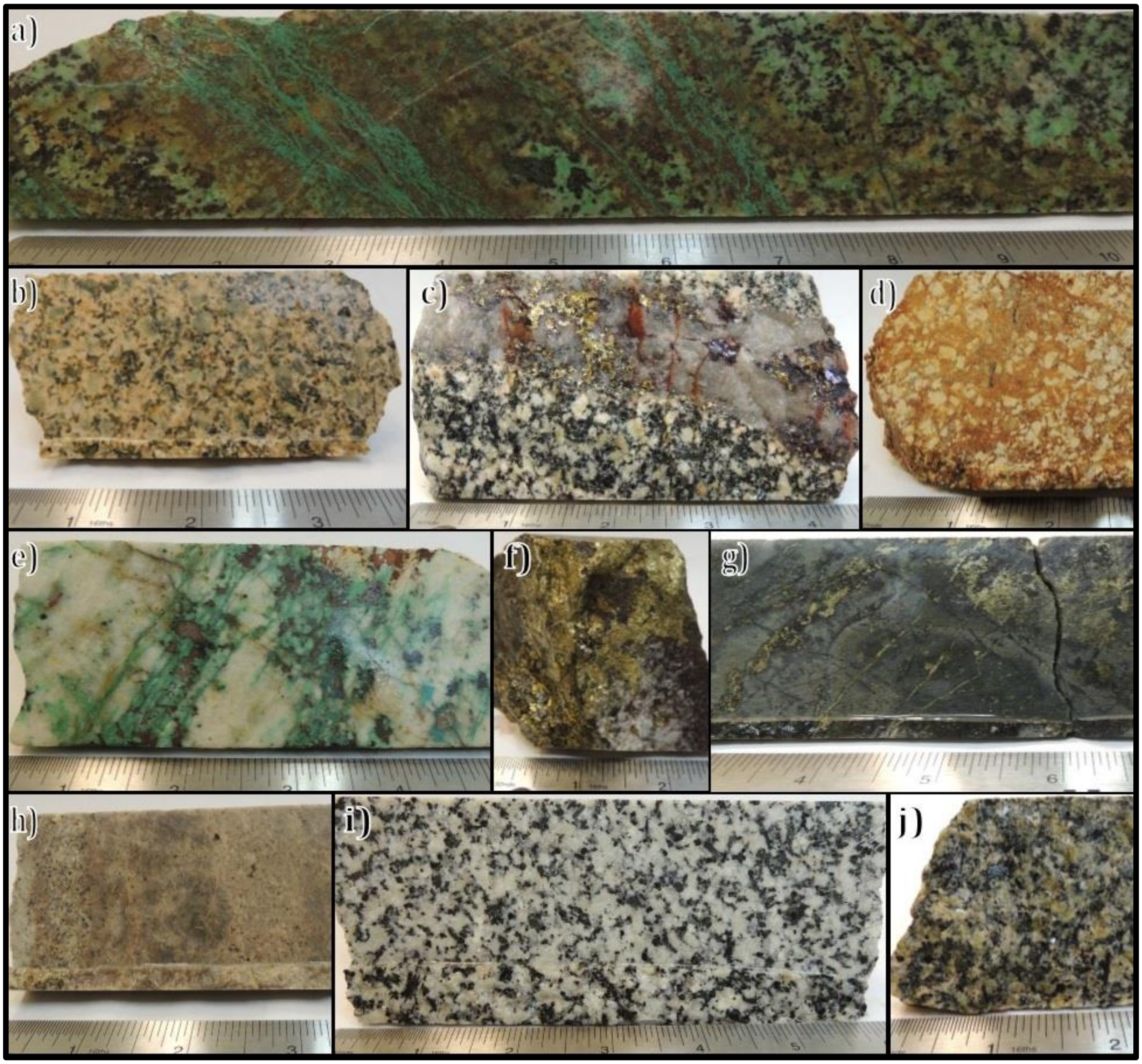
The most recent volcanics preserved in the region are rhyolite ignimbrites capping many of the low hills around the district. This unit comprises the flat top on Tabletop Mountain. The ignimbrites were deposited unconformably on older rocks, are welded, and have a basal vitrophyre.

The youngest units are the alluvium and colluvium filling the basin to the east.

7.3.2 Alteration and Mineralization

The description for the alteration types are from Richard H. Sillitoe's *Porphyry Copper Systems* (2010) and the U.S. Geological Survey's *Porphyry Copper Deposit Model, Chapter B of Mineral Deposit Models for Resource Assessment* (2010).

Figure 7-6 Selected Core Photographs of Alteration and Mineralization Types



Scales are in inches. a) *Copper oxides in veins and disseminated in granodiorite [EN-15 256'] b) Potassic alteration overprinting propylitic alteration. [EN-159 175'] c) Primary copper sulfide mineralization with quartz vein [EN-88 1460'] d) Iron oxides replacing mafics in leached granodiorite [EN-160 556'] e) Copper oxides and secondary sulfides in quartz vein [EN-15 254'] f) Massive primary copper sulfide [EN-46 554'] g) Primary sulfides in replacement skarn [EN-154 908'] h) Aplite dike rock [EN-158 591'] i) Unaltered granodiorite [EN-37 1081'] j) Propylitic altered granodiorite [EN-121 1891']*

7.3.2.1 Sodic-calcic

The primary minerals associated with sodic-calcic alteration are albite, actinolite and magnetite. Heavy sodic-calcic alteration adds new plagioclase while replacing other minerals in the granite. The rock takes on a very light color and resembles leucogranite dikes where pervasive and thick. This alteration is not associated with a sulfide mineral assemblage; however, it can overprint potassic alteration and preserve the sulfides.

7.3.2.2 Potassic

The primary minerals associated with potassic alteration are biotite and potassium feldspars. Potassium feldspars are dominant in granitic rocks. Alteration of plagioclase feldspars to alkali feldspar result in the rock having a pink hue. In the Contact deposit secondary pink plagioclase ranges from centimeter scale alteration selvages around small quartz veins and fractures to zones tens of feet wide. The sulfide assemblage can be pyrite-chalcopyrite, chalcopyrite ± bornite, bornite ± digenite ± chalcocite.

7.3.2.3 Propylitic

The primary minerals associated with propylitic alteration are chlorite, epidote, albite, and carbonates. The mafic minerals in granitoid rocks are altered into the above listed green minerals giving the rock a green color. The sulfide assemblage associated with propylitic alteration is pyrite.

7.3.2.4 Chlorite-sericite (Weak Sericitic)

The primary minerals associated with chlorite-sericite alteration are chlorite, sericite + illite, and hematite. Chlorite-sericite alteration shows up as green colored fine grained replacements of plagioclase. The sulfide assemblage of the chlorite-sericite zone is pyrite-chalcopyrite.

7.3.2.5 Sericitic

The primary minerals associated with sericitic alteration are quartz and sericite and hematite. Sericite forms selvages around late stage fracture controlled veins where all minerals are replaced with sericite.

7.3.2.6 Supergene

Supergene alteration occurs from meteoric water moving through sulfide rich rocks. The water oxidizes the sulfides and converts pyrite into sulfuric acid and aqueous iron sulfate, a solution capable of dissolving chalcopyrite. Copper from the oxidation of chalcopyrite may then precipitate as the mineral chalcocite, forming zones of supergene enrichment within a deposit. Supergene copper mineralization may oxidize to form other copper minerals in the form of oxides, silicates, and carbonates.

7.3.2.7 Skarn

Skarns are replacement of wall rocks with calcium-iron-magnesium minerals. Calcium-iron minerals are dominant in limestones.

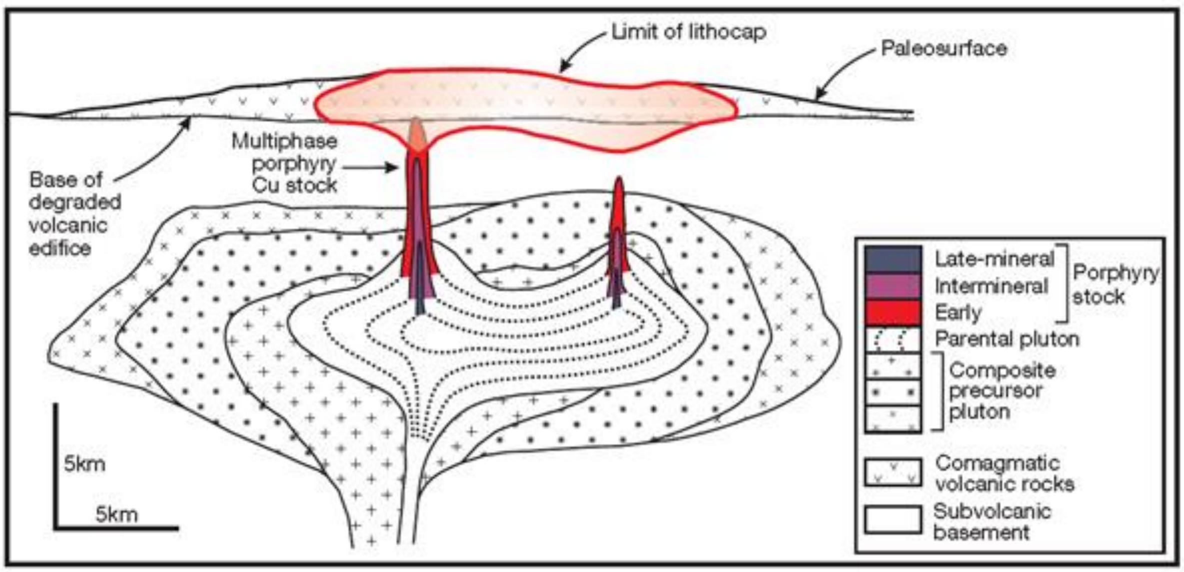
8. DEPOSIT TYPES

8.1 PORPHYRY COPPER SYSTEM

The model for mineralization at the Project is a porphyry copper system. Although a porphyry copper deposit has not been discovered in the area, the mineralization present is consistent with a porphyry copper model.

Porphyry copper systems involve the shallow emplacement of calc-alkaline multi-phase batholiths. These systems are commonly located in arcs associated with subduction and are also emplaced during extension following an orogeny. Metal enriched, late-stage differentiates cut through the precursor plutons in stocks from cupolas of the parental pluton as depicted in Figure 8-1 (Sillitoe, 2010, p. 6).

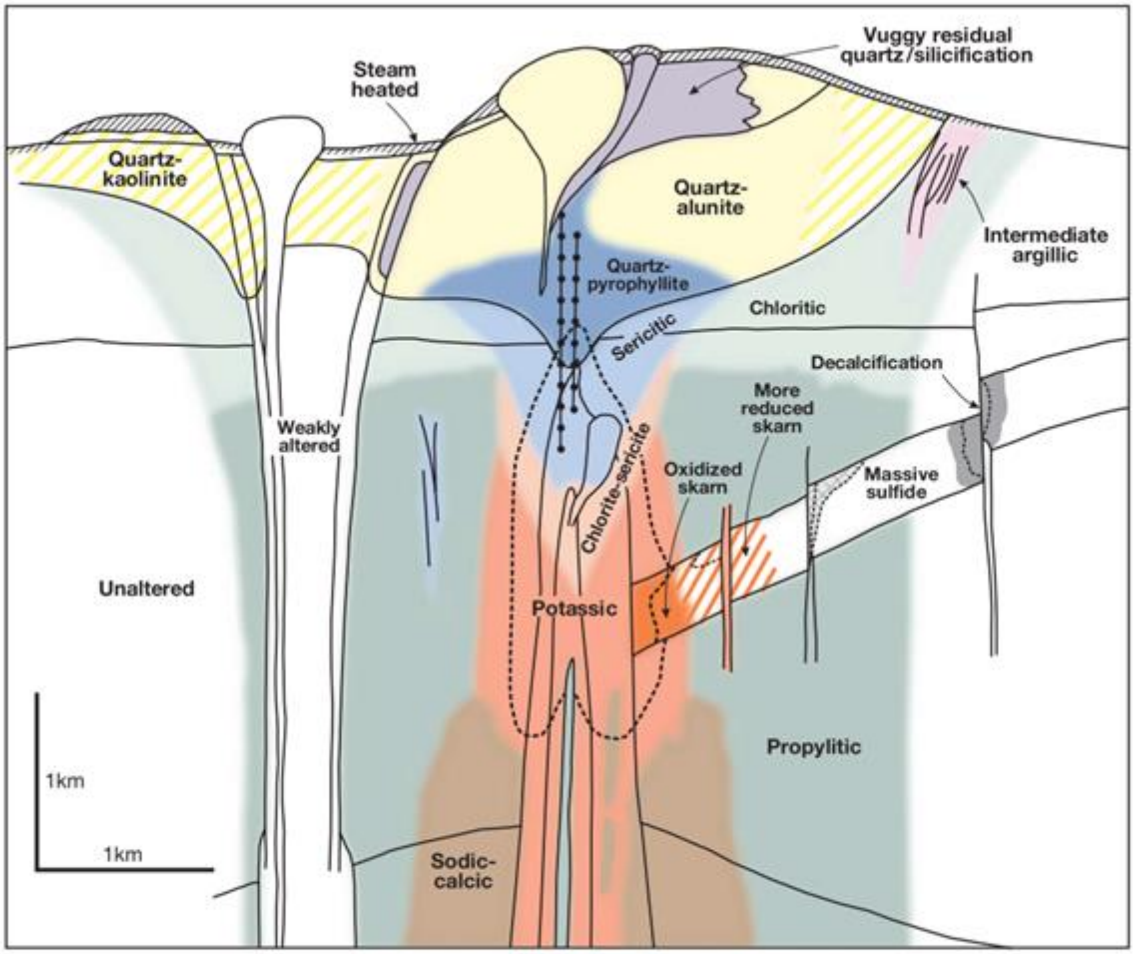
Figure 8-1 Spatial Relationships between Porphyry Related Rocks



Spatial Relationships between Porphyry Cu Stocks, Underlying Pluton, Overlying Comagmatic Volcanic Rocks, and the Lithocap. The precursor pluton is multiphase, whereas the parental pluton is shown as a single body in which the concentric dotted lines mark its progressive inward consolidation. The early, intermineral, and late-mineral phases of the porphyry Cu stocks, which span the interval during which the porphyry Cu deposits formed, originate from increasingly greater depths in the progressively crystallizing parental chamber. Note that subvolcanic basement rocks host much of the porphyry Cu deposit on the left, whereas that on the right is mainly enclosed by two phases of the precursor pluton (Sillitoe, 2010).

Alteration from the metal enriched fluids imprints a predictable pattern on the wall rock (Figure 8-2). The core of the alteration is potassic metasomatism. This core alteration is the main source for a porphyry copper deposit. Propylitic alteration extends laterally beyond the potassic alteration and is commonly barren with the exception of sub-epithermal veins. Above and centered on the potassic alteration zone are cones of chlorite-sericite and sericitic alteration which telescope down and overprint earlier alteration as the system matures and cools. Sodic-calcic alteration is located deep in the system and is likely sourced from sedimentary brines convecting heat during the cooling of the pluton (Sillitoe, 2010).

Figure 8-2 Generalized Alteration-mineralization Zoning Pattern for Telescoped Porphyry Cu Deposits



Note that shallow alteration-mineralization types consistently overprint deeper ones. Volumes of the different alteration types vary markedly from deposit to deposit (Sillitoe, 2010).

8.2 Porphyry Copper System Deposit Types

Descriptions for the porphyry copper system deposits are from the United States Geological Survey's *Porphyry Copper Deposit Model, Chapter B of Mineral Deposit Models for Resource Assessment* (US Geological Survey, 2010). Porphyry copper systems have three deposit types which are applied to the Project's deposit and the surrounding areas. These deposit types include porphyry copper, vein, and skarn replacement deposits.

8.2.1 Porphyry Copper Deposits

Porphyry copper deposits are centered on the porphyry stocks from late stage intrusions. Mineralization is centered on these stocks and occurs as disseminated sulfides, quartz veins, and fracture coatings. Deposits are generally low grade and high tonnage. Supergene processes mentioned above (Section 7.3.2) can both oxidize and enrich the deposit (US Geological Survey, 2010).

8.2.2 Vein Deposits

Vein deposits are quartz and sulfide veins deposited by the convection of fluids during the cooling of a porphyry copper deposit. These are ancillary to the porphyry copper deposit, and may follow a structural trend resulting from faulting, jointing, or emplacement of the intrusive with surrounding wall rocks. These deposits can form as stacked or sheeted systems or in stocks. The vein deposits may vary from moderate to high grade with moderate tonnage. Supergene processes can dilute the grade and disperse it into the wall rock. This process increases tonnage, decreases grade, and decreases the cost of recovery for the copper. Supergene processes can also increase copper content of the deposit (US Geological Survey, 2010).

8.2.3 Skarn Replacement Deposits

Skarn replacement deposits occur where metal enriched hydrothermal fluids replace a percentage of a carbonate wall rock with sulfides. Decarbonization and silicification alterations accompany skarn deposits. Generally, skarn deposits have irregular pod or tabular shapes, and create small deposits of high grade material (US Geological Survey, 2010).

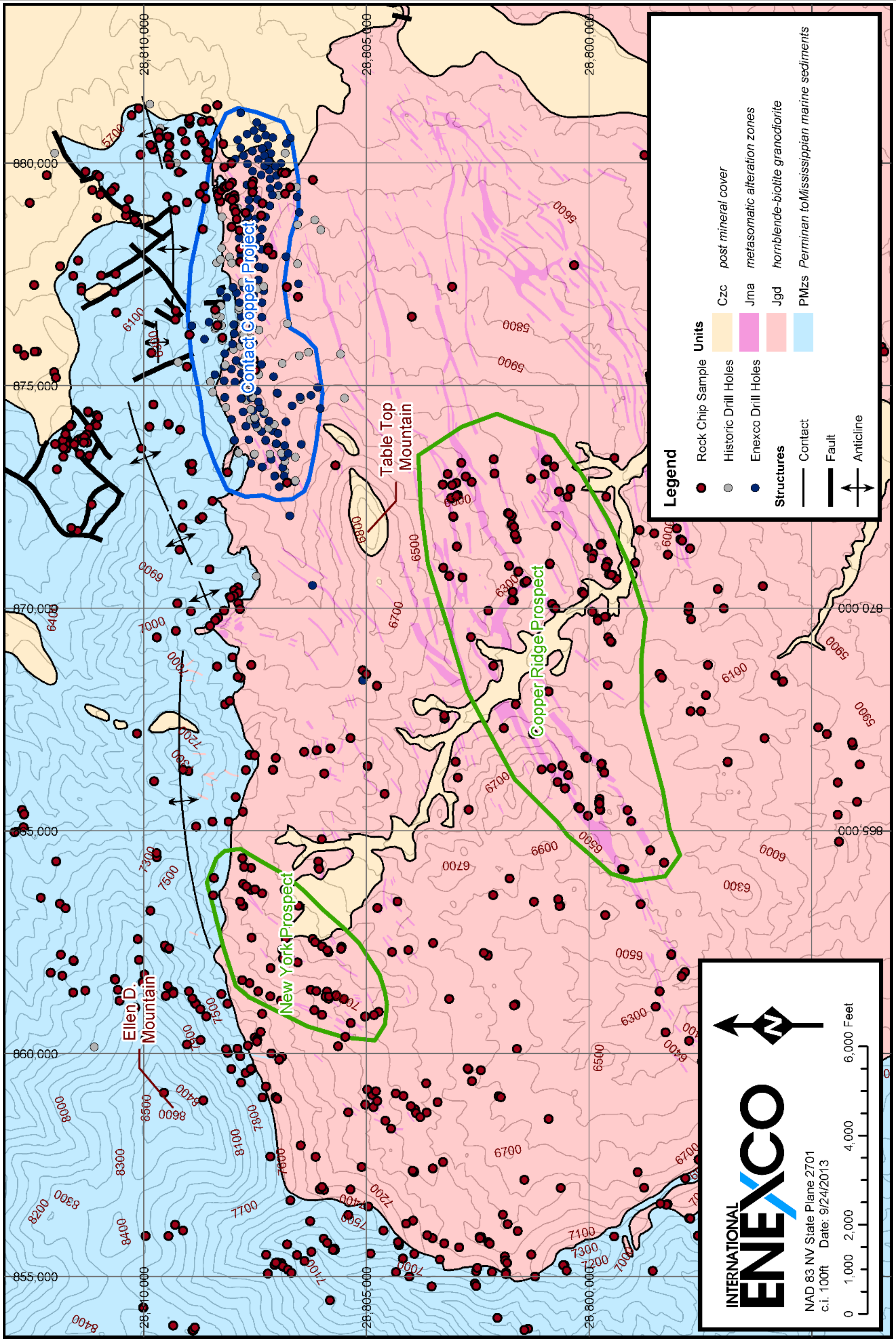
9. EXPLORATION

In 2011, Enexco's land holdings at the Project were increased through the acquisition of adjoining claims. Subsequently, surface sample coverage in the district was increased and additional geologic work was conducted. Enexco has identified additional areas where zones of copper oxide mineralization is present. The two most significant identified thus far are the Copper Ridge and the New York prospects (Figure 9-1).

The Copper Ridge prospect is located one mile southwest of the Project's primary resource area. Copper mineralization is present in exposed gossans, silicified quartz veins and veinlets, and leached zones with iron oxides. The area extends approximately 8,000 feet east-west by 2,000 feet north-south. Previous activity is limited to shallow prospect pits and the area has no previous drilling. The results of surface sampling program confirm the presence of copper across the area. Further geologic mapping and sampling is needed to identify drill targets.

The New York prospect is located two miles west of the Project's primary resource area on the southeast flank of Ellen D Mountain. Surface outcroppings of copper oxide minerals are present in zones of potassic alteration and silicification over an area approximately 2,500 feet by 250 feet in width that trends toward underground mine workings on Ellen D Mountain. The area has not been drill tested and requires additional mapping and sampling to identify drill targets.

Figure 9-1 Exploration Locations



Adapted from Gibbons (1973), Plate 1

10. DRILLING

10.1 PROJECT DRILLING HISTORY

Drilling at the Project can be divided into six phases of surface drilling (Table 10-1):

- 1967-1972 by Calta Resources Limited and Coralta Resource Limited
- 1973-1975 by Phelps Dodge Company
- 1998-2004 by Golden Phoenix Minerals, Inc.
- 2007-2009 by Enexco International, Inc.
- 2010 by Allied Nevada Gold Corp.
- 2010-2012 by Enexco International, Inc.

The drilling by Calta/Coralta (1967-1972) consisted of 56 core holes totaling 47,417 feet. The drilling focused on high grade veins, so intervals absent of visual copper tended not to be assayed. Assays were done at Bondar-Clegg & Company, Ltd. of Vancouver, BC. Core is available for drill holes (N-16 thru N-33). Fifty-five of these holes are within the current block model boundaries and 52 were used in the Mineral Resource estimation. The three omitted drill holes lack collar coordinates.

The drilling by Phelps Dodge Company (1973-1975) included 16 pre-collared holes, rotary drilled to depths of 600-1,000 feet and core drilled the remainder. Drilling totaled 30,594 feet. Eight holes were drilled in excess of 2,000 feet with a maximum of 3,515 feet. One hole, PD-4, intersected a significant intercept of high grade copper at depth believed to be a deep intercept of the Delano vein. Phelps Dodge's assays were performed at Rocky Mountain Geochemical Corp. of Salt Lake City, Utah. Six of these drill holes are within the current block model boundaries and were used in the Mineral Resource estimation.

The drilling by Golden Phoenix Minerals (1998-2004) consisted of 40 reverse circulation drill holes totaling 18,180 feet. The drill cuttings were generally sampled in five-foot intervals. Assays were done at N.A. Degerstrom, Inc. of Spokane, Washington. All of these drill holes are within the current block model boundaries and 38 are used in the Mineral Resource estimation. Two drill holes lacked collar coordinates and could not be located on the ground and were omitted from the Mineral Resource calculation. Chip trays are available for all of the Golden Phoenix drill holes.

Drilling by Enexco (2007-2009) included 153 core holes totaling 138,297 feet. The objective was to conduct infill drilling within an area recommended by Jobin-Bevans and Kelso (2006) and to confirm historical drilling. Ten of these core holes were drilled to obtain material for metallurgical testing. Assays were done at ALS Chemex, Reno Mineral Lab of Reno, Nevada, iPL/Inspectorate of Vancouver, BC and Sparks, Nevada and American Assay Laboratories of Sparks, Nevada. All drill holes are within the current block model boundaries and, with the exception of the ten metallurgical drill holes, were used in the Mineral Resource estimation.

Drilling by Allied (2010) consisted of four core holes, east of Enexco's 2010 resource area, totaling 2,670 feet. The drilling was intended to test for potential gold bearing zones. Assays were done at ALS Chemex, Reno Minerals Lab of Reno, Nevada. All four drill holes are within the current block model boundaries and were used in the Mineral Resource estimation.

Additional drilling by Enexco (2010-2012) included 27 core holes totaling 17,760 feet and 58 reverse circulation holes totaling 28,335 feet. Three holes in 2010-2011 were drilled to test the northern extent of the resource area. The remaining holes were drilled in 2011-2012 to test the east end of the resource area and infill areas as recommended in the 2010 PFS. Assays were done at ALS Chemex, Reno Minerals Lab of Reno, Nevada and SGS Canada, Inc. of Toronto, ON and Vancouver, BC. All 85 drill holes are within the current block model boundaries and were used in the Mineral Resource estimation.

Table 10-1 Contact Copper Project Drilling History

Start Year	End Year	Company	Drill Hole Series	Footage Drilled
1967	1972	Calta/Coralta	EK, N, C, BK, DDH	47,417
1973	1975	Phelps Dodge	PD	30,594
1998	2004	Golden Phoenix	CRC	18,180
2007	2010	Enexco	EN	141,959
2010	2010	Allied Nevada Gold	CON	2,670
2011	2012	Enexco	EN, ENR	42,434
			Total	**283,254**

10.2 Enexco Drilling and Sampling Procedures

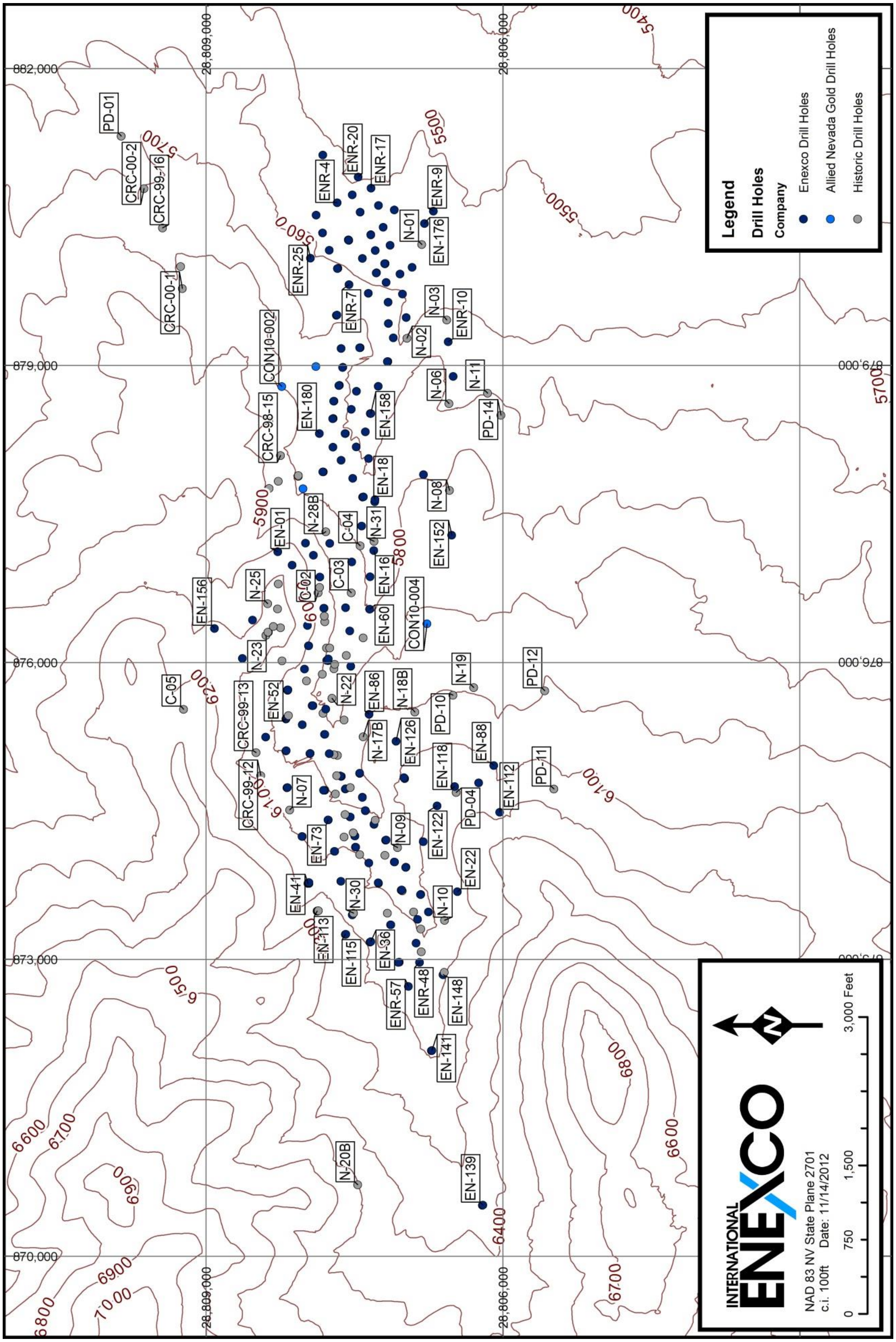
The core drilling conducted by Enexco utilized truck or skid mounted core drill rigs to drill holes EN-1 through EN-180. These holes were drilled primarily with HQ size core with reductions to NQ size core in areas where drilling depth or geologic conditions necessitated. The reverse circulation drilling was done with a track mounted reverse circulation drill utilizing a 5-¼ inch center return hammer bit or a center return tricone bit in areas were groundwater hindered drilling. The core holes were surveyed for azimuth and inclination at 200-foot intervals with a down-hole survey tool. The reverse circulation drill holes were surveyed for azimuth and inclination with a Brunton compass when collaring each hole. Following drill hole completion, holes were surveyed with a Trimble GeoXH GPS unit (accuracy of 2 feet horizontal and 4 feet vertical) and marked with rebar and aluminum tag indicating the drill hole name.

An Enexco representative was responsible for the core handling procedures at the drill rigs. The core was removed from the core barrel without any loss and was properly reassembled and placed in the core box in the correct orientation. Following each drill run, the depth of the hole was marked with a wooden block. Core boxes were marked with the drill hole number, box number and from-to footage noted on front and lid of the box. Full core boxes were securely covered and transported to Enexco's Filer, Idaho office for logging and splitting. Samples were selected for assay in the following manner: the core was continuously sampled in intervals of five feet to a depth of 500 feet in holes EN-1 through EN-156; below 500 feet in depth, areas with visible copper mineralization or copper detected above 0.1% with a Niton x-ray florescence tool were sampled in five-foot intervals. In holes EN-157 through EN-180, core was sampled continuously in intervals of two to eight feet, except in areas where post mineralized rock was encountered.

An Enexco representative was responsible for the sample procedures at the reverse circulation drill rig. The drill cuttings were separated with a cyclone splitter where 50% of the material was collected in a cloth bag for assay. The remaining cuttings were used for observation while drilling and to collect cuttings for chip trays, with the balance discarded to the sump. Samples were collected in five-foot intervals in bags labeled with hole number and from-to footage, a numbered sample tag was placed in the bag and a second sample tag stapled to the top of the bag. The full sample bags were placed in totes and transported to the assay laboratory. Representative cuttings of each sample were collected in 20-section plastic chip trays where each tray represented 100 feet of drilling, and were used for logging the geology.

The drill holes that comprise the Mineral Resource data base are listed in APPENDIX C DRILL HOLE COLLARS and are depicted in Figure 10-1 below. HRC's opinion is these drill holes have been drilled and sampled consistent with industry practices. Recovery though out all drilling has been good and averaged greater than 90%. There are no known factors that could materially impact the accuracy and reliability of the results. The interpretations of the results includes major structural features that dip 45-60 degrees south-southeast within the resource area. The widths of these structures, as intersected by the drilling, vary from hole to hole and are adjusted to true widths within the 3D block model. The Project is an advanced property under NI 43-101 guidelines. Drill results from previous operators are identified in Section 10.1.

Figure 10-1 Drill Hole Location Map



11. SAMPLE PREPARATION, ANALYSES AND SECURITY

Relevant information regarding sample preparation, assaying, and quality control measures is provided in the following sections.

11.1 SAMPLE PREPARATION

All diamond drill cores and reverse circulation drill chips from Enexco's drilling were logged, photographed, and tagged for sampling in the following manner.

In drill holes EN-1 through EN-153, the core was split in half using a diamond saw or manual impact splitter by Enexco employees in Filer, Idaho or contracted to Triad Labs of Twin Falls, Idaho. Intervals with visible copper mineralization were split with a diamond saw; other intervals were split by hand or sawed. After splitting, samples were tagged, bagged and transported to sample preparation laboratories IAS Environmental of Pocatello, Idaho, Triad Labs of Twin Falls, Idaho or American Assay of Elko, Nevada, where they were crushed and pulverized for analysis.

In drill holes CON10-001 through CON10-004, drilled by Allied, the drill core was logged and then transported to ALS Chemex, Elko Minerals Lab of Elko, Nevada. The sample preparation laboratory conducted the splitting of core, crushing and pulverizing for analysis.

In drill holes EN-154 through EN-180 the core was split in half using a diamond saw by Enexco employees in Filer, Idaho. Samples were tagged and bagged for transportation to a third party laboratory. In drill holes ENR-1 through ENR-58, the cuttings from reverse circulation drilling were split via a cyclone at the drill. Approximately 50% of the cuttings from each five-foot interval were collected, bagged and tagged on site, and shipped to the third party laboratory. The sample preparation laboratories where samples were crushed and pulverized for analysis were ALS Chemex, Elko Minerals Lab, and SGS North America, Inc. both of Elko, Nevada.

11.2 SAMPLE ANALYSES

Multiple assay laboratories were used for primary assays and check assays. Laboratory selections varied by work load and stage of the Project. American Assay Laboratories of Sparks, Nevada was the primary assay laboratory for drill holes EN-1 through EN- 8; ALS Chemex, Reno Minerals Lab of Reno, Nevada was the primary assay laboratory for drill holes EN -19 through EN-47, EN-154 through EN-156 and ENR-1 through ENR-3; iPL/Inspectorate of Vancouver, BC and Sparks, Nevada was the primary assay laboratory for drill holes EN-48 through EN-153; and SGS Canada, Inc. of Toronto, ON and Vancouver, BC was the primary assay laboratory for drill holes EN-157 through EN-180 and ENR-4 through ENR-58. ALS Chemex, Reno Minerals Lab of Reno, Nevada was the primary assay laboratory for drill holes CON10-001 through CON10-004 drilled by Allied. All laboratories used for analysis are independent of Enexco. Copper was determined by inductively coupled plasma atomic emission spectroscopy (ICP-AES) and atomic absorption spectroscopy (AA) methods. Other elements were determined by multi-element ICP for geochemical purposes.

All laboratories performing the analytical work were ISO certified at the time of assaying. In addition to its own internal programs, Enexco conducted a quality assurance and control (QA/QC) program using duplicate, standard and blank samples. For drill holes EN-1 through EN-47, check assays were performed on randomly selected pulps by ACME Analytical Laboratories Ltd. in Vancouver, BC. For drill holes EN-48 through EN-153, the QA/QC program was modified to include checks on pulps of blanks and standards that were systematically inserted into the sample stream by the third party sample preparation laboratories, and the check assays were performed by ACME Analytical Laboratories Ltd. of Vancouver, BC and ALS Chemex, Reno Minerals Lab of Reno, Nevada.

For drill holes ENR-154 through EN-156, standards and blanks were not used in the sample stream and check assays were performed by SGS Canada, Inc. of Vancouver, BC. The assays of drill holes EN-157 through EN-180 and ENR-1 through ENR-58 included standards and blanks inserted into the sample stream; check assays were performed by ALS Chemex, Reno Minerals Lab of Reno, Nevada. Drill holes CON10-001 through CON10-004 were not subjected to Enexco's QA/QC program and the results for these holes are as provided by Allied, the former property owner.

11.3 Sample Security

All drill cores from the 2007-2009 and 2010-2012 drilling were transported from the site by Enexco employees to a secure logging facility at Enexco's offices in Filer, Idaho. The reverse circulation drill samples from the 2011-2012 drilling were transported directly from the site to the sample preparation laboratory in Elko, Nevada. No employee, officer or director of Enexco conducted any part of the sample preparation with the exception of the core handling and splitting procedures described above. Bagged samples were transported by Enexco employees to the sample preparation laboratories. Prepared sample pulps were shipped by standard air or ground freight directly from the sample preparation laboratories to the assay laboratories.

11.4 Quality Control

Enexco's quality control programs included one of standard and blank insertion, one of check assays, and one of data entry. Table 11-1 and Figure 11-1 below summarize the nature, extent and results of the quality control procedures employed by Enexco. HRC's opinion is that the results fall within acceptable margins of laboratory error and provide adequate confidence in the data collection and laboratory methods.

11.4.1 Standards and Blanks

Four assay laboratories were utilized during Enexco's standards and blanks program. These laboratories were International Plasma Labs Ltd (iPL-now Inspectorate), Inspectorate, ALS Chemex, and SGS. Standards and blanks were inserted at a rate of approximately five percent each of samples assayed per drill hole. Failure limits for standards and blanks were based on whether an assay value fell outside two standard deviations of the population mean of the standard. The percentage of failures for standards was six percent and the percentage of failures for blanks was two percent.

Table 11-1 below lists the results for the program by assay laboratory. The failure rate for the standards sent to Inspectorate was higher than the other laboratories, so samples sent to this laboratory underwent additional data verification checks. Samples above the failure limits were checked for abnormalities.

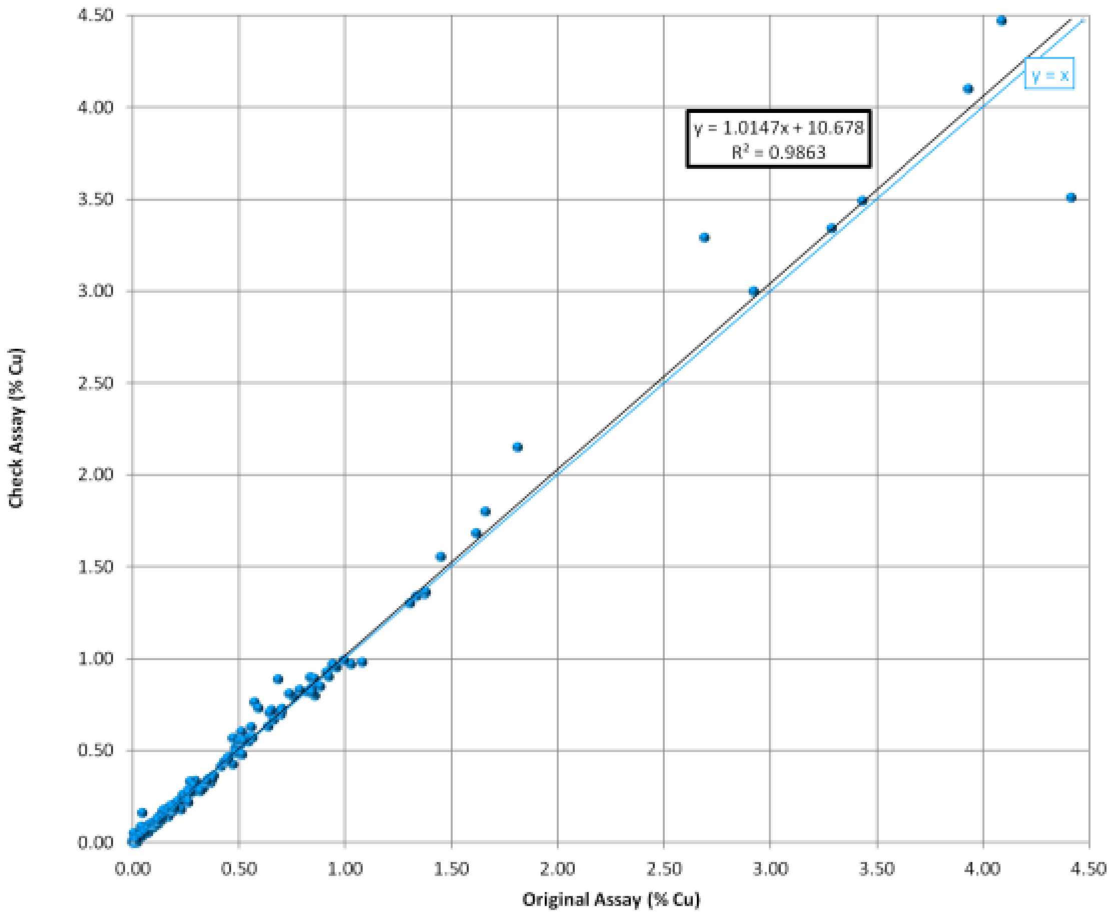
Table 11-1 QA/QC Program Results

	Inspectorate	iPL	Chemex	SGS	Total
Total Submitted Samples	2,894	4,138	7,503	7,708	22,243
Submitted Standards	148	175	58	216	597
Failed Standards	25	7	1	0	33
% Standards Failure	*17%*	*4%*	*2%*	*0%*	*6%*
Submitted Blanks	135	182	58	200	575
Failed Blanks	6	2	0	3	11
% Blank Failure	*4%*	*1%*	*0%*	*2%*	*2%*

11.4.2 Check Assay Program

A total of five assay laboratories were utilized in Enexco's check assay program. These laboratories were American Assay Laboratories (AAL), iPL, Inspectorate, ALS Chemex, and SGS. Check assays were selected on a hole-by-hole basis at a rate of approximately five percent of the samples assayed per drill hole. Detailed records were kept to assure check assays were not sent to the same laboratory as the one performing the original assays. Check assays were chosen on a random basis, with the exception of a number of samples selected from those assayed by Inspectorate. Check assay values ranged from 0% Cu to 4.4% Cu. For samples assayed by Inspectorate, all samples assaying higher than 1.0% Cu were selected for check assay. The scatter plot in Figure 11-1 shows the results of Enexco's check assay program. The slope of the line of best fit for the scatter plot is 1.0147 and the R^2 value for the line is 0.98. Both of these results indicate a strong one-to-one relationship between original assays and check assays. HRC' opinion is Enexco's check assay program provides additional confidence in the assay database.

Figure 11-1 Check Assay Program Results



11.4.3 Data Entry Validation Controls

All assay data compiled by Enexco is subject to data validation techniques. All data is stored in a secure database with built-in data entry validation controls. Any time a data validation control is breached, an error code is reported which allows the user to resolve the issue on the spot. Data validation controls include not allowing repeating drill holes or sample numbers, ensuring data is not duplicated. Footage "From-To" intervals are validated against each other and against a drill hole total depth. No "From-To" interval can be entered if the "From" or the "To" value is greater than the total depth of the drill hole or if the "From" value is greater than the "To" value. All geologic information is entered via a lookup table ensuring that only valid rock type names are entered and stored as numeric codes. Survey data entered is restricted to values between 0 and 360 degrees for azimuth and -180 and 180 degrees for inclinations. Down-hole surveys are validated against the total depth of drill holes, ensuring no survey depth exceeds the total depth of a drill hole. Assay values are imported directly into the database from the laboratory source files, eliminating errors in the assay data. Assays in the database are password-protected and locked from manual editing. All assays greater than 1.0% Cu are manually checked against their assay certificates.

The master database is exported to Maptek Vulcan's ISIS database program and subjected to further data validation. The ISIS database checks for overlapping intervals, missing intervals, and errors in collar elevations. The data is loaded into Maptek Vulcan's Envisage to visually check for errors, such as errors in drill hole location, alignment or length, or errors in lithological codes or assay values.

11.5 AUTHOR'S OPINION

HRC's opinion is the sample preparation, security and analytical procedures are correct and adequate for preparing this Report. The sample methods and density are appropriate and the samples are of sufficient quality to comprise a representative, unbiased database.

12. DATA VERIFICATION

Zachary Black of HRC conducted a visit to the Project on June 7, 2012, where he observed drilling in progress and examined the locations of drill sites. He also discussed geological features of the deposit, reviewed geological logs, and inspected drill core and reverse circulation cuttings at Enexco's storage facility in Filer, Idaho. The following sections discuss HRC's verification of data for the Project.

12.1 HISTORICAL DATA VERIFICATION

HRC did not collect independent samples to check historical data. Samples were collected by Ian Kelso in 2006 to check historical data. The sample results are given in the Jobin-Bevans and Kelso report (2006). This report was reviewed by HRC and HRC believes additional check analyses are not needed on previously collected samples.

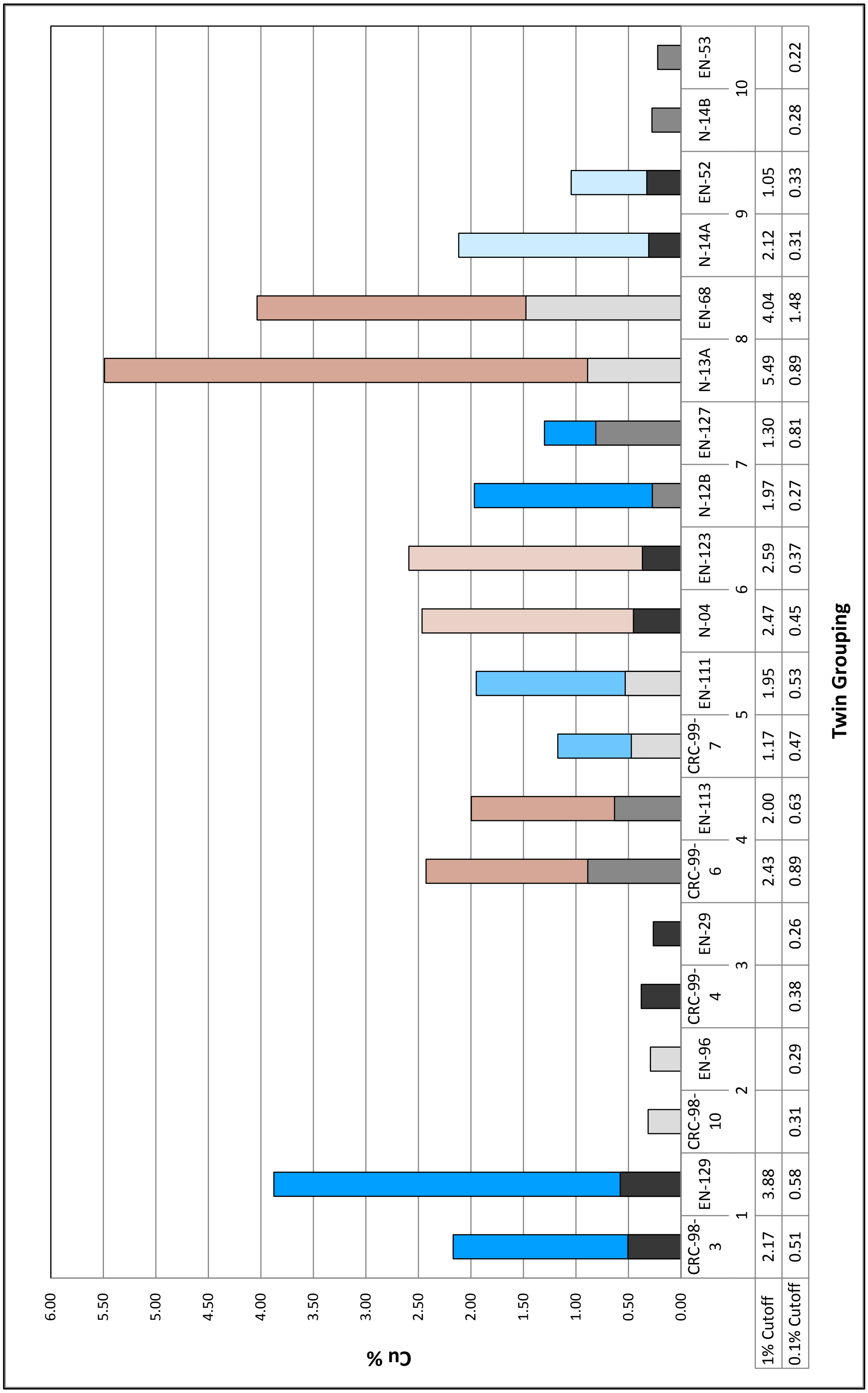
Enexco, in the course of drilling from 2007-2012, has drilled holes in close proximity to historical drill holes, Table 12-1 lists these holes.

Table 12-1 Historical Drill Hole Twin Sets

Twin Set	Drill Hole	Az.	Year Drilled	Incl.	Drill Hole	Az.	Year Drilled	Incl.
	Historical				Enexco			
1	CRC-98-3	323	1998	-45	EN-129	320	2008	-55
2	CRC-98-10	323	1998	-60	EN-96	320	2008	-55
3	CRC-99-4	321	1999	-70	EN-29	318	2007	-60
4	CRC-99-6	321	1999	-70	EN-113	320	2008	-55
5	CRC-99-7	N/A	1999	-90	EN-111	N/A	2008	-90
6	N-04	322	1969	-45	EN-123	320	2008	-55
7	N-12B	322	1969	-50	EN-127	320	2008	-55
8	N-13A	322	1969	-35	EN-68	321	2008	-55
9	N-14A	N/A	1969	-90	EN-52	N/A	2008	-90
10	N-14B	322	1969	-60	EN-53	316	2008	-55

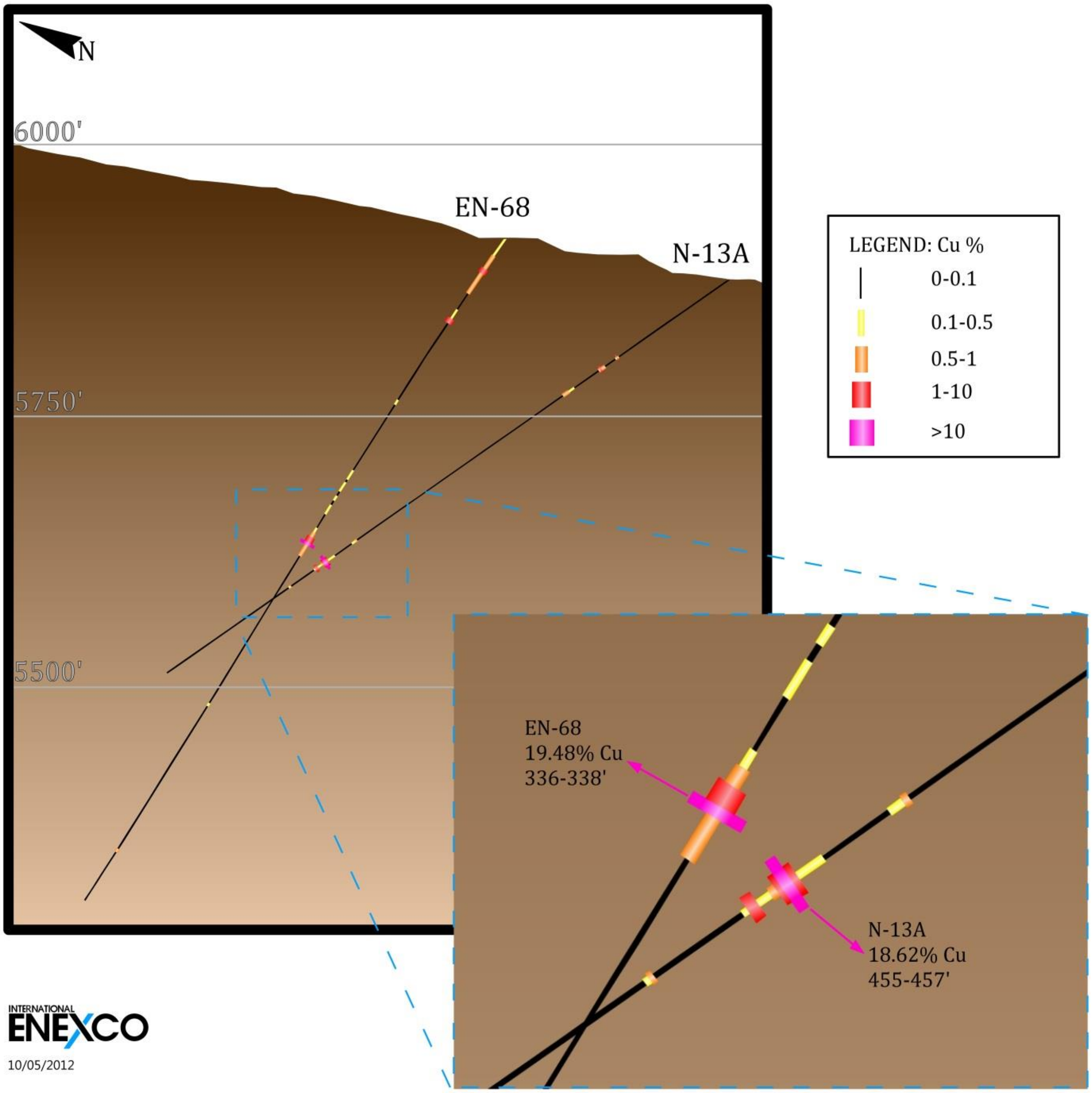
Enexco's holes were drilled in similar orientations to nearby historical holes, making it possible to compare intervals by weighted averages of copper grades. Figure 12-1 shows the twin sets compared at cut-offs of 0.1% and 1.0% Cu. The twin sets were also examined in cross-section. Figure 12-2 shows an example of a twin set in cross section. HRC's opinion is the historical data for the Project was collected following standard industry practices for drilling, sampling, and assaying at the time. Comparison of data from previous drill holes with recent drilling verifies that the historical data is accurate and adequate for preparing this Report.

Figure 12-1 Twinned Drill Hole Results



Twin Grouping	Hole	1% Cutoff	0.1% Cutoff
1	CRC-98-3	2.17	0.51
1	EN-129	3.88	0.58
2	CRC-98-10		0.31
2	EN-96		0.29
3	CRC-99-4		0.38
3	EN-29		0.26
4	CRC-99-6	2.43	0.89
4	EN-113	2.00	0.63
5	CRC-99-7	1.17	0.47
5	EN-111	1.95	0.53
6	N-04	2.47	0.45
6	EN-123	2.59	0.37
7	N-12B	1.97	0.27
7	EN-127	1.30	0.81
8	N-13A	5.49	0.89
8	EN-68	4.04	1.48
9	N-14A	2.12	0.31
9	EN-52	1.05	0.33
10	N-14B		0.28
10	EN-53		0.22

Figure 12-2 Group 8 (N-13A & EN-68) Cross-section



12.2 CURRENT DATA VERIFICATION

HRC did not collect independent samples to check current data. Mr. Black of HRC visited the site and observed the core handling, logging and sampling procedures of Enexco and concludes the procedures meet current industry standards. Locations and elevations of historical and current drill holes were checked on aerial photographs and 3D topographic surfaces.

HRC has reviewed Enexco's check assay programs and believes the programs provide adequate confidence in the data. HRC has reviewed the assay database and conducted spot checks on drill holes selected at random with drill logs and assay certificates, and found no errors. All drill cores and cuttings from Enexco's drilling have been photographed. Drill logs have been digitally scanned and archived. The split core and cutting trays have been securely stored and are available for further checks.

12.3 ADEQUACY OF DATA

HRC's opinion is the historical and current data is adequate for the purposes of preparing this Report. Historical data is consistent with the current data. Current data is subjected to ongoing data checks. The historical and current data is stored in a secured database.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

Dr. Deepak Malhotra, Registered Member of SME and President of Resource Development Inc. (RDI), is responsible for the metallurgical and mineral processing aspects within this section. Dr. Malhotra is a Qualified Person as defined by NI 43-101 and is independent of Enexco.

13.1 Metallurgical Summary

Metallurgical tests have been carried out on samples from the Project by various companies over a 40 year period. Table 13-1 below is a summary of all known test work.

Table 13-1 Summary of Metallurgical Test Work

Year	Company	Lab	Source	Tests	Samples
4/14/1970	Calta Mines	Gallagher Company	Test 1919 Report	Flotation and Vat leach	Composite core sample 2.37% Cu
6/16/2000	Golden Phoenix Minerals	McClelland Laboratories, Inc.	Contact Project Update Report Job 2769	Bottle roll and columns	Surface grab samples 0.25 – 4.67% Cu
2/7/2000	Golden Phoenix Minerals	Degerstrom	Columns Leach Test work on Contact Copper Granodiorite Samples	Column leach tests	Surface grab (bulk) samples 1.47-4.01% Cu
4/20/2007	Enexco	Dawson Metallurgical Laboratories, Inc.	Report Project P-2977	Flotation and vat leach tests	Core sample 1.94% Cu
9/2/2008	Enexco	CAMP, Butte	Diagnostic Copper Leach Testing Report	Bottle roll tests	28 core samples 0.15-2.77% Cu
12/3/2008	Enexco	Resource Development Inc. (RDI)	Preliminary Metallurgical Testing Report	Bottle roll and static leach tests	Core samples 0.14 – 6.1% Cu
5/27/2009	Enexco	McClelland Laboratories, Inc.	MLI Job 3311 Report	Bottle roll and column leach tests -	Composite core samples 0.18 – 0.99% Cu
12/15/2010	Enexco	McClelland Laboratories, Inc.	Materials Characterization Tests and Analyses Job 3311 Report	Leach Residue Analyses, Mod ABA static ARD potential tests, MWMP extraction tests	Composite core samples 0.18 – 0.99% Cu
8/11/2011	Enexco	Phillips Enterprises, LLC		Crushing and abrasion tests, SG determinations	Composite core samples 0.16-0.64% Cu
7/31/2012	Enexco	GeoSystems Analysis, Inc.	Job 91200-B Report	Permeability tests	Composite core samples 0.15-0.54% Cu
9/9/2012	Enexco	McClelland Laboratories Inc.	MLI Job 3581 Data	Column leach tests	Composite core samples 0.16 – 0.64% Cu
8/7/2013	Enexco	Metcon Research	Column Leach Study on GA, GB and MID Composite Samples	Bottle roll and column leach tests	Composite core samples 0.15-0.54% Cu
8/12/2013	Enexco	SGS Metcon	Column Leach Study on DEEP Composite Samples	Bottle roll and column leach tests	Composite core samples 0.26-0.69% Cu

13.2 Discussion

Work in the last 20 years has focused on tests to determine the amenability of the copper mineralization at the Project to leaching with sulfuric acid. The tests have included bottle roll, static, and open- and closed-cycle column leach tests. Tests by Enexco from 2008 to 2013 can be grouped into two programs:

1. 2008-2010 determined characteristics of vein and wall rock mineralization; and
2. 2011-2013 determined characteristics of weathered versus un-weathered mineralization.

The samples tested in both programs are representative of the predominant types and styles of mineralization identified at the Project, which consist of copper oxide minerals hosted in quartz veins and within veinlets, fracture coatings and disseminations in granodiorite. The most significant changes occur vertically by way of a zone of surficial and structural decomposition within the granodiorite that is up to 270 feet in thickness. The locations of drill holes used to obtain samples are shown in Figure 13-1. Relevant results from 2008-2010 included tests by RDI in Denver, Colorado on drill core from holes EN-72 and EN-74. RDI concluded from bottle roll and static leach tests that a crush size of (P100) 1-inch would be sufficient for heap leaching, and predicted copper extraction would be about 80%, with gangue acid consumption of 11 lb/ton. McClelland Laboratories, Inc. (McClelland) in Sparks, Nevada, subsequently performed column leach tests on composite samples of core from five drill holes, EN-70, EN-72, EN-74, EN-80, and EN-116 (Table 13-2). Copper extractions ranged from 67.6% to 85.9% and net acid consumptions ranged from 43.4 to 97.8 lb/ton (Table 13-3).

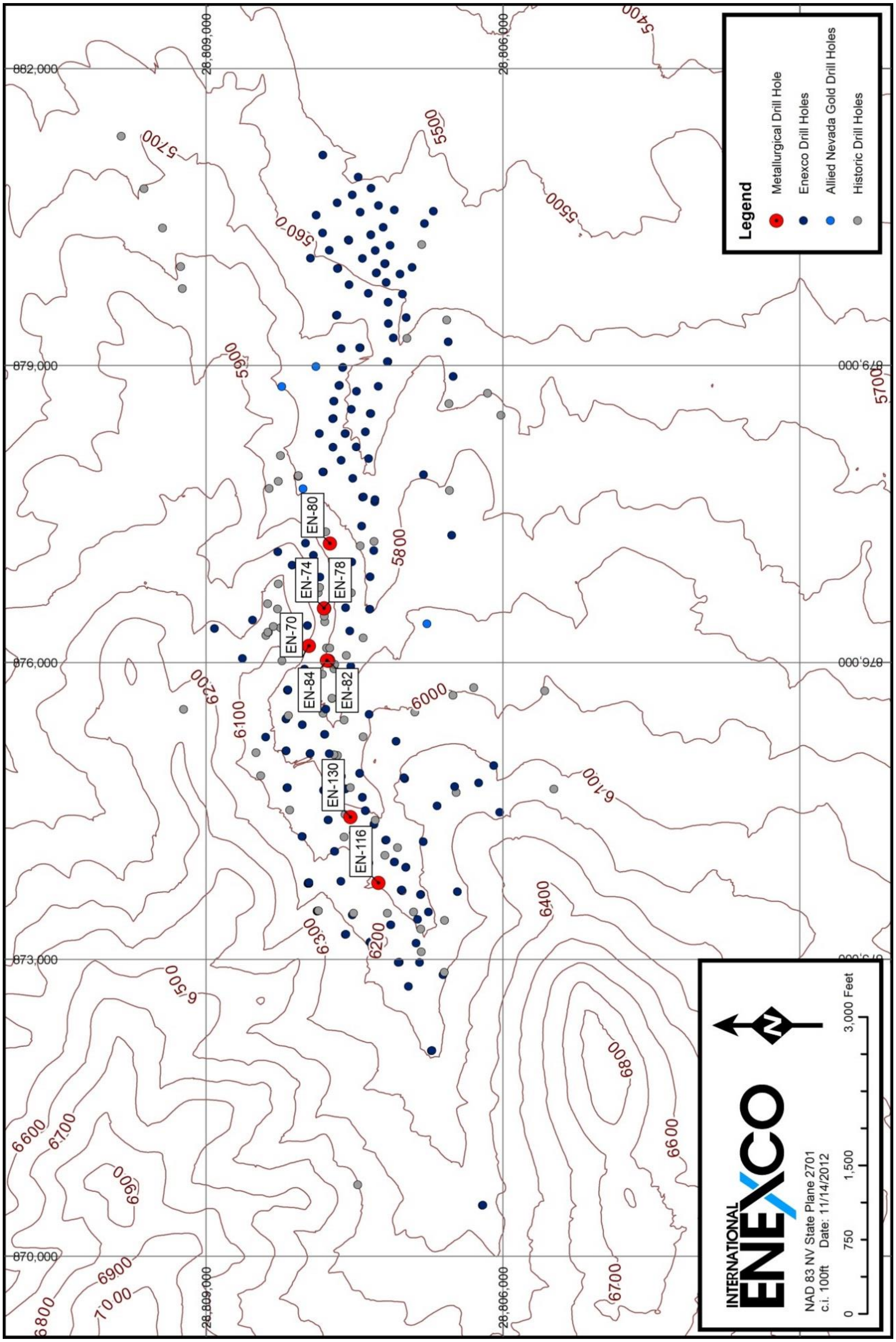
Table 13-2 Description of 2009 Composite Samples

Composite	Drill Holes	Sample Depth
Comp 1	EN-74/EN-80	500-650 feet
Comp 2	EN-72	200-400 feet
Comp 3	EN-70	0-300 feet
Comp 4	EN-116	0-420 feet

Table 13-3 Column Leach Results, McClelland 2009

Days under leach: 69-129	Feed Size (P100)					
	1 inch			1/2 inch		
Composite	% Cu Calculated Head	Cu Recovery, %	H_2SO_4 Consumed lb/ton	% Cu Calculated Head	Cu Recovery, %	H_2SO_4 Consumed lb/ton
Comp 1	0.99	85.9	95.0			
Comp 2	0.18	77.8	43.4	0.20	85.0	59.2
Comp 3	0.71	67.6	97.8			
Comp 4	0.43	81.4	88.4	0.38	76.3	65.6

Figure 13-1 Location of Core Holes for Metallurgical Samples



Relevant results from 2011-2013 were based on the recognition that surficial and structural decomposition of the granodiorite is a prominent feature in the deposit. These zones consists of friable decomposed granodiorite called gruss. Composite samples were prepared using core from drill holes EN-72, EN-82 and EN-84 (Table 13-4). Pictures of the 2011-2012 samples can be seen in Figure 13-2 below. Column testing at McClelland resulted in copper extractions of 58.5% to 75.0% with acid consumptions from 65.2 lb/ton to 90.8 lb/ton on (P100) 1-inch material. Testing on finer crush size of (P100) ½-inch resulted in copper extractions of 73.1 to 84.0% and acid consumptions of 75.4 to 101.0 lb/ton (Table 13-5).

Column leach tests at Metcon Research (Metcon) of Tucson, Arizona resulted in copper extractions of 68.1 to 79.2% and acid consumptions of 23.0 to 41.0 lb/ton on (P100) 1-inch size (Table 13-6) on GA, GB and MID samples, and a copper extraction of 57.3% to 57.4% on two different composites of DEEP material. The differences between the two laboratories are attributed to the management of sulfuric acid during leaching, with Metcon controlling free acid to prevent excessive dissolution of gangue minerals in the samples, i.e. biotite. However, the samples were cured before leaching. The test data indicated that higher the cure dosage, the higher the acid consumption. For example, acid consumption for sample GA was 83.0 lb/ton when cure dosage was 64.0 lb/ton and acid consumption was 40.6 lb/ton when cure dosage was 12.8 lb/ton. For sample DEEP, the acid consumption was 73.0 lb/ton when the cure dosage was 32.0 lb/ton; sample DEEP2 has an acid consumption of 34.1 lb/ton when the cure dosage was 10.0 lb/ton. These results indicate that cure may not be needed for the ore in order to minimize acid consumption.

Table 13-4 2011-2012 Metallurgical Sample Descriptions

Composite	Drill Holes	Sample Depth
GA	EN-72/EN-82/EN-84	10-95 feet
GB	EN-72/EN-82/EN-84	80-145 feet
MID	EN-72/EN-82/EN-84	185-435 feet
DEEP	EN-82/EN-84	680-745 feet
DEEP2	EN-9/EN-10/EN-12	710-820 feet
GA and GB=weathered oxide *MID and DEEP=un-weathered oxide*		

Table 13-5 Summary of Leach Test Results, McClelland 2012

Composite	Feed Size (P100)					
	1 inch			1/2 inch		
	% Cu Calculated Head	Cu Recovery, %	H2SO4 Consumed lb/ton	% Cu Calculated Head	Cu Recovery, %	H2SO4 Consumed lb/ton
GA	0.60	65.0	90.8	0.64	82.8	101.0
GB	0.16	75.0	89.6	0.16	75.0	95.0
MID	0.26	73.1	85.4	0.25	84.0	89.4
DEEP	0.53	58.5	65.2	0.52	73.1	75.4
Leach cycle: 108 days						

Table 13-6 Summary of Leach Test Results, Metcon 2012

Composite	Feed Size (P100)		
	1 Inch		
	% Cu Calculated Head	Cu Recovery, %	H2SO4 Consumed lb/ton
GA	0.54	73.0	41.0
GB	0.15	68.1	23.0
MID	0.27	79.2	23.8
DEEP	0.60	57.4	73.0
DEEP2	0.27	57.3	34.2
Leach cycle: 88 days on GA, GB, MID, 90 days on DEEP, 76 days on DEEP2 *Acid cure of 32.6 lb/ton on GA, 12.6 lb/ton on GB and MID, 32.0 lb/ton on DEEP, 10.0 lb/ton on DEEP2*			

The testing and analytical procedures from 2008-2012 are consistent with industry practices for assessing the amenability of a copper oxide deposit to heap leaching. The samples used in the tests are representative of the types and styles of mineralization within the deposit as a whole. Further testing is needed on composite samples representing specific production periods in the life of the Project. The reviewer's opinion is the samples and procedures in the metallurgical testing are representative and follow standard practice in the mining industry.

Copper extraction of 80% and acid consumption of 11 lb/ton were assumed in the 2010 PFS. Based on the data from recent test work, copper extractions of 70% for weathered material, 79% for un-weathered material, and 57% for deep material are assumed for a (P100) 1-inch crush size. Deep material was found to have little impact on overall extraction as the material occurs at or below the lowermost limits of the ultimate pit. Based on review of the column test work and neutralization potential data, acid consumption was calculated at 17.0 lb/ton of all ore types, and it was determined the ore requires no pre-curing with strong acid. Further work is needed to confirm the copper recovery, acid consumption and leaching characteristics. From the tests conducted, there are no processing factors or deleterious elements known that could have a significant effect on potential economic extraction.

Figure 13-2 2012 Metallurgical Samples

14. MINERAL RESOURCE ESTIMATES

This Report updates the Mineral Resource estimate in the 2012 RE by 3L Resources.

Zachary J. Black, E.I.T., Registered Member SME, Resource Geologist, HRC and Terre Lane, MMSA Qualified Person Member, Principal Mining Engineer are responsible for the estimation of the Mineral Resource herein. Mr. Black and Ms. Lane are Qualified Persons as defined by NI 43-101 and are independent of Enexco. Mr. Black and Ms. Lane were previously employed by Gustavson Associates and were contributors to previous resource estimates and the 2009 PFS, the 2010 PFS, and the 2012 RE.

HRC estimated the Mineral Resource for the Project from drill hole data, using controls from the main rock types and a single stage indicator approach to model the higher grade vein and lower grade disseminated styles of mineralization.

14.1 BLOCK MODEL PHYSICAL LIMITS

HRC created a three dimensional block model in Datamine. The un-rotated block model was created with individual block dimensions of 25 x 25 x 20 feet (XYZ). The block model extends from 868,600 east to 882,500 east, 28,804,000 north to 28,810,500 north, and 7,100 feet to 2,900 feet in elevation (model dimensions in NAD 83 Nevada State Plane 2701). The portion of each block lying below the surface topography was determined and utilized for tonnage calculations. All property and minerals within the block model extents fall entirely on Enexco's patented and unpatented claims.

14.2 DRILL HOLE SAMPLE STATISTICS

Statistics on grade-lithology relationships are calculated for the following rock types: quartzite, argillite, limestone/skarn, granodiorite, quartz filled fractures, and post-mineralized rocks (volcanics, silts and sands) as shown in Table 14-1 below. Cumulative frequency plots (CFP) for four major rock types encompassing the individual rock types were constructed and are shown in Figure 14-1.

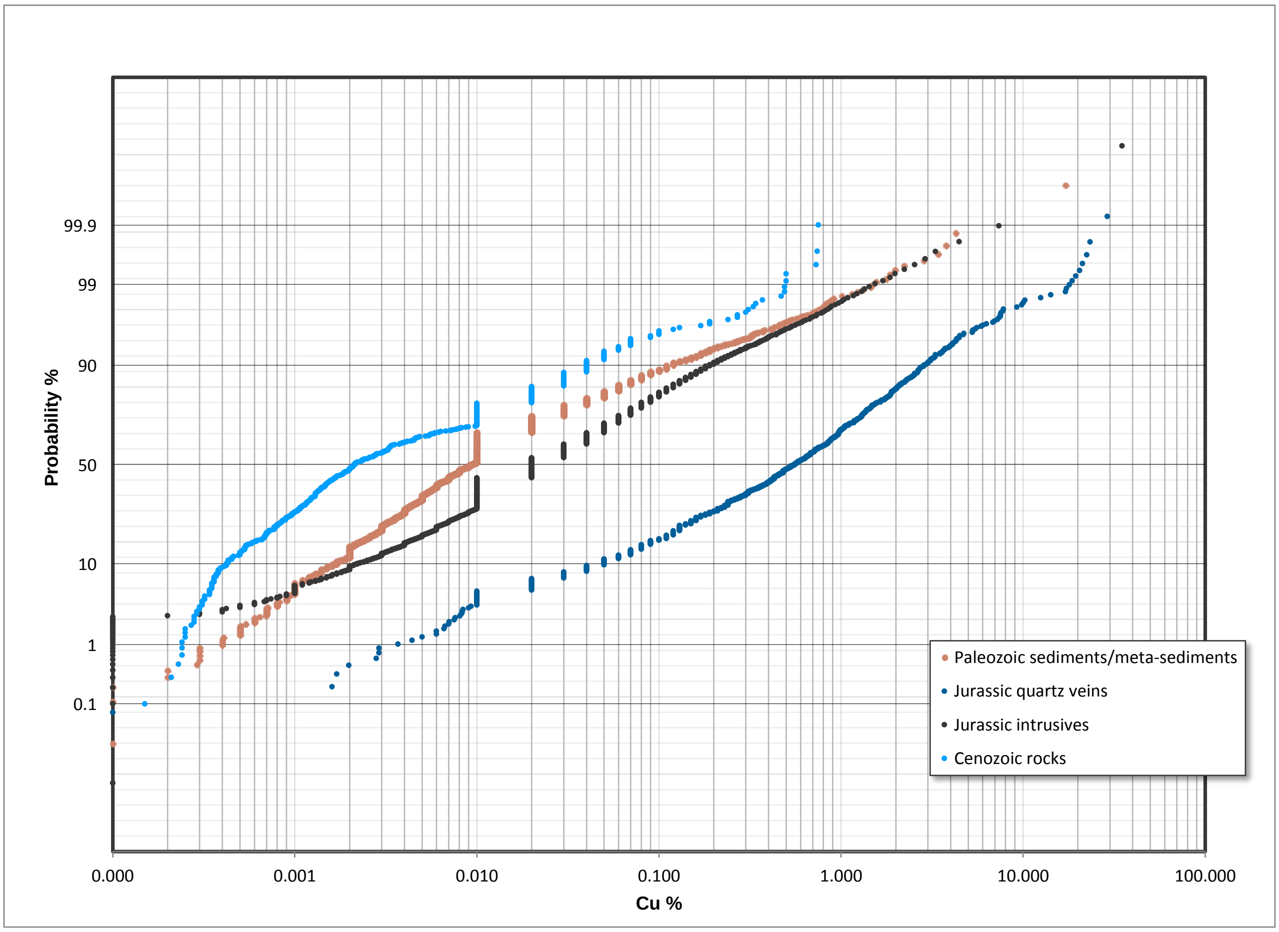
Table 14-1 Assay (Cu %) Statistics for Rock Type Data Groups

Lithology Grouping	Rock Type	# Samples	Min	Max	Mean	Median	Variance	Std. Dev.	Coef. Of Var.
Paleozoic sediments/ meta-sediments	Quartzite	161	0	1.500	0.053	0.010	0.028	0.166	3.115
	Argillite	387	0	0.770	0.023	0.010	0.005	0.070	3.012
	Limestone/skarn	662	0	4.220	0.100	0.010	0.122	0.349	3.494
	Undifferentiated	1,950	0	17.100	0.100	0.010	0.284	0.533	5.350
	All	*3,160*	*0*	*17.100*	*0.088*	*0.010*	*0.203*	*0.451*	*5.127*
Jurassic intrusive	Granodiorite	30,138	0	35.000	0.122	0.020	0.356	0.597	4.885
Jurassic quartz vein	Quartz filled fractures	736	0	29.000	1.351	0.560	7.811	2.795	2.069
Cenozoic rocks	Volcanics, silts, and sands	487	0	0.220	0.009	0.000	0.000	0.019	2.003
Total		***34,521***	***0***	***35.000***	***0.144***	***0.020***	***0.528***	***0.727***	***5.058***

The Jurassic quartz veins have a mean copper grade of 1.351% Cu whereas the Paleozoic sediments/meta-sediments and Jurassic intrusives have mean copper grades between 0.088 and 0.122% Cu. The Cenozoic rocks are post-mineralization and are mostly barren of copper values.

The average copper grade for the Jurassic quartz veins is more than 15 times greater than the other rock types. Coefficients of variation range from 2.069 for the Jurassic quartz veins to 5.127 for the Paleozoic sediments/meta-sediments, indicating the variation of grade is greatest in the Paleozoic sediments/meta-sediments and least in the Jurassic quartz veins. In CFPs, the Paleozoic sediments/meta-sediments and Jurassic intrusives have similar, overlapping distributions, indicating copper grades are distributed similarly within these units.

Figure 14-1 Lithology Grouping-CFP



14.3 GEOLOGIC MODELING

The three dimensional (3-D) geologic model for this study was constructed utilizing surface geologic mapping and drilling. Drill holes were logged geologically and intervals were flagged according to the Modeled Units listed in Table 14-2 below. Surface geology mapping was conducted from fall 2012 to spring 2013 and incorporated the same rock type units. The drill hole flags and surface mapping field observations were imported into Aranz Geo's Leapfrog Geo software and solids were generated which were then used to flag blocks by the centroid in the block model using the associated Block Code listed in Table 14-2.

The Paleozoic sediments/meta-sediments are classified into three units: Paleozoic quartzite 3 (PMq3), Paleozoic argillite 2 (PMa2), and Paleozoic undifferentiated sediments (PMs). PMa2 and PMq3 is described in Section 7.3.1. All older sediments are categorized into the group PMs.

Jurassic intrusives are subdivided according the chemical weathering they have undergone. Unweathered granodiorite (Jfi) was modeled through the use of drill hole data and cross sections. Small intercepts of Paleozoic sediments/meta-sediments in drill holes were ignored due to the brecciated nature of the contact discussed in section 7.3.1.2. Polylines were drawn in north-south and plan section to aid in interpreting the contact. Weathering of the granodiorite, referred to as gruss, was modeled due to its differing properties when compared to fresh granodiorite. Three gruss units were subsequently modeled: Cw, Cg2, and Cg4. The Cw unit is considered surficially weathered granodiorite. This unit was modeled as a surface related alteration that partially mimics the surface topography and varies in thickness from 10 to more than 100 feet. This surface was built using drill hole intercepts and north-south sections to manipulate the surface where drill hole data lacked. The Cg2 and Cg4 units are considered structural gruss units as observations have shown them to have similar attributes as the copper bearing structural trends that control mineralization. Because of this similarity, these units were modeled as interpolants using the same structural trend meshes built for the resource estimation, called anisotropic controls and discussed further in Section 14.3.1 below. This structural trend allows for dynamic search orientations throughout the model. The interpolant used data from rock quality description (RQD) measurements recorded in geologic logging of the core. RQD values less than 20 percent were used to build Cg2 and RQD values between 20 and 40 percent were used to build Cg4.

Cenozoic rocks were grouped together as Neogene volcanics, silts, and sands (Nvs). This unit was drilled extensively in the eastern end of the model area. Modeling was conducted using drill intercepts and surface geology mapping. The Nvs unit occurred post-mineralization, therefore, copper grade is purposefully excluded from this unit in the resource estimation.

Table 14-2 Rock Unit Coding

Lithology Grouping	Rock Type	Modeled Unit	Block Code
Paleozoic sediments/meta-sediments	Argillite	PMa2	32
	Quartzite	PMq3	33
	Limestone/skarn/undifferentiated	PMs	30
Jurassic intrusives	Granodiorite	Jfi	1
	Weathered gruss	Cw	20
	Structural gruss (<20)	Cg2	22
	Structural gruss (20-40)	Cg4	24
Cenozoic rocks	Volcanics, silts, and sands	Nvs	40

Figure 14-2 3-D Geologic Model

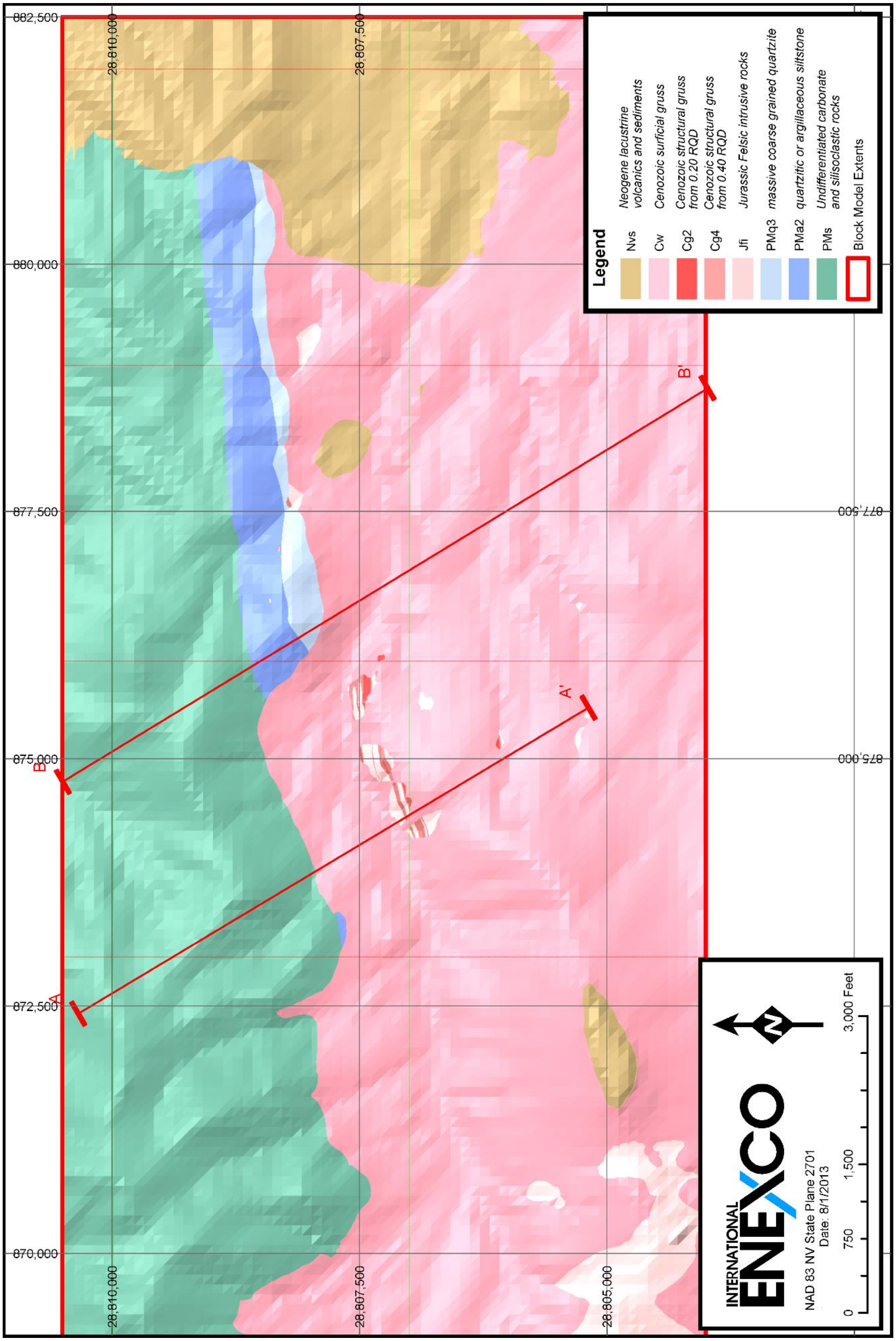


Figure 14-3 3-D Geologic Model Perspective A – A'

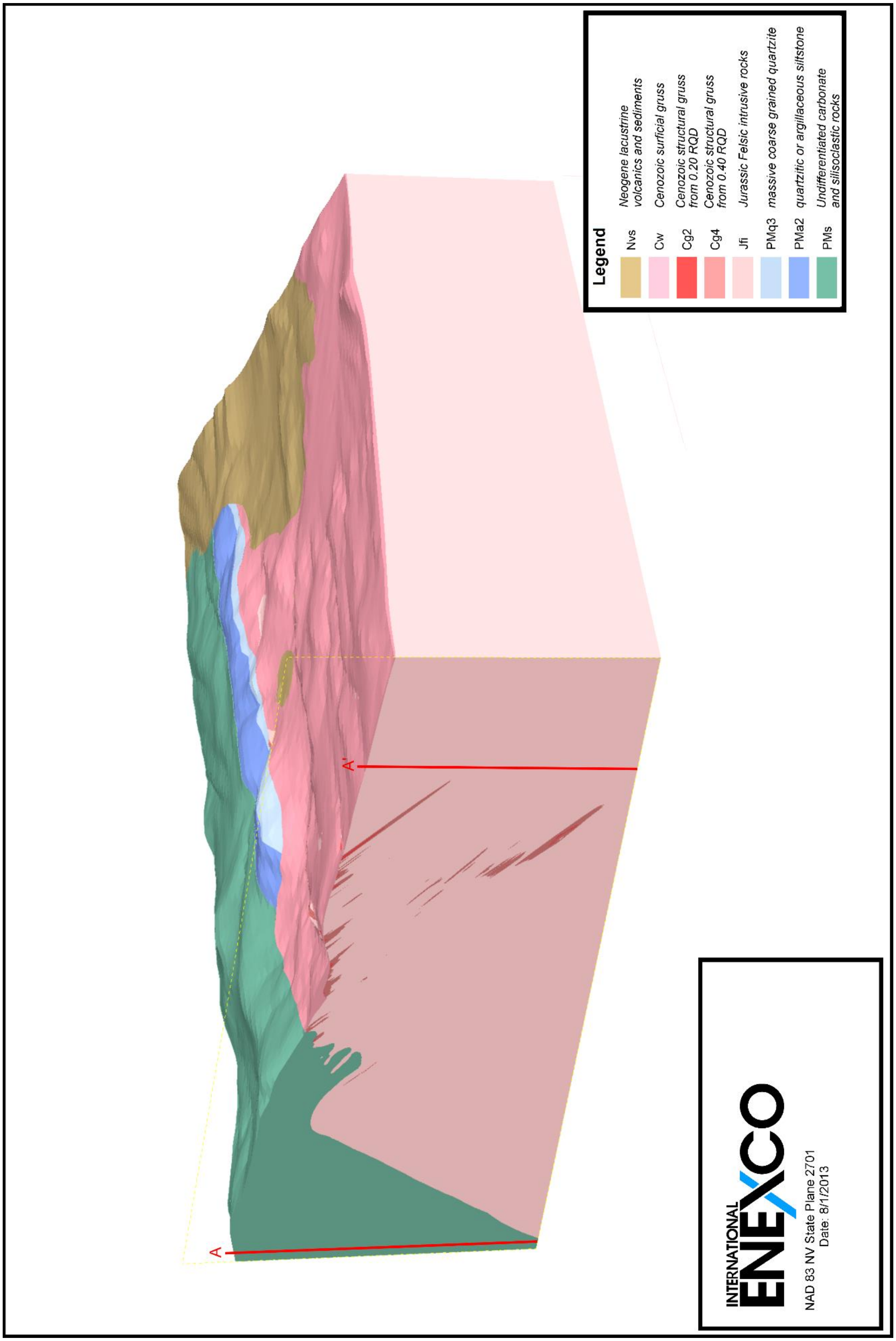
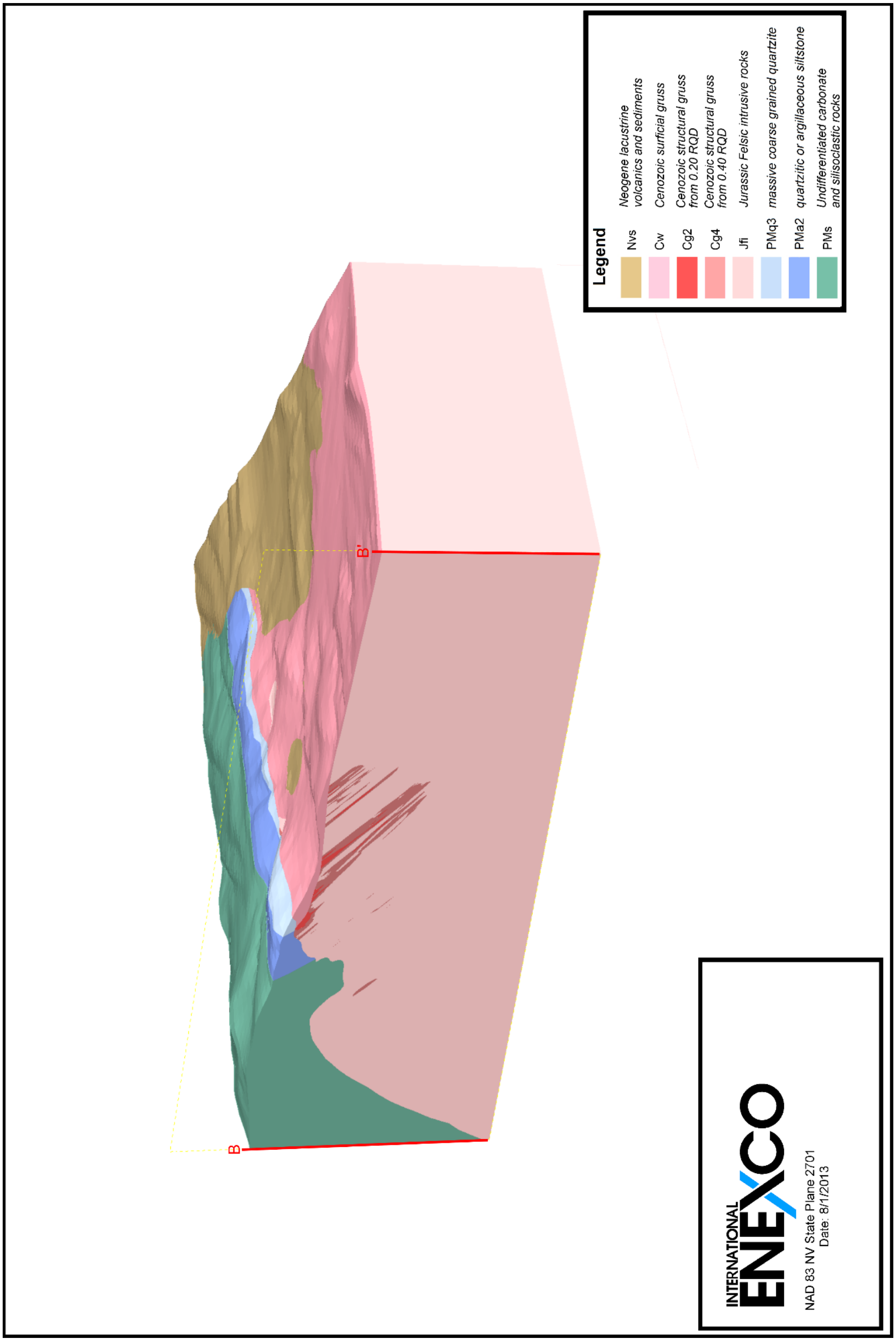


Figure 14-4 3-D Geologic Model Perspective B – B'



14.3.1 Modeling Jurassic Quartz Veins

Due to the narrow widths and high copper grades, the estimation method selected to model the Jurassic quartz veins is Dynamic Anisotropy. With this method, the orientation of the search ellipse changes on a block by block basis utilizing wireframes. In this model, five separate wireframes (called Anisotropic Control in Figure 14-7, Figure 14-8, and Figure 14-9) were created and utilized to model the structural fabric of the Jurassic quartz veins associated with the mineralization of the deposit. These wireframes were created based on surface geology maps adapted from Gibbons (1973) and from drill hole intercepts.

14.4 Combining Old and New Drill Hole Data

All drill hole data, including Enexco data and data from drilling by other companies, were included in the drill hole database. Validation of historical data is described in Section 12.1 of this Report. All assay data in the database was used for statistical analysis and variography of copper grades.

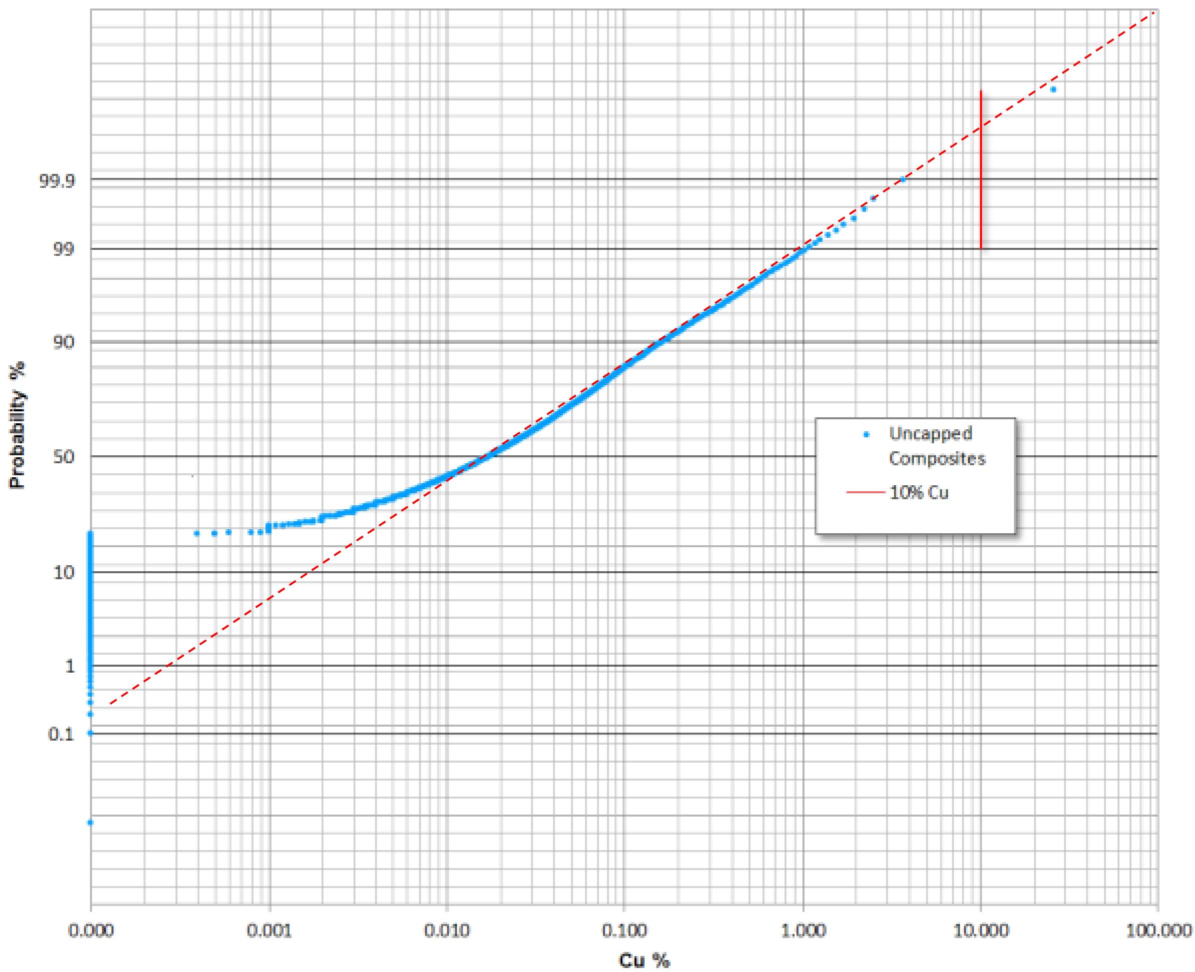
14.5 Compositing

HRC used down-hole compositing to standardize the input data set. Ten-foot down-hole composites were used in prior reports and are used for this resource update. Analysis of different composite lengths revealed larger composites decrease the detail and resolution of the mineralization in the model, and smaller composites (i.e. assay intervals) are too small for the size of blocks in the block model.

14.6 Capping Copper Grades

Grade capping is the practice of replacing any statistical outliers with a maximum value from the assumed sample distribution. This is done statistically to better understand the true mean of the sample population. The estimation of a highly skewed grade distribution can be sensitive to the presence of even a few extreme values. HRC utilized a log scale CFP of the composite assay data to identify the presence of any statistical outliers (Figure 14-5). From this plot, it was determined samples should be capped at 10% Cu at the break in the data of the CFP. The final dataset for grade estimation in the block model consists of 10-foot down-hole composites capped at 10% Cu.

Figure 14-5 10-Foot Composites-CFP



14.7 Bulk Density

Density tests were performed with core samples from 56 drill holes, including EN-35 through EN-83, EN-66B, EN-85, EN-87, EN-89, EN-91, EN-95, EN-167, and CON10-002. Density tests were also performed on 12 surface samples taken from the Paleozoic sediments/meta-sediments units. Table 14-3 below summarizes the basic statistics of the density data for these samples.

Table 14-3 Density Statistics by Lithology Grouping and Modeled Unit

Lithology Grouping	Modeled Unit	Mean (g/cm³)	Min (g/cm³)	Max (g/cm³)	Var.	St. Dev	No. of Samples
Paleozoic sediments/meta-sediments	PMa2	2.57	2.22	2.98	0.05	0.23	11
	PMq3	2.27	2.12	2.50	0.02	0.15	5
	PMs	2.85	1.78	3.62	0.05	0.23	257
Jurassic intrusives	Jfi	2.65	1.93	3.85	0.01	0.11	860
Cenozoic rocks	Nvs	1.53	1.44	1.68	0.02	0.13	3

The PMs unit has the highest density at 2.85 and the Nvs unit has the lowest density at 1.53. The mean density of the Jurassic intrusives is consistent with granitic rocks at 2.65. The resource model consists predominantly of Jurassic intrusives (see Table 14-4). The tonnage factors listed below were used to calculate tonnages for the Mineral Resource estimate based on the unit each block was coded by in the model.

Table 14-4 Density by Modeled Unit

Lithology Grouping	% of Resource	Density (g/cm^3)	Factor (tons/ft^3)
PMa2	2.4	2.57	0.080
PMq3	0.1	2.27	0.071
PMs	6.9	2.85	0.089
Jfi	90.6	2.65	0.083
Nvs	0	1.53	0.048

14.8 Variograms

A variography analysis was completed in the 2012 RE to establish spatial variability of copper values in the deposit. Variography establishes the appropriate contribution that any specific composite should have when estimating a block volume value within a model. This is performed by comparing the orientation and distance used in the estimation to the variability of other samples of similar relative direction and distance.

Variograms were created for horizontal and vertical orientations in increments of 30° horizontally and 15° vertically. Search ellipsoid axis orientations were based on the results of the analysis. The sill and nugget values were taken from the omnidirectional and down-hole variograms, respectively. Table 14-5 summarizes the variogram parameters used for the analysis. The resulting variograms were used to define the search ellipsoid responsible for the sample selection in the estimation of each block (Table 14-6). The ellipse orientations are rotated dynamically to better represent changes in the strike and dip of the veins. An example directional variogram from the study is shown in Figure 14-6. The composite grade data for Jurassic intrusives were also analyzed using indicator variograms at various cut-offs.

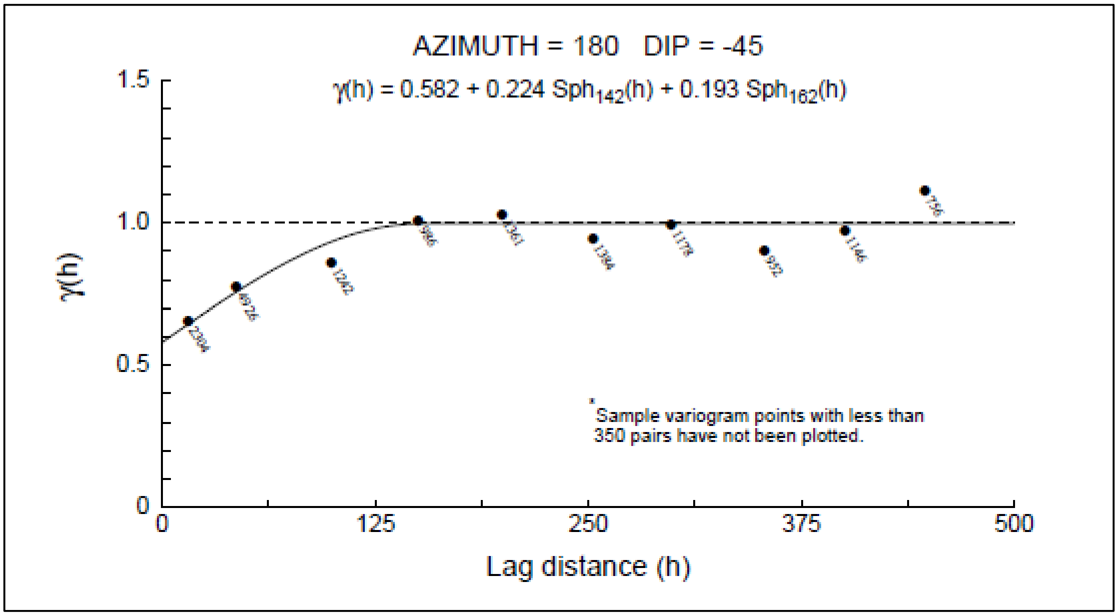
Table 14-5 Summary of Variogram Parameters

Nugget (C0)	C1	C2
0.582	0.224	0.193

Axis	Range (feet)	Azimuth	Dip
Z	24	165°	49°
X’	169	25°	34°
Y’	128	280°	21°

Modeling Criteria
Minimum number pairs required: 350
Max allowable drift on head and tail means: 5
Sample variogram points weighted by # pairs

Figure 14-6 Example Directional Variogram



In grade modeling, the variograms were used to establish search distances. Comparisons were made with ordinary kriging (OK) and inverse distance-squared (ID-2) methods. The ID-2 method was selected for reporting due to better fit with drill hole data throughout the model. The variogram parameters used for estimation are shown in Table 14-6 below. These parameters feature a major axis orientation striking 75 degrees and dipping 45 degrees to the southeast.

14.9 By-Product Metals

Within the Jurassic quartz veins, silver, molybdenum, and in a few samples, gold, occur at detectable levels. These metals were not modeled because they are not recovered in a heap leaching operation using sulfuric acid.

14.10 Grade Estimation

The Project's mineralization is characterized by high-grade copper bearing quartz filled fractures, or quartz veins, with a zone of relatively lower grade mineralization surrounding the veins, and low-grade copper disseminated between the veins. In conjunction with Dynamic Anisotropy (discussed in Section 14.3.1), HRC chose a single stage indicator approach to model the portion of high-grade vein material within each block, and then a separate grade estimate for the halo and inter-vein disseminated mineralization. This approach limits the extrapolation of high grade mineralization into the hanging wall and footwall units. Intervals with composite values greater than 1.0% Cu were designated as vein material for this estimate.

The copper grade was estimated from 10-foot down-hole composites using an ID-2 algorithm. Composites were assigned a 0 or 1 vein code, where a value of 1 means the composite is vein, 0 meaning it was not vein (wall rock). The estimate of vein percentage within each block was performed from the 0 and 1 vein codes. The grade of the vein portion of each block was estimated from the composites coded with a 1 (vein). The final estimate of the grade, the wall rock portion of the block, was estimated via ID-2 using only wall rock composites. The average grade of the block was then calculated by the weighted average of vein and wall rock components. All three lithologic units showed similar grade distributions, excluding the high grade

Jurassic quartz veins, so HRC chose to group all three units together and treat them as a single domain for modeling purposes.

14.11 Mineral Resource Classification

The Mineral Resource was placed into measured, indicated and inferred categories based on the modeling parameters listed below in Table 14-6. For a block to be included in the measured or indicated categories, it was required to be estimated from at least two different drill holes. The estimation variance was also used to place the Mineral Resource into categories. The CFP for the estimation variance was observed to have two clearly defined breaks at 0.58 and 0.85. Bench plans showed continuity between drill holes, thus, an estimation variance of less than 0.40 was determined for the measured classification. Blocks with an estimation variance between 0.40 and 0.85 were coded indicated. Blocks with a higher estimation variance were coded inferred.

Table 14-6 Modeling Parameters

Category	X Direction	Y Direction	Z Direction	Min. Samples per Estimate	Max. Samples per Estimate	Max. Samples per Drill Hole	Estimation Variance
Measured	170	130	25	4	9	2	>=0 and <0.4
Indicated	340	260	50	4	9	2	>=0.4 and <=0.85
Inferred	510	390	75	2	9	2	>0.85

14.12 Mineral Resource

A preliminary open pit optimization algorithim was run on the block model to constrain the resources and support the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") requirement the Mineral Resources have reasonable prospects for eventual economic extraction.

Table 14-7 shows the Mineral Resource for this Report. The Mineral Resource is pit-constrained and contained within a Lerchs-Grossman (LG) pit shell based on a copper price of $4.00 per pound. Operating costs used in generating the pit shell were preliminary estimates of $1.05 per ton for mining and $2.80 per ton for processing and G&A, and approximate those derived in Section 21. Other parameters were a copper recovery of 75% applied to all blocks and a 45-degree slope applied to all sectors in the pit. The pit search also gave value to inferred blocks based on copper grade for the purpose of determining a pit-constrained resource, although inferred blocks were given no value and treated as waste in the subsequent determination of the Mineral Reserve in Section 15. It is HRC's opinion that the material within the LG pit is compliant with NI 43-101 definitions for mineral resources and satisfies the recommended CIM expectation of "reasonable prospects for economic extraction". Modeled zones falling outside the LG pit may be economic under different extraction methods or conditions, but for the purposes of continuity with Section 15, the Mineral Resource for the Project is reported as constrained by the $4.00 per pound LG pit. The copper price of $4.00 per pound was selected for definition of mineral resources as the copper price under which the deposit has reasonable prospects for economic extraction.

The Mineral Resource in Table 14-7 is inclusive of the Mineral Reserve presented in Section 15. The reader is advised that mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of the Mineral Resource may be materially affected by social and economic factors, and environmental, permitting, and legal aspects, which are discussed in Sections 4 and 20 of this Report.

Table 14-7 Mineral Resource at 0.07% Cu Cut-off

Category	Cu %	Tons (000)	Pounds Cu (000)
Measured	0.21	75,473	313,968
Indicated	0.19	137,640	517,526
Total Measured + Indicated	**0.20**	**213,113**	**831,494**
Inferred	0.20	12,982	52,188

***Notes:**

(1) Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.
(2) Measured and Indicated Mineral Resources captured within the pit shell meet the test of reasonable prospect for economic extraction and can be declared a Mineral Resource.
(3) Inferred Mineral Resources are that part of the Mineral Resource for which the quantity and grade or quality are estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity.
(4) All resources are stated above a 0.05% Cu cut-off.
(5) Pit optimization is based on assumed copper price of US$4.00/lb.
(6) Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

The cut-off grade used for reporting is 0.07% Cu cut-off, based on the parameters in the determination of the Mineral Reserve in Section 16. At a 0.07% Cu cut-off grade, the measured category is 75.5 million tons at 0.21% Cu and the indicated category is 137.6 million tons at 0.19% Cu, for a total of 213.1 million tons at 0.20% Cu. At a 0.07% cut-off, the inferred category in the LG pit is 13.0 million tons at 0.20% Cu.

Within the Mineral Resource, Jurassic intrusive is the dominant rock type, accounting 89% of the Measured and Indicated Resources. With respect to surface weathering, gruss makes up 8% of the Measured and Indicated Resources.

14.13 Model Validation

The Mineral Resource estimate for the Project was performed using both OK and ID-2 estimation methods. To validate the resource model, cross sections similar to Figure 14-7, Figure 14-8, and Figure 14-9 for each of estimation method were examined visually and compared to the drill hole composite samples. Basic statistics (Table 14-8), CFPs, and swath plots (Figure 14-10, Figure 14-11, and Figure 14-12) were also used. Overall, there is good correlation between the grade models and the composite data, although deviations occur near the edges of the deposit and in areas where the density of drilling is less and material is classified as inferred resources. HRC's opinion is the grade estimations are valid for the purposes of this Report.

Figure 14-7 5,400 Level Section through Block Model

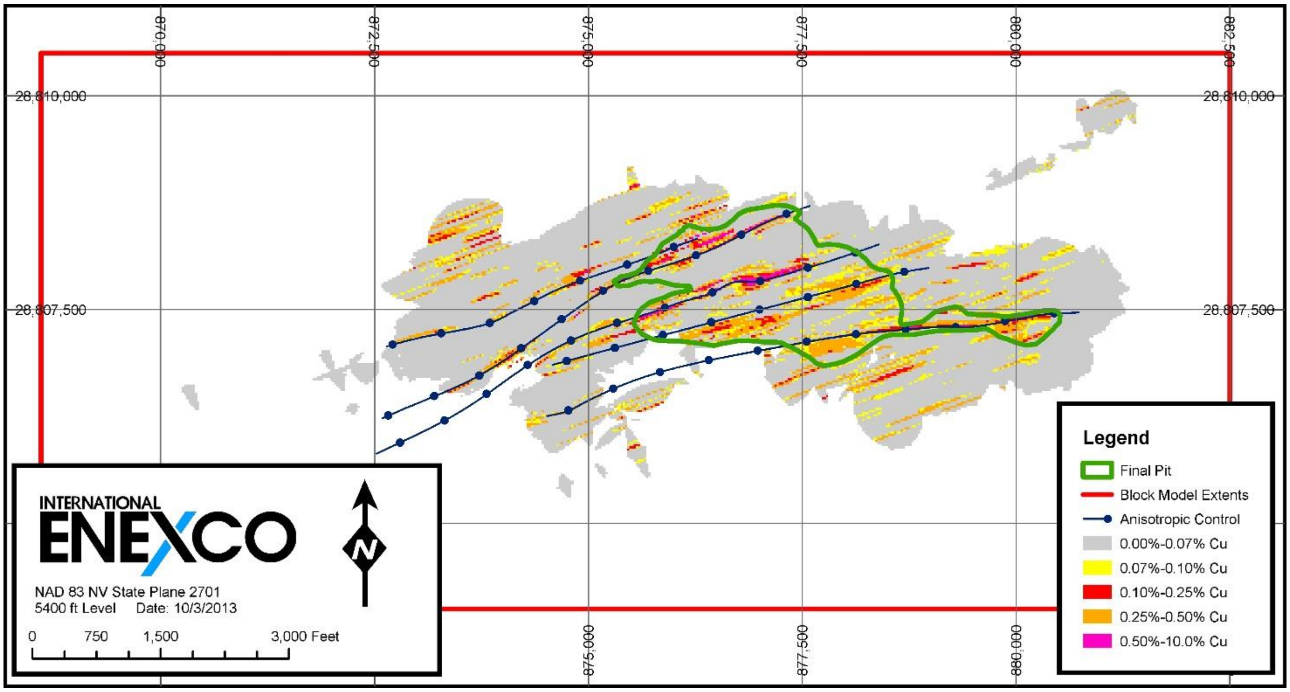


Figure 14-8 875,000 East, North-South Section through Block Model

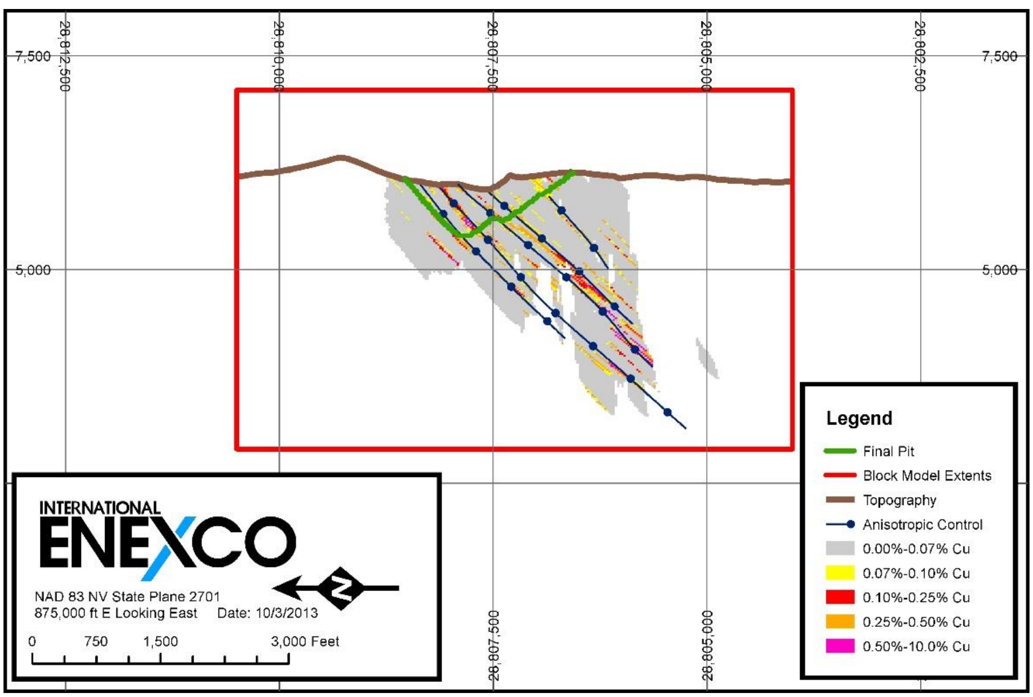


Figure 14-9 28,807,800 North, West-East Section through Block Model

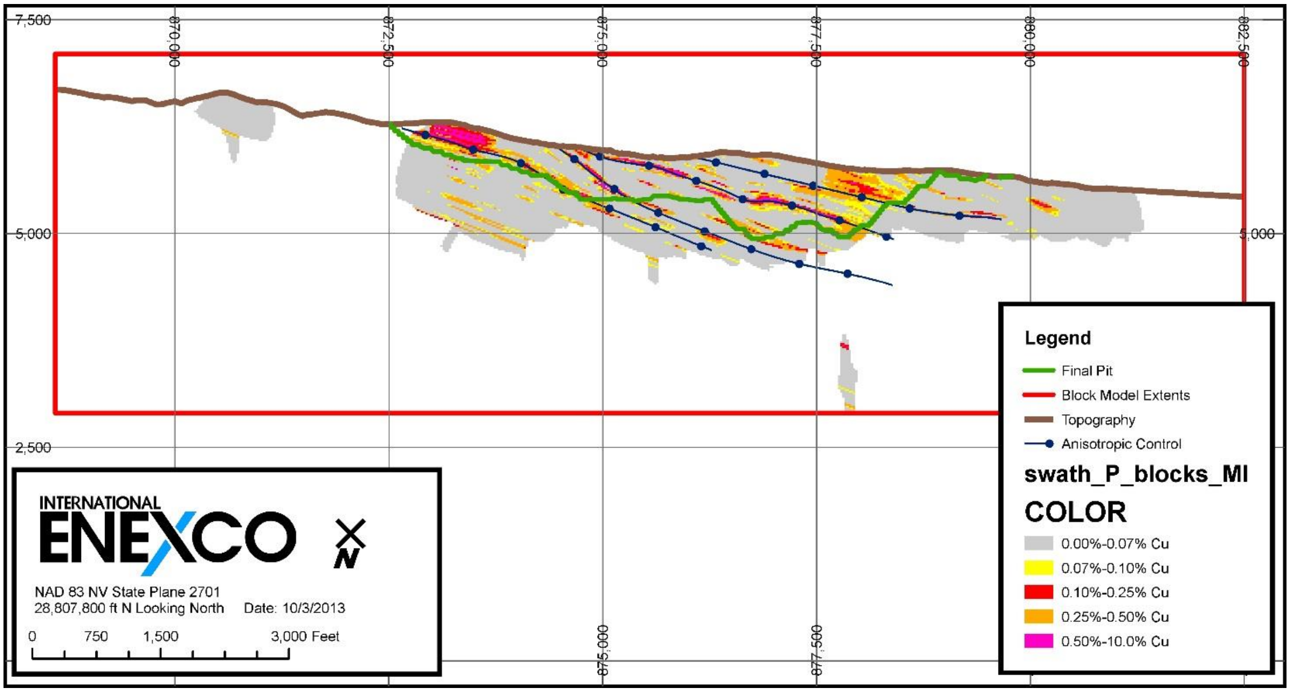


Table 14-8 Basic Statistics for Estimations and Composites

	ID-2	OK	Composites
Mean	0.044	0.045	0.080
Min	0.000	0.000	0.000
Max	8.981	8.775	9.992
Standard Deviation	0.123	0.115	0.262
Variance	0.015	0.013	0.069

Figure 14-10 East-West Swath Plot

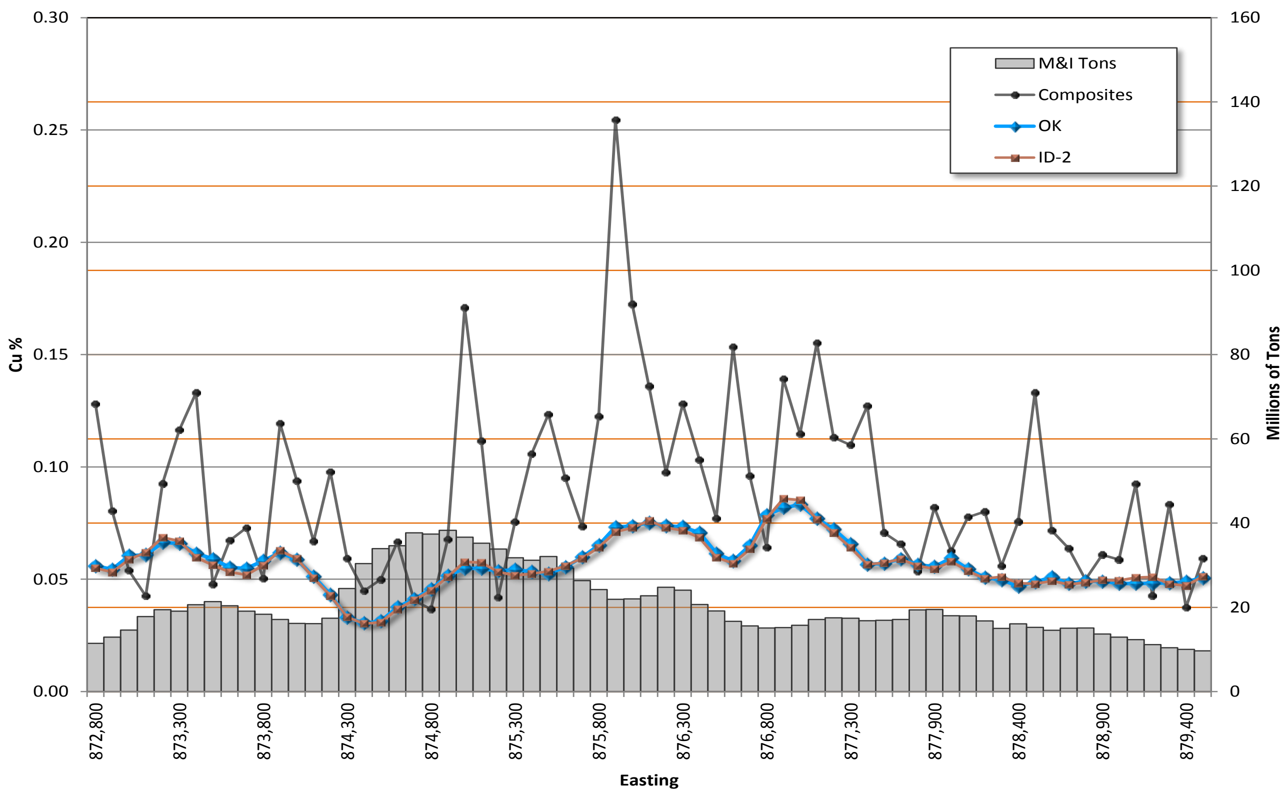


Figure 14-11 North-South Swath Plot

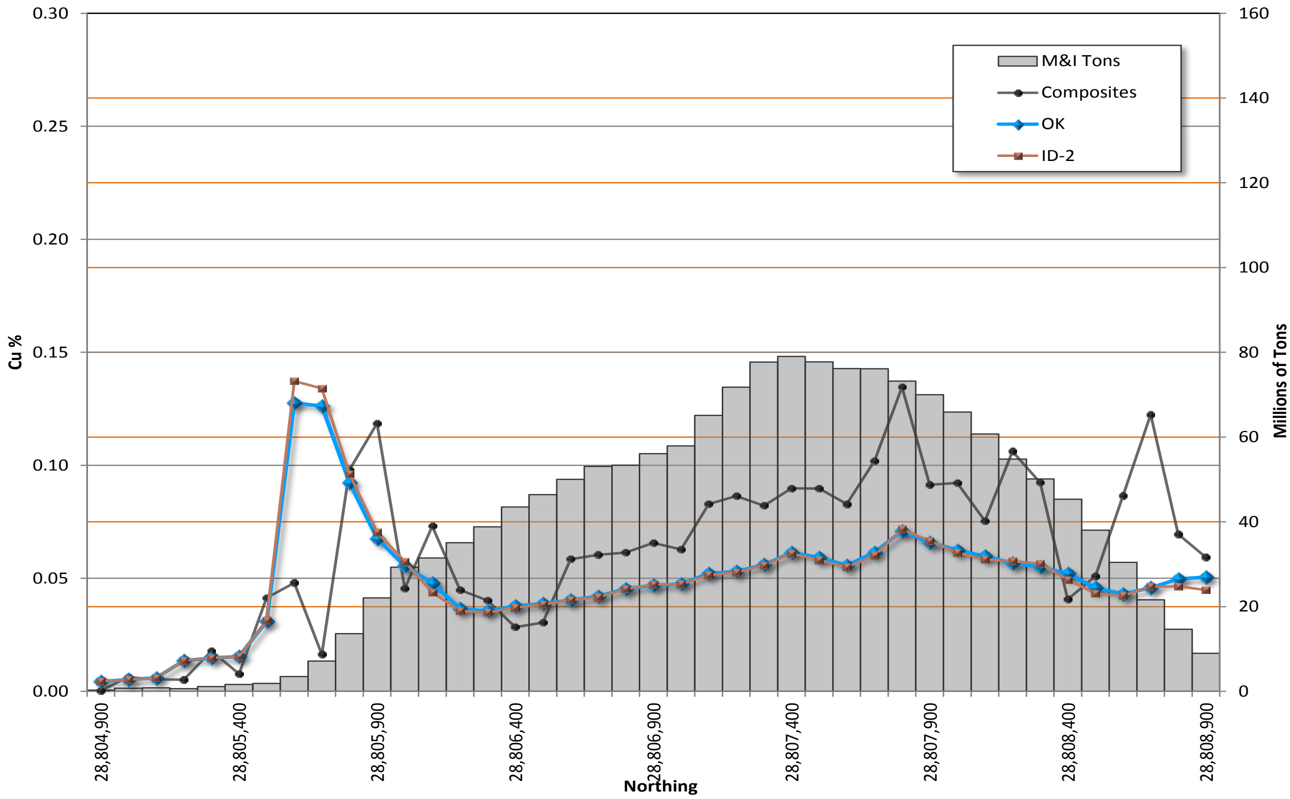
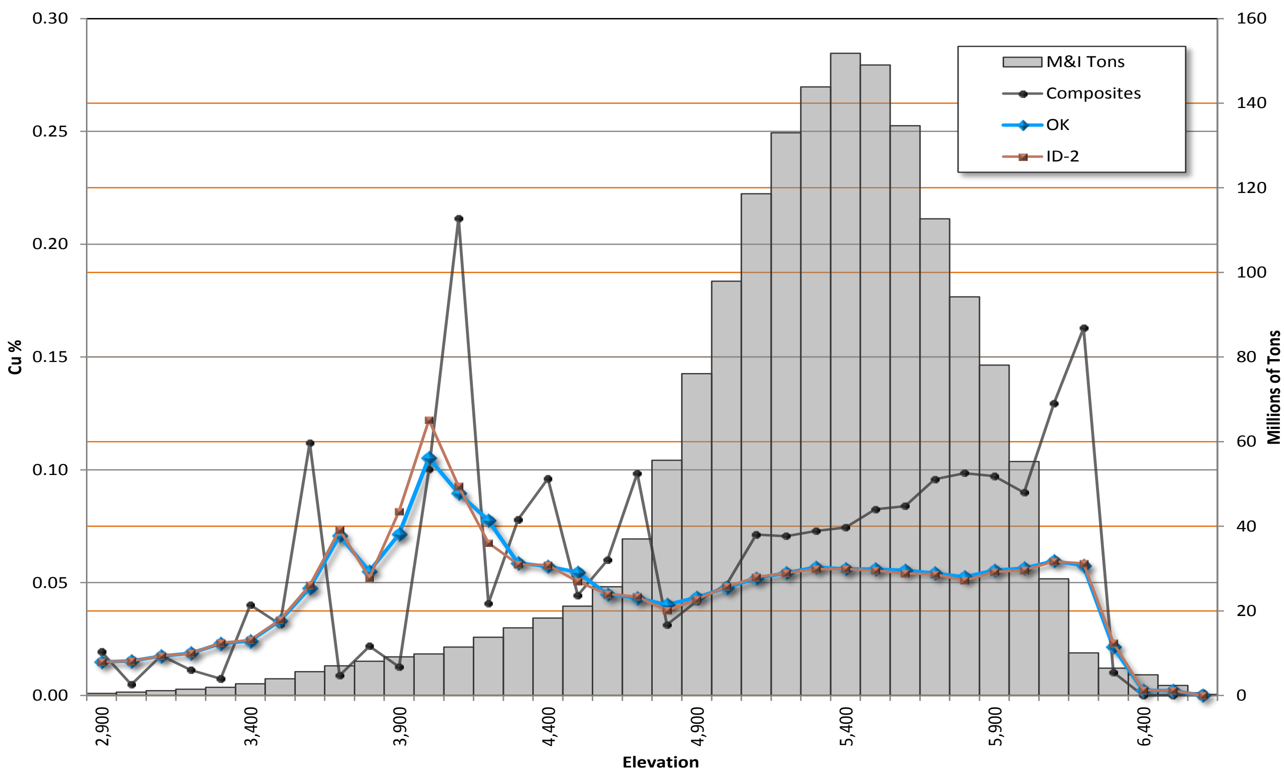


Figure 14-12 Level Swath Plot



15. MINERAL RESERVE ESTIMATES

The Mineral Reserve for the Project were determined by using Datamine's MaxiPit™ Lerchs Grossman pit optimizer to generate an ultimate pit shell.

Parameters for the design were $1.05 per ton of material moved for mining, $2.80 per ton for processing and general and administrative costs. Copper recoveries were assigned by block, with weathered and unweathered blocks assigned recoveries of 70 and 79 percent, respectively. Interramp slope angles were varied by sector in the pit, ranging from 40 to 52 degrees based upon results of the geotechnical study described in Section 16. Fluctuating copper prices were used in the optimization to generate pits of varying tonnage and grade. The ultimate pit shell was selected from the best observed combination of revenue and copper production, which occurred with a pit shell having an equivalent cut-off grade of 0.07% copper. Only Measured or Indicated blocks in the resource model were used to generate positive economic values in the optimization. Inferred blocks were given zero copper values and treated as waste.

The resulting ultimate pit shell was used for the final pit design. Haul roads adequate for 150-ton haul trucks were added, using a road width of 95 feet and 10 percent maximum slope angle. The mineable material for the ultimate pit is shown by reserve category and phase below in Table 15-1.

Table 15-1 Mineral Reserve by Pit Phase and Category Reported at a 0.07% Cu Cut-off

Phase	Category	Cu %	Tons (000)	Pounds Cu (000)
1	Proven	0.30	12,472	73,795
	Probable	0.28	11,018	61,472
	Proven + Probable	**0.29**	**23,490**	**135,267**
2	Proven	0.20	8,402	33,259
	Probable	0.21	12,049	51,464
	Proven + Probable	**0.21**	**20,451**	**84,723**
3	Proven	0.21	36,804	156,195
	Probable	0.20	60,349	235,563
	Proven + Probable	**0.20**	**97,153**	**391,758**
Total	Proven	0.23	57,678	263,249
	Probable	0.21	83,416	348,499
	Proven + Probable	**0.22**	**141,094**	**611,748**
Only measured and indicated resource categories have a sufficient level of confidence to be classified as Proven and Probable reserves. No Inferred Resources are included in the Mineral Reserves or are used in the economic analysis.				

A production schedule was then developed by sequencing the ultimate pit into three phases and allocating tonnages over time by bench within each phase. The pit phases vary in ore tons, copper grade and stripping ratio. Phase 1 is the highest grade and lowest stripping ratio, and mines outcropping oxide material at the west end of the ultimate pit. Phase 2 expands the pit east, with a lower grade and a higher strip ratio to access deeper ore in Phase 3. The Phase 1 through Phase 3 pits are shown in Figure 15-1, Figure 15-2, and Figure 15-3. The annual production schedule is shown in Table 16-4.

Figure 15-1 Phase 1 Pit Design

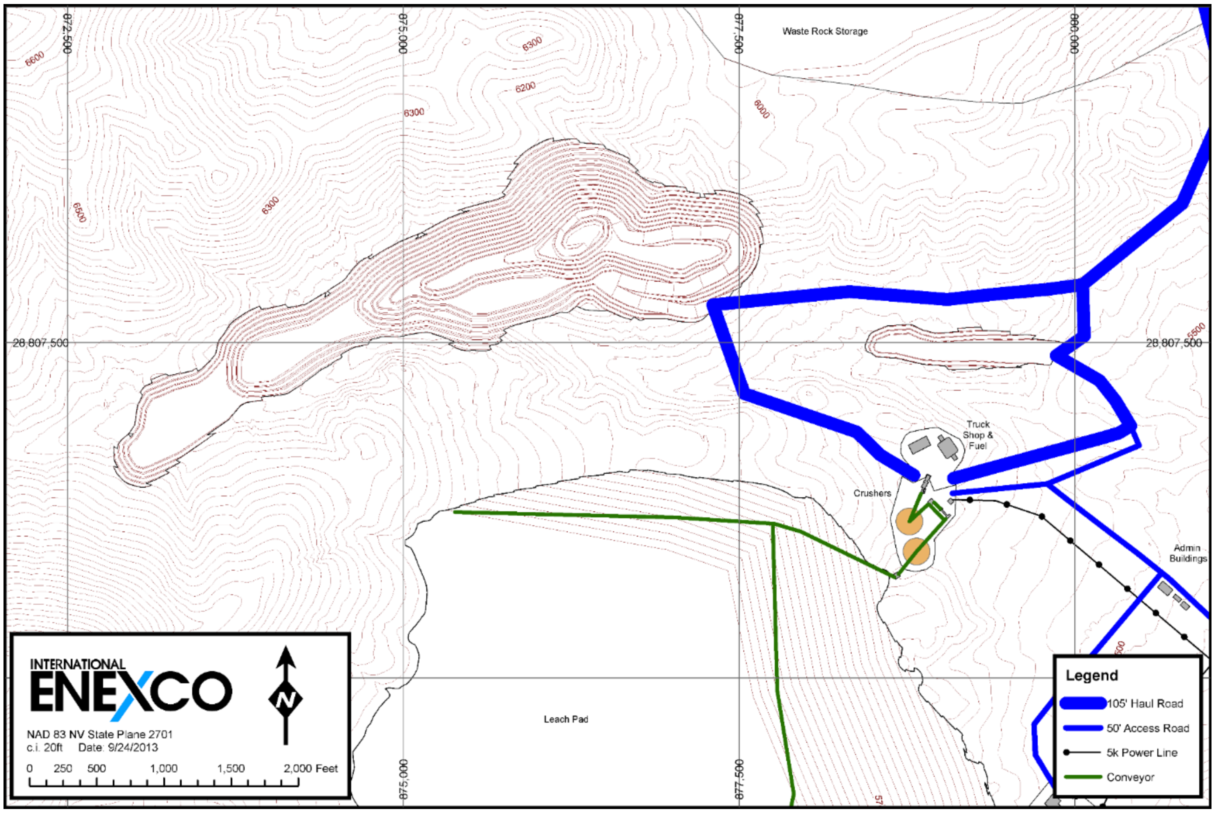


Figure 15-2 Phase 2 Pit Design

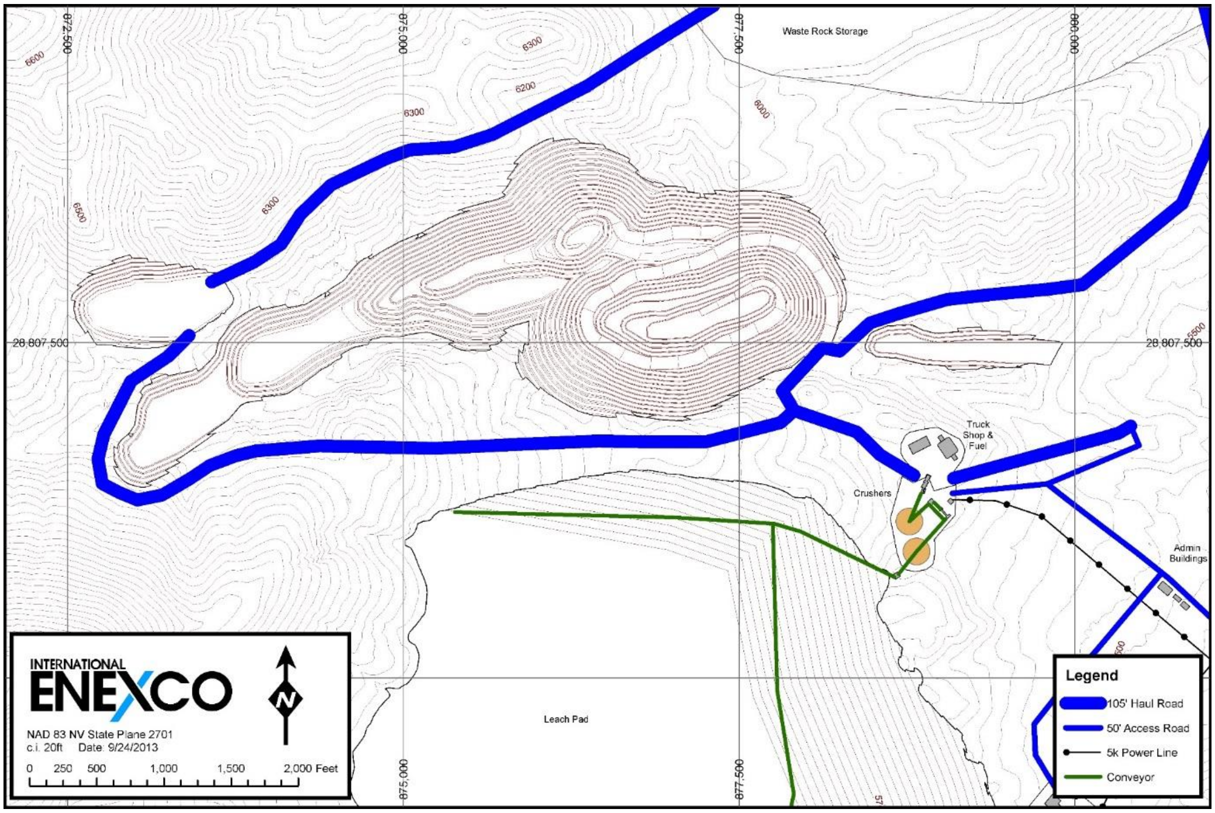
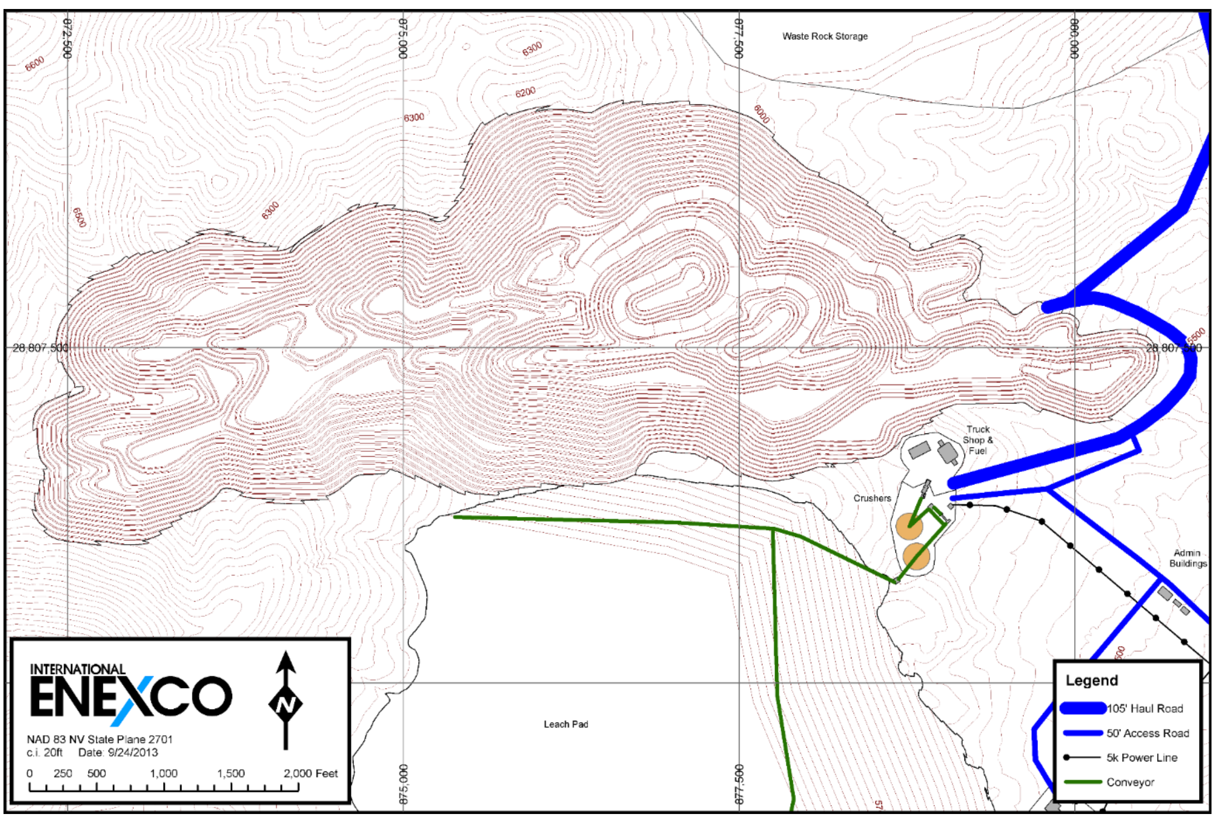


Figure 15-3 Phase 3 (Ultimate) Pit Design



16. MINING METHODS

Ore production will be by conventional open pit mining methods. The primary mining equipment selected consists of rotary blast hole drills, 19.5 cubic yard hydraulic shovels, and 150 ton haul trucks, supported by track dozers and other ancillary equipment. Drilling will be in 20-foot high benches, double benched with 25-foot wide catch benches on final slopes. Blasting will utilize ammonium nitrate-fuel oil (ANFO) supplemented with emulsion for explosives as most blast holes are expected to be dry. The ore will be hauled by trucks to the crushing area east of the pit, and then conveyed to the heap leach pad south of the pit. Material below the cut-off grade will be hauled by trucks to the waste rock storage facility north of the pit.

The open pit is designed and mined in three phases in order to maximize the grade during the initial years and to balance the required waste stripping over time. Mining in Phase 1 begins at the west end of the ultimate pit, between bench elevations 5,400 feet and 6,200 feet ASL, and progresses eastward to a final pit depth at elevation 4,940 feet ASL. When fully mined, the ultimate pit as planned extends 8,300 feet in length from west to east, and 2,800 feet in width from north to south.

Ore production is planned to vary up to 57,000 tons per day (18.9 million tons per year) in order to maintain a consistent copper production of 50 million pounds per year from the SX-EW plant. Total mine production of ore and waste will reach 226,000 tons per day (83 million tons) in year five. Mining will be on a seven day per week schedule, with two shifts per day.

16.1 PRE-PRODUCTION DEVELOPMENT

The pre-production requirements at the Project are minimal with mineable ore occurring near the surface. Access to the Phase 1 pit is over gentle terrain which will make the construction of initial haul roads inexpensive. An allowance of $1 million has been included in the initial capital cost to cover the initial haul road construction and any clearing or grubbing.

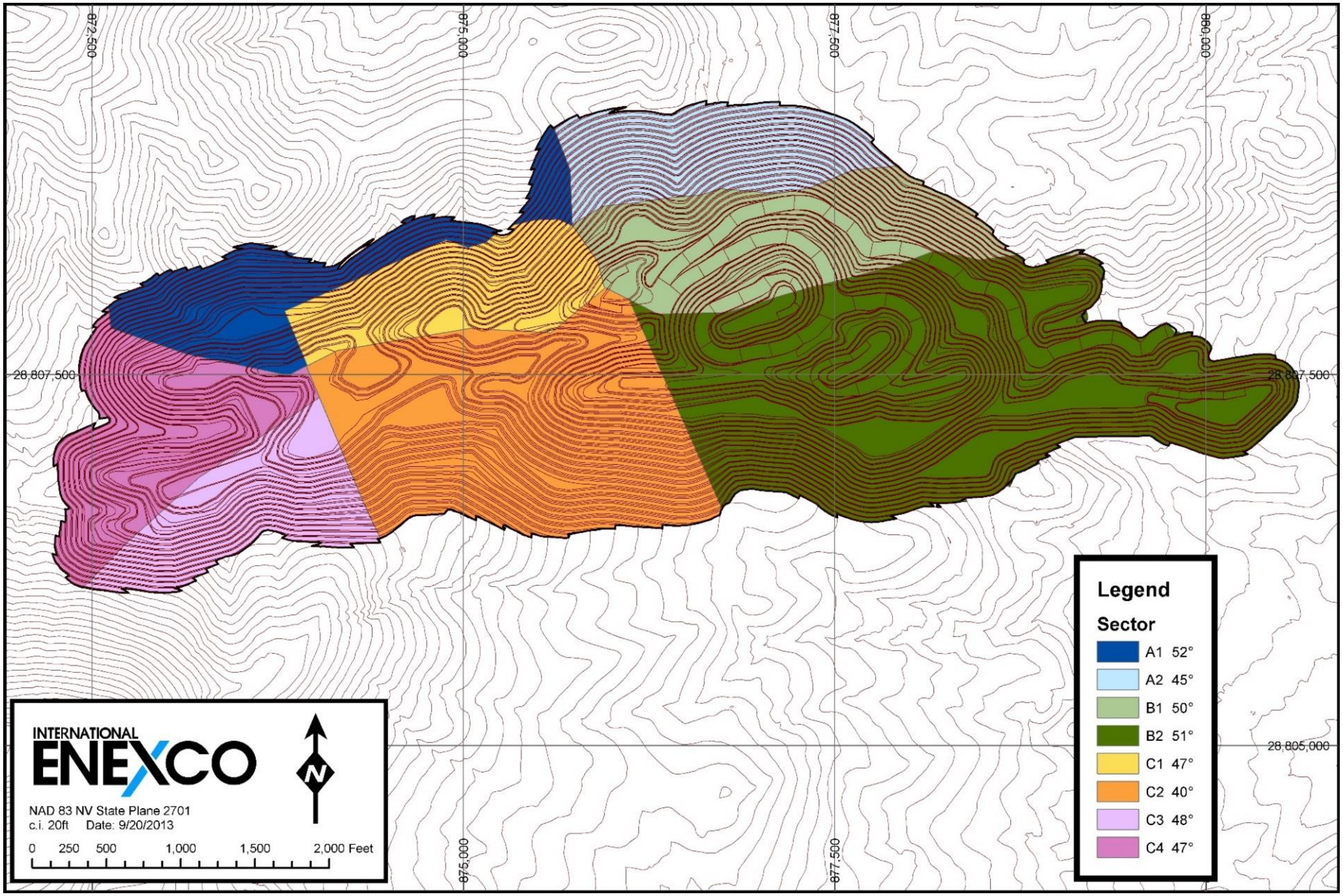
16.2 OPEN PIT MINE DESIGN

A geotechnical study to determine slope angles for the pit design was conducted in 2011 by The MINES Group, Inc. (The MINES Group, Inc.; Myers, Kenneth, 2011). The study used cell mapping of surface outcrops and trenches to record fractures throughout the mine area as well as laboratory testing of representative samples. From the cell data, three structural domains (A, B, and C) were identified based on rock type and location, and then subdivided into design sectors. Slope angles were determined from failure analysis of the fracture patterns and physical characteristics of the units. Table 16-1 below summarizes the recommended pit slopes from the study and Figure 16-1 shows the locations of the design sectors in relation to the ultimate pit design.

Table 16-1 Recommended Pit Slope Angles

Modeled Unit	Structural Domain	Design Sector	Pit Slope Angle (°)	Face Angle (°)
PMs	A	A1	52	78.6
		A2	45	66.5
PMa2, PMq3	B	B1	50	75.5
Jfi, Cw, Cg2, Cg4	B	B2	51	76.2
Jfi, Cw, Cg2, Cg4	C	C1	47	70.3
		C2	40	58.1
		C3	48	71
		C4	47	70.7

Figure 16-1 Pit Slope Design Sector Locations

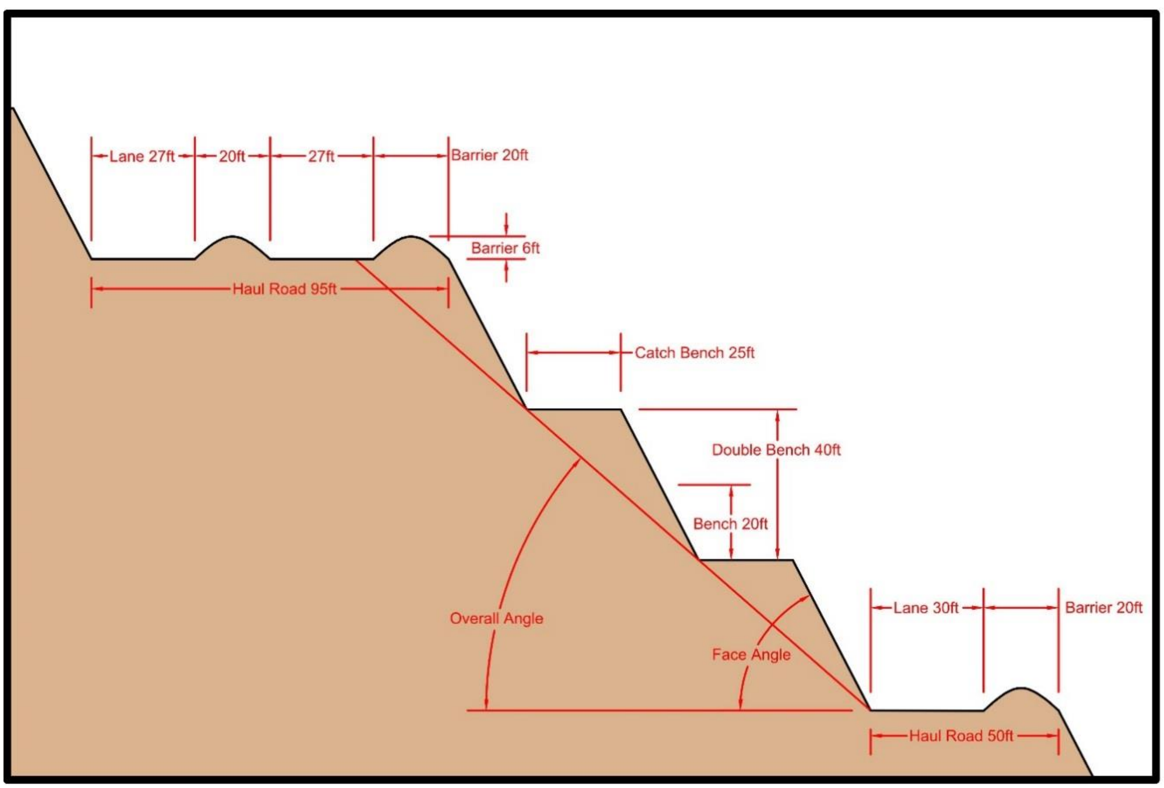


The optimized pit shells described in Section 15 were used as a basis for the three phases of the pit designs. A 20-foot bench height was selected to maximize selectivity of ore and waste, thereby minimizing dilution while still maintaining productivity in mining. All three phases of pit design utilize double benching resulting in vertical benches of 40 feet and bench widths of 25 feet. Surface and in-pit access roads are designed with a 95-foot width, which provides a ramp width to truck width ratio of 4.8:1 and safely allows 2-way truck haulage and berms. Maximum grade of the haul roads is 10%, except for the lowermost few benches where the grade is increased to 14% and the ramp width is narrowed to 50 feet to minimize excessive waste stripping. The pit design criteria are presented in Table 16-2 and depicted in Figure 16-2 Pit Design Elements.

Table 16-2 Pit Design Criteria

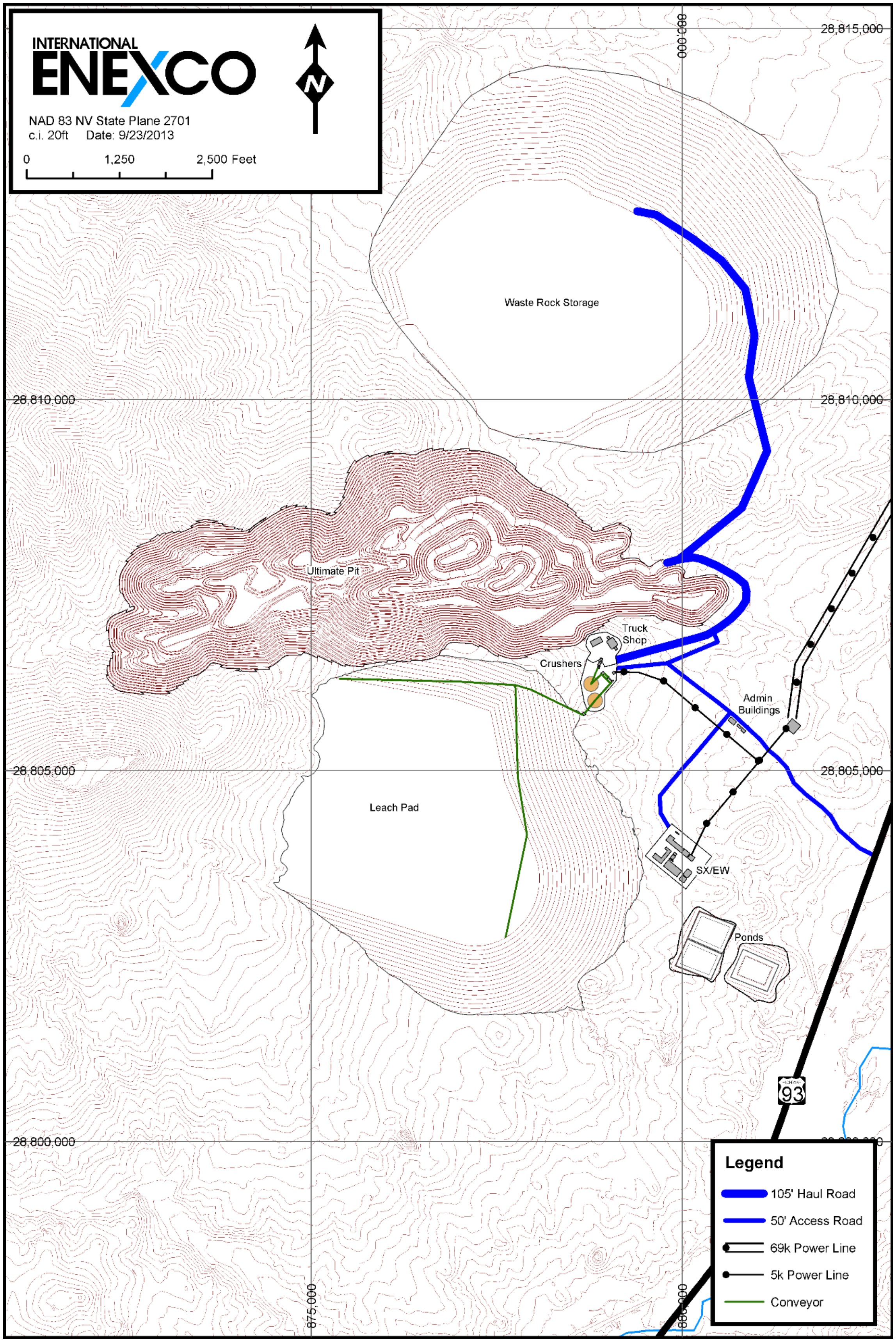
Pit Design Criteria	Parameter
Overall Slope Angles	See Table 16-1
Face Angles	See Table 16-1
Catch Bench Width	25 ft
Double Bench Height	40 ft
Minimum Turning Radius	40 ft
Haul Road Width	95 ft
Road Grade	10%
Haul Road Width Pit Bottom	50 ft
Road Grade Pit Bottom	14%

Figure 16-2 Pit Design Elements



The waste rock storage area is located north of the pit as shown in Figure 16-3, and is designed to contain 325 million tons. Testing has indicated the material is non-acid generating. The stockpile will be constructed with 3H:1V overall side slopes enabling it to remain stacked while requiring no re-grading or soil cover upon closure.

Figure 16-3 General Facilities Arrangement



16.3 PRODUCTION SCHEDULE

The mine production schedule is based on a seven day per week schedule with two 12 hour shifts per day. There are four crews planned to cover the rotating schedule. Each 12 hour shift includes 30 minutes down for blasting and miscellaneous delays, 30 minutes for shift start up and shutdown, and one hour for lunch and breaks for a total of 10 effective working hours. Table below shows typical yearly schedule parameters and hours scheduled.

Table 16-3 Mine Schedule Parameters

Mine Schedule	
Crews	4
Shifts/day	2
Hours/shift	12
Lunch, Breaks, etc. (hours)	1
Blasting, Misc. (minutes)	30
Startup & Shutdown (minutes)	30
Days/Year	365
Scheduled Hours/Year	8,760

The mine plan is developed to provide approximately 50 million pounds of recoverable copper to the leach pad each year. The mine production schedule has a 10-year mine life based on a 0.07% Cu cut-off grade as shown below in Table 16-4. Initial mine production is 36 million tons per year (nominal rate of 100,000 tpd), increasing to peak of 80 million tons per year (225,000 tpd) by Year 5 when the waste to ore ratio increases 3.6 to 1 in the Phase 2 pit. Phase 2 requires stripping Delano Hill to access deeper, higher-grade ore in Phase 3. The average life of mine strip ratio is 2.3:1 (waste:ore).

Table 16-4 Annual Mine Production Schedule

Year	Ore Tons (000)	Cu %	Waste Tons (000)	Total Tons (000)	Strip Ratio	Total Cu Pounds (000)	Recovered Cu Pounds (000)
Year 1	11,401	0.25	25,225	36,626	2.2	56,577	42,772
Year 2	10,627	0.31	17,757	28,384	1.7	65,292	49,360
Year 3	15,377	0.21	33,399	48,776	2.2	64,347	48,647
Year 4	14,781	0.23	37,727	52,507	2.6	68,198	51,558
Year 5	17,895	0.18	64,367	82,262	3.6	65,005	49,143
Year 6	18,953	0.18	63,472	82,424	3.3	67,010	50,659
Year 7	17,206	0.20	34,011	51,217	2.0	67,110	50,735
Year 8	16,397	0.20	22,793	39,190	1.4	67,110	50,735
Year 9	14,038	0.24	24,028	38,066	1.7	67,294	50,874
Year 10	4,420	0.27	5,140	9,560	1.2	23,806	17,998
Life-of-Mine	141,094	0.22	327,919	469,013	2.3	611,748	462,481

The amount of equipment required to meet the scheduled tonnages is calculated based on the mine schedule, equipment availabilities, usages, and haul and loading times for the equipment. Equipment mechanical physical availabilities start at 94% for the trucks, drills, and loading units. For each year of production, the mechanical physical availabilities decrease by one percent. The use of availability for all of the equipment is calculated at 83% based on the breaks and down time in the schedule parameters. An additional 85% efficiency factor is applied to all of the equipment for calculating the total units of equipment required. Table below lists the annual equipment availability parameters.

Table 16-5 Annual Equipment Availabilities

Equipment Availabilities	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Physical Availability	94%	93%	92%	91%	90%	89%	88%	87%	86%	85%
Use of Availability	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%
Efficiency	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%

16.4 Drill and Blast Parameters

The design parameters used to define drill and blast requirements are based on a 6.75 inch blast hole on a 14-foot by 16-foot pattern in the ore zones and a 15-foot by 17-foot pattern in the waste zones. Benches will be blasted and mined on 20-foot levels with three feet of sub-drill. Buffer rows are planned to allow for controlled blasting and to minimize damage to the highwalls. The number of blast holes and blast hole drills required each month or year is calculated based on the parameters shown in Table 16-6 and are also used in calculating the operating costs. The initial mine production requires four rotary production drills and three additional drills are purchased as the strip ratio increases.

Table 16-6 Drill and Blast Parameters

DRILLING & BLASTING PARAMETERS	Units	Production Pattern		Wall Control Pattern	
		Ore Rock	*Waste Rock*	*Buffer*	*Buffer*
Tonnage Factor	t/ft³	0.080	0.080	0.080	0.080
Blast Pattern Details					
Bench Height	ft	20.00	20.00	20.00	20.00
Sub Drill	ft	3.00	3.00	3.00	0.00
Diameter of Hole	in	6.75	6.75	6.75	6.75
Staggered Pattern Spacing	ft	14.00	15.00	12.00	10.00
Staggered Pattern Burden	ft	16.00	17.00	14.00	12.00
Drill Equivalent Square Pattern	ft	15.00	16.00	13.00	11.00
Hole Depth	ft	23.00	23.00	23.00	20.00
Height of Stemming or Unloaded Length	ft	12.00	12.00	15.00	15.00
Material Quantity					
Volume Blasted/Hole	ft³	4,500	5,120	3,380	2,420
Tons Blasted/Hole	tons	360	410	270	194
Powder Factor					
Percent Emulsion		5%	5%	5%	5%
Percent ANFO		95%	95%	95%	95%
Density of Powder	lb/ft³	54.16	54.16	54.16	54.16
Loading Density	lb/ft	13.46	13.46	13.46	13.46
Powder/hole	lb/hole	148.04	148.04	107.67	67.29
Powder Factor	lb/t	0.41	0.36	0.40	0.35
Powder Factor	lb/ft³	0.03	0.03	0.03	0.03
Drill Productivities					
Penetration Rate	ft/hr	165.00	165.00	165.00	165.00
Penetration Rate	ft/min	2.75	2.75	2.75	2.75
Cycle Time Estimate					
Drilling Time	min	8.36	8.36	8.36	7.27
Steel Handling Time	min	0.00	0.00	0.00	0.00
Set up Time	min	1.75	1.75	1.75	1.75
Add Steel	min	0.00	0.00	0.00	0.00
Pull Rods	min	0.50	0.50	0.50	0.50
Total	min	*10.61*	*10.61*	*10.61*	*9.52*
Drilling Factors for Wall Control					
Buffer Holes - 2 Rows					
Wall Control Drill Holes Required	***Perimeter Blast***				
Buffer Holes - 2 Rows	holes/ft		0.17		
Material to Remove from Production Blast	t/ft		48.00		

16.5 Load and Haul Parameters

The design parameters used to define the loading and hauling requirements are shown in Table 16-7 below. The main loading units will be two 19.5 yd^3 front shovels with a 17 yd^3 front end loader as a backup unit. The shovels were chosen over front end loaders as the main loading unit due to their higher loading rate versus the loaders. This will be advantageous given the short cycle times of the trucks. The 150 ton haul trucks are the main hauling unit. The shovel is calculated to require five passes to load the trucks and the loader will require six passes. The 150 ton trucks were also evaluated in the schedule, however, the 150 ton trucks were chosen because they were found to be more cost effective than the 100 ton trucks. Haulage profiles for the ore and waste material from each pit phase were generated and used to calculate the truck cycle times which were used in the equipment requirement calculations.

Table 16-7 Load and Haul Parameters

Parameter	Unit	WA 900 Loader HD1500	PC 3000 HD1500
Bucket Capacity (heaped)	yd3	17.00	19.50
Bank Material Weight Dry	tons/bcy dry	2.24	2.24
Bank Material Weight Wet	tons/bcy wet	2.31	2.31
Bulk Factor (Swell Factor)		1.35	1.35
Loose Material Weight Dry	t/lcy dry	1.70	1.70
% Moisture		3.00%	3.00%
Bucket Fill Factor		0.90	0.95
Effective Bucket Capacity	yd3	15.30	18.53
Wet Material Weight (LCM)	wt/lcy	1.71	1.71
Dry Material Weight (LCM)	dt/lcy	1.66	1.66
Tonnes/Pass	wt	26.15	31.66
Truck Size Capacity (volume)	yd^3 heaped	102.00	102.00
Truck Size Capacity (ton)	wt	158.00	158.00
Theoretical Passes (volume)	passes	6.67	5.51
Theoretical Passes (ton)	passes	6.04	4.99
Actual Passes	passes	6.00	5.00
Truck Load - Volume (volume)	yd3	91.80	92.60
Truck Load - Volume (ton)	wt	156.90	158.30
Truck Load for Productivity	dt	152.30	153.70
Truck Capacity Utilized (ton)	by weight	99.30%	100.20%
Truck Capacity Utilized (volume)	by volume	90.00%	90.80%
Average Cycle Time	sec	35.00	30.00
Truck Spot Time	sec	45.00	45.00
Load Time per Truck	min	4.25	3.25
Maximum Productivity	trucks/hr	14.10	18.50
Insitu Volume/Hour	bcy/hr	960.00	1,266.70
Tons/Hour	dt/hr	2,150.40	2,837.30

16.6 Mine Equipment

The initial mine production equipment will include two 19.5 yd^3 shovels. A 17 yd^3 front end loader will function as a backup loading unit and infill for production when needed. Initially 12 150 ton haul trucks are required to meet the production schedule, during the end of Year 1 an additional truck will be added to meet production requirements, during Year 3 more trucks will be added and during Year 5, four more trucks will be added for a total of 20 trucks. Four production drills will also be purchased initially with three more required as strip ratios increase. Support equipment will consist of three dozers Cat D8, D9 and D10. A 16-foot wide road grader will maintain the haul roads along with a 10,000 gallon water truck. A 148 hp excavator will be purchased for scaling highwalls and other miscellaneous projects around the mine site. Six mobile light plants will be purchased for lighting the working areas during nighttime production. A maintenance service truck with a mobile crane will be purchased for field maintenance and a self-contained fuel lube truck will be purchased for infield fueling.

Table 16-8 lists the initial and additional equipment requirements.

Table 16-8 Equipment Purchases

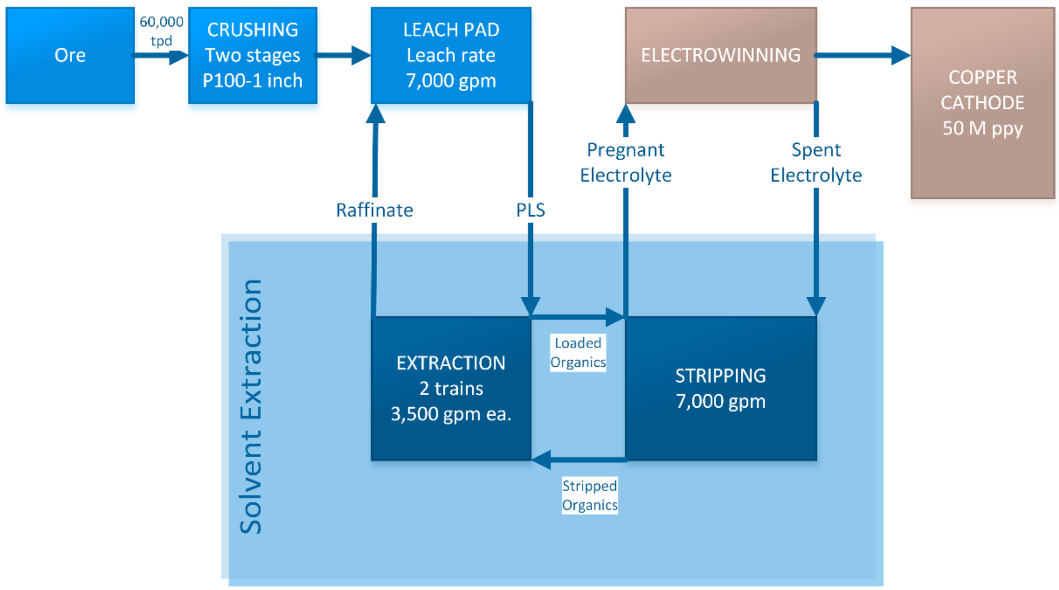
Description	Initial Units	Additional Units	Total Units	$/Unit	Initial Capital Cost	Additional Cost	Total Capital Cost
19.5 yd^3 Front Shovel	2	3	5	$5,000,000	$10,000,000	$15,000,000	$25,000,000
17 yd^3 Loader	1	1	2	2,000,000	2,000,000	2,000,000	4,000,000
Production Drill	4	3	7	950,000	3,800,000	2,850,000	6,650,000
150 ton Haul Truck-	12	8	20	2,350,000	28,200,000	18,800,000	47,000,000
16' Grader	1	0	1	850,000	850,000	0	850,000
Water Truck	1	0	1	850,000	850,000	0	850,000
448hp Dozer	1	0	1	970,000	970,000	0	970,000
347hp Dozer	1	0	1	665,000	665,000	0	665,000
580hp Dozer	1	0	1	1,400,000	1,400,000	0	1,400,000
Lube/Fuel/Service	3	0	3	125,000	375,000	0	375,000
Light Plants	6	0	6	22,000	132,000	0	132,000
Small Excavator 148hp	1	0	1	190,000	190,000	0	190,000
Misc. Equip	1	0	1	500,000	500,000	0	500,000
Pickups	10	0	10	40,000	400,000	0	400,000
Total	45	15	60		$50,332,000	$38,650,000	$88,982,000

17. RECOVERY METHODS

17.1 PROCESSING

The Contact Copper Project is designed as an open-pit, heap leach operation. Processing will begin with primary and secondary crushing, followed by stacking the ore on a heap leach pad. Copper will be leached by sulfuric acid solution and processed through a solvent extraction-electrowinning plant (SX-EW) to produce high-purity copper cathodes on site. A flow sheet for ore processing is shown in Figure 17-1.

Figure 17-1 Flow Diagram



17.1.1 Crushing and Conveying

Ore will be reduced to a size of 100%-minus one inch using two-stage crushing operating in open-cycle. Ore from the mine will be hauled to the crushing area, located at the east end of the pit, where it will be dumped into a 300-ton hopper and apron feeder which will feed the primary gyratory crusher. From the primary crusher, the ore will be conveyed to a 40,000 ton coarse ore stockpile and withdrawn to secondary cone crushers and then conveyed to a 40,000 ton fine ore stockpile. From the fine ore stockpile, the ore will then be conveyed to the leach pad using a series of mobile conveyors and a radial stacker which will place the ore on the pad for leaching in 10 to 20-foot lifts. The crushing, conveying and stacking circuit is sized to a capacity of 60,000 tons per day.

17.1.2 Leaching

Crushed ore, once stacked on the leach pad, will be leached with a weak sulfuric acid solution. The leach pad is located on the south side of the open pit on a side slope that drains towards the east and toward the processing plant (Figure 16-3). Solution will be distributed to the pad through piping and emitters at a nominal rate of 0.005 gpm per square foot of surface area, and recovered from the leach pad through a system of drains which direct the solutions to a collection pond at a design rate of 7,000 gpm. Four ponds are required to support the leaching operations.

1. Collection pond containing intermediate leach solution (ILS) that is recycled to the heap to build up copper grade
2. Pond containing the pregnant leach solution (PLS) that is the feed solution to the SX circuit
3. Pond containing the copper-depleted solution (raffinate) returned from the SX circuit
4. An event pond for stormwater collection from the site.

Design of the leach pad and ponds is described in Section 18.

17.1.3 Solvent Extraction and Electrowinning

The solvent extraction circuit will consist of two parallel sets of cells, each with two extraction cells in series or parallel followed by stripping cell. In the circuit, the PLS flows counter-current through the extraction stages where it is contacted with an organic solvent in mixer-settler tanks. Hydrogen ions in the organic exchange with copper ions in the PLS to produce a copper-loaded organic and raffinate. The organic is immiscible in the raffinate and is separated by flowing over weirs at the ends of the settler tanks. Once stripped of copper, the raffinate flows to the raffinate pond and then on to the ILS pond where fresh acid is added before pumping to the heap leach pad. The copper-loaded organic is pumped to the stripping cells where the organic is stripped of copper by strong sulfuric acid and recycled back to the extraction cells. The acid solution leaving the stripping cells is filtered and pumped to the tankhouse where the copper is plated on stainless steel cathodes in electrowinning cells. The cathodes are removed from the cells every seven to nine days by overhead crane and the plated copper is stripped from the cathodes by machine and bundled for shipping.

18. PROJECT INFRASTRUCTURE

The Contact Copper Project is well located with respect to access, community services, power and water.

18.1 ACCESS

The Project site is readily accessible from the towns of Jackpot, Nevada and Twin Falls, Idaho to the north, and Wells, Nevada, to the south where U.S. Highway 93 intersects U.S. Interstate 80. Access from U.S. Highway 93 to the Project is via an existing gravel road maintained by Elko County. Facilities anticipated at the Project include:

- Access roads
- Power lines and distribution
- Administration and other Buildings
- Communications
- Water Supply
- Leach pad and ponds
- Waste rock storage

18.2 ACCESS ROADS

Several all-weather gravel roads provide good access within the property. The access road from the highway to the administration area and plant site will be paved for ¾ mile to reduce maintenance and provide dust control.

18.3 POWER LINES AND DISTRIBUTION

Two high-tension power lines are located north of Contact; one, a 345 KV line operated by Sierra Pacific Power Company; and the other, a 138 KV line operated by Idaho Power Company and Wells Rural Electric Company. Discussions with the local utility company indicates upgrades to the line may be needed to provide adequate capacity for the Project via the 138 KV line. A 1.8 mile transmission line will be needed to bring power from the 138 KV line north of the Project to a 10MW substation, from where power will be reduced in voltage and distributed to the mine, crushing area, SX-EW plant, and administration buildings.

18.4 ADMINISTRATION AND OTHER BUILDINGS

Project support buildings will be located near the SX-EW plant (see Figure 16-3). Administration offices, safety and change rooms will be built from modular units, and plumbed with potable water and septic systems. A maintenance shop and warehouse will be steel frame buildings with concrete slab floors.

18.5 COMMUNICATIONS

Phone, cellular and internet services exist on site. Enexco has installed and licensed a VHF repeater on Ellen D Mountain for radio communications around the site.

18.6 WATER SUPPLY

HRC anticipates sufficient water will be obtained from a supply well or wells in the alluvial basin east of the plant site. The Project is estimated to require 700 gpm of water, to be used in make-up water for the processing and heap leach operations, and in drilling and dust control in the mining operation. Confirmation of the water supply will be needed prior to completion of a feasibility study.

18.7 HEAP LEACH PAD AND PONDS

18.7.1 Heap Leach Pad

MWH Global, Inc. (MWH) provided Enexco with a design of the heap leach pad (Contact Copper Project Pre-feasibility Design Report for Heap Leach Pad, Waste Stockpile, and Ancillary Facilities, 2013). The location of the heap leach pad, as shown in Figure 16-3, is south of the ultimate pit and was selected from a review of possible locations conducted by MWH earlier in 2013. The heap leach pad is sized to accommodate 125-175 million tons of ore, to be constructed in three phases consisting of 40-60 million tons per phase. The heap leach pad was located to limit the obstruction of major stormwater runoffs, and on suitably flat ground close to the pit. The event pond and process ponds were located downslope of the heap leach pad in close proximity to the SX-EW plant.

MWH prepared a preliminary design, including the specifications for the pad layout, liner, drainage and collection systems (Figure 18-1), construction and operating methods, ponds and diversion system, and closure and reclamation. MWH provided cost estimates which were incorporated into the Project operating and capital costs. The design was prepared considering the Nevada Department of Environmental Protection (NDEP) regulations and statutes. As part of the NDEP, The Bureau of Mining Regulation and Reclamation (BMRR) branch regulates mining in Nevada under the authority of the Nevada Revised Statutes (NRS) 445A.300-NRS 445A.730 and the Nevada Administrative Code (NAC) 445A.350-NAC 445A.447. The applicable NRS and NACs were reviewed and incorporated in the design. MWH notes that the design is preliminary and based upon information available. MWH notes that further work is needed in the feasibility stage through the collection of site specific geotechnical data, including foundation materials, hydrology and seismic information, to confirm the suitability of the site. MWH analysis, described later in this section, determined that a waste rock buttress will be required along the base perimeter of the heap leach to maintain stability. Construction of the waste rock buttress is included in the operating cost estimates. Key elements in the construction of the heap leach pad are:

- Geosynthetic clay liner (GCL)
- LLDPE liner (80-mil textured)
- Overliner of crushed ore minus-1 inch ore, 2 feet in thickness
- 12 inch diameter HDPE collection headers
- 12 inch, 8 inch, 4 inch CPT N12 collection pipe (30 feet spacing for 4 inch collection pipes)

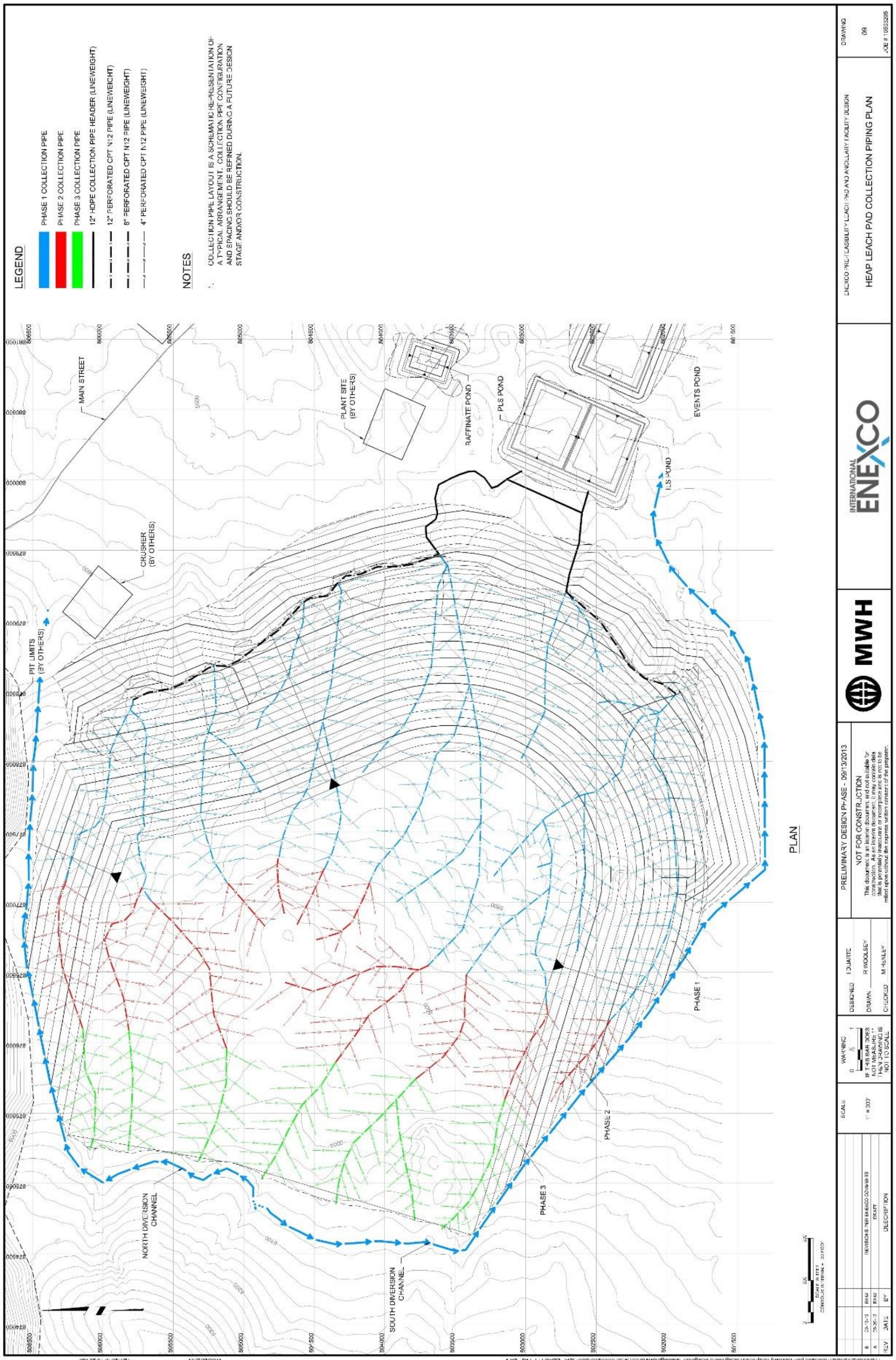


Figure 18-1 Heap Leach Pad

18.7.2 Ponds

The events and process ponds were placed downstream of the heap leach pad in close proximity to plant operations. The ponds were sized based on the estimates for water management described in Section 20. Cut and fill volumes required for construction of the ponds were balanced to reduce construction costs. Following regulatory requirements and best management practices, process ponds are double-lined and include a Leak Collection and Recovery System (LCRS), and non-process ponds are single-lined. The design includes two process ponds, a PLS (pregnant leach solution) and ILS (intermediate leach solution) pond, both capable of containing 20.3-million gallons. The ponds were designed with the capacity to contain the operating level of 5-million gallons, per Enexco, plus the 10-year wet season rainfall volume, plus an 8-hour drain down of the heap, plus freeboard. The events pond was designed to contain 25.1-million gallons which is adequate storage for the 100-year, 24-hour storm event less the 10-year wet season, plus freeboard. The raffinate pond, which is also double-lined with a LCRS, was designed to contain the 5-million gallon operational volume plus freeboard.

18.7.3 Heap Leach Pad Stability

Laboratory testing was conducted in 2012 under the direction of MWH on residue samples from column tests CL-10, CL-11, and CL-12, described in Section 13. The tests included physical, hydraulic, and geotechnical properties, and were conducted at METCON Research (METCON) and Geosystems Analysis, Inc. (GSA), both in Tucson, Arizona, and Ninyo and Moore in Phoenix, Arizona.

METCON measured the particle size distributions of the crushed samples before and after column leaching. The ore head samples are relatively coarse-grained, with less than five percent fines (passing 0.075 mm). The leach residue samples show slight to moderate shifts toward finer particle sizes, but little increase in fines content. MWH's judgment is this indicates a small amount of particle break-down of the ore under acid leaching, and ore breakdown in acid will not be sufficient enough to cause a large impact on ore hydraulic characteristics. MWH recommends further testing to evaluate ore decrepitation under long-term exposure to overburden pressures and acid leach solution for a feasibility study.

GSA measured hydraulic properties of the crushed ore samples. The hydraulic properties testing included dual-wall saturated hydraulic conductivity (Ksat) at different simulated heap loads and moisture retention characteristics (MRC), using modified flex wall permeameter procedures in ASTM D5084-03.Testing was conducted on the same material used in the metallurgical column testing at METCON. GSA used a simulated raffinate solution to perform their tests. In the Ksat testing at GSA, samples are loaded in a 6-inch (15 cm) diameter by 12 inch (30 cm) column. The column is constructed with a flexible membrane wall that allows different side-wall pressures to be applied which mimic lateral earth pressures of different heap heights. The testing predicts the changes in Ksat and bulk density of the ore under the loads applied by the heap. The testing results showed only slight reductions in Ksat values with loads. Although the final build on the heap will have thickness of greater than 200 feet, MWH concludes the results indicate permeability should be sufficient for effectively leaching the ore.

The MRC testing at GSA followed the pressure plate procedure in ASTM C199-09. This test is run using a rigid 6-inch (15 cm) diameter Tempe cell fitted with a high pressure (1 bar) porous ceramic plate. The soil matric suction is applied using either a hanging column or a compressor, depending on the soil potential being applied. Based on the results of the MRC tests, the ore is expected to drain rapidly with only a small amount of retained solution.

Ninyo and Moore performed geotechnical testing on a composite residue sample columns CL-11 and CL-12. Testing included standard Proctor testing (ASTM D696, Method B) and consolidated-undrained (CU) triaxial testing (ASTM D4767-11). The results show a Mohr-Colomb failure envelope with a cohesion of zero and a friction angle of 36 degrees. MWH recommends further testing to better characterize material strength properties at critical interfaces (e.g., with the liners) and for subgrade materials.

Nevada ranks third behind Alaska and California as the most seismically active states. Because of this seismic setting, even though the site is located within a relatively low seismic hazard area of the state compared to other parts of Nevada, the potential exists for moderate to large earthquakes (M5 to M7) to occur along nine mapped faults within a 100 kilometer radius of the site (41.770° N, -114.775° W).

Several normal faults are mapped within a few miles of the site to the west, north and north east. No seismogenic faults are mapped within the limits of the heap leach and pond locations at the site (USGS NSHMP fault data base).

The U.S. Geological Survey (USGS) National Seismic Hazard Mapping Project (NSHMP) provides probabilistic estimates of ground motions and spectral accelerations (Seismic Hazard Maps and Data, 2013). To estimate ground motions, MWH used the USGS 2008 NSHMP and selected a peak ground acceleration (PGA) with a two percent chance of exceedance in 50 years, corresponding to a mean return period of 2,475 years. The ground motion probabilities are computed for rock site conditions (Vs30 = 760 m/s) which appear to be appropriate for the portion of the site mapped as granodiorite. The deaggregated earthquake hazard contribution of PGA that has a two percent chance of being exceeded in 50 years is greater than or equal to 0.13537 g, or about 0.14 g. The PSDs of the residue samples show slight to moderate shifts toward finer particle sizes, but little increase in fines content. This indicates a small amount of particle break-down of the ore under acid leaching. MWH's judgment is that ore breakdown in acid will not be sufficient enough to cause a large impact on ore hydraulic characteristics. MWH recommends further testing to evaluate ore decrepitation under long-term exposure to overburden pressures and acid leach solution.

MWH performed a slope stability analysis to limit equilibrium methods in the SLOPE/W version 8.11 model (Geoslope International, Ltd, 2013). MWH selected the Morgenstern-Price method with a half-sine function for interslice forces for the analysis method (within the Slope/W model). This method uses both circular and non-circular shear surfaces and satisfies both moment and force equilibrium. The entry and exit method was used to define the extent of the slip surfaces to be considered in the analysis. For all of the analyses, the factor of safety (FOS) values reported reflect the calculated FOS values associated with optimized slip surfaces. Factor of safety is the ratio of resisting forces to driving forces. Optimization of the slip surface is an iterative procedure that is performed internally within SLOPE/W by altering segments of the initially calculated slip surface to find the surface with the lowest FOS. Two section geometries through the heap leach pad when fully constructed were analyzed for slope stability under various loading conditions. The sections were selected where the existing ground has the steepest slope. The analyses considered short term (ST), long term (LT), and post-earthquake (PE) loading conditions, using material properties for the ore, liner and bedrock.

The FOS values were computed for all model simulations of failure surfaces. Based on the stability modeling results, the short term, long term, and post-earthquake static stability requirements were all satisfied. This could improve if future testing data shows that the assumed parameters are overly conservative. Furthermore, according to the State of Nevada Bureau of Mining and Regulation and

Reclamation (Bureau of Mining Regulation and Reclamation, 1994), recommended factor of safety is 1.05 for pseudo static analysis for heap leach pads which is less stringent that the 1.2 criteria used in this study. In general, the seismic performance of a heap leach can be quantified by allowable permanent displacement. MWH believes that the Project's heap leach facility can tolerate large displacements (i.e. several feet) without compromising the integrity of the fluid management system or causing an uncontrolled release of contaminants. Newmark-type analyses and seismic deformation modeling (i.e. FLAC modeling) can be performed in future studies, if required, to estimate the permanent displacements induced by the design seismic event.

MWH notes that the results are preliminary and based on the material properties used. The stability models can be refined once more data is available from field and laboratory testing.

18.7.4 Waste Rock Storage

The location for waste rock storage is north of the mine area. Based on a 2.3:1 overburden waste to ore ratio provided by Enexco, the total anticipated waste rock tonnage is 325 million tons. An average waste rock density of 115-pounds per cubic foot (lb/ft3) was assumed and should be confirmed at a later design stage. MWH assumed the waste stockpile material is non-acid-generating and will not require an impermeable liner. Based on direction from Enexco, MWH assumed the waste stockpile would be constructed with 3H:1V overall side slopes; therefore, during closure the stockpile can remain stacked and will not require regrading or a soil cover.

19. MARKET STUDIES AND CONTRACTS

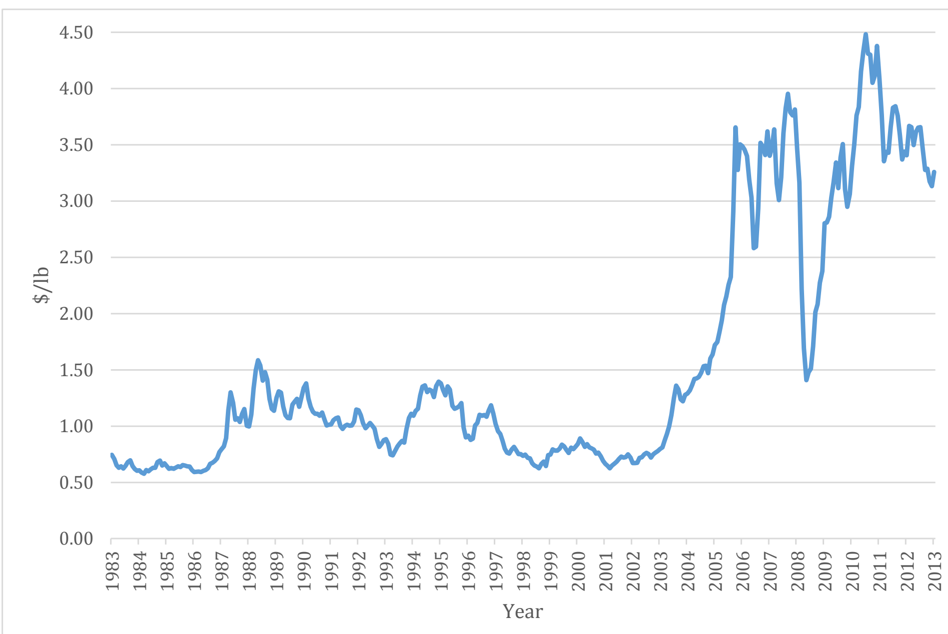
No market study has been performed for the Project. The U.S. is a net importer of refined copper. According to the USGS (US Geological Survey; Edelstein, D.L., 2013) , the U.S in 2012 mined an estimated 1.15 million tonnes of copper and consumed 1.78 million tonnes of refined metal. Imports of refined copper were 0.6 million tonnes. Cathode copper within the U.S. is readily marketable at prevailing copper prices. As such, no market study is deemed necessary.

Copper prices are affected by worldwide trends in supply and demand, and determined by trading on the major metals exchanges, including the New York Mercantile Exchange (COMEX) and the London Metals Exchange (LME).Figure 19-1 shows the trend in copper prices since 1983.Trailing average prices are one approach to establish the price of copper for use in evaluating projects. As of the date of this report, over the last five years the price of copper has ranged in monthly average spot price from $1.41 per pound to $4.48 per pound, with an average of $3.27 per pound. Over the last three years, the average monthly spot price is $3.71 per pound.

Copper from SX/EW plants typically carries a premium to quoted cash prices, once quality has been established and registered with COMEX or LME, and is sold Free Carrier (FCA) at the site. Copper which does not meet premium quality is typically discounted from the quoted cash price.

As of the date of this study, Enexco has not entered into any contracts for the development of this Project, for the purchase of supplies and services or for the sale of any product. Enexco has not yet entered into any discussion with potential consumers regarding off-take or other agreements. To the extent possible, all estimates of costs used in this study have been benchmarked against prevailing industry rates.

Figure 19-1 30-year Copper Grade A Cathode Monthly Price



Source: (COMEX-CME Group, 2013)

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL LIABILITIES AND PERMITTING

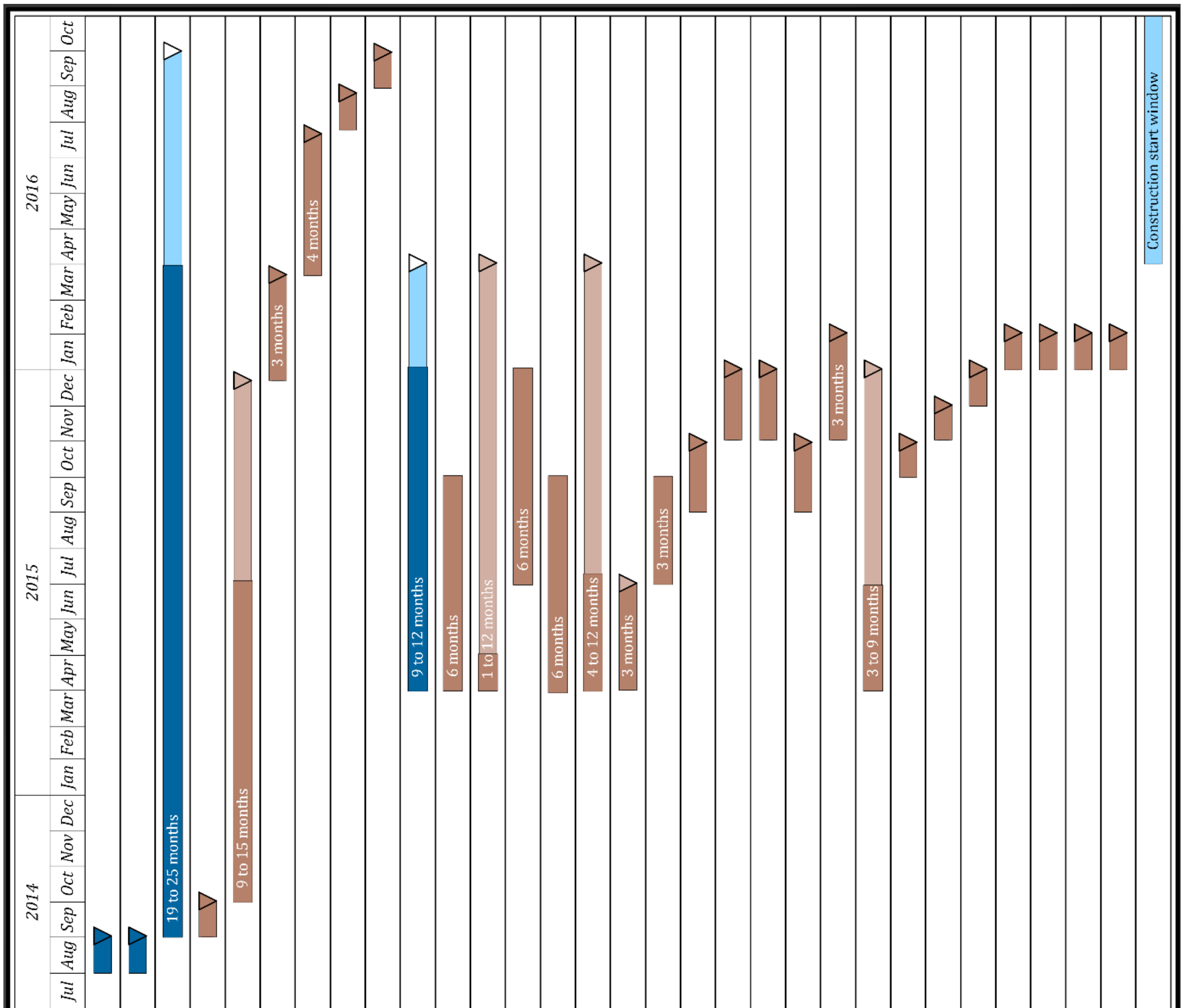
The Project is subject to no known environmental liabilities. There are no mine workings, rock piles or tailings of significance within Enexco's claims.

Various permits and plans are required to meet and maintain regulatory compliance. Environmental permitting requirements for the Project are expected to be similar to other mines in Nevada. Permitting includes consideration of reclamation, surface water, groundwater and air pollution prevention plans, and other items common to mining operations in the State of Nevada. Permits and plans will include all applicable monitoring, reporting schedules, bonding and fees. Such plans and permits are expected to include the following in order of importance:

- Plan of Operations (POO), State of Nevada and U.S. National Environmental Policy Act (NEPA) compliance
- Use of BLM-Administered Land, Compliance with Title 43 Code of Federal Regulations (CFR) Subpart 3809 Surface Management
  - Environmental Assessment (EA), or
  - Environmental Impact Study (EIS)
- Mining Reclamation Permit
- Water Pollution Control Permit
- Stormwater NPDES General Permit
- Activities in Wetlands or Waters of the U.S.
- Air Quality Operating Permit
- Permit to Appropriate Public Waters
- Industrial Artificial Pond Permit
- Hazardous Materials Permit
- Fire and Life Safety
- General Local Permits

These permits are not obtained at this time and specific reporting and planning requirements will be identified through the permitting process. Figure 20-1 below outlines the intended permitting schedule.

Figure 20-1 Project Permitting Schedule



20.2 ENVIRONMENTAL STUDIES

Enexco has initiated environmental studies with regard to the potential development of the Project. A variety of permits will be required from Federal, State, and county agencies for the Project as listed in Section 20.1 above. In order to secure these permits, data from numerous disciplines have been collected to assist with mine development, operations, and closure planning. This information will be included with ongoing studies. The following sections outline the studies, baseline data, and additional work for permitting.

Environmental data have been obtained from the following sources:

- Baseline Survey Summary, JBR Environmental Consultants, Inc. (JBR), September 2008
- Vegetation Community Types and Reference Area Establishments, JBR, October 2008
- Vegetation Baseline Report, JBR, February 2009
- Wildlife Baseline Survey and Threatened, Endangered, Sensitive, and Candidate Wildlife Species Survey, JBR, December 2008
- Soils Literature Review, JBR, March 2009
- Jurisdictional Waters Review and Seep and Spring Survey, JBR, March 2009
- Quarterly ground and surface water quality sampling
- Acid-base accounting on ore and waste rock, SVL, 2010
- Waste rock characterization studies, McClelland, December 2011
- Process leach residue characterization studies, McClelland, December 2010

The results of the baseline studies to date indicate no known issues that negatively impact the ability to extract the Mineral Resources.

20.2.1 Vegetation Baseline

JBR performed a baseline study of the Project area in 2008. Four vegetation communities were identified as listed in Table 20-1. Approximately 25% of the survey area burned in 2007 and contains BLM fire rehabilitation seeded vegetation.

Table 20-1 Vegetation Communities

Plant Community Name	Elevation Range (feet)	Acres	Reference Area
Big sagebrush steppe	5,400 – 6,800	2,365	1
Mixed sagebrush shrubland	5,800 – 6,800	1,385	2
Low sagebrush shrubland	5,500 – 6,800	900	3
Wildfire rehabilitation area	5,400 – 6,600	1,600	*

** Fire rehabilitation data obtained from BLM* *(JBR Environmental Consultants, Inc., 2009)*

20.2.2 Wildlife Baseline

JBR conducted surveys for potential threatened, endangered, sensitive, and candidate (TESC) wildlife species that may be present in the Project area. The wildlife baseline surveys were conducted from April to September 2008 and a wildlife species list was developed. No occurrences of threatened or endangered wildlife species were reported. Sensitive species observed during field work include eight species of bats, four species of birds, and one species of reptile. All species of bats living in the state of Nevada are considered sensitive species by the BLM. The Project area according to the Nevada Department of Wildlife may provide habitat for a number of bat species due to the old mine workings in the area.

20.2.3 Soils Baseline

JBR performed a review of soils. The majority of soils in the deposit area are shallow (less than 20 inches to bedrock or other restricting layer) with coarse fragment content more than 20%. The high content of coarse fragments and steep slopes will be limiting factors in salvaging this material for reclamation. JBR estimates 87,388 to 174,778 cubic yards of good quality soil material, and 1,751,811 cubic yards of medium to poor quality material are salvageable in the Project area.

20.2.4 Waste Rock

Enexco conducted acid-base characterization tests at SVL Analytical, Inc. of Kellogg, Idaho, on 170 samples of waste rock and mineralized material from the Project (Contact ABA Samples Report, 2010). The results indicate the majority of samples tested have no potential for acid generation. Enexco conducted humidity cell tests on five samples of waste rock at McClelland Metallurgical Laboratories of Reno, Nevada. The meteoric water mobility procedure was used over a one-year period. The results confirmed the samples tested have no potential for acid generation.

20.2.5 Water Sources

JBR performed a review of jurisdictional waters. Salmon Falls Creek flows north past the Project towards the Idaho border. The Project lies in the Salmon Falls Creek Area sub-basin (Hydrologic Unit Code 17040213).

USGS maintains a stream gage (#13105000, “SALMON FALLS CREEK NR SAN JACINTO NV”) approximately 12 miles north (downstream) of the Project. Daily flow and discharge data is available from 1910 to the present; however, this data is affected to an unknown degree by diversions. Annual peak flows usually are at or below 1,500 cubic feet per second (cfs). The average annual flow is 140 cfs, with a high of 439 cfs in 1984 and a low of 45.4 cfs in 1934. Water quality data for Salmon Falls Creek is available through the Bureau of Water Quality Planning under the Nevada Department of Environmental Protection (NDEP). Salmon Falls Creek is monitored at the Salmon Falls Creek location ID# E8.

There are domestic water wells existing near the Project site, and the presence of year-round flow in Salmon Falls Creek east of the site provides evidence that water will be available for the Project. Installation of monitoring wells for the establishment of a water quality baseline and determination of water supply for the Project will be included at the feasibility level. Enexco samples domestic water wells, seeps and springs, and Salmon Falls Creek on a quarterly basis.

20.2.6 Precipitation Data

Precipitation data is available through the NOAA Hydrometeorological Design Center, NOAA Atlas 14 (National Oceanic and Atmospheric Administration, 2004). In 1994, NOAA published Hydrometeorological Report No. 57, “Probable Maximum Precipitation - Pacific Northwest States, Columbia River (including portions of Canada), Snake River and Pacific Coastal Drainages”, which includes the Project area (National Oceanic and Atmospheric Administration, 1994). This provides all-season general-storm probable maximum precipitation (PMP) estimates for durations from 1 to 72 hours for several basins, including the Snake River Basin. Additionally, this detailed report discusses seasonal variations, depth-area-duration relations, and a host of other storm analyses. This report estimates the 10 square miles 24-hour PMP in the Project area as 2.27 inches.

20.2.7 Evaporation Data

Pan evaporation data is available through the Western Regional Climate Center, for the “Twin Falls WSO” station in Twin Falls County, Idaho (station # 109303. The station is approximately 60 miles north-northeast of the Project, at an elevation of 4,026 feet above mean sea level, and latitude 42º35’, longitude 114º21’) (Western Regional Climate Center, 2009). This data is summarized in Table 20-2 below.

Table 20-2 Monthly Average Pan Evaporation Rate

Station	Period of Record	J	F	M	A	M	J	J	A	S	O	N	D	Total (Inches)
Twin Falls	1963-2005	NM	NM	NM	5.80	8.09	9.15	10.24	9.09	6.65	4.25	0.77	NM	54.04

NM: Pan Evaporation was not measured from the months of December through March. *(Western Regional Climate Center, 2009)*

20.2.8 Flooding

All areas within the Project are designated by the Federal Emergency Management Association (FEMA) as areas of minimal flooding.

20.2.9 Water Management

MWH estimated the 100-year, 24-storm event and used this to evaluate the required stormwater channel flows to be diverted around the leach pad and waste rock storage. The 100-year, 24-hour storm event precipitation was estimated to be 2.31 inches, based on National Oceanic and Atmospheric Administration (NOAA) Point Precipitation Frequency Estimate for the Project’s location. The SCS curve number was estimated to be 81, which is an empirical coefficient used to estimate runoff from storms for semiarid rangeland with low-growing brush. Stormwater channels and approximate watershed areas were preliminarily delineated based on the Project’s topography. Stormwater flows were estimated using the rational method. The flows and site topography (slopes) were used to size channels and riprap. MWH assumed stormwater channels would be routed around the leach pad or waste stockpile, where required, and release clean water into existing washes. Dewatering in open pit mining is expected to be minimal. Exploration and definition drilling has not encountered significant water in drill holes within the resource area.

20.2.10 Water Balance Model

MWH developed a water balance model to estimate the water storage requirements for the heap leach ponds. The model computed monthly water contributions from precipitation as the monthly precipitation depth multiplied by the catchment area of the heap, process and storm events ponds, and outside contribution catchment area. Monthly precipitation depths were developed from historical climate summaries at the Contact, Nevada (COOP ID 261905), Jackpot, Nevada (COOP ID 264106) and Gibbs Ranch, Nevada (COOP ID 263114) weather stations. Monthly water losses from evaporation as the monthly evaporation depths multiplied by the catchment area of the heap and process ponds. Monthly evaporation depths were estimated using the Hargreaves equation (Hargreaves, 1994; Jensen, 1997) and temperature data from the Contact, Nevada weather station. The amount of solution held hydrostatically by the ore was estimated from the water retention relationships for the Project’s ore measured by GSA in samples CL-10, CL-11, and CL-12.

Results from the water balance model were used for sizing the ponds. The PLS pond and the ILS pond each require a maximum operational volume of five million gallons, plus an additional combined volume to store an 8-hour drain down from the heap, and an additional combined volume to store excess water equivalent to the wet season with a 10-year return period. The combined volume of the PLS pond, ILS pond, and storm events pond must also have a combined volume necessary to store excess water equivalent to the greater of the 100-year, 24-hour storm event volume and the wet season with a 100-year return period (one percent annual return probability), and all ponds must have two feet of freeboard above the water storage requirements. Based on the model results, the required combined volume of the PLS and ILS ponds is 30 million gallons, and the required size of the storm events pond is 22 million gallons.

20.2.11 Closure Plan

MWH evaluated methods for closure of the heap leach facility, waste rock storage and ponds. The heap leach pad closure will include the following, re-grading the top surface to gradually slope at about 0.5% grade towards the stormwater diversion channels on the west side of pad. Outslopes and benches will be re-graded to 3H:1V where necessary. Then a 24-inch layer of evapotranspirative (ET) soil cover will be placed on the top surface and a 24-inch soil cover and rock armor for erosion protection on outslopes followed by seeding all surfaces for vegetative growth. A water treatment plant will treat solutions during rinsing and draindown of the heap leach facility. The plant will include a lime slaking facility for hydration of quicklime [CaO], and a solids contact reactor/clarifiers for pH neutralization and mineral precipitation. On-site disposal to the mine area will be used for disposal of treated drain-down solution and precipitated mineral solids. No specific reclamation activities are planned for closure of the waste rock stockpile because the waste rock is non-acid generating. The waste rock stockpile will be constructed with 3H:1V outslopes, this slope and configuration is assumed to be stable. Ponds will be reclaimed by cutting and removing liners, re-grading for positive drainage, placing erosion resistant cover and reseeding. The design includes constructing the stormwater diversion channels sized for post-closure at the start of the Project so additional work should be limited and should be inspected and maintained as necessary during closure activities.

21. CAPITAL AND OPERATING COSTS

21.1 CAPITAL

The capital costs are developed from estimates of the major project areas. The mining capital is developed through the generation of a major equipment list with quotes from manufactures. The leach pad capital is developed from the quantity and cost estimates provided by MWH Global, Inc. The plant capital is developed from equipment quotes, factored estimates, and comparisons with other recently constructed projects. The initial capital costs are estimated to be $188.9 million including contingency. The costs are summarized in Table 21-1, and described further in the sections below.

Table 21-1 Capital Cost Summary

Description	Cost (000)
Direct Costs	
Site Preparation	$2,688
Mining Equipment	50,332
Crushing	11,533
Conveying	6,838
Pad & Ponds	26,146
SX-EW Plant	36,339
Infrastructure	11,050
Reagents & Initial Fills	2,532
Direct Costs Total	$147,459
Indirect Costs	
Construction Indirects	$2,838
Contingency (@ 20%)	19,425
Contingency Mine Equip. (@ 10%)	5,033
EPCM	7,095
Freight, Mobilization	2,365
Owners Costs	4,730
Indirect Costs Total	$41,486
Capital Costs Total	**$188,945**

21.1.1 Site Preparation

Site preparation costs include improving the access road to the plant site, yard gate, lighting, clearing and grading the plant site for construction, initial haul road construction, and a water diversion ditch around the waste stockpile. Cost estimates are summarized in Table 21-2.

Table 21-2 Estimated Site Preparation Costs

Description	Cost (000)
Haul Roads	$1,021
Plant Site, Preparation 10A@ 25k/A	255
Site Access Road	179
Waste Stockpile Storm Management	1,208
Yard Gate, Lighting	26
Total	$2,688

21.1.2 Mining Equipment

The initial major mining equipment consists of four drills, two 19.5 yd^3 shovels, one 17 yd^3 loader and 12 150 ton haul trucks and other support equipment as described in Section 16. Quotes for the purchase price of major equipment were obtained from Komatsu Equipment Company, who also provided options for lease/purchase agreements. The initial fleet has been included in the economic analysis as a capital lease. Table 21-3 lists the initial capital purchase price for the equipment. Sustaining capital for additional mine equipment is estimated at $38.6 million and is listed in detail in Section 16.

Table 21-3 Mining Equipment

Description	Cost (000)
Production Drill (4)	$3,800
19.5 yd^3 Front Shovel (2)	10,000
17 yd^3 Loader (1)	2,000
150 ton Haul Truck (12)	28,200
Support Equipment	6,332
Total	$50,332

21.1.3 Crushing and Conveying Equipment

Crushing equipment includes the purchase and installation of a gyratory crusher and two secondary crushers. Costs were obtained from InfoMine USA, Inc.'s CostMine *Mine and Mill Equipment Costs* (2012). Conveying equipment was quoted by Superior Industries, LLC and includes overland conveyors, mobile conveyors and a stacking conveyor to place ore on the leach pad. Estimated costs are listed in Table 21-4.

Table 21-4 Estimated Crushing and Conveying Costs

Description	Cost (000)
Earth/Concrete/Mechanical Installation	$2,041
Primary Crusher/Gyratory	7,451
Secondary Crusher	2,041
Conveying Equipment	6,838
Total	$18,372

21.1.4 Leach Pad and Ponds

The capital estimate for leach pad and ponds was developed by MWH. The leach pad is constructed in three phases of 40 to 60 million tons each. The total area required for the leach pad is 18 million square feet. The estimate below in Table 21-5 consists of the cost of the first phase of the leach pad. The costs for the additional phases are included in the plant sustaining capital in Section 21.1.10.

Table 21-5 Estimated Leach Pad and Ponds Costs

Description	Cost (000)
Leach Pad subgrade and liner	$16,851
Leach Pad Overliner	1,124
Leach Pad Collection Pipe	668
Leach Pad Storm Water Diversion	2,435
PLS, ILS and Raffinate Ponds	1,789
Events Pond	3,279
Total	$26,146

21.1.5 SX-EW Plant

The costs for the SX-EW plant were developed from quotations of major components and general plant costs, as well as additional allowances to include installation, freight and electrical components. The general plant costs include vehicles, utility equipment, tools and inventory of parts and repair items. A summary of the SX-EW plant costs is listed in Table 21-6 below.

Table 21-6 Estimated SX-EW Plant Costs

Description	Cost (000)
Acid Storage	$765
Electrowinning	14,889
Solvent Extraction	13,974
Tank Farm	4,261
Plant General	1,531
Water Supply System	919
Total	$36,339

21.1.6 Infrastructure

The infrastructure costs allow for an administration building, an equipped assay lab, a warehouse and shop for the plant, related support items, and electrical. The buildings are modular or steel frame. Electrical costs cover the main substation and connection to the 138 KV main transmission line. Distribution and electrical components for the SX-EW plant, crushing and conveying, and mine are included in the costs of the respective areas. Infrastructure costs are summarized in Table 21-7 below.

Table 21-7 Estimated Infrastructure Costs

Description	Cost (000)
Administration Building	$357
Assay Laboratory	1,021
Plant Warehouse/Shop	638
Ambulance Garage	49
Computers Software	102
First Aid Facility	77
Plant Communications	37
Safety Supplies	153
Security Office	28
Septic System	51
Site Fencing	123
Truck Scale	51
Truck Shop	2,552
Water Supply System	306
Electrical	
Grounding	61
Power distribution- emer. Generator	3,332
Power Line -2 miles	559
Substation	1,501
Total	$11,049

21.1.7 Reagents and Initial Fills

Reagents and initial fill costs cover start-up of the leach operation and initial stock of reagents for the SX-EW plant. These costs are in addition to the operating costs covered by working capital and amount to $2,532,000.

21.1.8 Indirect Costs

Indirect costs consist of engineering, construction, owner's costs, and contingency. Table 21-8 below summarizes the estimated indirect costs.

An allowance for project engineering, procurement and construction management (EPCM) is included at 7.5% of the direct costs for a total cost of $7.1 million. The allowance covers work involved in the detailed design and construction of the project.

Construction indirect costs consist of a three percent allowance on the direct costs. Freight and mobilization consists of an allowance of 2.5% on the direct costs.

The owner's costs consist of a five percent allowance on the direct costs and include allowances for additional metallurgical testing, a feasibility study, permitting, and other Enexco costs up through construction. Costs related to reclamation are handled separately in the cash flow analysis.

Contingency is included in the capital costs to allow for uncertainty in the estimates. An allowance of 20% has been applied to the direct capital costs with the exception of the mine equipment to which an allowance of 10% was applied. HRC believes these allowances are appropriate for the Project at the pre-feasibility level.

Table 21-8 Estimated Indirect Costs

Description	Cost (000)
EPCM	$7,095
Construction Indirects	2,838
Freight, Mobilization	2,365
Owner's Costs	4,730
Contingency (@ 20%)	19,425
Contingency Mine Equip. (@ 10%)	5,033
Total	$41,486

21.1.9 Working Capital

An allowance of $10,973,000 is included to provide the operation with sufficient working capital during start-up. The working capital allowance is established prior to production and is shown recouped in the final year of the cash flow model.

21.1.10 Sustaining Capital

Sustaining capital includes the purchase of additional mine equipment as described in Section 16, the phase two and three expansions of the leach pad, and the costs for major repairs on mining and plant equipment (Table 21-9). The costs include indirect costs and contingencies of 10% on mining equipment and 20% on leach pad expansions and plant sustaining capital.

Table 21-9 Estimated Sustaining Capital Costs

Description	Cost (000)
Sustaining Capital - Mine	$88,551
Sustaining Capital - Leach Pad	16,837
Sustaining Capital - Plant	20,999
Total	$126,387

21.1.11 Closure Cost

Total costs of $25 million are estimated by MWH for closure and reclamation of the leach pad, ponds and related facilities.

21.2 OPERATING COST

The operating costs are developed based upon an average production rate of 50 million pounds per year of copper cathode. Costs were developed from estimates of personnel and consumable items for the operation and from comparison with operating costs at similar operations and InfoMine USA, Inc.'s CostMine *Mine and Mill Equipment Costs* (2012) . The following cost estimates and assumptions were used:

- Labor: prevailing rates for the area
- Fuel: delivered, basis of diesel at $3.00 per gallon
- Power: all-in rate of $0.05 per kWh

- Sulfuric Acid: delivered, $120 per ton

With the above conditions, the cash operating costs are estimated at approximately $797 million over the 9.4 year production life, for an average cash cost of $1.72 per pound of copper produced as shown in Table 21-10 below. With the mine labor changing as stripping ratios and average grades fluctuate, the staffing for the Project ranges from 235 employees to 309 employees (Table 21-11).

Table 21-10 Estimated Operating Costs

Operating Cost	Total Cost (000)	$/lb Cu	$/ton Ore
Mining	$424,936	0.92	3.01
Processing	325,359	0.70	2.31
G&A	30,001	0.06	0.21
Property Tax	16,913	0.04	0.12
Cash Operating Costs		*1.72*	*5.65*
Royalties		0.01	0.03
Total	$797,209	$1.73	$5.68

Table 21-11 Site Labor Requirements

Department	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mine Operations	128	123	153	161	197	193	161	149	157	141
Mine Eng & Geo	8	8	8	8	8	8	8	8	8	8
Plant Operations	85	85	85	85	85	85	85	85	85	85
G&A	19	19	19	19	19	19	19	19	19	19
Total Property	240	235	265	273	309	305	273	261	269	253

21.2.1 Mining

Mine operating costs are based on scheduled production, equipment requirements, operating hours, hourly equipment operating costs, and manpower requirements. Labor estimates include salaried supervision (exempt from overtime pay) and hourly (non-exempt) personnel. The production schedule was based upon the sequence of mining developed for the Phase 1 through Phase 3 pits. The schedule was balanced for consistent production of copper from the SX-EW plant over time. The mining operating costs cover drilling, blasting, loading and hauling of ore and waste from the pit, and mine support. The total mine operating costs are shown in Table 21-12. Mining operating costs are estimated to average $45.3 million per year, for a unit cost of $0.91 per ton of material mined.

Table 21-12 Mining Operating Costs

Department	Average Yearly Cost	Life-of-Mine Cost	$/lb Cu	$/ton Ore	$/ton Mined
Mine G&A	$1,336	$12,515	$0.03	$0.09	$0.03
Drilling	3,551	33,272	0.07	0.24	0.07
Blasting	10,641	99,692	0.22	0.71	0.21
Loading	7,207	67,521	0.15	0.48	0.14
Haulage	16,679	156,261	0.34	1.11	0.33
Roads & Dumps	3,332	31,216	0.07	0.22	0.07
Dewatering	37	342	0.00	0.00	0.00
Mine Mtce.	1,217	11,405	0.02	0.08	0.02
Engineering	710	6,648	0.01	0.05	0.01
Geology	647	6,064	0.01	0.04	0.01
Total	$45,357	$424,936	$0.92	$3.01	$0.91

The projected labor requirements for the mine operations are shown in Table 21-13. Allowances are added to the base rates for labor at 35% for salaried personnel and 40% for hourly, which covers insurance, benefits, vacation, and sick leave. A five percent allowance is included for hourly personnel for overtime pay. Mine labor ranges from 135 to 205 employees over the production life. Table 21-14 shows the average annual costs breakdown for labor, materials, and supplies for activities within the mine.

Table 21-13 Mining Labor Requirements

Title	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mine Superintendant	1	1	1	1	1	1	1	1	1	1
Mine Foreman	4	4	4	4	4	4	4	4	4	4
Blasting Foreman	1	1	1	1	1	1	1	1	1	1
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1
Maintenance Foreman	4	4	4	4	4	4	4	4	4	4
Driller	13	11	18	20	28	28	20	16	16	12
Blaster	2	2	2	2	2	2	2	2	2	2
Blaster Helper	2	2	2	2	2	2	2	2	2	2
Loader Operator	4	4	4	4	4	4	4	4	4	8
Shovel Operator	8	7	13	12	20	20	12	12	12	4
Truck Driver	45	40	53	60	80	76	60	52	60	52
Dozer Operator	8	8	8	8	8	8	8	8	8	8
Grader Operator	4	4	4	4	4	4	4	4	4	4
Utility Operator	4	4	4	4	4	4	4	4	4	4
Lead Mechanic	4	4	4	4	4	4	4	4	4	4
Heavy Equipment Mechanic	4	4	4	4	4	4	4	4	4	4
Light Vehicle Mechanic	1	2	2	2	2	2	2	2	2	2
Welder/Mechanic	6	9	12	12	12	12	12	12	12	12
Apprentice/Fueler	8	8	8	8	8	8	8	8	8	8
Planner	1	2	2	2	2	2	2	2	2	2
Electrician	2	2	2	2	2	2	2	2	2	2
Sr Mining Engineer	1	1	1	1	1	1	1	1	1	1
Jr Mining Engineer	1	1	1	1	1	1	1	1	1	1
Chief Surveyor	1	1	1	1	1	1	1	1	1	1
Surveyor	1	1	1	1	1	1	1	1	1	1
Sr Geologist	1	1	1	1	1	1	1	1	1	1
Ore Control Geologist	1	1	1	1	1	1	1	1	1	1
Sampler	2	2	2	2	2	2	2	2	2	2
Total	135	132	161	169	205	201	169	157	165	149

Table 21-14 Mining Operating Costs by Category

Department	Area	Average Yearly Cost	Life-of-Mine Costs	$/ton Ore	$/ton Mined	$/lb Cu
Mine G&A	Energy	11,474	107,500	0.00	0.00	0.00
	Fuel & Lubes	113,408	1,062,496	0.01	0.00	0.00
	Labor & Benefits	722,461	6,768,571	0.05	0.01	0.01
	Materials/Supplies	261,975	2,454,383	0.02	0.01	0.01
	Services	203,853	1,909,854	0.01	0.00	0.00
Drilling	Fuel & Lubes	1,331,968	12,478,902	0.09	0.03	0.03
	Labor & Benefits	1,545,257	14,477,155	0.10	0.03	0.03
	Materials/Supplies	674,186	6,316,292	0.04	0.01	0.01
Blasting	Fuel & Lubes	25,186	235,963	0.00	0.00	0.00
	Labor & Benefits	309,374	2,898,453	0.02	0.01	0.01
	Materials/Supplies	10,276,402	96,277,223	0.68	0.21	0.21
	Services	29,946	280,562	0.00	0.00	0.00
Loading	Fuel & Lubes	4,158,205	38,957,254	0.28	0.08	0.08
	Labor & Benefits	1,437,624	13,468,767	0.10	0.03	0.03
	Materials/Supplies	1,565,997	14,671,466	0.10	0.03	0.03
	Services	45,206	423,528	0.00	0.00	0.00
Haulage	Fuel & Lubes	7,801,606	73,091,436	0.52	0.16	0.16
	Labor & Benefits	4,506,191	42,217,457	0.30	0.09	0.09
	Materials/Supplies	4,004,179	37,514,224	0.27	0.08	0.08
	Services	366,965	3,438,009	0.02	0.01	0.01
Roads & Dumps	Fuel & Lubes	1,114,769	10,444,010	0.07	0.02	0.02
	Labor & Benefits	1,332,688	12,485,643	0.09	0.03	0.03
	Materials/Supplies	884,433	8,286,050	0.06	0.02	0.02
Dewatering	Materials/Supplies	21,747	203,744	0.00	0.00	0.00
	Services	14,779	138,464	0.00	0.00	0.00
Mine Mtce.	Fuel & Lubes	145,429	1,362,488	0.01	0.00	0.00
	Labor & Benefits	679,847	6,369,331	0.05	0.01	0.01
	Materials/Supplies	370,718	3,473,171	0.02	0.01	0.01
Engineering	Fuel & Lubes	7,596	71,167	0.00	0.00	0.00
	Labor & Benefits	344,351	3,226,141	0.02	0.01	0.01
	Materials/Supplies	60,932	570,862	0.00	0.00	0.00
	Services	272,133	2,549,548	0.02	0.01	0.01
Geology	Fuel & Lubes	3,798	35,584	0.00	0.00	0.00
	Labor & Benefits	406,115	3,804,797	0.03	0.01	0.01
	Materials/Supplies	81,016	759,017	0.01	0.00	0.00
	Services	131,828	1,235,063	0.01	0.00	0.00
Total		$45,263,644	$424,064,573	$3.01	$0.90	$0.92

21.2.2 Processing

Costs for processing and administration are estimated to average $37.7 million per year over the 9.4 years of the leach operation, for a total of $2.50 per ton of ore treated (Table 21-15). Costs associated with closure of the operation are covered in the economic analysis in Section 22.1.8.

Table 21-15 Processing and Administration Costs

Department	Area	Average Yearly Cost	Life-of-Mine Cost	$/ton Ore	$/lb Cu
Plant G&A	Energy	43,373	406,351	0.00	0.00
	Fuel & Lubes	1,944	18,210	0.00	0.00
	Labor & Benefits	162,047	1,518,184	0.01	0.00
	Materials/Supplies	147,916	1,385,789	0.01	0.00
	Services	260,792	2,443,302	0.02	0.01
Primary Crushing	Energy	182,215	1,707,134	0.01	0.00
	Fluids	311,369	2,917,144	0.02	0.01
	Fuel & Lubes	43,542	407,934	0.00	0.00
	Labor & Benefits	743,826	6,968,729	0.05	0.02
	Maint Labor	99,777	934,791	0.01	0.00
	Materials/Supplies	148,197	1,388,422	0.01	0.00
	Wear Parts	1,261,646	11,820,065	0.08	0.03
Secondary Crushing	Energy	239,488	2,243,706	0.02	0.00
	Fluids	99,931	936,232	0.01	0.00
	Maint Labor	354,848	3,324,491	0.02	0.01
	Wear Parts	535,048	5,012,743	0.04	0.01
Conveying	Energy	546,519	5,120,209	0.04	0.01
	Fluids	642,208	6,016,700	0.04	0.01
	Labor & Benefits	1,451,678	13,600,433	0.10	0.03
	Maint Labor	323,085	3,026,907	0.02	0.01
	Materials/Supplies	51,469	482,203	0.00	0.00
	Wear Parts	825,936	7,738,001	0.05	0.02
SX-EW	Energy	2,961,850	27,748,886	0.20	0.06
	Fluids	92,669	868,193	0.01	0.00
	Fuel & Lubes	355,422	3,329,866	0.02	0.01
	Labor & Benefits	1,809,200	16,949,977	0.12	0.04
	Maint Labor	34,118	319,642	0.00	0.00
	Reagents	1,846,182	17,296,454	0.12	0.04
	Wear Parts	42,647	399,552	0.00	0.00
Leaching	Energy	648,238	6,073,191	0.04	0.01
	Fluids	7,165	67,125	0.00	0.00
	Fuel & Lubes	11,176	104,705	0.00	0.00
	Labor & Benefits	665,680	6,236,597	0.04	0.01
	Maint Labor	57,744	540,993	0.00	0.00
	Materials/Supplies	1,476,712	13,834,972	0.10	0.03

	Sulfuric Acid	15,028,705	140,800,447	1.00	0.30
	Wear Parts	17,229	161,419	0.00	0.00
Plant Mtce.	Fuel & Lubes	7,206	67,510	0.00	0.00
	Labor & Benefits	427,418	4,004,383	0.03	0.01
	Materials/Supplies	49,996	468,401	0.00	0.00
Assay Lab	Labor & Benefits	452,162	4,236,200	0.03	0.01
	Materials/Supplies	188,250	1,763,675	0.01	0.00
Administration		3,040,158	28,482,530	0.20	0.01
Total		$37,696,782	$353,172,396	$2.50	$0.76

Operating costs include crushing and handling of ore from the primary crushers to the leach pad. The crushing and conveying circuit is sized to handle the maximum daily ore production of approximately 60,000 tons per day. The crushing and conveying costs amount to $0.52 per ton, which includes wear and maintenance items, labor and power.

A major component in the operating costs is sulfuric acid. Sulfuric acid will be transported via tanker truck from Salt Lake City, Utah and has been estimated at a delivered cost of $120 per ton for sulfuric acid. At the projected consumption of 17 pounds per ton of ore, the cost of sulfuric acid amounts to $15 million annually. Other costs for the leach operation include allowances for tubing, pipes and pump repair parts, and general labor to layout and maintain the leach lines.

The other major components in the operating costs are power, labor and reagents. Power costs for the operation as a whole are based on average annual consumption of 93.0 million kWh at $0.05 per kWh, which includes a demand charge of $7 per kW and a use charge of $0.04 per kWh (Table 21-16). Costs for reagents are $1.8 million annually.

Table 21-16 Estimated Power Requirements

Area	Average Total (kW hr/yr)
Mine G&A	229,000
Admin	394,000
Plant G&A	867,000
Primary Crushing	3,644,000
Secondary Crushing	4,790,000
Conveying	10,930,000
SX/EW	59,237,000
Leaching	12,965,000
Total	93,056,000

Plant G & A and maintenance costs include allowances for mechanical and electrical supplies to service the SX-EW plant and general support equipment at the plant site. The plant operating costs include allowance for a laboratory to provide assay coverage for the mine and SX-EW plant.

Administration costs include allowances to cover general office, safety and property insurance costs. The major tax liability is the state Net Proceeds of Minerals Tax, which is covered in the economic analysis in Section 22.

The projected labor requirements for the plant personnel are shown in Table 21-17 below and the administration personnel are shown in Table 21-18. A total of seven exempt and 78 non-exempt employees are required for the plant operations and a total of eight exempt and 11 non-exempt employees are required for the general administration. Allowances are added to the base rates for labor at 35% for salaried personnel and 40% plus a five percent allowance for overtime pay for hourly personnel.

Table 21-17 Plant Labor

Type	Employees	Title
Salary Personnel	1	Plant Superintendent
	4	SX/EW Foreman
	1	Leaching Foreman
	1	Crush & Convey Foreman
Hourly Personnel	4	Crush & Convey Leadman
	4	Crusher Operators
	8	Conveyor Operators
	8	Laborers
	4	Stacker Operator
	4	Pad Operator
	4	Pad Helper
	8	SX/EW Operator
	12	SX/EW Helper
	4	Assayer
	2	Sample Prep
	8	Mechanic
	4	Mechanic Helper
	4	Electrician
Total	85	

Table 21-18 Administration Labor

Type	Employees	Title
Salary	1	General Manager
	1	Environmental Manager
	1	HR Manager
	1	Safety Superintendent
	1	Controller
	1	Purchasing Manager
	1	Environmental Tech
	1	IT Tech
Hourly	1	Payroll
	1	Accounts Payable
	1	Admin Assistant
	2	Janitor
	4	Safety/Security
	2	Warehousemen
Total	19	

22. ECONOMIC ANALYSIS

The economic analysis presented provides the Internal Rate of Return (IRR), Net Present Value (NPV), payback period for the project. The annual cash flows were based on the production schedule and capital and operating costs in Sections 16 and 21. The analysis includes sensitivity of the Project to variations in copper price, capital cost and operating cost.

The economic analysis was based on a copper price of $3.20 per pound, and uses Proven and Probable Mineral Reserves, only. No inferred resources were included in the analysis. Costs and revenues in the cash flow analysis were un-inflated. The analysis was unlevered with the exception of a portion of the mining equipment, for which a manufacturer provided a quotation for the initial fleet on a lease/purchase basis. The results for the analysis are shown in Table 22-1.

Table 22-1 Cash Flow Schedule

	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Life-of-Mine
MINE PRODUCTION															
Tons Ore Mined			11,401,133	10,626,585	15,376,926	14,780,513	17,895,098	18,952,510	17,206,210	16,397,147	14,037,872	4,420,008	-	-	141,094,003
Cu%			0.25	0.31	0.21	0.23	0.18	0.18	0.20	0.20	0.24	0.27	-	-	0.22
Waste			25,224,715	17,757,396	33,399,231	37,726,587	64,367,016	63,471,951	34,010,883	22,793,285	24,028,243	5,139,626	-	-	327,918,932
Total Tons Mined			**36,625,848**	**28,383,981**	**48,776,157**	**52,507,100**	**82,262,114**	**82,424,461**	**51,217,093**	**39,190,432**	**38,066,115**	**9,559,634**	**0**	**0**	**469,012,935**
SR			**2.2**	**1.7**	**2.2**	**2.6**	**3.6**	**3.3**	**2.0**	**1.4**	**1.7**	**1.2**	**0.0**	**0.0**	**2.3**
PROCESS RODUCTION															
Tons Ore Processed			**11,401,133**	**10,626,585**	**15,376,926**	**14,780,513**	**17,895,098**	**18,952,510**	**17,206,210**	**16,397,147**	**14,037,872**	**4,420,008**	**-**	**-**	**141,094,003**
Cu%			0.25	0.31	0.21	0.23	0.18	0.18	0.20	0.20	0.24	0.27	-	-	0.22
Income Statement															
REVENUE															
Contained Copper to Pad			56,576,766	65,291,559	64,347,497	68,198,133	65,004,563	67,009,780	67,109,788	67,109,788	67,293,651	23,806,407	-	-	611,747,934
Copper Produced			40,911,356	48,457,125	51,410,682	51,557,788	49,023,433	50,314,517	50,582,460	50,627,383	50,896,896	18,699,798	-	-	462,481,438
Cummulative Recovery			72%	73%	76%	76%	76%	75%	75%	75%	75%	76%	76%	76%	76%
Royalties			$327,291	$387,657	$411,285	$412,462	$392,187	$402,516	$404,660	$405,019	$407,175	$149,598	$0	$0	$3,699,852
Net Revenue			**$130,589,048**	**$154,675,142**	**$164,102,896**	**$164,572,461**	**$156,482,798**	**$160,603,939**	**$161,459,213**	**$161,602,608**	**$162,462,891**	**$59,689,755**	**$0**	**$0**	**$1,476,240,750**
OPERATING EXPENSES															
Total Mining			**36,036,873**	**31,412,473**	**44,394,138**	**46,328,916**	**64,422,480**	**62,930,702**	**45,949,512**	**39,958,755**	**41,720,203**	**11,781,948**	**0**	**0**	**424,935,999**
Total Processing			**25,058,353**	**28,414,498**	**35,821,433**	**34,928,623**	**39,349,643**	**41,039,954**	**38,456,392**	**37,251,421**	**33,772,540**	**11,266,502**	**0**	**0**	**325,359,358**
Total G&A			**3,117,334**	**3,112,435**	**3,112,435**	**3,112,435**	**3,117,334**	**3,112,435**	**3,112,435**	**3,112,435**	**3,117,334**	**1,973,921**	**0**	**0**	**30,000,530**
Cash Operating Costs	**0**	**0**	**65,835,551**	**64,665,279**	**85,104,864**	**86,213,636**	**108,745,560**	**108,917,859**	**89,255,443**	**81,969,459**	**80,130,030**	**26,371,186**	**0**	**0**	**797,208,867**
EBITDA	**0**	**0**	**64,753,497**	**90,009,863**	**78,998,032**	**78,358,825**	**47,737,238**	**51,686,080**	**72,203,771**	**79,633,149**	**82,332,861**	**33,318,568**	**0**	**0**	**679,031,883**
Depreciation	0	0	27,925,604	42,292,835	37,114,040	32,387,198	36,416,795	33,964,271	28,597,590	27,723,332	24,855,461	24,054,922	0	0	315,332,048
Reclamation Deduction	0	0	0	0	0	1,601,795	0	0	0	2,366,509	0	0	20,727,555	0	24,695,859
Net Proceeds of Mines Tax			2,571,022	3,927,827	3,292,530	3,166,182	1,550,396	1,664,519	2,641,889	2,962,450	3,051,805	584,124	0	0	25,412,746
Interest Expense	0	550,122	2,455,961	2,047,827	1,616,670	1,161,194	680,025	204,441	0	0	0	0	0	0	**8,716,239**
Income - before NOL & Perc Depletion	**0**	**-550,122**	**31,800,910**	**41,741,374**	**36,974,792**	**40,042,456**	**9,090,022**	**15,852,848**	**40,964,291**	**46,580,858**	**54,425,595**	**8,679,522**	**-20,727,555**	**0**	**304,874,991**
Net Operating Loss Adjustment	0	550,122	-14,550,122	0	0	0	0	0	0	0	0	0	0	0	-14,000,000
Depletion	0	0	-8,625,394	-20,870,687	-18,487,396	-20,021,228	-4,545,011	-7,926,424	-20,482,146	-23,290,429	-24,369,434	-4,339,761	0	0	-152,957,910
Federal Income Tax	0	0	-3,018,888	-7,304,740	-6,470,589	-7,007,430	-1,590,754	-2,774,249	-7,168,751	-8,151,650	-10,519,656	-1,518,916	7,254,644	0	-48,270,978
AMT Tax	0	0	-431,270	-1,043,535	-924,369	-1,001,061	-227,250	-396,321	-1,024,107	-1,164,522	-365,463	-216,988	-3,109,133	0	-9,904,020
Taxable Income, less Tax	**0**	**0**	**5,175,236**	**12,522,412**	**11,092,438**	**12,012,737**	**2,727,007**	**4,755,854**	**12,289,287**	**13,974,257**	**19,171,042**	**2,603,857**	**-16,582,044**	**0**	**79,742,083**
Cash Flow Calculation															
Adjustments for Non Cash Items															
Depreciation & Reclamation Deduction	0	0	27,925,604	42,292,835	37,114,040	33,988,993	36,416,795	33,964,271	28,597,590	30,089,841	24,855,461	24,054,922	20,727,555	0	340,027,907
Net Operating Loss Adjustment	0	-550,122	14,550,122	0	0	0	0	0	0	0	0	0	0	0	14,000,000
Depletion	0	0	8,625,394	20,870,687	18,487,396	20,021,228	4,545,011	7,926,424	20,482,146	23,290,429	24,369,434	4,339,761	0	0	152,957,910
Equipment Financing	0	49,432,000	0	0	0	0	0	0	0	0	0	0	0	0	49,432,000
Principal Payments	0	-1,494,793	-7,235,412	-7,643,547	-8,074,703	-8,530,179	-9,011,349	-7,442,017	0	0	0	0	0	0	-49,432,000
Total Adjustments for Non Cash Items	**0**	**47,387,085**	**43,865,707**	**55,519,975**	**47,526,733**	**45,480,042**	**31,950,457**	**34,448,679**	**49,079,736**	**53,380,270**	**49,224,895**	**28,394,683**	**20,727,555**	**0**	**506,985,817**
Capital															
Investment - Mine	3,431,398	50,857,179	823,712												55,112,289
Investment - Plant	255,162	62,115,066	22,306,388												84,676,617
Investment - G&A	1,114,043	5,054,528	1,501,612												7,670,183
Capital Indirects & Contingency	863,829	31,717,614	8,904,249												41,485,692
Total Capital	**5,664,432**	**149,744,387**	**33,535,960**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**188,944,780**
Sustaining Capital - Mine			3,406,361	3,659,657	23,654,909	4,926,877	21,140,420	5,666,908	4,952,290	4,681,211	4,826,510	3,586,135	0	0	80,501,278
Sustaining Capital - Plant			0	1,454,804	5,583,047	5,505,152	2,404,299	2,542,220	5,821,962	5,716,294	1,900,522	601,587	0	0	31,529,886
Sustaining Capital - Indirects& Contingency			340,636	656,926	3,482,100	1,593,718	2,594,902	1,075,135	1,659,621	1,611,380	862,755	478,931	0	0	14,356,105
Reclamation Closure Costs		0	0	0	0	1,601,795	0	0	0	2,366,509	0	0	20,727,555	0	24,695,859
Salvage Value			0	0	0	0	0	0	0	0	0	0	0	-8,898,200	-8,898,200
Total Capital & Sustaining	**5,664,432**	**149,744,387**	**37,282,957**	**5,771,387**	**32,720,056**	**13,627,542**	**26,139,621**	**9,284,263**	**12,433,874**	**14,375,394**	**7,589,788**	**4,666,652**	**20,727,555**	**-8,898,200**	**331,129,707**
Working capital		0	10,973,000	0	0	0	0	0	0	0	0	-10,973,000	0	0	0
Total Capital & Working Capital	**5,664,432**	**149,744,387**	**48,255,957**	**5,771,387**	**32,720,056**	**13,627,542**	**26,139,621**	**9,284,263**	**12,433,874**	**14,375,394**	**7,589,788**	**-6,306,348**	**20,727,555**	**-8,898,200**	**331,129,707**
Beginning Cash	**0**	**-5,664,432**	**-108,021,735**	**-107,236,749**	**-44,965,749**	**-19,066,634**	**24,798,603**	**33,336,447**	**63,256,717**	**112,191,867**	**165,171,000**	**225,977,149**	**263,282,037**	**246,699,993**	
Period Net Cash Flow	**-5,664,432**	**-102,357,303**	**784,986**	**62,271,000**	**25,899,115**	**43,865,237**	**8,537,844**	**29,920,270**	**48,935,150**	**52,979,133**	**60,806,149**	**37,304,888**	**-16,582,044**	**8,898,200**	
Ending Cash	**-5,664,432**	**-108,021,735**	**-107,236,749**	**-44,965,749**	**-19,066,634**	**24,798,603**	**33,336,447**	**63,256,717**	**112,191,867**	**165,171,000**	**225,977,149**	**263,282,037**	**246,699,993**	**255,598,193**	**255,598,193**

22.1 Cash Flow Schedule

22.1.1 Production Schedule

Development is assumed to extend through the end of 2016 for construction and the start of operation. The critical item in the schedule is permitting, which is difficult to forecast for mining projects in the United States. As discussed in Section 20, permitting for the Project is expected to take 19 to 25 months. Ore production to the leach pad is projected at an initial rate of 30,000 tons per day, increasing to peak production of 52,000 tons per day by in year six of operation. Copper production from the SX-EW plant is projected at an overall recovery of 75.6%, beginning at 41 million pounds per year in Year 1, and increasing to average 49.2 million pounds per year thereafter over a 9.4 year mine life, for total production of 462 million pounds of copper cathode.

22.1.2 Copper Price

A price of $3.20 per pound was selected for use as the copper price over the duration of the project life and represents 98% of the 5-year trailing price for copper as of the date of this Report. The sensitivity analysis shows the effect of variations in copper price in the range of $2.80 to $3.50 per pound.

The price received for copper may be at a discount or a premium to the quoted cash price. Copper cathodes typically receive a premium to cash prices once quality has been established, and are sold FCA at the plant. Registration to qualify for premium pricing may take two years from the start of production, during which time the price may be discounted depending on quality. Charges for insurance and freight may also apply if demand is weak. For this analysis, the base case price is used without adjustments for premiums, discounts or charges.

22.1.3 Royalties

Royalties for the Project are discussed in Section 4. A 0.25% NSR applies to certain patented claims acquired prior to 2008. A 1.75% NSR applies to two patented claims from which production occurs near the end of the mine life. In the cash flow model, a 0.25% NSR is assumed throughout on all production.

22.1.4 Operating Expenses

Operating expenses were developed over the production schedule from the estimates in Sections 16 and 21, and range from $65 million to $109 million per year. Included in the cash operating costs is county property tax, at 2.94% on the taxable assets, which are calculated at 35% of the Project's capital cost.

22.1.5 Taxes

Federal income tax is calculated at the greater of a 35% rate for regular income tax, or a 20% rate for the alternate minimum tax (AMT). Both calculations allow for depreciation and loss carry forward. In the regular income tax calculation, the capital costs are assumed depreciated at 200% declining balance over seven years. In the AMT calculation, capital costs are depreciated at 150% declining balance over 10 years. Depletion in the regular income tax is calculated as the less of 15% of gross sales or 50% of the net income from production. In the AMT calculation, the amount of depletion eligible for deduction is assumed negligible.

The major component in state tax is the Nevada Net Proceeds of Minerals Tax, which is an ad valorem property tax assessed on minerals mined or produced in Nevada when they are sold or removed from the state. If the net proceeds in the taxable year total $4 million or more, the tax rate is five percent. If the net proceeds are less than $4 million, the tax is a graduated rate. For purposes of the cash flow analysis, a flat five percent rate is assumed.

22.1.6 Initial Capital Expenditures

The capital costs were estimated in Section 21 and total $189 million, distributed over two years of pre-production and the first three months of start-up. Included is $49 million for the initial mining fleet, which a major manufacturer quoted on either a straight purchase or a lease/purchase basis. In the cash flow analysis, the lease-purchase agreement is assumed in the base case in the economics results, which includes financing over 72 months at 5.5% interest. Subsequent mining equipment in the cash flow analysis is assumed purchased without leasing.

22.1.7 Sustaining Capital

Sustaining capital costs were estimated over the life of the Project in Section 21, and range from $6 million to $26 million. The costs include $26 million in Year 3 and $23 million in Year 5 for the additional mining fleet to accommodate the increase in stripping ratios, and $21 million in Year 11 in closure costs. Total sustaining capital over the life of the Project is $126 million.

22.1.8 Reclamation

Total costs of $25 million are included in the cash flow schedule to cover the estimated costs of reclaiming the leach pad and site.

22.1.9 Working Capital, Salvage and Net Operating Loss

A working capital fund of $11 million, equivalent to two months of operating costs in Year 1, is included in the analysis. The working capital fund is shown as an expense in Year 1 of operation, and a credit in the last year of production when the fund is depleted. An allowance of 10% on mining equipment is made in the last year of the project for salvage value. The cash flow model includes Enexco's Net Operating Loss (NOL) to date in the tax treatment, which is $14 million in exploration costs on the Contact Copper Project.

22.2 ECONOMIC ANALYSIS RESULTS

22.2.1 Cash Flows, IRR, and NPV

The cash flow model is shown in Table 22-2. At a copper price of $3.20 per pound, the project generates total before-tax cash flows of $304 million and total after-tax cash flows of $256 million. The payback period is 3.4 years. The after-tax IRR is 25.9% and the NPV at eight percent discount rate is $107 million.

Table 22-2 Cash Flow Model

Project Valuation Overview	Before Tax Analysis	After Tax Analysis
Total Cash flow (millions)	$303.9	$255.6
NPV @ 5.0%; (millions)	$183.8	$149.1
NPV @ 8.0%; (millions)	$135.5	$106.7
NPV @ 10.0%; (millions)	$110.1	$84.5
Internal Rate of Return	30.4%	25.9%
Payback Period	3.0	3.4
Payback Multiple	3.8	3.4
Total Initial Capital (millions)	$188.9	$188.9
Max Neg. Cash flow (millions)	-$108.0	-$108.0

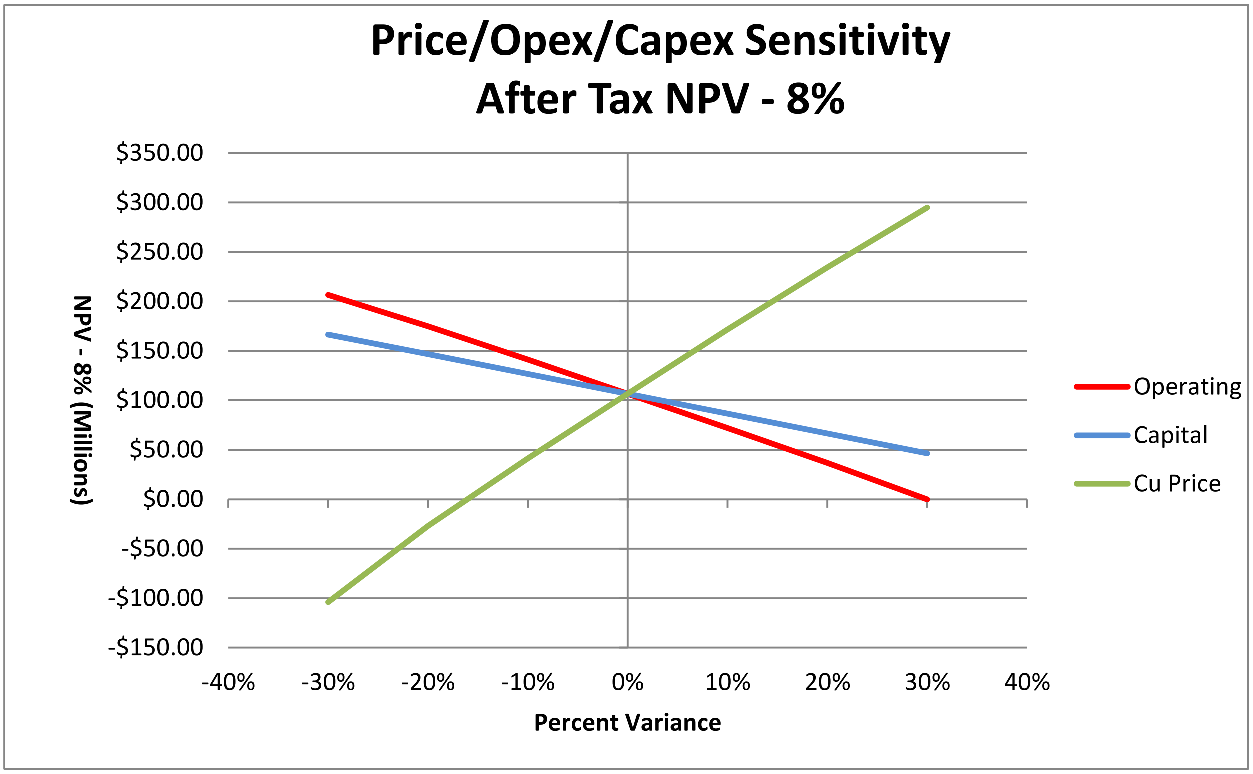
22.2.2 Sensitivities

Sensitivity of the cash flow model for changes in copper price, capital costs, and operating costs are shown in Figure 22-1. The Project is highly sensitive to changes in copper price, ranging from 15.9% IRR and $45 million NPV-8% at a copper price of $2.90 per pound to 35% IRR and $167 million NPV-8% at a copper price of $3.50 per pound (Table 22-3).

Table 22-3 NPV 8% Sensitivities

After Tax NPV @ 8% (millions)			
Change	Capital Costs	Operating Costs	Copper Price
-30%	$166.5	$206.5	-$104.0
-20%	$146.7	$175.0	-$27.1
-10%	$126.7	$141.1	$41.2
0%	$106.6	$106.6	$106.6
10%	$86.6	$71.9	$171.5
20%	$66.5	$36.9	$234.4
30%	$46.4	-$0.20	$294.9

Figure 22-1 Sensitivities



23. ADJACENT PROPERTIES

There are no adjacent properties to describe in the context of the Project. No Mineral Resources or Mineral Reserves on adjacent properties have been projected, estimated or otherwise included within this Report.

24. OTHER RELEVANT DATA AND INFORMATION

No additional information or explanation is necessary to make this Report understandable and not misleading.

25. INTERPRETATION AND CONCLUSIONS

HRC was selected by Enexco to update the Mineral Resource and Mineral Reserve estimates and provide an updated economic analysis of the Contact Copper Project. The estimates for Mineral Resources and Mineral Reserves disclosed in this Report have been prepared in accordance with NI 43-101 and 43-101F1.

This Report replaces an existing Mineral Resource and Mineral Reserve estimate and economic analysis provided in the 2010 PFS and the Mineral Resource estimate in the 2012 RE. The changes documented in this Report are the result of land acquisition, new drilling, and changes in scope and economic conditions. This Report has been prepared to support public disclosure of the current Mineral Resource and Mineral Reserve estimates and updated economic analysis.

HRC concludes the Contact Copper Project has potential to be developed by open pit mining followed by heap leaching and solvent extraction and electrowinning. At a copper price of $3.20 per pound, the estimated Project cash flow generates an after-tax IRR of 25.9% with an after-tax NPV-8% of $107 Million.

HRC finds the density of data adequate for the Project to advance to a feasibility study. Areas of uncertainty identified in this Report include the Inferred Resources within the Mineral Resource estimates. No Inferred Resources have been included in the Mineral Reserve or the economic analysis in this Report. HRC notes in addition to the Inferred Resources, additional metallurgical tests are required to confirm and optimize the parameters for heap leaching, as well as further study at the feasibility level to confirm the foundation for the heap leach pad, groundwater conditions around the site, and the capital and operating cost estimates for the Project.

HRC notes areas to enhance the Project further, including:

- Evaluate run-of-mine leaching as an alternative to crushing marginal-grade material
- Source and evaluate re-conditioned used equipment for mining and processing
- Further optimize mine and leach pad designs to minimize costs and maximize copper production early in the mine life
- Further geotechnical study of pit slopes, in particular, on the south side of the pit where the assumed pit slopes may be conservative
- Add copper oxide reserves by drilling extensions and exploration targets on the property to extend the mine life

Project risks are identified as follows:

- Further metallurgical testing may indicate lower copper recoveries or higher acid consumptions
- Cost of sulfuric acid, power, or other key operating components could increase over time
- Capital cost estimates for initial and sustaining capital may increase
- Price of copper could decrease below the price used for the analysis
- Permitting could take longer than anticipated

26. RECOMMENDATIONS

HRC recommends Enexco:

1. Continue exploration in areas outside of the current reserves. Additional reserves will extend the mine life and should enhance the project.
2. Perform metallurgical tests on composite samples representing specific production periods and optimize operating parameters for heap leaching.
3. Obtain geotechnical data on foundations of leach pads and ponds, and confirm water supply.
4. Perform further engineering design and cost estimates on mining, crushing and stacking, leach pads and ponds, processing plant and infrastructure.
5. Utilize this Report in a feasibility study.

Table 26-1 Estimated Costs for Contact Feasibility Study

		$ (x 1000)
A. Metallurgical Studies	Optimize operating parameters	250
B. Geotechnical Studies	Pad & pond foundations, confirm water supply	250
C. Project Engineering & Report	Mine, Processing Plant, Infrastructure, Economics	750
	Total	**1,250**

27. REFERENCES

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APPENDIX A. DATE AND SIGNATURE PAGES

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.

Jeffrey Choquette, PE, MMSA Qualified Person Member, is specifically responsible for Sections 15 and 16, and Sections 18 through 26.

Terre Lane, MMSA Qualified Person Member, is specifically responsible for Sections 15 through 22 and is responsible for the overall content and organization of the entire report.

Zachary J. Black, Registered Member SME, is specifically responsible for Sections 1 through 12, and 14.

Deepak Malhotra, Ph.D., MMSA Qualified Person Member, is specifically responsible for Sections 13 and 17.

This Report is effective as October 1, 2013

<u>(Signed) "Jeffrey Choquette" October 1, 2013</u>
Jeffrey Choquette, MMSA-QP

<u>(Signed) "Zachary J. Black" October 1, 2013</u>
Zachary J. Black, QP SME-RM

<u>(Signed) "Terre A. Lane" October 1, 2013</u>
Terre A. Lane, MMSA-QP

<u>(Signed) "Deepak Malhotra" October 1, 2013</u>
Deepak Malhotra, SME-RM

Jeffery W. Choquette, P.E.
Principal Engineer
Hard Rock Consulting, LLC
1746 Cole Blvd, Ste. 140
Lakewood, Colorado 80401
Telephone: 720-648-2625
Email: jchoquette@hardrock-consulting.com

CERTIFICATE of QUALIFIED PERSON

I, Jeffery W. Choquette, P.E., do hereby certify that:

1. I am currently employed as Principal Engineer by:
   Hard Rock Consulting, LLC
   1746 Cole Blvd, Ste. 140
   Lakewood, Colorado 80401 U.S.A.

2. I am a graduate of Montana College of Mineral Science and Technology and received a Bachelor of Science degree in Mining Engineering in 1995.

3. I am a Registered Professional Engineer in the State of Montana (No. 12265).

4. I am a QP Member in Mining and Ore Reserves in good standing of the Mining and Metallurgical Society of America (No. 01425QP).

5. I have practiced mining engineering and project management for seventeen years. I have worked for mining and exploration companies for sixteen years and as a consulting engineer for two and a half years.

6. 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.

7. I personally inspected the Contact Copper Project on August 1st and 2nd, 2013.

8. I am responsible for the preparation of the report titled "NI 43-101 PRE-FEASIBILITY STUDY ON THE CONTACT COPPER PROJECT," dated October 1, 2013, with an effective date of October 1, 2013 (the "Technical Report"), with specific responsibility for sections 15-16, and 18-26.

9. I have had no prior involvement with the property that is the subject of this Technical Report.

10. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
11. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

12. 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.

13. 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 1st day of October, 2013.

/s/ Jeffery W. Choquette

Signature of Qualified Person

"Jeffery W. Choquette"

Print name of Qualified Person

MONTANA
JEFFERY
CHOQUETTE
No. 12265 PE
LICENSED
PROFESSIONAL ENGINEER

Zachary J. Black, SME-RM
Resource Geologist
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1746 Cole Blvd, Ste. 140
Lakewood, Colorado 80401
Telephone: 720-648-2625
Email: zjblack@3lresources.com

CERTIFICATE of QUALIFIED PERSON

I, Zachary J. Black, SME-RM, do hereby certify that:

1. I am currently employed as Resource Geologist by:
   Hard Rock Consulting, LLC
   1746 Cole Blvd, Ste. 140
   Lakewood, Colorado 80401 U.S.A.

2. I am a graduate of the University of Nevada, Reno with a Bachelor of Science in Geological Engineering, and have practiced my profession continuously since 2005. Engineering in 1995.

3. I am a registered member of the Society of Mining and Metallurgy and Exploration (No. 4156858RM).

4. I have worked as a Geological Engineer/Resource Geologist for a total of eight years since my graduation from university; as an employee of a major mining company, a major engineering company, and as a consulting geologist. I have 8+ years of experience working on resource and reserve estimates in Mexico and the United States.

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 personally inspected the Contact Copper Project on June 7, 2012.

7. I am responsible for the preparation of the report titled “NI 43-101 PRE-FEASIBILITY STUDY ON THE CONTACT COPPER PROJECT,” dated October 1, 2013, with an effective date of October 1, 2013 (the “Technical Report”), with specific responsibility for sections 1-12, and 14.

8. I contributed to a previous technical report by 3L Resources, Ltd. on the property that is the subject of this Technical Report.

9. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

11. 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.

12. 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 1st day of October, 2013.

/s/ Zachary J. Black

Signature of Qualified Person

"Zachary J. Black"

Print name of Qualified Person

Terre A Lane, MMSA #01407QP

Consulting Mining Engineer
Hard Rock Consulting, LLC
1746 Cole Blvd, Ste. 140
Lakewood, Colorado 80401
Telephone: 720-648-2625
Email: tal0362@gmail.com

CERTIFICATE of AUTHOR

I, Terre A. Lane do hereby certify that:

1. I am currently employed as Consulting Mining Engineer by Hard Rock Consulting, LLC at:
   1746 Cole Blvd, Ste. 140
   Lakewood, Colorado 80401

2. I am a graduate of the Michigan Technological University of Michigan with a Bachelor of Science degree in Mining Engineering (1982).

3. I am a qualified professional member in good standing of the Mining and Metallurgical Society of America, MMSA #01407QP.

4. I have worked as a Mine Engineer for over 25 years since my graduation from university; as an employee of several mining companies, an engineering company, a mine development and mine construction company, an exploration company, and as a consulting engineer.

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

6. I am responsible for the preparation of the technical report titled "NI 43-101 Technical Report on the Contact Copper Project Nevada, USA" with an effective date of October 1, 2013 (the Report) with specific responsibility for Sections 15 through 22 and overall content and organization of the entire report. I have also personally completed an independent overall review and analysis of the data and written information contained in this Report.

7. I have prior involvement with International Enexco Ltd. and previously worked for Gustavson Associates and 3L Resources on Enexco's Contact Copper Project subject of this and the previous Reports.

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

9. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a two kilometer distance of any of the subject properties.

10. I am independent of International Enexco Ltd. in accordance with Section 1.5 of NI 43-101.

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

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

Dated this 1st day of October, 2013.

/s/ Terre A. Lane

Signature of Qualified Person

"Terre A. Lane"

Print name of Qualified Person

DEEPAK MALHOTRA, PHD
President
Resource Development, Inc.
11475 West I-70 Front Road North
Wheat Ridge, Colorado 80033
Telephone: (303) 422-1176 Facsimile: (303) 424-8580
Email: dmalhotra@aol.com

CERTIFICATE OF AUTHOR

I, Deepak Malhotra, PhD do hereby certify that:

1. I am President of:

   Resource Development, Inc. (RDi)
   11475 W. I-70 Frontage Road North
   Wheat Ridge, CO, USA, 80033

2. I graduated with a degree in Master of Science from Colorado School of Mines in 1973. In addition, I have obtained a PhD in Mineral Economics from Colorado School of Mines in 1977.

3. I am a registered member of the Society of Mining, Metallurgy and Exploration, Inc. (SME), member No. 2006420RM.

4. I have worked as a mineral processing engineer and mineral economist for a total of 40 years since my graduation from university; as an employee of several mining companies, an engineering company, a mine development and mine construction company, an exploration company and as a consulting engineer. I have experience in projects similar to Contact Copper Project inclusive of those in the Western United States.

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 Sections 13 and 17 of the technical report entitled "NI 43-101 PRE-FEASIBILITY STUDY ON THE CONTACT COPPER PROJECT," dated October 1, 2013, with an effective date of October 1, 2013 (the "Technical Report").

7. I personally inspected the Contact Copper Project on August 1st, 2013.

8. I have had prior involvement with the Contact Copper Project that is the subject of this Technical Report. I was responsible for Section 18 of the technical report titled "NI 43-101 Technical Report on th Contact Copper Project" dated July 31, 2009.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10 . I have read National Instrument 43-101 and Form 43-101, 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.

12. 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.

Dated this 1st day of October, 2013.

/s/ Deepak Malhotra
Signature of Qualified Person

"Deepak Malhotra"
Print name of Qualified Person

APPENDIX B. LIST OF CLAIMS

a. Located Claims

Claim Name			NMC Number	
Prefix	*From*	*To*	*From*	*To*
AA	1	15	NMC 949487	NMC 949501
AA	26	27	NMC 949512	NMC 949513
CR-	1	44	NMC 1028472	NMC 1066246
CR-	47	57	NMC 1078937	NMC 1078947
CT	1	8	NMC 902279	NMC 902286
CT	12	21	NMC 902290	NMC 902299
CT	23	25	NMC 902301	NMC 902303
CT	33	62	NMC 902311	NMC 902340
CT	64	90	NMC 902342	NMC 902368
CT	92	99	NMC 902370	NMC 902377
CTA	5	6	NMC 976759	NMC 976760
CTX	12		NMC 965469	
CTX	14		NMC 965471	
CTX	21	26	NMC 965476	NMC 965481
CTX	31	46	NMC 965484	NMC 965499
JULIE			NMC 949515	
RANDI		1	NMC 949516	NMC 949517
SHERI			NMC 949514	
TG	101		NMC 963449	
TG	103		NMC 963451	
TG	105	136	NMC 963453	NMC 963484
TG	167	173	NMC 963515	NMC 963521
TG	175	184	NMC 963523	NMC 963532
TG	186		NMC 963534	
TG	188		NMC 963536	
TG	190		NMC 963538	
TG	205	228	NMC 963553	NMC 963576
TG	236	243	NMC 963584	NMC 963591
TG	254	267	NMC 963602	NMC 963615
		Total		***288***

b. Patented Claims

Claim Name	Mineral Survey Number
Alberta[3]	USSN 3959
Alice[2]	USSN 4120
Alice.	USSN 4234
Allen No. 2[2]	USSN 3854
Arkansas[2]	USSN 3854
Badger[2]	USSN 4677
Bagger	USSN 4234
Bella No. 13	USSN 4554
Big Hope	USSN 4233
Blue	USSN 4235
Blue Bird	USSN 3959
Blue Lead	USSN 4554
Blue Rock No. 1	USSN 4261
Blue Rock No. 3	USSN 4261
Blue Rock No. 4	USSN 4261
Blue Rock No. 5	USSN 4261
Blue Rock No. 6	USSN 4261
Blue Rock No. 7	USSN 4261
Blue Rock No. 8	USSN 4261
Blue Rock No. 9	USSN 4261
Bobs	USSN 4574
Bonanza No. 5	USSN 4267
Bonanza No. 6	USSN 4267
Bonanza No. 8	USSN 4267
Bonanza No. 9	USSN 4267
Brooklyn[2]	USSN 1972
Brooklyn Ext[2]	USSN 1972
Brooklyn Fr[2]	USSN 1972
Bryan	USSN 3959
Calcite[2]	USSN 1996
Calcite Fr[2]	USSN 1996
Canalise Smith	USSN 4233
Caution[3]	USSN 3959
Columbia[1]	USSN 4584
Columbia Fr[1]	USSN 4584
Comstock[3]	USSN 3959
Copper King[2]	USSN 3854
Copper King 2[2]	USSN 3854
Copper Shield	USSN 4554

Copper Shield No. 2	USSN 4554
Copper Wedge	USSN 4576
Deer[2]	USSN 4677
Delano 1[2]	USSN 3854
Delano 2[2]	USSN 3854
Della	USSN 3959
Demorest[2]	USSN 4677
Deposit	USSN 4241
Diamante[2]	USSN 4593
Effie	USSN 4241
Effie Fay No. 1	USSN 4554
Effie Fay No. 3	USSN 4554
Effie Fay No. 4	USSN 4554
Effie Fay No. 6	USSN 4554
Effie Fay No. 7	USSN 4554
Elizabeth[2]	USSN 4121
Emerald[2]	USSN 4677
Emma[2]	USSN 1999
Emma Fr[2]	USSN 1999
Empire[2]	USSN 1999
Empire Ext[2]	USSN 1999
Estelle No. 1	USSN 4575
Estelle No. 2	USSN 4575
Estelle No. 3	USSN 4575
Estelle No. 4	USSN 4575
Estelle No. 5	USSN 4575
Estelle No. 6	USSN 4575
Gate City[3]	USSN 3959
Grace Copper	USSN 4575
Great Hope	USSN 4575
Green Monster[2]	USSN 1995
Helen B. Smith	USSN 4235
Hidden Treasure[2]	USSN 1999
Highland	USSN 4234
Highland Ext.	USSN 4234
Highland No. 1	USSN 4234
Highland No. 2	USSN 4234
Highland No. 3	USSN 4234
Highland No. 4	USSN 4235
Highland No. 5	USSN 4235
Highland No. 6	USSN 4235

Hornet[2]	USSN 1999
Ilo[2]	USSN 3959
J.G.B. Smith	USSN 4233
Jules Verne[2]	USSN 2003
Junction	USSN 4241
Kentuck[2]	USSN 4117
Lamont[2]	USSN 4118
Lenore	USSN 4122
Lina[2]	USSN 1999
Lincoln[2]	USSN 4233
Lincoln Ext.	USSN 4233
London	USSN 3959
Lucie No. 1	USSN 4239
Lucie No. 11	USSN 4239
Lucie No. 12	USSN 4239
Lucie No. 2	USSN 4239
Lucie No. 3	USSN 4239
Lucie No. 5	USSN 4239
Lucie No. 6	USSN 4239
Lucie No. 7	USSN 4239
Lucie No. 8	USSN 4239
Lucie No. 9	USSN 4239
Lucky Boy No. 2[2]	USSN 4677
Maggie	USSN 4234
Magnolia	USSN 4234
Mammoth	USSN 4235
Mammoth Ext.	USSN 4235
Maresa[3]	USSN 3959
Masher	USSN 4233
Mattie	USSN 4576
McDermott[3]	USSN 3959
Mint	USSN 4586
Monitor	USSN 4554
Monterey[2]	USSN 4120
New York	USSN 4241
New York Ext.	USSN 4241
Old Abe	USSN 4241
Olinda	USSN 4199
Olinda No. 1	USSN 4199
Olinda No. 2	USSN 4199
Olinda No. 3	USSN 4199

Olinda No. 3	USSN 4199
Palo Alto	USSN 3959
Panther City	USSN 4198
Panther City No. 1	USSN 4198
Panther City No. 2	USSN 4198
Portland Fraction	USSN 4259
Portland No. 1	USSN 4259
Portland No. 2	USSN 4259
Portland No. 3	USSN 4259
Portland No. 4	USSN 4259
Portland No. 5	USSN 4259
Portland No. 6	USSN 4259
Portland No. 7	USSN 4259
Portland No. 8	USSN 4259
Potomac[2]	USSN 1999
Rattler	USSN 4233
Reed[3]	USSN 3959
Reliance[2]	USSN 1999
Reliance Ext.[2]	USSN 1999
Round Top No. 1[2]	USSN 4677
Silver Bow[2]	USSN 1999
Smith[3]	USSN 3959
St. Louis No. 2	USSN 4122
St. Louis No.1	USSN 4122
Sunrise	USSN 4241
Valley Veiw[2]	USSN 1972
Viola	USSN 4235
Walter	USSN 4241
Washington[3]	USSN 3959
Waterloo[2]	USSN 4233
Waterloo Ext.	USSN 4233
Wellomo[3]	USSN 3959
Western Union[2]	USSN 3854
Winona Fraction	USSN 4267
Xenophanes[2]	USSN 4119
Total	**156**

Note:
[1] 87.5% Enexco owned(surface & mineral rights), 1.75% NSR
[2] 100% Enexco owned(surface & mineral rights), 0.25% NSR
[3] 100% Enexco owned mineral, <100% Enexco owned surface

APPENDIX C. DRILL HOLE COLLARS

Collar locations are in NAD 83 Nevada State Plane 2701.

Drill Hole	Easting	Northing	Elevation	Year	Hole Type	Company
BK-01	876249	28807412	5834	1971	Core	Calta
C-03	876703	28807533	5837	1971	Core	Calta
C-04	877178	28807442	5801	1971	Core	Calta
CON10-001	878992	28807891	5746	2010	Core	Allied Nevada Gold
CON10-002	878788	28808235	5795	2010	Core	Allied Nevada Gold
CON10-003	877757	28808019	5789	2010	Core	Allied Nevada Gold
CON10-004	876391	28806769	5913	2010	Core	Allied Nevada Gold
CRC-00-1	879780	28809240	5812	2000	Rock Chip	Golden Phoenix
CRC-00-2	880789	28809629	5718	2000	Rock Chip	Golden Phoenix
CRC-04-01	876395	28808360	6126	2004	Rock Chip	Golden Phoenix
CRC-04-02	876246	28808432	6130	2004	Rock Chip	Golden Phoenix
CRC-04-03	876243	28808445	6130	2004	Rock Chip	Golden Phoenix
CRC-98-1	876351	28808247	6152	1998	Rock Chip	Golden Phoenix
CRC-98-2	875881	28807825	5898	1998	Rock Chip	Golden Phoenix
CRC-98-3	875485	28807818	5901	1998	Rock Chip	Golden Phoenix
CRC-98-4	875937	28807707	5858	1998	Rock Chip	Golden Phoenix
CRC-98-5	876542	28808278	6140	1998	Rock Chip	Golden Phoenix
CRC-98-6	876794	28808271	6116	1998	Rock Chip	Golden Phoenix
CRC-98-7	876411	28807801	5952	1998	Rock Chip	Golden Phoenix
CRC-98-8	876760	28807854	5938	1998	Rock Chip	Golden Phoenix
CRC-98-9	873307	28806830	6097	1998	Rock Chip	Golden Phoenix
CRC-98-10	873692	28807018	6062	1998	Rock Chip	Golden Phoenix
CRC-98-11	874052	28807194	6043	1998	Rock Chip	Golden Phoenix
CRC-98-12	874408	28807290	6009	1998	Rock Chip	Golden Phoenix
CRC-98-13	874736	28807544	5970	1998	Rock Chip	Golden Phoenix
CRC-98-14	877832	28808270	5845	1998	Rock Chip	Golden Phoenix
CRC-98-15	878090	28808248	5825	1998	Rock Chip	Golden Phoenix
CRC-98-16	877888	28808069	5790	1998	Rock Chip	Golden Phoenix
CRC-98-17	875068	28807708	5940	1998	Rock Chip	Golden Phoenix
CRC-99-1	873480	28806903	6084	1999	Rock Chip	Golden Phoenix
CRC-99-2	873078	28806825	6118	1999	Rock Chip	Golden Phoenix
CRC-99-3	874058	28807448	6099	1999	Rock Chip	Golden Phoenix
CRC-99-4	874280	28807512	6070	1999	Rock Chip	Golden Phoenix
CRC-99-5	872869	28806593	6198	1999	Rock Chip	Golden Phoenix
CRC-99-6	873491	28807870	6249	1999	Rock Chip	Golden Phoenix
CRC-99-7	873493	28807867	6249	1999	Rock Chip	Golden Phoenix
CRC-99-8	873467	28807168	6146	1999	Rock Chip	Golden Phoenix
CRC-99-9	873768	28807258	6081	1999	Rock Chip	Golden Phoenix
CRC-99-10	874465	28807595	6036	1999	Rock Chip	Golden Phoenix

CRC-99-11	874236	28807606	6088	1999	Rock Chip	Golden Phoenix
CRC-99-12	874857	28808449	6086	1999	Rock Chip	Golden Phoenix
CRC-99-13	875090	28808497	6056	1999	Rock Chip	Golden Phoenix
CRC-99-14	877760	28808362	5875	1999	Rock Chip	Golden Phoenix
CRC-99-15	880000	28809257	5811	1999	Rock Chip	Golden Phoenix
CRC-99-16	880392	28809440	5771	1999	Rock Chip	Golden Phoenix
EK-01	875982	28807702	5852	1967	Core	Calta
EK-02	876073	28807583	5833	1971	Core	Calta
EK-03	876147	28807783	5904	1971	Core	Calta
EN-01	877120	28808276	6014	2007	Core	Enexco
EN-02	877207	28807994	5956	2007	Core	Enexco
EN-03	875933	28808005	5973	2007	Core	Enexco
EN-04	876169	28807967	5991	2007	Core	Enexco
EN-05	876983	28808131	6058	2007	Core	Enexco
EN-06	877085	28807917	5952	2007	Core	Enexco
EN-07	876865	28807850	5941	2007	Core	Enexco
EN-08	876037	28807767	5887	2007	Core	Enexco
EN-09	876320	28807548	5826	2007	Core	Enexco
EN-10	876555	28807590	5857	2007	Core	Enexco
EN-11	875960	28807538	5849	2007	Core	Enexco
EN-12	875960	28807538	5849	2007	Core	Enexco
EN-13	877017	28807529	5838	2007	Core	Enexco
EN-14	876867	28807343	5791	2007	Core	Enexco
EN-15	876373	28807976	6040	2007	Core	Enexco
EN-16	877132	28807305	5773	2007	Core	Enexco
EN-17	877132	28807305	5773	2007	Core	Enexco
EN-18	877647	28807293	5728	2007	Core	Enexco
EN-19	877625	28807295	5727	2007	Core	Enexco
EN-21	873684	28806460	6170	2007	Core	Enexco
EN-22	873684	28806460	6170	2007	Core	Enexco
EN-23	873404	28806864	6092	2007	Core	Enexco
EN-24	873404	28806864	6092	2007	Core	Enexco
EN-25	873404	28806864	6092	2007	Core	Enexco
EN-26	873404	28806864	6092	2007	Core	Enexco
EN-27	873404	28806864	6092	2007	Core	Enexco
EN-28	873164	28806879	6125	2007	Core	Enexco
EN-29	874247	28807497	6070	2007	Core	Enexco
EN-30	874246	28807498	6071	2007	Core	Enexco
EN-31	874247	28807496	6070	2007	Core	Enexco
EN-32	873790	28807639	6161	2007	Core	Enexco
EN-33	873790	28807639	6161	2007	Core	Enexco
EN-34	873179	28807335	6214	2007	Core	Enexco
EN-35	873174	28807342	6214	2007	Core	Enexco

EN-36	873179	28807335	6214	2007	Core	Enexco
EN-37	873774	28807968	6148	2007	Core	Enexco
EN-38	873776	28807960	6161	2007	Core	Enexco
EN-39	873776	28807960	6161	2007	Core	Enexco
EN-40	873768	28807960	6165	2008	Core	Enexco
EN-41	873766	28807970	6148	2008	Core	Enexco
EN-42	875723	28808175	5963	2008	Core	Enexco
EN-43	875722	28808175	5962	2008	Core	Enexco
EN-44	875723	28808174	5962	2008	Core	Enexco
EN-45	875721	28808176	5967	2008	Core	Enexco
EN-46	875721	28808177	5966	2008	Core	Enexco
EN-47	875721	28808177	5966	2008	Core	Enexco
EN-48	875721	28808177	5969	2008	Core	Enexco
EN-49	877671	28807417	5736	2008	Core	Enexco
EN-50	877673	28807411	5731	2008	Core	Enexco
EN-51	875719	28808175	5967	2008	Core	Enexco
EN-52	875430	28808193	5971	2008	Core	Enexco
EN-53	875433	28808190	5969	2008	Core	Enexco
EN-54	875429	28808193	5971	2008	Core	Enexco
EN-55	877379	28807429	5777	2008	Core	Enexco
EN-56	877379	28807429	5777	2008	Core	Enexco
EN-57	877379	28807429	5777	2008	Core	Enexco
EN-58	875108	28808191	5986	2008	Core	Enexco
EN-59	875108	28808191	5986	2008	Core	Enexco
EN-60	876540	28807346	5797	2008	Core	Enexco
EN-61	875107	28808193	5984	2008	Core	Enexco
EN-62	876539	28807346	5799	2008	Core	Enexco
EN-63	874736	28808179	6048	2008	Core	Enexco
EN-64	876539	28807347	5799	2008	Core	Enexco
EN-65	874735	28808182	6043	2008	Core	Enexco
EN-66	875564	28807921	5913	2008	Core	Enexco
EN-67	874243	28808028	6089	2008	Core	Enexco
EN-68	875564	28807922	5914	2008	Core	Enexco
EN-69	874243	28808030	6090	2008	Core	Enexco
EN-71	874410	28807766	6031	2008	Core	Enexco
EN-73	874410	28807768	6039	2008	Core	Enexco
EN-75	874709	28807800	6011	2008	Core	Enexco
EN-77	874710	28807812	6017	2008	Core	Enexco
EN-79	875081	28807947	5993	2008	Core	Enexco
EN-81	875080	28807950	5994	2008	Core	Enexco
EN-83	875371	28808029	5951	2008	Core	Enexco
EN-85	875371	28808029	5955	2008	Core	Enexco
EN-86	875476	28807353	5996	2008	Core	Enexco

EN-87	875275	28807800	5930	2008	Core	Enexco
EN-88	874958	28806093	6100	2008	Core	Enexco
EN-89	875274	28807804	5928	2008	Core	Enexco
EN-90	873985	28807096	6039	2008	Core	Enexco
EN-91	874854	28807642	5974	2008	Core	Enexco
EN-92	873984	28807099	6038	2008	Core	Enexco
EN-93	874848	28807635	5961	2008	Core	Enexco
EN-94	873702	28807023	6060	2008	Core	Enexco
EN-95	874848	28807634	5961	2008	Core	Enexco
EN-96	873700	28807029	6060	2008	Core	Enexco
EN-97	874503	28807390	6001	2008	Core	Enexco
EN-98	874135	28807489	6091	2008	Core	Enexco
EN-99	874504	28807391	6000	2008	Core	Enexco
EN-100	874134	28807490	6092	2008	Core	Enexco
EN-101	874207	28807181	6024	2008	Core	Enexco
EN-102	874784	28806245	6146	2008	Core	Enexco
EN-103	874204	28807186	6025	2008	Core	Enexco
EN-104	873974	28807355	6098	2008	Core	Enexco
EN-105	873974	28807355	6098	2008	Core	Enexco
EN-106	874092	28807700	6098	2008	Core	Enexco
EN-107	874091	28807702	6102	2008	Core	Enexco
EN-108	873350	28807134	6164	2008	Core	Enexco
EN-109	873349	28807136	6164	2008	Core	Enexco
EN-110	873450	28807522	6166	2008	Core	Enexco
EN-111	873490	28807877	6247	2008	Core	Enexco
EN-112	874486	28806034	6179	2008	Core	Enexco
EN-113	873488	28807879	6247	2008	Core	Enexco
EN-114	873254	28807588	6222	2008	Core	Enexco
EN-115	873252	28807592	6222	2008	Core	Enexco
EN-117	873775	28807262	6080	2008	Core	Enexco
EN-118	874746	28806488	6161	2008	Core	Enexco
EN-119	874881	28807447	5977	2008	Core	Enexco
EN-120	874880	28807448	5976	2008	Core	Enexco
EN-121	874725	28807591	5972	2008	Core	Enexco
EN-122	874551	28806666	6155	2008	Core	Enexco
EN-123	874723	28807593	5972	2008	Core	Enexco
EN-124	874366	28807302	6022	2008	Core	Enexco
EN-125	874364	28807303	6020	2008	Core	Enexco
EN-126	875203	28807082	6057	2008	Core	Enexco
EN-127	875078	28807755	5949	2008	Core	Enexco
EN-128	875529	28807791	5892	2008	Core	Enexco
EN-129	875529	28807791	5892	2008	Core	Enexco
EN-131	875247	28808398	6019	2008	Core	Enexco

EN-132	874439	28807545	6038	2008	Core	Enexco
EN-133	875247	28808398	6019	2008	Core	Enexco
EN-134	878043	28807635	5733	2008	Core	Enexco
EN-135	878060	28807357	5697	2008	Core	Enexco
EN-138	870515	28806209	6346	2009	Core	Enexco
EN-139	870517	28806203	6346	2009	Core	Enexco
EN-140	872076	28806723	6194	2009	Core	Enexco
EN-141	872079	28806718	6192	2009	Core	Enexco
EN-142	872078	28806720	6191	2009	Core	Enexco
EN-143	874837	28807002	6113	2009	Core	Enexco
EN-144	874837	28807000	6112	2009	Core	Enexco
EN-145	874826	28806993	6111	2009	Core	Enexco
EN-146	874190	28806806	6110	2009	Core	Enexco
EN-147	878790	28807262	5640	2009	Core	Enexco
EN-148	872844	28806605	6195	2009	Core	Enexco
EN-149	879731	28807360	5613	2009	Core	Enexco
EN-150	879173	28807636	5723	2009	Core	Enexco
EN-151	879173	28807636	5723	2009	Core	Enexco
EN-152	877286	28806519	5863	2009	Core	Enexco
EN-153	877898	28806803	5791	2009	Core	Enexco
EN-154	876042	28808633	6122	2010	Core	Enexco
EN-155	876428	28808531	6088	2010	Core	Enexco
EN-156	876344	28808915	6115	2010	Core	Enexco
EN-157	877861	28807519	5725	2011	Core	Enexco
EN-158	878516	28807340	5666	2012	Core	Enexco
EN-159	878517	28807337	5666	2012	Core	Enexco
EN-160	878330	28807390	5686	2012	Core	Enexco
EN-161	879041	28807164	5632	2012	Core	Enexco
EN-162	879040	28807168	5627	2012	Core	Enexco
EN-163	879280	28807108	5608	2012	Core	Enexco
EN-164	879723	28807015	5600	2012	Core	Enexco
EN-165	879927	28807040	5572	2012	Core	Enexco
EN-166	880322	28807335	5548	2012	Core	Enexco
EN-167	880321	28807336	5548	2012	Core	Enexco
EN-168	880163	28807290	5553	2012	Core	Enexco
EN-169	879938	28807278	5578	2012	Core	Enexco
EN-170	879936	28807282	5578	2012	Core	Enexco
EN-171	880031	28807195	5561	2012	Core	Enexco
EN-172	880573	28807097	5515	2012	Core	Enexco
EN-173	880266	28807557	5570	2012	Core	Enexco
EN-174	880266	28807557	5570	2012	Core	Enexco
EN-175	880266	28807557	5570	2012	Core	Enexco
EN-176	880435	28806793	5577	2012	Core	Enexco

EN-177	879485	28806979	5585	2012	Core	Enexco
EN-178	874637	28807420	5972	2012	Core	Enexco
EN-179	874637	28807420	5972	2012	Core	Enexco
EN-180	878313	28807856	5735	2012	Core	Enexco
EN-66B	875563	28807929	5913	2008	Core	Enexco
ENR-1	879992	28806918	5587	2011	Rock Chip	Enexco
ENR-2	879841	28807180	5586	2011	Rock Chip	Enexco
ENR-3	880084	28807421	5566	2011	Rock Chip	Enexco
ENR-4	880646	28807675	5520	2011	Rock Chip	Enexco
ENR-5	881127	28807822	5514	2011	Rock Chip	Enexco
ENR-6	880165	28807755	5584	2011	Rock Chip	Enexco
ENR-7	879817	28807555	5623	2011	Rock Chip	Enexco
ENR-8	880324	28807336	5546	2011	Rock Chip	Enexco
ENR-9	880560	28806703	5562	2011	Rock Chip	Enexco
ENR-10	879240	28806554	5650	2011	Rock Chip	Enexco
ENR-11	878891	28806502	5674	2011	Rock Chip	Enexco
ENR-12	879641	28807160	5627	2011	Rock Chip	Enexco
ENR-13	879426	28807158	5627	2011	Rock Chip	Enexco
ENR-14	880399	28807210	5533	2011	Rock Chip	Enexco
ENR-15	880551	28807444	5540	2011	Rock Chip	Enexco
ENR-16	880619	28807258	5522	2011	Rock Chip	Enexco
ENR-17	880793	28807334	5510	2011	Rock Chip	Enexco
ENR-18	880725	28807522	5509	2011	Rock Chip	Enexco
ENR-19	880215	28807142	5543	2011	Rock Chip	Enexco
ENR-20	880907	28807462	5495	2012	Rock Chip	Enexco
ENR-21	880521	28807888	5564	2012	Rock Chip	Enexco
ENR-22	880340	28807822	5583	2012	Rock Chip	Enexco
ENR-23	879984	28807667	5607	2012	Rock Chip	Enexco
ENR-24	879982	28807672	5608	2012	Rock Chip	Enexco
ENR-25	880087	28807945	5605	2012	Rock Chip	Enexco
ENR-26	880030	28807190	5562	2012	Rock Chip	Enexco
ENR-27	880028	28807192	5561	2012	Rock Chip	Enexco
ENR-28	879508	28807683	5686	2012	Rock Chip	Enexco
ENR-29	879511	28807678	5686	2012	Rock Chip	Enexco
ENR-30	879181	28807444	5716	2012	Rock Chip	Enexco
ENR-31	879180	28807448	5716	2012	Rock Chip	Enexco
ENR-32	878983	28807618	5711	2012	Rock Chip	Enexco
ENR-33	878981	28807622	5711	2012	Rock Chip	Enexco
ENR-34	878642	28807708	5717	2012	Rock Chip	Enexco
ENR-35	878641	28807710	5717	2012	Rock Chip	Enexco
ENR-36	878558	28807533	5693	2012	Rock Chip	Enexco
ENR-37	878800	28807654	5715	2012	Rock Chip	Enexco
ENR-38	878798	28807658	5716	2012	Rock Chip	Enexco

ENR-39	878741	28807482	5689	2012	Rock Chip	Enexco
ENR-40	878314	28807592	5709	2012	Rock Chip	Enexco
ENR-41	878313	28807596	5710	2012	Rock Chip	Enexco
ENR-42	878467	28807716	5717	2012	Rock Chip	Enexco
ENR-43	878464	28807722	5717	2012	Rock Chip	Enexco
ENR-44	878180	28807484	5706	2012	Rock Chip	Enexco
ENR-45	877927	28807816	5756	2012	Rock Chip	Enexco
ENR-46	877926	28807821	5757	2012	Rock Chip	Enexco
ENR-47	878175	28807719	5729	2012	Rock Chip	Enexco
ENR-48	872970	28806844	6122	2012	Rock Chip	Enexco
ENR-49	872973	28807049	6139	2012	Rock Chip	Enexco
ENR-50	872969	28807058	6140	2012	Rock Chip	Enexco
ENR-51	873932	28806980	6043	2012	Rock Chip	Enexco
ENR-52	873930	28806983	6042	2012	Rock Chip	Enexco
ENR-53	873657	28806831	6080	2012	Rock Chip	Enexco
ENR-54	873655	28806834	6080	2012	Rock Chip	Enexco
ENR-55	873479	28806754	6091	2012	Rock Chip	Enexco
ENR-56	872728	28806957	6147	2012	Rock Chip	Enexco
ENR-57	872728	28806957	6147	2012	Rock Chip	Enexco
ENR-58	874640	28807420	5972	2012	Rock Chip	Enexco
N-01	880223	28806822	5556	1969	Core	Calta
N-02	879275	28806971	5617	1969	Core	Calta
N-03	879462	28806568	5625	1969	Core	Calta
N-04	874736	28807544	5970	1969	Core	Calta
N-05	874672	28807692	5985	1969	Core	Calta
N-06	878617	28806546	5705	1969	Core	Calta
N-07	874508	28808156	6082	1969	Core	Calta
N-08	877739	28806541	5805	1969	Core	Calta
N-09	874130	28807066	6032	1969	Core	Calta
N-10	873395	28806591	6118	1969	Core	Calta
N-11	878720	28806156	5678	1969	Core	Calta
N-19	875747	28806297	5995	1969	Core	Calta
N-22	875637	28807728	5878	1971	Core	Calta
N-23	876275	28808396	6132	1971	Core	Calta
N-24	876148	28807751	5886	1971	Core	Calta
N-25	876593	28808378	6116	1971	Core	Calta
N-26	874856	28807680	5966	1971	Core	Calta
N-27	876468	28807805	5950	1972	Core	Calta
N-29	877881	28808069	5792	1972	Core	Calta
N-30	873467	28807512	6165	1972	Core	Calta
N-31	877224	28807304	5770	1972	Core	Calta
N-12A	875062	28807672	5940	1969	Core	Calta
N-12B	875062	28807672	5940	1969	Core	Calta

N-12C	875062	28807672	5940	1969	Core	Calta
N-12D	875062	28807672	5940	1969	Core	Calta
N-13A	875639	28807729	5876	1969	Core	Calta
N-13B	875639	28807729	5876	1969	Core	Calta
N-13C	875639	28807729	5876	1969	Core	Calta
N-13D	875639	28807729	5876	1969	Core	Calta
N-14A	875465	28808168	5966	1969	Core	Calta
N-14B	875465	28808168	5966	1969	Core	Calta
N-14C	875465	28808168	5966	1969	Core	Calta
N-15A	875814	28807985	5938	1969	Core	Calta
N-15B	875814	28807985	5938	1969	Core	Calta
N-16A	875420	28807605	5909	1969	Core	Calta
N-16B	875420	28807605	5909	1971	Core	Calta
N-17A	875248	28807413	5981	1969	Core	Calta
N-17B	875248	28807413	5981	1969	Core	Calta
N-18A	875500	28806890	6019	1969	Core	Calta
N-18B	875500	28806890	6019	1969	Core	Calta
N-20A	870722	28807469	6480	1969	Core	Calta
N-20B	870722	28807469	6480	1969	Core	Calta
N-21A	876017	28808232	6042	1972	Core	Calta
N-21B	876017	28808232	6042	1971	Core	Calta
N-28A	877320	28807790	5848	1972	Core	Calta
N-28B	877320	28807790	5848	1972	Core	Calta
PD-01	881318	28809857	5692	1973	Core	Phelps Dodge
PD-04	874688	28806473	6173	1973	Core	Phelps Dodge
PD-10	875670	28806506	6011	1974	Core	Phelps Dodge
PD-11	874721	28805487	6151	1974	Core	Phelps Dodge
PD-12	875713	28805578	6007	1974	Core	Phelps Dodge
PD-14	878498	28806022	5722	1974	Core	Phelps Dodge