JOHNSON CAMP MINE PROJECT

FEASIBILITY STUDY

Cochise County, Arizona

TECHNICAL REPORT

PURSUANT TO NATIONAL INSTRUMENT 43-101

OF THE CANADIAN SECURITIES ADMINISTRATORS

Prepared For

NORD RESOURCES CORP.

Prepared By

Bikerman Engineering & Technology Associates, Inc.

Old Lyme, Connecticut

September 2007

BIKERMAN ENGINEERING & TECHNOLOGY ASSOCIATES, INC.

Technical Report / Form 43-101F1

Johnson Camp Mine Project
Feasibility Study

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GLOSSARY OF NON-GEOLOGICAL TERMS AND ABBREVIATIONS

“AIF”	Annual Information Form
“BETA”	Bikerman Engineering & Technology Associates, Inc.
“BLM “	U.S. Bureau of Land Management
“NORD”	Nord Resources Corp.
“CIM”	Canadian Institute of Mining, Metallurgy, and Petroleum
“IBLA”	Internal Board of Land Appeals
“JCM”	Johnson Camp Mine
"TSE"	The Toronto Stock Exchange

GLOSSARY OF TERMS RELATING TO MINERAL PROPERTIES

“AA”	Atomic Absorption analytical method
“AN”	Ammonium Nitrate
“ANFO”	Ammonium Nitrate – Fuel Oil Explosive
"ASL"	Above mean sea level expressed in metres (feet)
“Cu”	Copper
"DDH"	Diamond drill hole
"EM"	Electromagnetic geophysical survey method
“EW”	Electrowinning

"FA"	Fire Assay analytical method
"g/t"	grams per ton
“G&A”	General and Administrative Costs
“gpm”	gallons per minute
“HG”	high grade ore (>0.15%TCU)
“HP”	horsepower
“ILS”	intermediate leach solution
“IRR”	internal rate of return
“lbs”	pounds
“LG”	low grade ore (<0.15%TCU)
“NPV”	net present value
“PLS”	pregnant leach solution
"ppb"	parts per billion
"ppm"	parts per million
“QP”	Qualified Person
“SX”	Solvent Extraction
“TCU”	total copper %

CONVERSION FACTORS

Length
1 micron = 1 micrometer
1 millimeter = 1000 micrometers = 0.0394 inches
1 centimeter = 0.394 inch
1 meter = 3.281 feet
1 kilometer = 0.6214 miles

Area
1 hectare = 100m x 100m = 10,000m2
1 square kilometer = 100 hectares = 247.1 acres = 0.386 square miles

Mass
1 kiloton = 1,000 metric tons
1 metric ton = 0.984 long tons
1 metric ton = 1.1023 short tons (1 short ton = 2,000 lbs)
1 metric ton = 1,000 kilograms
1 kilogram = 1,000 grams
1 kilogram = 2.205 pounds
1 kilogram = 35.274 ounces = 32.151 troy ounces

1 troy ounce = 31.103 grams
1 troy ounce per short ton = 34.286 grams per ton

1 part per million = 1 gram per ton = 1,000 parts per billion
1 milligram = 0.001 gram = 35.274 x 10-6 ounces
1 milliliter = 0.001 liter = 0.352 fluid ounces

3.0 SUMMARY

This report, Johnson Camp Mine Project Feasibility Study, conforms to Form 43-101F1 requirements for technical reports.

NORD Resources Corp. (NORD) of Tucson, Arizona, USA commissioned BIKERMAN ENGINEERING & TECHNOLOGY ASSOCIATES, INC. (BETA), of Old Lyme, CT, USA to complete an independent Technical Report on the copper resource estimation and feasibility for the Johnson Camp Mine copper deposit in accordance with industry standard practices and in compliance with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves and Canadian National Instrument 43-101 (NI 43-101) and, for reserves, in compliance with United States SEC Guide 7.

Nord Resources Corporation ("Nord"), formerly Nord Copper, has produced copper from existing heaps at the Johnson Camp Mine on a limited basis without mining additional ore and later kept the property on a care and maintenance status while seeking capital investment to develop the property. Current Nord operations are on a care and maintenance status. Major facilities consist of two open pits; one waste rock dump; three leach pads; various solution ponds; two raffinate ponds; one solvent extraction (SX) plant; one electrowinning (EW) plant; one tank farm; seven storm water collection ponds; and several production and monitor wells. Buildings include maintenance, laboratory, storage and office complexes.

During production, open-pit mining is performed by drilling and blasting defined ore zones, excavation, truck haulage from the pit to the crushing circuit for crushing and agglomeration, and then overland conveyor transport of prepared ore to the leach pads.

Ore is placed in layers and subjected to leaching using a dilute sulfuric acid solution. Leached copper in solution (Pregnant Leach Solution (PLS)) flows to the leach pad base where it is collected and directed to flow into PLS Ponds. These solutions are transferred to the SX Plant where they are processed through mixer-settlers and mixed first with an organic solution to extract the copper from the PLS, then across a strong sulfuric acid solution to extract the copper from the organic solution to make a rich copper-bearing electrolyte that is transferred to the EW Plant for processing into cathode copper.

Copper-barren raffinate solution from the SX Plant returns to the raffinate ponds, and is pumped back onto the leach pads for use in repeated leaching of existing and newly emplaced copper ores. Make-up sulfuric acid is added to the raffinate as necessary to enhance copper recovery from the leach pads.

Organic solutions within the SX plant are recycled within the SX circuit. Within the EW plant, copper contained within the electrolyte is electrowon onto stainless steel sheets through application of current, and plated copper cathodes stripped off and bundled for shipment offsite.

Water for operations is supplied from area wells and historic mine shafts on the property. Haul roads exist from the pit bases to the leach pad and waste dump locations, and operational roads are located across the property between site facilities, to well locations and to storm water management facilities.

3.1 LOCATION, HISTORY, AND PROJECT DESCRIPTION

The Johnson Camp property is located approximately 65 miles east of Tucson in Cochise County, Arizona, one mile north of the Johnson Road exit off of Interstate Highway 10 between the towns of Benson and Willcox (see Figure 3.1). This is on the eastern slopes of the Little Dragoon Mountain range. The property consists of 59 patented lode mining claims comprising 872 acres, 511 acres of fee simple lands, and 102 unpatented mining claims that total 1,604 acres, all in the Johnson Camp Mining District. Total project area consists of approximately 3,092 acres, or roughly 4.83 square miles.

Since the 1880s the property has been the site of intermittent underground mining and various types of mineral processing, and from 1975 to 1997 supported open pit mining, heap leaching and SX/EW operations. Although mining ceased in 1997, the Johnson Camp leach pads and SX/EW operation remained active until 2003, producing approximately 6.7 million pounds of copper cathode from residual copper in the heaps over the period 1998 to 2003.

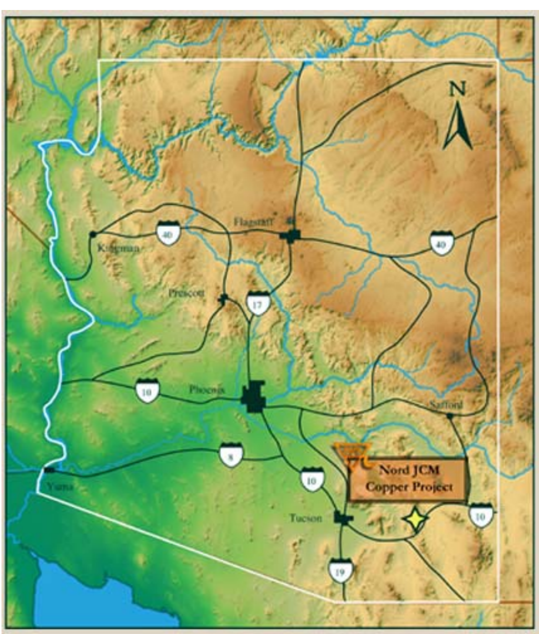


Figure 3-1 – Location Map

3.2 INFRASTRUCTURE

The property currently contains capital improvements in the form of buildings and equipment briefly described as follows:

- Truck shop
- Core storage building
- Office and warehouse
- Laboratory
- Plant mechanical shop

- A 4,000-gpm solvent extraction plant consisting of four extraction mixer/settlers currently operated in parallel and two strip mixer-settlers;
- A tank farm for intermediate storage of electrolyte in front of the tank house;
- A 20 million pound-per-year capacity electrowinning tank house consisting of 74 electrowinning cells with a full complement of cathodes and anodes;
- Four solution storage ponds with a total capacity of approximately 8 million gallons;
- Miscellaneous piping to and from the above facilities;
- A compliment of vehicles, pumps, and other equipment and items.

The above capital items are part of a currently inactive mineral processing facility capable of producing approximately twenty million pounds of copper cathode annually.

The property has supported a nearly continuous active open pit and heap leach mining and processing operation since 1975. Since the 1880's, the property also has been the site of intermittent underground mining and various types of mineral processing.

3.3 GEOLOGY AND MINERALIZATION

Regional Geology

The Johnson Camp mine is located along the east flank of the Little Dragoon Mountains in southeast Arizona. The Little Dragoon Mountains consist of a core of middle Precambrian granites and schists that are unconformably overlain by younger Precambrian metasediments and Paleozoic and Mesozoic sediments. This entire sequence of rocks was intruded by a stock of quartz monzonite composition in late Cretaceous or early Tertiary time, contemporaneous with what is widely referred to as the Laramide orogeny in the southwestern United States. This intrusion has created a contact metamorphic aureole consisting of hornfels and garnet/marble tactite up to several thousand feet in width. Mild folding of the pre-intrusion sediments and metasediments along northwest-trending axes accompanied the Laramide orogeny.

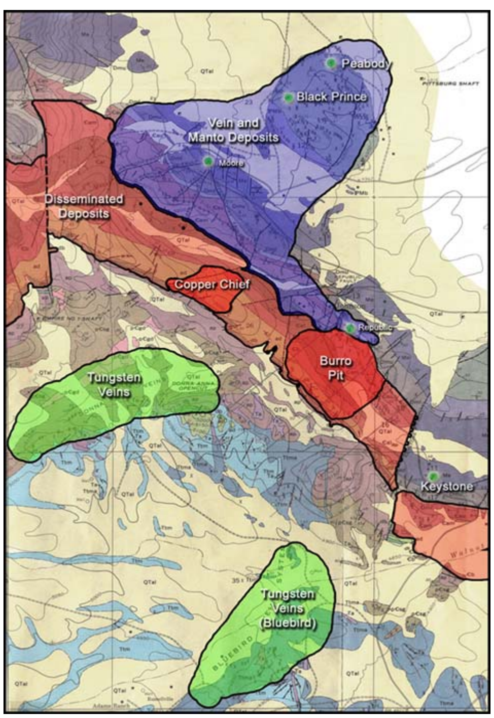
Extensional faulting during the Tertiary and Quaternary periods produced the prominent mountain ranges and intervening alluvium-filled valleys that comprise the topography evident today. Drill hole penetrations in the alluvium indicate that valley fill along the flanks of the mountain ranges exceed 1,000 feet. Thirty to forty degrees rotation of the lithologies accompanied extensional faulting, producing the current bedding orientations measured in the Little Dragoon Mountain area.

Deposit Geology

Large disseminated copper deposits occur in the lower Abrigo Formation and also in the underlying Bolsa Quartzite and Diabase units. The first bulk-mineable deposit was identified by Cyprus Mines Corporation in the 1960's. Because the Bolsa material being mined by an independent leaseholder displayed an impressive amount of oxide copper, a drilling program was launched to define the resource. Drilling by Cyprus outlined the Burro copper oxide reserve, mostly within the Lower Abrigo Formation. Cyprus commenced open pit mining of this deposit in 1975, and subsequently shut down the operation in 1986 due to low metal prices, after having produced approximately 15 million tons of ore grading approximately 0.6 percent total copper.

As can be seen in the Burro pit, copper formed primarily on bedding planes as veins and replacements along with quartz and pyrite, and along fractures which parallel the major fault sets. Extending from the surface to depths ranging from 100 to 150 feet, (but still above the water table), oxide copper consists primarily of copper in limonite and in manganiferous wad. These oxides transition into chrysocolla and malachite which dominate from a depth of 150 feet down to the water table at roughly the 4,560-foot elevation. Some native copper occurs disseminated throughout this range also. Chalcocite appears as coatings on pyrite below the 4,600-foot elevation, and continues as a secondary mineral, replacing sulfides down to the maximum depths drilled. Pyrite, bornite and chalcopyrite, all with chalcocite coatings, are evident below the 4,600-foot elevation, generally increasing in abundance to at least the 4,460-foot elevation.

Arimetco resumed mining the Burro deposit in 1990, after purchasing the property from Cyprus, and produced an additional 15 million tons of material with an average grade of 0.35 percent total copper. This ore came primarily from the Bolsa Quartzite and the diabase sills, which were considered by Cyprus to be too low in grade and were thus left unmined. Copper in these units is found mostly as exotic accumulations on fractures, presumably derived from dissolution of copper in the immediately overlying Lower Abrigo Formation.



Additional mineable material and in-situ resources remain below the current Burro pit bottom in both the Lower Abrigo and Bolsa Formations, and represent the main focus of this study.

Like the Burro deposit, the Copper Chief deposit is a disseminated bulk-mineable copper deposit. It is situated approximately 1,500 feet along strike to the north of the Burro pit, and is hosted primarily by the diabase and Lower Abrigo Formation. Unlike in the Burro deposit, the intervening Bolsa Quartzite is mostly barren. Copper occurs in limonite, goethite, and manganiferous wad, and as disseminations of chrysocolla, malachite, and lesser native copper in the diabase and along fractures within the diabase and the underlying Pioneer Shale down to the water table at approximately the 4,600-foot elevation. Chalcocite, bornite, and chalcopyrite increase in abundance with increasing depth below the water table, as does pyrite.

3.4 MINERAL RESOURCES

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Mineral resources, as reported, include minable reserves, i.e. reserves are a subset of resources.

The Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) defines a Mineral Resource as follows:

"A Mineral Resources is a concentration or occurrence of natural, solid, inorganic, or fossilized organic material in or on the Earth's crust in such a form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge." Using this definition of a Mineral Resource, BETA was able to identify the following resources:

Burro Pit Resource Summary for Total Copper

TCu	Measured			Indicated			Inferred		
Cutoff	Tons	TCu (%)	lbs. Cu	Tons	TCu (%)	lbs. Cu	Tons	TCu (%)	lbs. Cu
0.00	64,833,000	0.28	366,825,100	37,911,200	0.25	191,982,300	16,881,800	0.24	79,952,200
0.10	53,848,000	0.33	355,289,100	29,860,400	0.31	183,880,300	12,582,400	0.30	76,224,200
0.20	39,128,200	0.40	313,260,400	19,889,400	0.39	155,654,400	8,164,200	0.39	63,925,700
0.30	24,848,200	0.49	243,065,100	12,522,400	0.48	119,739,200	5,035,600	0.48	48,261,200
0.40	14,408,000	0.60	171,973,900	7,458,000	0.57	85,095,800	3,269,400	0.55	36,139,900
0.50	8,625,200	0.70	120,890,800	4,065,800	0.68	55,148,500	1,837,600	0.64	23,495,550
0.60	5,302,800	0.80	84,834,200	2,354,200	0.78	36,603,100	975,200	0.73	14,175,500
0.70	3,307,800	0.90	59,229,450	1,489,200	0.85	25,456,400	491,200	0.81	7,994,800
0.80	2,219,800	0.97	43,139,600	779,000	0.96	14,975,500	194,200	0.92	3,587,262
0.90	1,344,400	1.06	28,436,750	433,600	1.07	9,240,000	44,000	1.23	1,079,232
1.00	802,400	1.14	18,264,200	194,000	1.23	4,774,000	32,000	1.35	863,168

Copper Chief Resource Summary for Total Copper

TCu	Measured			Indicated			Inferred		
Cutoff	Tons	TCu (%)	lbs. Cu	Tons	TCu (%)	lbs. Cu	Tons	TCu (%)	lbs. Cu
0.00	65,135,200	0.19	247,513,760	49,139,400	0.16	152,332,140	149,567,400	0.15	433,745,460
0.10	43,012,400	0.26	223,664,480	28,405,200	0.23	128,959,608	86,432,400	0.21	368,202,024
0.20	23,314,000	0.36	167,394,520	12,403,600	0.34	84,592,552	33,258,200	0.33	222,164,776
0.30	12,073,600	0.47	114,216,256	5,009,600	0.49	49,494,848	14,376,200	0.46	133,123,612
0.40	6,568,000	0.58	76,582,880	2,710,400	0.62	33,663,168	6,885,800	0.58	80,426,144
0.50	3,807,800	0.69	52,395,328	1,662,800	0.74	24,443,160	3,672,000	0.71	51,922,080
0.60	2,023,400	0.82	33,143,292	1,042,600	0.85	17,724,220	2,412,000	0.79	38,206,080
0.70	1,275,000	0.92	23,562,000	760,000	0.93	14,120,800	1,368,000	0.91	24,897,600
0.80	819,000	1.02	16,756,740	524,000	1.01	10,616,240	832,000	1.02	16,889,600
0.90	544,000	1.11	12,076,800	316,000	1.12	7,103,680	432,000	1.18	10,221,120
1.00	320,000	1.23	7,852,800	224,000	1.10	4,910,080	296,000	1.29	7,636,800

3.5 MINE RESERVE PARAMETERS

Mining of the Johnson Camp Mine Project is by open-pit methods utilizing mid-size earth moving equipment. Feasible pit shapes complete with haul-road designs have been modeled based on: the disposition of grade values in the resource model; economic parameters such as copper price and mining and operating costs; and technical parameters such as pit slopes and copper recovery.

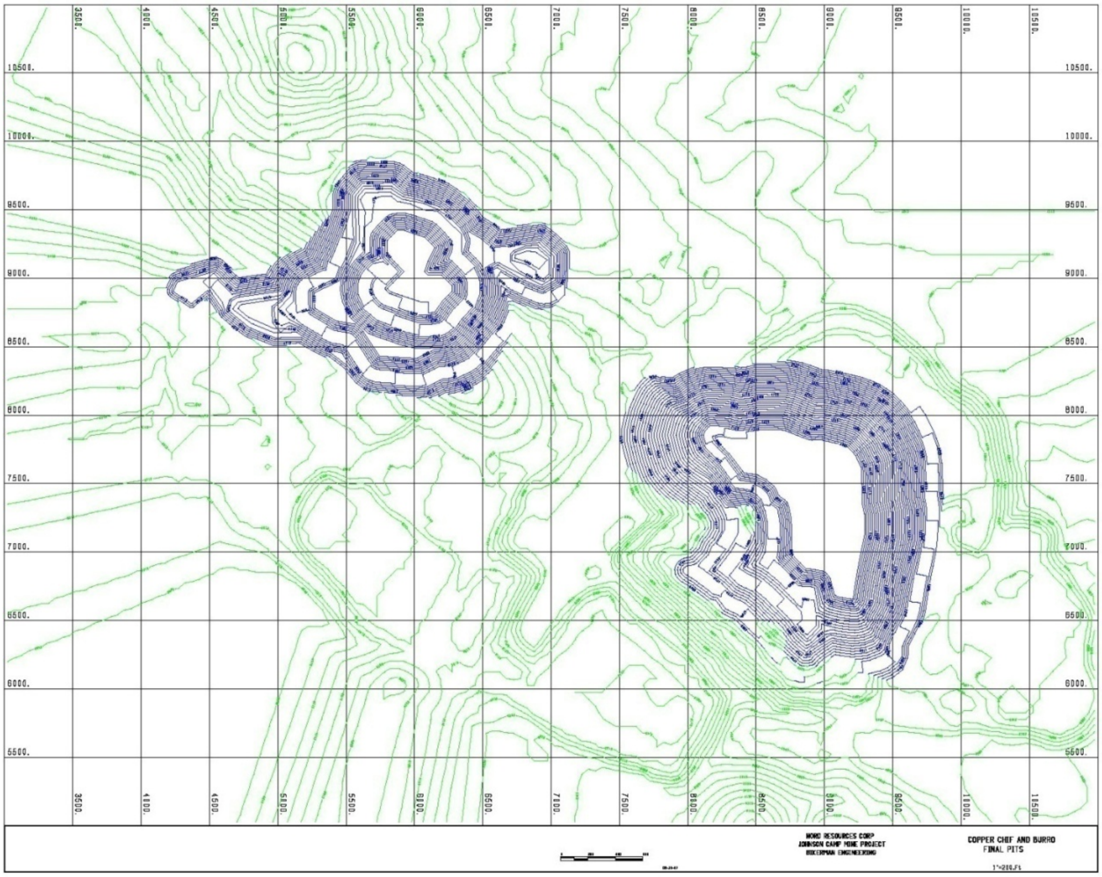
In preparing estimates of proven and probable reserves for the Johnson Camp property, Bikerman Engineering and Technology Associates, Inc. used the geologic resource model and resource estimates prepared by The Winters Company as reported in their feasibility study called "Nord Copper Corporation Feasibility Study, Johnson Camp Copper Project, Cochise County, Arizona", dated March 2000. The Winters Company no longer exists, and was independent to Nord at the time the resource estimates were made. Bikerman Engineering and Technology Associates Inc. reviewed the resource model and estimates as prepared by The Winters Company, and have concluded that they are compliant with the Securities and Exchange Commission Guide 7 and Canadian National Instrument NI 43 101, and are reasonable to incorporate into Bikerman Engineering and Technology Associates, Inc's (September 2007) feasibility study and technical report.

Minable reserves for the Johnson Camp project are based upon the measured and indicated resources in the computerized 3-D block model described herein. Minable pit shapes optimize the extraction of the mineral inventory given the economic and technical parameters determined for this feasibility. The pit optimization procedures utilized in definition of the final pit design take the following factors and assumptions into consideration:

- copper price of $US 1.50 per pound;
- process recovery of contained copper values dependent on rock type;
- mining cost of $US 1.51 per ton of ore moved;
- mining cost of $US 1.61 per ton of waste moved;
- crushing cost of $US 0.637 per ton of ore;
- processing and laboratory cost of $US 0.285 per pound of copper produced;
- G & A and social cost of $US 0.35 per ton of ore;
- environmental cost of $US 0.03 per ton of ore;
- reserves are block diluted;
- overall pit slope of 45 degrees on footwall and 55 degrees on hanging wall;
- minimum pit bottom of 60 feet;
- twenty-foot bench mining heights;
- bench face slope of 63 degrees;
- ultimate haul road grade of no greater than 10%; and
- total haul road width of 80 feet with berms.

A Lerchs-Grossman algorithm was utilized to optimize the pit.

3.5.1 Pit Design



3.5.2 Mineral Reserves

Total proven and probable minable reserves are 73.4 million tons of ore at an average total copper grade of 0.335 percent total copper (TCU), containing 492 million pounds of copper and over 374 million pounds of recoverable copper, minable at a strip ratio of 0.66 to 1. Proven Reserves total 55.0 million tons grading 0.338% containing over 318 million pounds of copper of which 245 million pounds are recoverable. Probable reserves total 18.4 million tons grading 0.327% copper containing 173 million pounds of copper of which 129 million pounds are recoverable. (Table 3-3). The estimate is based on diluted, proven and probable reserves located within the Burro and the Copper Chief pits using a 0.065 percent total copper internal cutoff. Recovery is based on rock type, as discussed in this section, and presented as average recovery by bench. Production and equipment requirements are based on a mining and processing schedule of approximately 25 million pounds of copper produced per year during the mine's 16 year life. The ore reserve estimates are compliant with CIM and SEC Guide 7 guidelines.

Table 3-3
Proven and Probable Minable Reserves

	Tons	Grade	Pounds Copper Contained	Pounds Copper Recoverable
Proven	54,978,000	0.338	318,540,300	245,279,300
Probable	18,410,400	0.327	173,481,600	128,862,500
Proven + Probable	73,388,500	0.335	492,021,800	374,141,700

A low-grade component of the minable reserves will be separated and processed differently. Material grading between 0.065% and 0.150% total copper will be trucked directly to the leach pads and leached run-of-mine. This production schedule, separating low grade ore from crusher grade ore, is shown in Table 3-4.

Table 3-4
Production Schedule

	ORE TO CRUSHING PLANT				LOW GRADE ORE RUN OF MINE TO PADS				WASTE	STRIP	Total
	TONS	TCU	LBS CU	LBS CU	TONS	TCU	LBS CU	LBS CU	TONS	RATIO	Tons
	(000s)		Contained	Recoverable	(000s)		Contained	Recoverable	(000s)		(000s)
YEAR 1	3,331	0.427	28,435	22,154	574	0.099	1,142	571	472	0.12	4,377
YEAR 2	2,590	0.564	29,209	22,369	29	0.107	63	31	199	0.08	2,818
YEAR 3	3,956	0.366	28,949	22,707	590	0.117	1,386	693	1,170	0.26	5,717
YEAR 4	4,628	0.318	29,439	23,041	531	0.068	718	359	7,162	1.39	12,320
YEAR 5	4,524	0.301	27,220	21,594	1,834	0.098	3,611	1,806	7,911	1.24	14,269
YEAR 6	4,788	0.291	27,847	21,628	1,712	0.104	3,544	1,772	5,174	0.80	11,674
YEAR 7	4,871	0.316	30,770	24,279	637	0.113	1,441	721	1,888	0.34	7,396
YEAR 8	4,100	0.383	31,366	24,707	242	0.121	586	293	651	0.15	4,993
YEAR 9	3,396	0.457	31,009	24,789	188	0.112	422	211	277	0.08	3,862
YEAR 10	3,376	0.481	32,491	24,964	38	0.095	72	36	83	0.02	3,497
YEAR 11	3,355	0.477	32,025	24,949	32	0.158	101	51	164	0.05	3,550
YEAR 12	4,319	0.374	32,275	24,181	769	0.106	1,637	819	5,201	1.02	10,289
YEAR 13	4,411	0.353	31,144	23,436	1,489	0.105	3,127	1,564	8,636	1.46	14,537
YEAR 14	4,682	0.341	31,967	24,064	918	0.102	1,872	936	6,508	1.16	12,108
YEAR 15	4,360	0.372	32,443	24,245	694	0.109	1,510	755	3,130	0.62	8,183
YEAR 16	2,007	0.330	13,256	9,944	416	0.114	946	473	596	0.25	3,019

Ore will be mined from two pits – the Burro and Copper Chief pits. A summary of proven and probable minable reserves by pit are presented separated by grade category in the following tables.

Table 3-5
Crusher grade (>0.15%) Proven and Probable Minable Reserves

	Tons	Grade	Pounds Copper Contained	Pounds Copper Recoverable
Burro	38,417,500	0.399	306,818,400	239,418,300
Copper Chief	24,276,000	0.335	162,528,500	123,385,900
Proven + Probable	62,693,400	0.374	469,346,900	362,804,250

Table 3-6
Low Grade (<0.15%) Proven and Probable Minable Reserves

	Tons	Grade	Pounds Copper Contained	Pounds Copper Recoverable
Burro	5,627,700	0.104	11,721,900	5,860,900
Copper Chief	5,065,750	0.108	10,953,100	5,476,500
Proven + Probable	10,693,400	0.106	22,675,000	11,337,500

3.5.3 Mine Operating Cost

Table 3-7
Operating Cost Summary

Production	Operating Costs					
	G & A	Ore	Waste	Crush & Stack	Process	Environment
(lb/year)	$US/ton Ore	$US/ton Ore	$US/ton Waste	$US/ton Ore	$US/lb Cu	$US/ton Ore
25,000,000	0.350	1.509	1.603	0.637	0.285	0.035

3.5.4 Mine Capital Costs

Table 3-8
Capital Cost Summary

Cost (000)	Total
Initial plant capital cost, $	$ 26,684
Mine software, hardware & surveying equipment, $	$ 125
Environmental monitoring, $	$ 240
Removal of Pad 1 liner, $	$ 620
New leach pads, $	$ 11,540
Infrastructure for conveyor relocation, $	$ 100
Mine contractor demob, $	$ 375
Plant sustaining capital, $	$ 450
Total Capital, $	$ 40,134

3.6 METALLURGY AND PROCESSING

Currently, the existing Johnson Camp leach dumps are being managed to control pond inventories and there is no copper production. Sections of the SX/EW facility are under rehabilitation that must be completed prior to restarting cathode production.

Nord intends to resume mining and leaching to produce approximately 25.0 million pounds of copper per year. In order to resume full operation, Nord will reline an existing solution pond, construct three new lined ponds, will prepare a new stand-alone lined leach pad facility for approximately 60 percent of the new ore that will be leached and will install a three-stage crushing circuit. The SX/EW plant will be rehabilitated to meet production goals and the EW section expanded.

Copper production will originate from both an active leach program of newly mined ore and the residual leaching of the existing old dumps. Dependent on TCu grade, the newly mined ore will be divided into two sub-categories, i.e., the higher grade HG ore (+0.15% TCu recoverable) will be crushed and subsequently stacked, by conveyors. The low grade LG ore at approximately 0.1% average TCu recoverable will be direct truck dumped on the existing leach pads. Drawing 8304-C-001 is a plot plan for the overall project. Both crushed HG and LG leach ore will be placed on top of the old heaps during start-up. They will be stacked in separate areas of the existing leach pads. NORD will immediately commence scheduling the ore deposition plan. This plan will define the timing of the construction of the new leach pads and their required size. It is NORD's plan to stack the LG ore exclusively on the existing pads and stack the majority of the HG crushed ore on the new proposed leach pad. It is anticipated that the existing leach pads, with the implementation of this stacking regime, will be continuously under leach. Leaching and subsequently rinsing of these existing leach pads will continue as long as an economic PLS grade is produced.

The operating plan for the LG ore is simply extraction and direct truck haulage from the mine for dumping on the existing leach pads. Recoveries from this LG ore are projected to be in excess of +50%.

The operating plan for the HG ore includes mining, crushing this ore to minus one-inch, acidulating and drum agglomerating the crushed ore with sulfuric acid, and conveying the acidulated ore through a series of movable conveyors to the new leach pad. That ore will be acid–cured with a 144-gram-per-liter raffinate solution before conventional leaching commences. The ore will be stacked in 30-foot lifts on both the old heaps and the new pad. This new ore will be leached with a combination of low-grade leach solution (intermediate leach solution – ILS) and raffinate. The highest grade PLS from the new leach pad system will report to the SX plant. Raffinate from the SX plant will be applied to the existing old leach dumps and LG ore for both new and residual copper recovery. Copper will be recovered from the PLS solution utilizing the existing SX circuit and cathode copper will be produced from the expanded EW circuit using stainless steel blanks. In the past, the EW plant has produced copper of five 9s (99.999 percent copper) quality. The future operation will be at low current densities (22 to 23 amps per square foot) as compared to most operations and this should continue to ensure a high cathode quality.

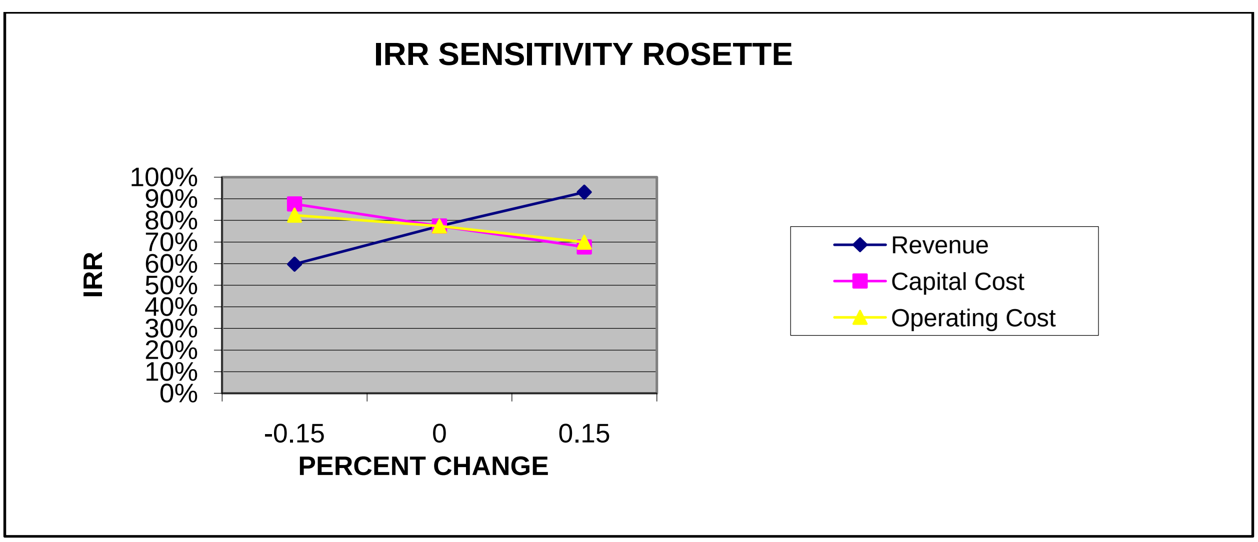
The new, lined leach pad will be subdivided into several smaller areas so that solution from the dumps can be segregated by copper grade. This will provide the operation with flexibility in managing PLS solution flows and grades. The stacking/loading plan remains pending. The circulation of solutions through the old heaps and LG ore, to the new heaps, will also aid in producing/controlling the PLS grade.

3.7 PROJECT ECONOMICS

BETA ran cash flows and sensitivity analyses to determine the project's net present value and internal rate of return under varying assumptions. The Base Case analysis utilizes a copper price of $2.45 per pound. BETA performed cash flow analyses for changes of plus and minus fifteen percent to the Base Case for Revenue (copper price), Capital costs, and Operating cost.

Figure 3-1 shows the Base Case results and the impact of changes of plus and minus 15% to revenue, capital and operating cost to the internal rate of return. This table is summarized as follows: The IRR of the base case is 77%. A 15% decrease in capital cost results in an IRR of 88%, whereas an increase of 15% in capital results in an IRR of 68%. A 15% decrease in operating cost results in an IRR of 82%, whereas an increase of 15% in operating cost results in an IRR of 70%. A decrease in copper price of 15% from $2.45 to $2.13 results in an IRR of 60%, whereas an increase of 15% to $2.82 results in an IRR of 93%.

Figure 3-1
IRR Sensitivity Rosette



BETA performed a further sensitivity analysis reflecting project economics at higher copper prices approaching current price levels. Results of runs including copper prices of $3.19 and $3.55 per pound are shown below.

Table 3-9
Sensitivity Analysis – IRR versus Copper Price

	IRR				
Cu Price	$ 2.13	$ 2.45	$ 2.82	$ 3.19	$ 3.55
IRR	60%	77%	93%	108%	122%

Figure 3-2 shows the impact of changes of plus and minus 15% to revenue, capital and operating cost to the net present value at an 8% discount rate. This graph is summarized as follows: The NPV of the base case at 8% discount rate is $176.4 million. A 15% decrease in capital cost results in an NPV of $180.3 million, whereas an increase of 15% in capital results in an NPV of $171.3 million. A 15% decrease in operating cost results in an NPV of $202.2 million, whereas an increase of 15% in operating cost results in an NPV of $149.5 million. A decrease in copper price from $2.45 to $2.13 results in an NPV of $117.7 million, whereas an increase to $2.82 results in an NPV of $242.7 million.

Figure 3-2
NPV Sensitivity Rosette

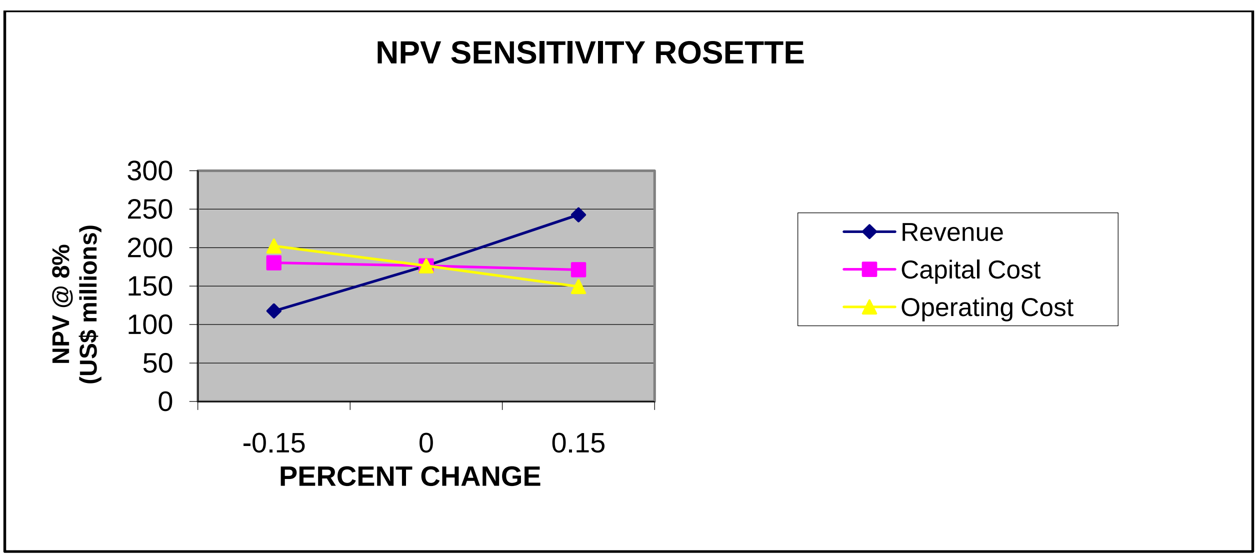


Table 3-10
Sensitivity Analysis – NPV versus Copper Price
($ millions)

NPV @ 8%										
Cu Price	$	2.13	$	2.45	$	2.82	$	3.19	$	3.55
NPV @ 8%	$	117.7	$	176.4	$	242.7	$	309.6	$	376.6

3.8 OTHER RELEVANT DATA

Working capital is estimated at $683,000, to be repaid at the end of mine life.

The financial projection assumes a salvage value of the mining, process and service equipment of $2,512,000.

Mine closure costs have been estimated to total $1,850,000.

A one percent (1%) marketing and delivery charge as applied to all copper sales.

The Arimetco royalty cost is calculated as $0.02 per pound produced when the copper price is equal to or greater than $1.00 per pound, subject to a cap of $1 million in aggregate.

3.9 BETA COMMENTS AS INDEPENDENT REVIEWER

Overall, the feasibility study report addresses all of the topics that need to be addressed for a full feasibility study, and is in compliance with NI 43-101.

BETA is of the opinion that the mineral resource and mineral reserve statements included in this report are accurate, well with normal limits required by a feasibility study. BETA reviewed the resource block model, was satisfied with overall results of the review and incorporated the resource block model into this report.

4.0 INTRODUCTION AND TERMS OF REFERENCE

This report, entitled Johnson Camp Mine Project Feasibility Study 2007, conforms to Form 43-101F1 requirements for technical report. As of this writing, NORD has made the decision to commence development activities as soon as possible for the Johnson Camp Mine Project. In order to facilitate expedited financing, NORD requested that a NI43-101 report be prepared by an independent consultant.

NORD Resources Corp. (NORD) of Tucson, Arizona, USA commissioned BIKERMAN ENGINEERING & TECHNOLOGY ASSOCIATES, INC. (BETA), of Old Lyme, CT, USA to complete an independent Technical Report on the copper resource estimation and feasibility for the Johnson Camp Mine copper deposit in accordance with industry standard practices and in compliance with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves and Canadian National Instrument 43-101 and United States SEC Guide 7.

The Qualified Persons (QP) responsible for the preparation of this report are: Dr. Michael Bikerman, P.Geo, David Bikerman, Eng. Mines, and Thomas McGrail, Eng. All QPs are independent to Nord under NI 43-101.

Dr. Michael Bikerman, David Bikerman, and Tom McGrail conducted a site visit of the Johnson Camp Mine and inspected related documentation in Nord's Tucson offices from June 12-15, 2007

BETA acknowledges that this report and other technical information will be presented by NORD for the acquisition of financial resources, and to fulfill the requirements under Canadian securities law.

NORD supplied documentation as noted within the report which forms the basis of significant portions of this report. Over the years, a number of engineering companies and consulting firms have completed studies that have been reviewed by BETA and incorporated in this Feasibility Study. These and other studies have also supplied baseline and technical information for the permitting documents. The complete reports are available for inspection as technical-supporting documents to this report at the offices of Nord Resources Corporation, 1 W. Wetmore, suite 203, Tucson, Arizona, 85705, USA..

5.0 RELIANCE ON OTHER EXPERTS

In compiling this Feasibility Study for the Johnson Camp Project, BETA relied on work of its own origin (ore reserve estimates, mine schedules, portions of the processing capital cost estimate, economic analysis) and reviewed the estimates and conclusions of several third party consultants and engineering companies. After reasonable due diligence, BETA has accepted and incorporated portions of these reports, third party estimates and conclusions as part of the basis of this study as BETA deemed warranted. The data, maps, and other records on which these estimates are based have been, after careful review, accepted by BETA as true and accurate.

BETA has assumed that Nord and their third party consultants and engineers did not withhold from BETA any facts or information requested by BETA. BETA relied on the work of experts in areas that are not specifically in the realm of expertise of the authors. The work relied upon, and the details of such, are tabulated below.

1. Environmental and Permitting. BETA has not audited the status of the permitting process at the Johnson Camp Mine and has relied upon the estimates and conclusions of Dale A. Deming, PE.

2. Taxes – BETA did not employ a tax specialist in preparation of the cash flows, and as such presents the pre-tax cash flow implications herein and therefore relies on the description of taxes as prepared by Nord. .

3. BETA has not audited the land status and has relied upon the title opinions performed by others, the most recent of which was performed by John C. Lacy of DeConcini McDonald Yewtin & Lacy, Attorneys at Law of Tucson, AZ.

6.0 PROPERTY DESCRIPTION AND LOCATION

6.1 LOCATION

The Johnson Camp property is located approximately 65 miles east of Tucson, in Cochise County, Arizona. The JCM mine site is approximately one mile north of Interstate Highway 10 between the towns of Benson and Willcox (see Figure 6.0-1). The Johnson Camp Mine property is located 15 miles east of the town of Benson and one mile north of Interstate Highway 10 in Cochise County, Arizona. The average elevation at the site is about 5,000 feet above sea level.

6.2 PROPERTY DESCRIPTION

Since the 1880s the property has been the site of intermittent underground mining and various types of mineral processing, and from 1975 to 1997 supported open pit mining, heap leaching and SX/EW operations. Although mining ceased in 1997, the Johnson Camp leach pads and SX/EW operation remained active until 2003, producing approximately 6.7 million pounds of copper cathode from residual copper in the heaps over the period 1998 to 2003.

The property consists of 59 patented lode mining claims comprising 872 acres, 511 acres of fee simple lands, and 102 unpatented mining claims that total 1,604 acres, all in the Johnson Camp Mining District. Total project area consists of approximately 3,092 acres, or roughly 4.83 square miles. No material issues of conflict regarding Nord's land position are known to exist.

Aside from the land summarized above, the property currently contains capital improvements in the form of buildings and equipment briefly described as follows:

- Truck shop
- Core storage building
- Office and warehouse
- Laboratory
- Plant mechanical shop
- A 4,000-gpm solvent extraction plant consisting of four extraction mixer/settlers currently operated in parallel and two strip mixer-settlers;
- A tank farm for intermediate storage of electrolyte in front of the tank house;
- A 20 million pound-per-year capacity electrowinning tank house consisting of 74 electrowinning cells with a full complement of cathodes and anodes;

- Four solution storage ponds with a total capacity of approximately 8 million gallons;
- Miscellaneous piping to and from the above facilities;
- A compliment of vehicles, pumps, and other equipment and items.

The above capital items are part of a currently inactive mineral processing facility capable of producing approximately twenty million pounds of copper cathode annually. The property has supported a nearly continuous active open pit and heap leach mining and processing operation since 1975. Since the 1880's, the property also has been the site of intermittent underground mining and various types of mineral processing.

6.2.1 Property Royalties and/or Encumbrances

The only royalty and/or encumbrance known to BETA is the Arimetco royalty. This royalty is calculated as $0.02 per pound produced when the copper price is equal to or greater than $1.00 per pound, subject to a cap of $1 million in aggregate.

6.3 MINERAL DISPOSITIONS

6.3.1 Land Status

The land status of the Johnson Camp project is described in reports prepared by an independent lands specialist contracted by Summo to review the Johnson Camp lands during that company's due diligence period in 1998-1999. These reports are available for inspection at Nord Resources Corporation's head office Based on these reports, and additional information, the current land status of the Johnson Camp project can be summarized as follows:

The property consists of 59 patented lode mining claims comprising 872 acres and fee simple lands comprising 511 acres, surrounded by 102 unpatented lode mining claims that total 1,604 acres (see Figure 3.3-1). All existing mining operations and surface facilities are situated on either the patented mining claims or the fee simple lands. Claim maintenance filing fees have been paid covering the 88 unpatented lode mining claims through the Assessment Year ending September 1, 2005. Most of these properties were acquired by Arimetco from Cyprus Mines Corporation by two Special Warranty Deeds dated August 8, 1989 and November 15, 1991, and a Quitclaim Deed dated August 8, 1989. Additional fee simple lands were acquired by Nord Copper by a Special Warranty Deed dated June 7, 1999, from Cyprus Mines Corporation. In addition to the real estate included in the property are several access rights-of-way, four water wells, and an agreement which allows ingress and egress to the well located in Section 19. Land Status Map. Nord Copper was merged into Nord Resources Corporation in February 2001.

In an April 1999 report on the Johnson Camp Project that was prepared for Summo, several possible issues of conflict regarding the Johnson Camp land status were noted. Two of these issues

have been resolved. These include the Cyprus/Arimetco Boundary Dispute in Township 15 South, Range 22 East, Section 25, Lots 15,16, and 19, which was resolved by the June 7, 1999 Special Warranty Deed to Nord Copper from Cyprus, and the issue of delinquent Cochise County property taxes, which was resolved by Arimetco before the property was transferred to Nord Copper. All of the remaining land conflict issues are minor, and none of these disputes affect the current operations or planned operations activities in the foreseeable future.
These issues are summarized as follows:

6.3.2 Claim Overlap

There is a limited amount of claim overlap in the extreme northern portion of the property where a group of claims, junior to the claims of the Johnson Camp property, extend over the perimeter boundary of the Johnson Camp claims. The overlapping portions of these third party junior claims are invalid.

6.3.3 State/BLM Dispute

Several of the unpatented claims in Section 36 lie on land involved in a dispute between the State of Arizona and the U.S. Bureau of Land Management (BLM). The majority of these claims were staked around 1900. The State of Arizona contends that all of Section 36 was granted to the State upon its entry into the Union (February 12, 1912). The BLM contends that the portion of Section 36 covered by these unpatented mining claims was not conveyed to the State. Mineral Patent No. 02-73-0024 was issued by the BLM to the State of Arizona on September 1, 1972 for that portion of Section 36 not covered by pre-existing mining claims. This patent was appealed (by the State), but the Internal Board of Land Appeals (IBLA) dismissed the appeal.

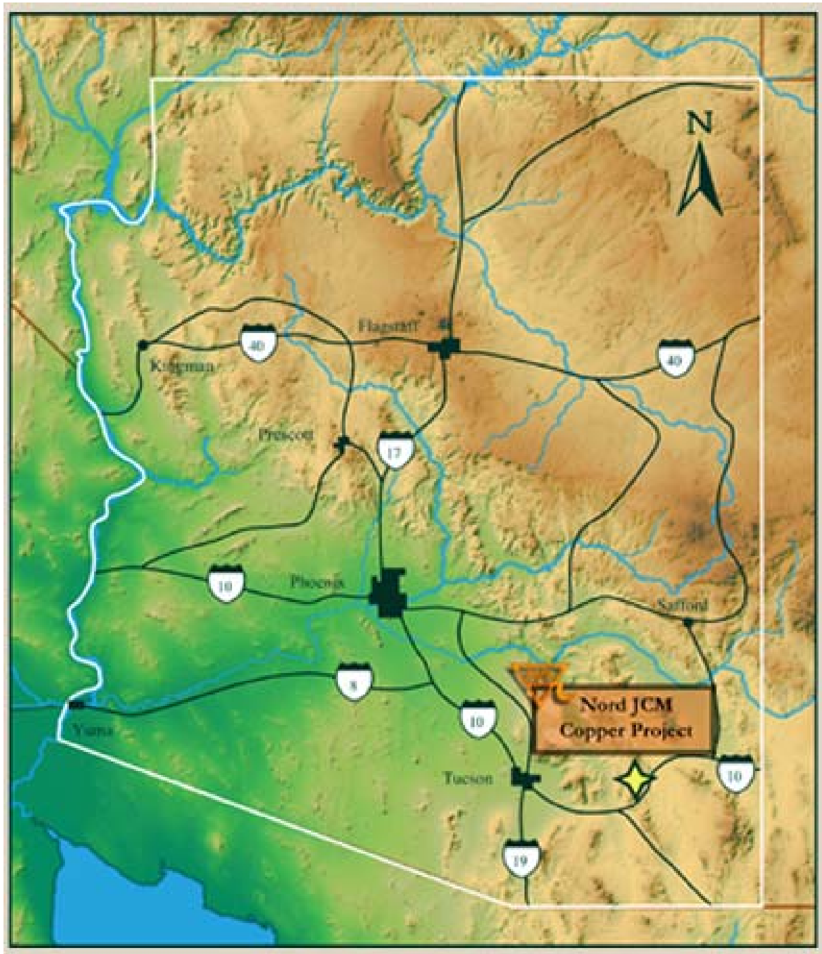
In practice, the State has treated the entire section as if it were State land since 1912, including issuing a State prospecting permit for the entire section some years ago. This permit was later rescinded. According to current State administrators, the State Lease now held by James Sullivan covers only the patented acreage, and specifically excludes the disputed area.

6.3.4 State Fractions

Within the disputed area and adjacent to the St. George patented claim, are what appear to be two small fractions of State land. During its tenure, Arimetco did not have a State lease covering this ground, but because it was within Arimetco's "permitted area", the State did not require a lease

Figure 6-1

Location Map



6.4 SITE DESCRIPTION

6.4.1 Existing Environmental Liabilities

This topic is completely discussed in Section 23: ENVIRONMENTAL PERMITTING and more specifically in Sub-Section 23.2.

6.4.2 Permitting Requirements

The permitting process for mine construction and development is discussed in Section 23: ENVIRONMENTAL PERMITTING and more specifically in Sub-Section 23.3.

7.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

7.1 ACCESSIBILITY

The Johnson Camp Mine property is located 15 miles east of the town of Benson and one mile north of Interstate Highway 10 in Cochise County, Arizona. Due to its location just one mile north of Interstate 10 (the major east-west transportation route across the southern U.S.), the Johnson Camp site provides excellent access for transportation and delivery of bulk supplies and shipment of copper cathodes. Major bulk consumables are readily available from nearby Tucson. Sulfuric acid is produced at three copper smelters within a 200-mile radius, including Asarco at Hayden (127 miles to the north), Phelps Dodge at Miami (167 miles to the north), and Grupo Mexico at Nacozari (130 miles to the south). Diesel fuel is readily available from Willcox, 17 miles to the east on Interstate 10.

Additionally, the mine is situated in close proximity to the Union Pacific Railway mainline through Dragoon; this has previously provided Nord with the option of direct shipping cathode to customers by truck or rail.

7.2 CLIMATE AND METEOROLOGY

The average elevation at the site is approximately 5,000 feet above sea level. The climate of the region is arid, where summers are hot and winters are cool.

7.2.1 Precipitation and Evaporation

Climatic data gathered by Cyprus during its operating period shows average annual precipitation to be approximately 15 inches, ranging from a low of 10 inches to a high of 25 inches. Precipitation occurs mostly as rain, although small accumulations of snow can occur during the winter months (December through March). These accumulations typically melt within 24 hours. Winter precipitation provides one-quarter to one-half of the annual precipitation. The seasonal monsoon rains of July through September provide another half to one quarter of the precipitation. The remainder is scattered throughout the year in small rain showers.

7.2.2 Temperature

The mean maximum annual temperature at the site averages 75 degrees Fahrenheit, with June through August averaging 94 degrees as the hottest months and December through March as the coldest months, averaging 62 degrees. The mean minimum temperature during the winter months is 38 degrees. Freezing rarely occurs on site.

7.2.3 Wind

Winds blow primarily from the west to the east, as part of weather patterns derived from either the California coast or the Gulf of California, Mexico. Monsoon weather patterns in late summer come up from the Gulf of Mexico and generally sweep northward across the property.

7.3 LOCAL RESOURCES

Demographics

The primary impact area, of the project, encompasses a small residential community of approximately 20 family homes near the mine site and the communities of Benson and Willcox, all in Cochise County, Arizona. The populations of Benson and Willcox are approximately 5,000 and 4,000 residents respectively. The area has been the site of on-going mining operations for more than 140 years. The reopening of the Johnson Mine Camp project will assuredly provide employment opportunities but will not significantly impact the demographics of the area.

Mine Labour Force

This topic is discussed in Section 24: Human Resources and Administration.

Economy and Infrastructure

In addition to the mines operating in the area, the economies of the communities within the project area are primarily cattle ranching and agriculture. Irrigated farming is evident in the Willcox area. Tourism is also evident as Benson acts as the gateway to the Kartchner Caverns State Park

Health Services

Each of Wilcox and Benson has a small community hospital and both Benson and Wilcox have resident doctors, for primary medical care. Tucson, however, is the nearest medical center for advanced health care requirements.

7.4 EXISTING LAND USES

Currently, a contractor is re-processing the waste dump to supply decorative stone and aggregate to the Tucson and southern Arizona area.

Cattle are grazed actively throughout the property and adjacent lands, and currently dominate the faunal assemblage in the area. Vegetation found on the property is typical of the upper Sonoran Desert and intermediate elevations of the Basin and Range country of Arizona. The basins and lower slopes support bunchgrasses and a variety of other plants, including mesquite, catclaw, greasewood, yucca, Spanish bayonet, prickly pear, and cholla cactus. At higher elevations there are live oak and juniper, with dense stands of pinion pine common on north-facing slopes.

Wildlife across the area is diverse and consists of a wide variety of small birds (including sparrows and thrushes), small mammals (including cottontail and jackrabbits, mice, and pack rats), and larger mammals (including mule deer, coyotes, and mountain lions) in the surrounding mountain areas.

7.4.1 Mineral Exploration and Development

This topic is covered extensively in Section 8: History and in Section 15: Exploration and Development.

7.5 PHYSIOGRAPHY

This is on the eastern slopes of the Little Dragoon Mountain range. The average elevation is approximately 5,000 feet above sea level and the region is arid and demonstrates the characteristics of the northern section of the Sonoran Desert. Dendritic drainage patterns are evident in aerial photographs of the area. The nearest lake of any significance is located at Willcox.

8.0 HISTORY

The Johnson Camp project area has been an active mining district for more than 100 years. The following summary of the history of mining at Johnson Camp through 1957 was compiled from United States Geological Survey (USGS) Professional Paper 416 (1964), Geology and Ore Deposits of the Dragoon Quadrangle, by J.R. Cooper and L.T. Silver.

The earliest known mining in the Johnson Camp area was done by Mexican miners prior to 1880. By the time of the completion of the Southern Pacific Railway in 1881 through Dragoon just south of Johnson Camp, a number of mining claims had been patented, including the Peabody, Republic, and Mammoth claims, which are all part of the current Johnson Camp property. The Peabody mine (owned by the Russell Gold and Silver Mining Company of Philadelphia) near the present location of the Black Prince shaft on the north end of the Johnson Camp property was one of the earliest producers, supporting a small smelter. The mine eventually closed, only to be purchased and reopened in 1899 by the Dragoon Mining Company, a subsidiary of the Federal Copper Company of New York. In 1907 the Bonanza Belt Copper Company assumed control of the mine, followed a short time later by the Consolidated Copper Company. After closure, the mine remained idle until 1957, when it was purchased by the Coronado Copper and Zinc Company, operators of the nearby Republic and Moore mines. Subsequent to 1902, recorded production averaged 7.4 percent copper and 4.2 ounces of silver per ton.

The second producer of significant size in the Johnson Camp area was the Black Prince Copper Company, which was formed by Denver-based investors in 1901. By 1903 the company was proceeding with underground development of the Republic and Mammoth mines beneath existing surface cuts to the south of the Peabody mine. After relinquishing control of the Republic and Mammoth mines to Arizona Consolidated Mining Company, the company commenced sinking of the Black Prince shaft in 1905, completing it to a depth of almost 1,000 feet by 1911.

With the Black Prince shaft in progress and the Republic and Mammoth mines in operation, access to the Johnson Camp area improved. In 1906 the Johnson Dragoon and Northern Railroad Company began construction of a standard-gauge railroad spur between Johnson Camp and the Southern Pacific main line at Dragoon. The spur was completed in November 1909, but saw limited use prior to World War I. That same year the Arizona Consolidated Mining Company was reorganized as the Arizona United Mining Company. Soon after, the company constructed a 125-ton smelter at the Republic mine to treat low-grade sulfide ore. However, smelting of the low-grade ores was unsuccessful due to lack of suitable flux, and the smelter was soon abandoned.

The period during World War I saw a resurgence of mining at Johnson Camp, with the railroad spur contributing to the success of a number of mines in the area. By 1916, Johnson Camp had a population of approximately 1,000, and more than 80,000 tons of ore were shipped via rail from the Republic mine and four or five smaller mines.

The Copper Chief mine was one of these, reaching its production peak during this period when it produced approximately ten percent of the Republic mine's total. The Peabody mine continued to contribute to the total production, and small amounts of copper were produced from the Keystone, Black Prince, and Johnson Copper Development operations.

In 1920, the fall in the price of copper forced the Republic mine to close, along with the smaller operations. Deprived of ore from its biggest customer, the railroad spur closed down, and in 1925, the tracks were removed. For the next 20 years, almost no mining took place in the district. The Republic mine was allowed to flood, and Johnson Camp was abandoned.

The Coronado Copper and Zinc Company, a wholly owned subsidiary of Cyprus Mines Corporation, dewatered the Republic mine in 1942 while operating under a lease and purchase option. After significant exploration work was completed, the company exercised the option to purchase the mine and constructed a 200-tpd selective flotation concentrator at the mine site. Beginning in May 1945, the mill operated almost without interruption for 12 years, processing ore from the Republic mine (1945-1952), the Mammoth mine (1945-1949), and the Moore mine (1951-1957). In 1949, Coronado Copper and Zinc added the Black Prince mine to its collection of properties. In 1957, low metal prices forced the closure of the Moore mine and the mill.

Cyprus Mines Corporation maintained the property through the late 1950's and 1960's, leasing portions of the property to various contractors. These included McFarland and Hullinger, who resumed underground mining in the Moore mine, and Ira Mosely, who mined oxide copper in silica for smelter flux in an open footwall cut at the Republic mine. In the mid to late 1960's, Cyprus became very interested in the oxide copper exposed in the open cut being mined by Mosely, and commenced an extensive diamond drilling program around the silica flux mine. The drilling encountered thick, widespread copper oxide and sulfide mineralization, which became known as the Burro deposit, located in the footwall of the old Republic Mine.

Cyprus terminated Mosely's lease and developed a large-scale open pit mine, heap leach and SX/EW processing complex in 1974. Operating as Cyprus Johnson Copper Company, Cyprus began mining in the Burro pit in 1975 using a five cubic- yard shovel and a fleet of four 50-ton trucks. Production continued until 1986, when Cyprus closed the operation after mining approximately 15 million tons of ore and approximately 12 million tons of waste rock from the Burro pit. In total, Cyprus produced approximately 107 million pounds of cathode copper by SX-EW methods. After closure, Cyprus dismantled the SX-EW plant and moved it to the Cyprus Sierrita mine south of Tucson. Cyprus maintained the Johnson Camp property until 1989, when it sold its holdings in the district to Arimetco, Inc.

Arimetco began construction of a 35,000-ppd SX-EW plant in June 1990 and rehabilitated the leach systems on the existing Cyprus pads and the collection, raffinate and plant feed ponds. Construction of the new SX/EW plant and related facilities was completed in September 1990, and mining resumed in the Burro pit in 1991 with a fleet of four 120-ton trucks and related loading equipment.

The leach pads were expanded in 1993 and 1994 to accommodate additional new ore. A 350-tph two-stage crushing plant was commissioned in late 1995, followed by construction of a reusable leach pad on top of the original Cyprus leach pad in 1996 to accelerate recovery of copper from new ore, and expansion of the plant to 50,000 tpd capacity. Arimetco also began limited open-pit mining from the Copper Chief deposit in 1996. Mining from both deposits continued until 1997, when Arimetco terminated ore production and the mining fleet was transferred to its Yerington property in Nevada. During the period 1991-1997 Arimetco mined a total of approximately 16 million tons of ore and 12 million tons of waste rock, primarily from the Burro pit.

In September 1998, Summo U.S.A. Corporation (Summo), entered into a Sale and Purchase Agreement with Arimetco to acquire the Johnson Camp property, subject to successful completion of due diligence work on the part of Summo. As part of that due diligence effort, Summo commissioned a feasibility study for the resumption of mining and SX-EW processing at Johnson Camp. That study (the Summo study), which was completed in April 1999, estimated mineable reserves in the Burro and Copper Chief deposits totaling 29,149,000 tons containing 0.402 percent total copper at a waste to ore ratio of 0.5:1. Although shown to be feasible at $0.85 copper, Summo elected to pursue other projects, and assigned its rights to the Sale and Purchase Agreement to Nord Copper in June 1999. Nord Copper subsequently completed its purchase of the project from Arimetco, and elected to update the feasibility study commissioned by Summo. The update was completed in 2000 and was further updated in late 2005 (the "NORD updated study"). Nord Copper was merged into Nord Resources Corp in February 2001.

The mine continued to produce copper from inventory in the heap on a reduced basis. The low production rate was primarily due to a lack of make-up sulfuric acid for the heaps. Up until the transfer of the property to Nord Copper, Arimetco had produced and shipped a total of approximately 50 million pounds of copper cathode from the Johnson Camp property. There remains a significant inventory of copper in the heaps and in the ground in the Burro and Copper Chief deposits.

Table 8.1: Johnson Camp Mine, Historical Reported Mine Production

Cyprus Production

Production Year	Mine Production Ore Tons	Cu Grade AsCu %)	Contained Cu Pounds	Shipped Cu Pounds
1975	2,132,260	0.496	21,152,019	6,143,024
1976	1,821,476	0.357	13,005,339	10,059,807
1977	1,563,030	0.399	12,472,979	10,327,424
1978	1,202,500	0.426	10,245,300	10,205,142
1979	1,588,400	0.522	16,582,896	10,032,003
1980	1,499,600	0.411	12,326,712	10,320,407
1981	1,551,500	0.470	14,584,100	10,693,485
1982	1,894,700	0.322	12,201,868	9,702,272
1983	1,962,600	0.504	19,783,008	9,717,616
1984	52,100	0.713	742,946	8,803,361
1985	-	-	-	6,200,836
1986	-	-	-	4,854,796
Sub-Total	15,268,166	0.004	133,097,167	107,060,173

Arimetco Production

Production Year	Mine Production Ore Tons	Cu Grade TCu(%)	Contained Cu Pounds	Shipped Cu Pounds
1991	750,100	0.340	5,100,680	5,549,725
1992	2,516,320	0.480	24,156,672	8,156,435
1993	3,259,320	0.340	22,163,376	7,386,504
1994	2,719,690	0.290	15,774,202	5,618,012
1995	2,995,592	0.290	17,374,434	6,345,518
1996	3,084,254	0.350	21,589,778	9,921,576
1997	1,254,971	0.370	9,286,785	4,747,995
1998	-	-	-	2,181,304
Sub-Total	16,580,247	0.348	115,445,927	49,907,069

Total Property Production (1975 - 1998)

	Mine Production Ore Tons	Cu Grade TCu(%) (1)	Contained Cu Pounds (1)	Shipped Cu Pounds
Total	31,848,413	?	248,543,094	156,967,242

Note: (1) Cyprus Cu grade was annually reported as % soluble copper and Arimetco reported % total copper, so unable to calculate Cu grade in terms of TCu or AsCu. The contained Cu lbs are based on TCu and AsCu grades.

9.0 GEOLOGICAL SETTINGS

The description of the geology presented here and resource estimate included in this report were reviewed by BETA and incorporated from prior engineering reports, published sources, and an on-site visit. There has not been any additional resource drilling results or mining in the Burro and Copper Chief deposits since the publication of the prior engineering reports.

9.1 REGIONAL GEOLOGY

The Johnson Camp mine is located along the east flank of the Little Dragoon Mountains in southeast Arizona (see figure 6-1). The Little Dragoon Mountains consist of a core of middle Precambrian granites and schists that are unconformably overlain by younger Precambrian metasediments and Paleozoic and Mesozoic sediments. This entire sequence of rocks was intruded by a stock of quartz monzonite composition in late Cretaceous or early Tertiary time, contemporaneous with what is widely referred to as the Laramide orogeny in the southwestern United States. This intrusion has created a contact metamorphic aureole consisting of hornfels and garnet/marble tactite up to several thousand feet in width. Mild folding of the pre-intrusion sediments and metasediments along northwest-trending axes accompanied the Laramide orogeny.

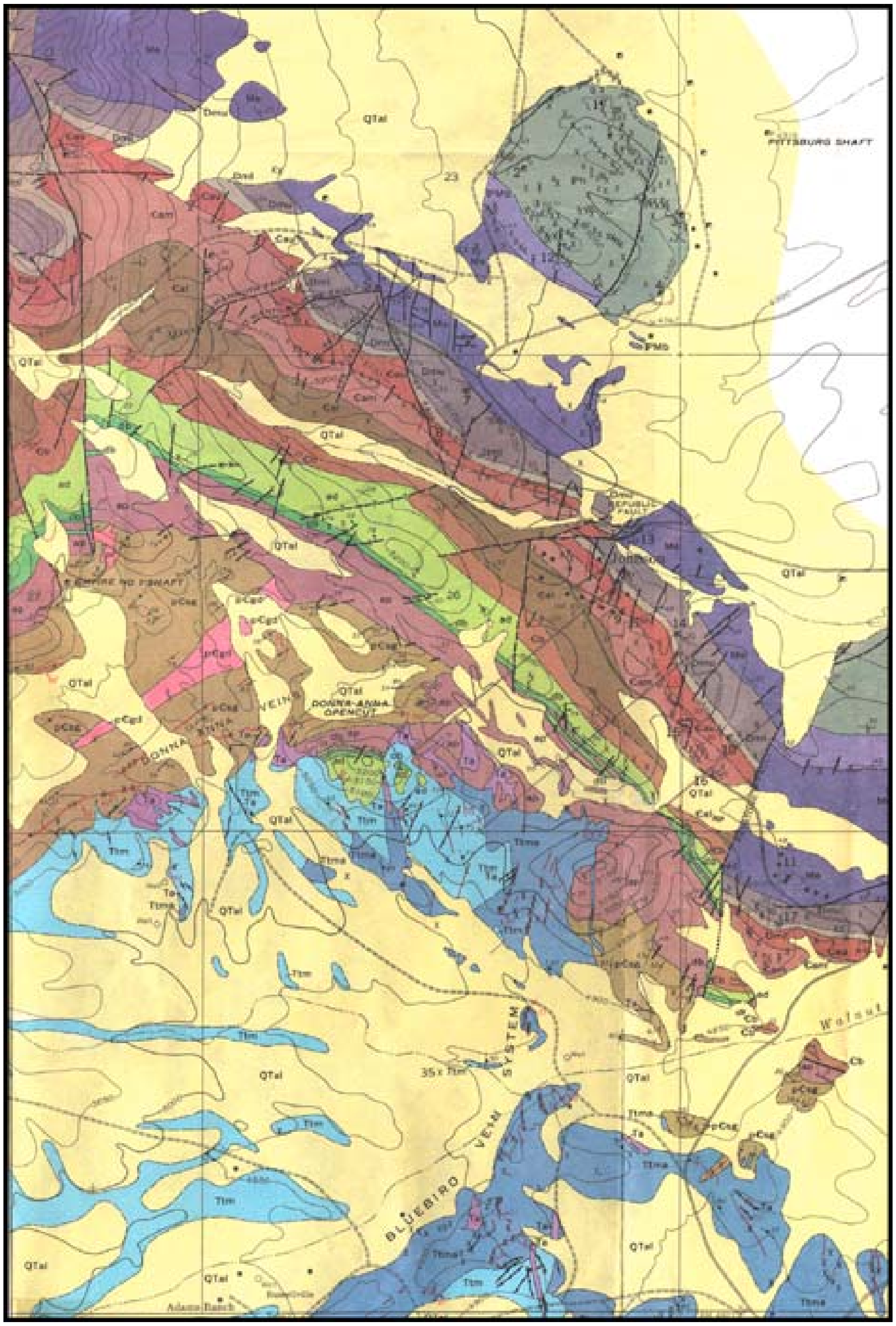
Extensional faulting during the Tertiary and Quaternary periods produced the prominent mountain ranges and intervening alluvium-filled valleys that comprise the topography evident today. Drill hole penetrations in the alluvium indicate that valley fill along the flanks of the mountain ranges exceeds 1,000 feet. Thirty to forty degrees rotation of the lithologies accompanied extensional faulting, producing the current bedding orientations measured in the Little Dragoon Mountain area.

9.2 DEPOSIT GEOLOGY

The rock units exposed on the Johnson Camp mine property range from basal Precambrian Pinal Schist at the west end of the property to Laramide quartz monzonite porphyry. The general geology of the Johnson Camp mine (figure 9-1) is taken from Cooper and Silver, (1964, "Geologic Map of the Area near Johnson, Arizona" Plate 6).

Figure 9-1

Geology of the Johnson Camp Mine region
from: Cooper and Silver, 1964, USGS Professional Paper 416



A brief description of the major rock units follows:

Precambrian Pinal Schist

The Pinal Schist forms the base of the section on the Johnson Camp property. The unit consists of metasedimentary rocks, now strongly foliated, mostly intercalated sericite schist, metagraywacke, metasandstone, and metashale units, locally interleaved with thin metavolcanic units. The rocks have been intensely deformed and subject to low-rank (greenschist facies) regional metamorphism. The general strike of bedding and schistosity is northeast, with steep dips primarily to the northwest. Although structural complications may cause apparent thickening of the section locally, the Pinal Schist appears to be at least 2,000 feet thick on the property before being truncated by the Laramide quartz monzonite intrusion, and may reach over 10,000 feet in thickness elsewhere in the area. The Pinal Schist is not known to be mineralized at Johnson Camp.

Precambrian Pioneer Shale

The Pioneer Shale unconformably overlies the Pinal Schist. The unit dips primarily to the northeast at 30 to 40 degrees, although local attitudes do vary across minor folds such as those mapped by Cooper and Silver (1964) before the leach pads on the property covered these exposures. The Pioneer Shale is primarily silty shale containing sparse quartzite interbeds. On the property, the 150 to 300-foot thick unit is pervasively contact metamorphosed as a result of its proximity to the Laramide quartz monzonite intrusion, consisting of gray and green spotted hornfels with pronounced gray and purple color banding. It is weakly mineralized in the vicinity of the Copper Chief deposit at the base of the known mineralized section, and also locally at the northwest end of the Burro deposit.

Precambrian Dripping Springs Quartzite

The Dripping Springs Quartzite overlies the Pioneer Shale, but in the Johnson Camp mine area, this unit has not been distinguished from the Cambrian Bolsa Quartzite, which unconformably overlies it.

Precambrian Diabase Sills

Two closely-spaced, parallel to sub-parallel diabase sills intrude the Precambrian section at and just below the contact between the Pioneer Shale and Drippings Springs Quartzite. The sills are generally dark green, schistose, and consist primarily of a fine-grained matrix with occasional amygdaloidal features at or near the upper contacts. They range in thickness from less than 10 feet to more than 100 feet, with the lower sill generally the thinner of the two, and are separated by a few tens of feet of Pioneer Shale or Dripping Springs/Bolsa Quartzite. The sills dip parallel with bedding, primarily to the northeast, at 30 to 40 degrees. Although they constitute the principal host to mineralization at the Copper Chief deposit, in the Burro deposit they represent only a minor host to mineralization. Southeast of the Burro deposit, the diabase sills are mostly barren of mineralization.

Cambrian Bolsa Quartzite

The Bolsa Quartzite is a prominent rock unit on the property. It occurs as a massive red-brown to tan-colored unit consisting primarily of metamorphosed massive sandstone beds with occasional conglomerate interbeds, and rare shale lenses, in sharp contrast to the dark green diabase and the overlying medium green lower Abrigo Formation. The unit dips 30 to 40 degrees to the northeast and ranges in thickness from roughly 300 feet in the Copper Chief deposit area to approximately 200 feet in the Burro deposit area. The Bolsa Quartzite is intensely iron-stained in the Burro deposit area, most likely as a result of the oxidation of abundant disseminated sulfides and sulfide bearing quartz veins. These veins are evidence of fracturing, parallel old bedding planes, and provide a significant host for ore in the Burro deposit. The unit is mostly barren in the Copper Chief deposit.

Cambrian Abrigo Formation

The Upper Cambrian Abrigo Formation lies conformably on top of the Bolsa Quartzite, and is divided into lower, middle and upper members, with increasing carbonate content up section, described in more detail as follows:

Lower Abrigo Formation.

Recent geologic mapping in the Burro pit by Dr. Jon Thorson for Summo identified three sub-members of the lower Abrigo Formation which are consistent and recognizable both across the exposures in the pit and in drill core and cuttings. The lower, approximately 120 feet thick, sub-member of the Lower Abrigo Formation consists primarily of thin-bedded green diopside hornfels with minor dark gray biotite hornfels. There is a strong retrograde alteration assemblage of chlorite+epidote with minor amounts of nontronite clay, which accentuates the green coloration of the rock unit. Quartz veins, primarily parallel to bedding, are locally abundant, as are sulfides below the water table or the remnants of sulfides above the water table. The middle sub-member is approximately 80 feet thick, and consists of white to light brown, bleached and strongly iron-stained hornfels with abundant oxidized quartz veins. It stands in sharp contrast to the overlying garnetite and the underlying green hornfels. The upper sub-member consists of a garnetite zone 10 to 40 feet thick, which separates the Lower Abrigo from the Middle Abrigo Formation. This unit is composed of thin-bedded to massive-bedded garnet occurring in crystals ranging from 1/16 to 1/8-inch in size, and gossanous zones and pods of yellow to yellow-green nontronite clay.

The Lower Abrigo Formation is the dominant host to mineralization in the Burro deposit, and is also an important host to mineralization in the Copper Chief deposit.

Middle Abrigo Formation.

This light brown weathering unit is approximately 300 feet thick in the Burro deposit area. It consists of a sequence of white to light gray or green thick-bedded calcareous hornfels or calcsilicate marble, with partings of thin-bedded greenish-gray to medium gray hornfels. The hornfels partings are more prominent in the lower portion of the Middle Abrigo, near its transition from the Lower Abrigo. Although the USGS divides the middle member into four sub-units, previous Johnson Camp geologists have treated the member as a single unit. The semimassive manto replacement deposits mined historically at the Republic, Mammoth, Moore and Copper Chief mines were situated at the contact between the middle and upper members of the Abrigo Formation.

Upper Abrigo Formation.

The Upper Abrigo Formation consists primarily of white to light gray massive beds of marble and thin beds of quartzite. It ranges up to 150 feet in total thickness. It is not known to be mineralized anywhere on the Johnson Camp property.

Devonian Martin Formation

The slope-forming Martin Formation disconformably overlies the Abrigo Formation and consists primarily of thin to medium-bedded limestone ranging up to 200 to 300 feet in total thickness. Dips range from 30 to 40 degrees to the northeast. The Martin limestones are not known to be mineralized on the property, although they are weakly mineralized at the Keystone Mine and do constitute a significant host to mineralization on the Sullivan property immediately to the south of Johnson Camp. Although the unit is a primary host to the major deposits at the Magma Mine, near Superior, Arizona, it has not received much exploration attention at Johnson Camp.

Mississippian Escabrosa Limestone

The Escabrosa Limestone overlies the Martin Formation, and is distinguished from it by its more massive, blocky, coarse-grained nature and its abundant crinoid fossil fragments and chert nodules, pods, and lenses. The unit is in excess of 700 feet thick in the Little Dragoon Mountains. It is not known to be a host to mineralization on the Johnson Camp property, although it is a significant host for copper mineralization on the Sullivan property immediately to the south.

Laramide Quartz Monzonite Intrusion

A large stock, approximately three miles wide and four to five miles long, was intruded into the Precambrian and Paleozoic section during the Laramide orogeny. Dates obtained by K-Ar radiometric methods suggest an approximate age of 53 million years. The intrusion consists primarily of two to five centimeter-long potassium feldspar phenocrysts and biotite in a light-colored groundmass of quartz and sodium feldspar. Several northeast trending alteration zones consisting primarily of multiple quartz veins are evident near the eastern end of the intrusion. These alteration zones are shaped like a champagne glass on its side, and suggest that fluids migrated from the center of the intrusion towards the eastern margin, where they dispersed and altered a large volume of rock. Given that the entire section appears to be tilted 30 to 40 degrees to the east, these champagne-glass shapes may represent the tops of the hydrothermal systems responsible for mineralization at not only the Johnson Camp property, but also other known mineral occurrences in the general area.

The quartz monzonite intrusion is not known to be mineralized, other than weakly within the alteration zones described above. However, it does appear to represent the source of mineralizing fluids at Johnson Camp, and other known mineral occurrences in the area, and may host unexposed mineralization at depth or just beneath alluvial cover.

Alluvium

A wedge of alluvium covers the dipping sequence of rocks to the east of Johnson Camp. Based on sparse water well data, the wedge appears to thicken from west to east towards the Gunnison Hills Fault, a steeply west-dipping normal fault exposed approximately 2.5 miles east of the Burro pit area. The thickness of the alluvium against the fault locally exceeds 1,000 feet. The alluvium consists primarily of sand and gravel derived from out-wash of the Precambrian granite and Paleozoic rock units exposed in the Little Dragoon Mountains and surrounding areas.

9.3 STRUCTURAL GEOLOGY

Structural disruptions of the host rocks in the Johnson Camp area are relatively minor. The Precambrian rocks were subject to compressional deformation in the middle Precambrian, producing foliation primarily sub-parallel to bedding. Because these rocks are generally unmineralized, the Precambrian deformation has not been studied in detail.

Younger Precambrian and Paleozoic sedimentary units have been affected by mild northeast-southwest compression, which produced broad open folds with axes oriented northwest. This compressional event is presumed to be Laramide in age. Old reports on the localization of the manto (replacement) deposits mined extensively in the first half of this century suggests some of these mantos were controlled by fold axes (Cooper and Silver, 1964). These manto deposits lie outside of the projected ultimate pit. No effects of folding have been observed in the existing Burro pit, drill core, or outcrop exposures. These folds do not appear to affect the localization of copper in the bulk-mineable deposits that are the focus of this study.

The remaining structural manifestations in the area appear to consist primarily of brittle deformation features (faults). Two major sets of faults are recognized in the immediate area of the Burro and Copper Chief pits. The most prominent set strikes 10 to 30 degrees northeast, with dips 70 to 75 degrees southeast. These faults display minor normal displacement on the order of 10 to 50 feet, with displacements only occasionally reaching 100 feet. The faults appear to have been channel ways for mineralizing solutions, and many of the manto deposits in the Middle Abrigo Formation are controlled by these structures. The faults do not appear to have a significant influence on the distribution of the disseminated copper mineralization in the Burro or Copper Chief deposits.

A second set of faults strikes generally 60 degrees northeast to 60 degrees southeast. Dips range from 30 to 50 degrees south on the faults that display normal movement while the steeper faults (75 degrees south) have reverse throws. Collectively these faults appear to have influenced the localization of manto deposits in the Middle Abrigo Formation. No evidence of direct controls on the distribution of mineralization in the bulk-mineable deposits has been identified. However, these faults are believed to have been conduits for the general migration of mineralizing solutions into the host section.

Both sets of faults appear to cut one another, and are therefore assumed to be contemporaneous. Faulting appears to have begun while the sedimentary rock units were still horizontal, prior to late Cretaceous or Tertiary rotation and tilting. The structural fabric indicates that the faults both pre-date and are contemporaneous with emplacement of the Laramide-aged quartz monzonite intrusion. Some dikes associated with the intrusion appear to fill pre-existing northeasterly fault structures. The intrusion itself does not appear to be cut by these fault sets.

10.0 DEPOSIT TYPE

The Johnson Camp property has an extensive history of production from a number of mines. Mineable ore reserves remain on the property, and numerous additional mineralized prospects and other geologic potential remain to be explored and tested. These include:

Large Disseminated Deposits.

Large disseminated copper deposits occur in the lower Abrigo Formation and also in the underlying Bolsa Quartzite and Diabase units. The first bulk-mineable deposit was identified by Cyprus Mines Corporation in the 1960's (see Section 3.2). Because the Bolsa material being mined by an independent leaseholder displayed an impressive amount of oxide copper, a drilling program was launched to define the resource. Drilling by Cyprus outlined the Burro copper oxide reserve of 22 million tons grading 0.85 percent total copper (Clayton, 1978), mostly within the Lower Abrigo Formation. Cyprus commenced open pit mining of this deposit in 1975, and subsequently shut down the operation in 1986 due to low metal prices, after having produced approximately 15 million tons of ore grading approximately 0.6 percent total copper.

As can be seen in the Burro pit, copper formed primarily on bedding planes as veins and replacements along with quartz and pyrite, and along fractures which parallel the major fault sets. Extending from the surface to depths ranging from 100 to 150 feet, (but still above the water table), oxide copper consists primarily of copper in limonite and in manganiferous wad. These oxides transition into chrysocolla and malachite which dominate from a depth of 150 feet down to the water table at roughly the 4,560-foot elevation. Some native copper occurs disseminated throughout this range also. As noted by Clayton in "Alteration and Mineralization of the Cyprus Johnson Deposit, Cochise County, Arizona", chalcocite appears as coatings on pyrite below the 4,600-foot elevation, and continues as a secondary mineral, replacing sulfides down to the maximum depths drilled. Pyrite, bornite and chalcopyrite, all with chalcocite coatings, are evident below the 4,600-foot elevation, generally increasing in abundance to at least the 4,460-foot elevation.

Arimetco resumed mining the Burro deposit in 1990, after purchasing the property from Cyprus, and produced an additional 15 million tons of material with an average grade of 0.35 percent total copper. This ore came primarily from the Bolsa Quartzite and the diabase sills, which were considered by Cyprus to be too low in grade and were thus left unmined. Copper in these units is found mostly as exotic accumulations on fractures, presumably derived from dissolution of copper in the immediately overlying Lower Abrigo Formation.

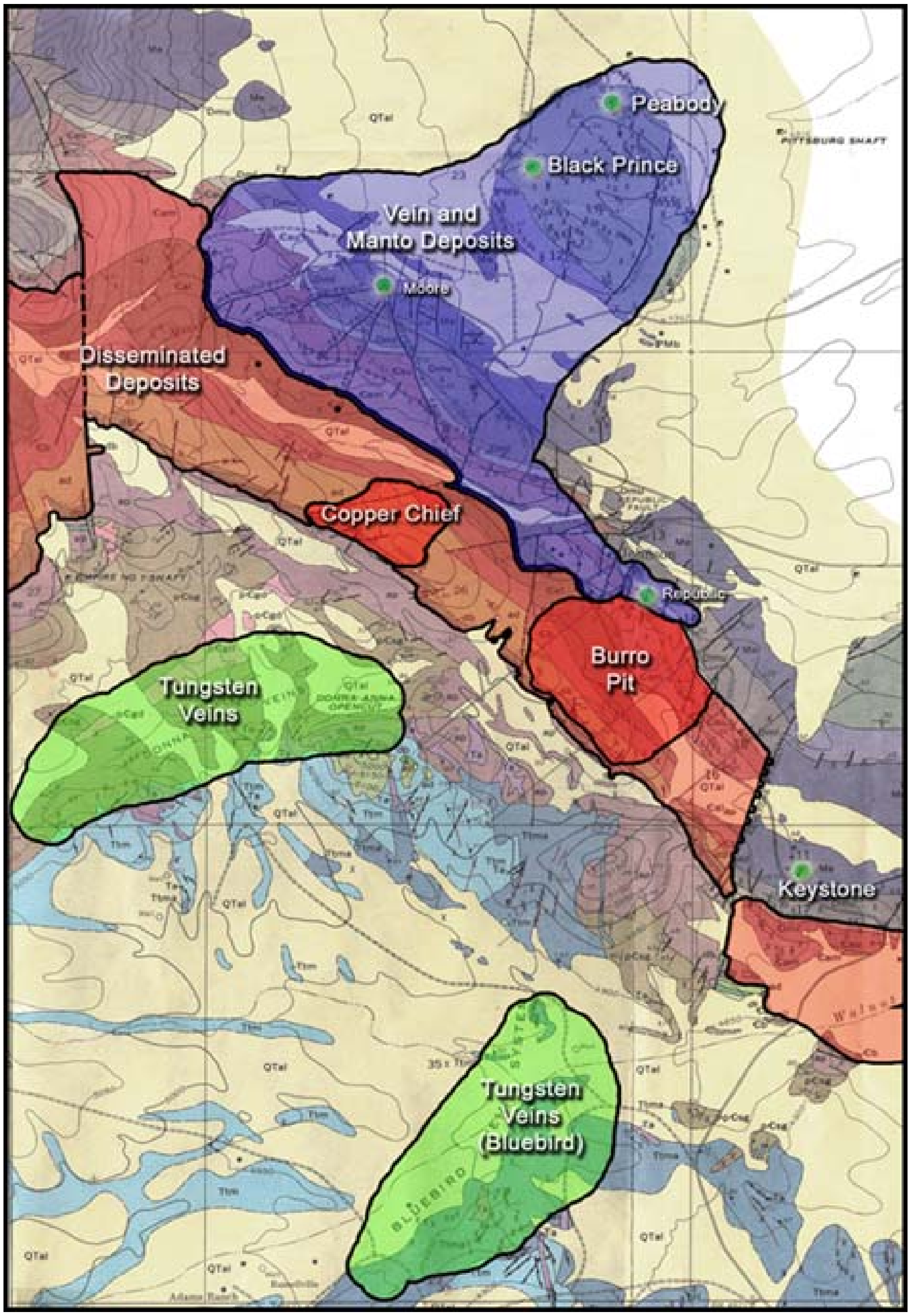
Additional mineable material and in-situ resources remain below the current Burro pit bottom in both the Lower Abrigo and Bolsa Formations, and represent the main focus of this study.

Like the Burro deposit, the Copper Chief deposit is a disseminated bulk-mineable copper deposit. It is situated approximately 1,500 feet along strike to the north of the Burro pit, and is hosted primarily by the diabase and Lower Abrigo Formation. Unlike in the Burro deposit, the intervening Bolsa Quartzite is mostly barren. Copper occurs in limonite, goethite, and manganiferous wad, and as disseminations of chrysocolla, malachite, and lesser native copper in the diabase and along fractures within the diabase and the underlying Pioneer Shale down to the water table at approximately the 4,600-foot elevation. Chalcocite, bornite, and chalcopyrite increase in abundance with increasing depth below the water table, as does pyrite.

Manto (Replacement) Deposits.

Historically, mine production at Johnson Camp has come from underground mining of massive and semi-massive sulfide replacement deposits in the upper portion of the Middle Abrigo Formation (figure 10-1). The Republic, Moore, Mammoth, Copper Chief, and other mines produced in aggregate roughly one million tons of ore grading approximately two percent copper and five percent zinc. These deposits are all situated in the hanging wall rocks (to the east-northeast) of the disseminated mineralization found in the Lower Abrigo Formation, which is one of the subjects of this study. The deposits include both tabular bodies that are oriented parallel to bedding, and chimneys, or mantos. Both types occur within garnetized limestone beneath tactite beds. These bodies have a bearing and plunge within the plane of bedding which coincides with local fold axes. However, the folds are too subtle to be of much use from an exploration perspective (Cooper and Silver, 1964). The Republic and Moore mines were the deepest and best developed of the mines that exploited the manto deposits. The Republic mine was developed to a depth of 1,000 feet below the surface (1,500 feet down the rake of the deposit), while the Moore mine was developed to a depth of 700 feet below the surface (1,500 feet down the rake of the deposit). The workings of both mines are flooded below the 4,700-foot elevation, and represent water supply sources for Nord Copper's planned mine operation.

Figure 10-1
Zoning of the Johnson Camp Mine



- **Purple" Manto Deposits**
- **7.3 to 9.9% copper and 3.1 to 5 oz/ton Silver (~1MM Tons)**
- **"Red" Disseminated Deposits 0.4% copper (29 MM Tons, current)**
- **"Green" Tungsten Deposits**

Lead-Zinc Replacement Deposits in Limestone.

A number of small mines and prospects were developed in the overlying Martin and Escabrosa Limestones east of the Republic and Moore mines and the Burro and Copper Chief deposits. These mineral occurrences probably reflect a zoning of lead and zinc outward (or upward) from the copper and copper-zinc ores that were deposited further to the west and closer to the Laramide intrusion. Very little underground development has been done on these prospects, and that which exists is no longer accessible.

Precious Metal Occurrences.

Only minor gold or silver production has taken place from the area surrounding the Johnson Camp property. Cooper and Silver in USGS Professional Paper 416 list recorded totals of 202 ounces of gold and 645,537 ounces of silver produced from the mines in the Johnson Camp/Gunnison Hills/Centurion areas. Approximately 78 percent of the silver and 85 percent of the gold was produced between 1909 and 1918 and 1949 and 1957, the main periods of activity. Most of this precious metal production apparently came from the major mines operating during these periods, including the Black Prince, Peabody, Republic, and Moore mines. However, the overall average precious metal grades collectively produced from these operations were low, averaging just less than one ounce of silver per ton and only trace amounts of gold. In the Copper Chief mine, records indicate that the ore averaged just 0.5 ounces of silver per ton. However, Cooper and Silver report higher average silver grades on the order of four to five ounces of silver per ton in the Black Prince, Peabody, and Climax mines approximately one mile north of Johnson Camp proper.

11.0 MINERALIZATION

Although evidence strongly suggests that the Johnson Camp deposits are porphyry-related, no dikes or protrusions of Laramide-aged igneous rocks into Paleozoic meta-sediments have been identified on surface or intersected in drilling or underground workings on the Johnson Camp property. Nevertheless, the style of mineralization and the type of alteration present, along with porphyry-related copper mineralization on the adjacent property to the south, all suggest the possible presence beneath the property of a mineralized apophysis or protrusion of the Laramide quartz monzonite stock that is exposed on the western end of the property.

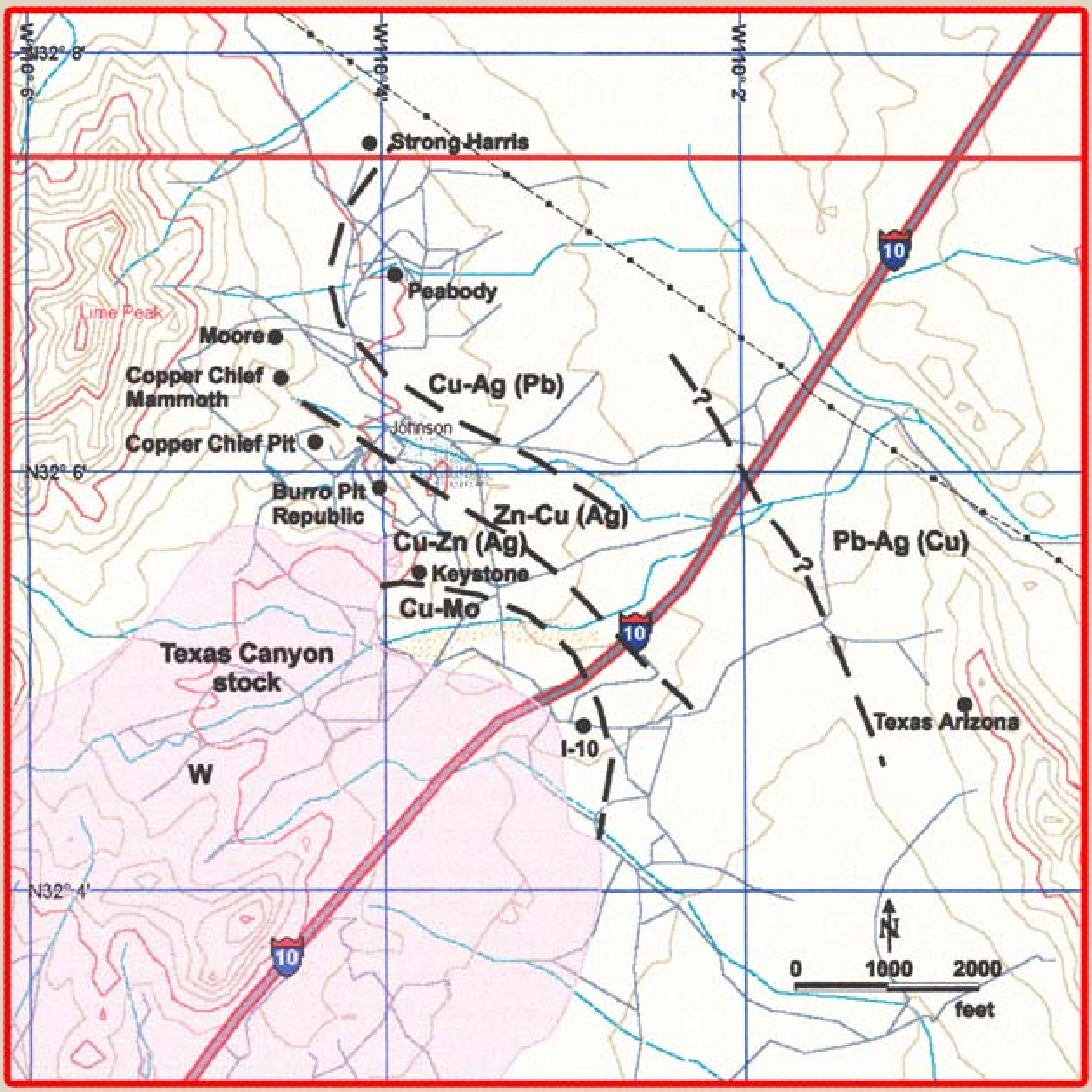
Recent detailed mapping in the present Burro pit has identified pronounced potassic alteration in a portion of the Bolsa Quartzite, consisting of masses of secondary biotite replacing a matrix of quartz and feldspar accompanied by coarse pink k-feldspar (Thorson, 1998). Petrographic examinations of altered Bolsa have confirmed the presence of potassic alteration products (Hansley and Cookro, 1998). Potassic alteration is not commonly found outside of a narrow halo peripheral to porphyry copper deposits. The occurrence of potassic alteration in the bottom of the present Burro pit suggests the possible presence of a buried porphyry intrusion deeper in the section. Intense quartz veining, primarily trending parallel to bedding, is also found in the bottom of the Burro pit, in proximity to the potassic alteration. The veins consist of disseminated sulfides (now oxidized) in a dominant quartz matrix. Vein density reaches a high of more than ten per foot in the northwest corner of the pit, immediately above the area of potassic alteration (Thorson, 1998).

In addition to the alteration evidence, a prominent magnetic low anomaly is present between the Burro pit and the Copper Chief deposit. This anomaly supports the possible presence of an extension or apophysis of the intrusion at depth. Also, according to press releases and other public information issued by JABA, Inc. relative to the adjacent property to the south, quartz monzonite was found in core from a drill hole that intersected a large (+5 billion pound) copper deposit located beneath Interstate 10 (Jaba, 1998).

Other bulk mineable targets lie along strike from the Copper Chief and Burro deposits. The North target lies mostly within the Middle Abrigo Formation, extending up-dip from the old underground mine workings developed along the manto deposits in the Middle Abrigo Formation. Copper oxides are evident along a sparsely prospected slope below the old workings. Similarly, the Keystone-Walnut target lies along strike to the southeast of the Burro deposit, up-dip from the old underground workings of the Keystone Mine. Drilling at the Keystone Mine by the U.S. Bureau of Mines in the late 1940's (Romslo, 1949) cut bedded replacement and disseminated copper over widths exceeding 50 feet in the Middle Abrigo Formation in a number of holes. These holes did not go deep enough to test the Lower Abrigo Formation, which is the dominant host for bulk-mineable deposits on the Johnson Camp property. Surface exposures of Middle Abrigo Formation, carrying copper oxides and chrysocolla suggest the presence of a bulk-mineable target up-dip from the USBM drill intercepts from the Keystone Walnut target.

Several small lead and zinc mines and prospects are found in the Martin and Escabrosa limestones east of the Burro and Copper Chief deposits. The location and distribution of these lead-zinc prospects could be instrumental in determining a property-wide metal zoning pattern (figure 11-1), which may be useful in identifying the locus and source of mineralizing fluids, and possibly targets for other copper deposits on the property.

Figure 11-1
Zoning map showing the locations of the Burro and Copper Chief pits.



District zoning pattern for the Johnson Camp mining district

Large disseminated copper deposits occur in the Lower Abrigo Formation and also in the underlying Bolsa Quartzite and Diabase units. Two of these deposits, the Burro and Copper Chief, have past open pit production and remaining copper resources and ore reserves. From 1975 to 1986, Cyprus produced approximately 15.3 million tons of ore grading approximately 0.6 percent total copper from the Burro deposit. From 1991 to 1997, Arimetco mined approximately 16.6 million additional tons of ore from this deposit, as well as a very small tonnage from the Copper Chief deposit. The copper mineralization that was the focus of this bulk mining consists of malachite, chrysocolla, copper in limonite and in manganiferous wad, chalcocite, and lesser native copper. Other bulk mineable targets lie along strike from the Copper Chief and Burro deposits. These include the North target and the Keystone-Walnut target, which is situated along strike to the southeast of the Burro deposit.

The style of mineralization and the potassic alteration recently mapped on the northern lower benches of the Burro pit suggest the possible presence beneath the property of a mineralized apophysis or protrusion of the nearby Laramide quartz monzonite stock. In addition to the alteration evidence, a prominent magnetic low anomaly is present between the Burro pit and Copper Chief deposit supporting the possible presence of a porphyry-type deposit at depth.

12.0 EXPLORATION AND DEVELOPMENT

Many competent geologists have worked this district including: 1940 Arthur Baker PhD Thesis; 1960-84 Cyprus geologists including Ken Walther, PhD and R.L. Clayton. The USGS Prof. Paper 416 (Cooper & Silver 1964) is the definitive work on the Dragoon, Arizona Quadrangle. Summo, in 1998, retained Jon Thorson, PhD to map the Burro Pit in detail and investigate the copper porphyry link. In 1999-2000, Nord retained a consulting geologist Thornwell Rodgers to work on the north and Keystone areas distal to the pits.

In 2005, NORD contracted an independent consultant to digitize all the relevant geology in the Burro and Copper Chief pit areas based on the previous Cyprus and Summo work. The correlation between the 1960's Cyprus work and the 1998 Summo work was reported by the consultant to be excellent. The consultant confirmed that the geologic mapping was carried out to the standards of the times, when mapped and that no issues with the geologic mapping have been discovered during the three years the consultant has been associated with the Johnson Camp property. BETA has reviewed the consultant's work and concurs with his conclusions.

The Johnson Camp project area has been an active mining district for more than 100 years. The following summary of the history of mining at Johnson Camp through 1957 was compiled from United States Geological Survey (USGS) Professional Paper 416 (1964), Geology and Ore Deposits of the Dragoon Quadrangle, by J.R. Cooper and L.T. Silver.

The earliest known mining in the Johnson Camp area was done by Mexican miners prior to 1880. By the time of the completion of the Southern Pacific Railway in 1881 through Dragoon just south of Johnson Camp, a number of mining claims had been patented, including the Peabody, Republic, and Mammoth claims, which are all part of the current Johnson Camp property. The Peabody mine (owned by the Russell Gold and Silver Mining Company of Philadelphia) near the present location of the Black Prince shaft on the north end of the Johnson Camp property was one of the earliest producers, supporting a small smelter. The mine eventually closed, only to be purchased and reopened in 1899 by the Dragoon Mining Company, a subsidiary of the Federal Copper Company of New York. In 1907 the Bonanza Belt Copper Company assumed control of the mine, followed a short time later by the Consolidated Copper Company. After closure, the mine remained idle until 1957, when it was purchased by the Coronado Copper and Zinc Company, operators of the nearby Republic and Moore mines. Subsequent to 1902, recorded production averaged 7.4 percent copper and 4.2 ounces of silver per ton.

The second producer of significant size in the Johnson Camp area was the Black Prince Copper Company, which was formed by Denver-based investors in 1901. By 1903 the company was proceeding with underground development of the Republic and Mammoth mines beneath existing surface cuts to the south of the Peabody mine. After relinquishing control of the Republic and Mammoth mines to Arizona Consolidated Mining Company, the company commenced sinking of the Black Prince shaft in 1905, completing it to a depth of almost 1,000 feet by 1911.

With the Black Prince shaft in progress and the Republic and Mammoth mines in operation, access to the Johnson Camp area improved. In 1906 the Johnson Dragoon and Northern Railroad Company began construction of a standard-gauge railroad spur between Johnson Camp and the Southern Pacific main line at Dragoon. The spur was completed in November 1909, but saw limited use prior to World War I.

That same year the Arizona Consolidated Mining Company was reorganized as the Arizona United Mining Company. Soon after, the company constructed a 125-ton smelter at the Republic mine to treat low-grade sulfide ore. However, smelting of the low-grade ores was unsuccessful due to lack of suitable flux, and the smelter was soon abandoned. The period during World War I saw a resurgence of mining at Johnson Camp, with the railroad spur contributing to the success of a number of mines in the area. By 1916, Johnson Camp had a population of approximately 1,000, and more than 80,000 tons of ore were shipped via rail from the Republic mine and four or five smaller mines.

The Copper Chief mine was one of these, reaching its production peak during this period when it produced approximately ten percent of the Republic mine's total. The Peabody mine continued to contribute to the total production, and small amounts of copper were produced from the Keystone, Black Prince, and Johnson Copper Development operations.

In 1920, the fall in the price of copper forced the Republic mine to close, along with the smaller operations. Deprived of ore from its biggest customer, the railroad spur closed down, and in 1925, the tracks were removed. For the next 20 years, almost no mining took place in the district. The Republic mine was allowed to flood, and Johnson Camp was abandoned.

The Coronado Copper and Zinc Company, a wholly owned subsidiary of Cyprus Mines Corporation dewatered the Republic mine in 1942 while operating under a lease and purchase option. After significant exploration work was completed, the company exercised the option to purchase the mine and constructed a 200-tpd selective flotation concentrator at the minesite. Beginning in May 1945, the mill operated almost without interruption for 12 years, processing ore from the Republic mine (1945-1952), the Mammoth mine (1945-1949), and the Moore mine (1951-1957). In 1949, Coronado Copper and Zinc added the Black Prince mine to its collection of properties. In 1957, low metal prices forced the closure of the Moore mine and the mill.

Cyprus Mines Corporation maintained the property through the late 1950's and 1960's, leasing portions of the property to various contractors. These included McFarland and Hullinger, who resumed underground mining in the Moore mine, and Ira Mosely, who mined oxide copper in silica for smelter flux in an open footwall cut at the Republic mine. In the mid to late 1960's, Cyprus became very interested in the oxide copper exposed in the open cut being mined by Mosely, and commenced an extensive diamond drilling program around the silica flux mine. The drilling encountered thick, widespread copper oxide and sulfide mineralization, which became known as the Burro deposit, located in the footwall of the old Republic Mine.

Cyprus terminated Mosely's lease and developed a large-scale open pit mine, heap leach and SX/EW processing complex in 1974. Operating as Cyprus Johnson Copper Company, Cyprus began mining in the Burro pit in 1975 using a five cubic- yard shovel and a fleet of four 50-ton trucks. Production continued until 1986, when Cyprus closed the operation after mining approximately 15 million tons of ore and approximately 12 million tons of waste rock from the Burro pit. In total, Cyprus produced approximately 107 million pounds of cathode copper by SX-EW methods. After closure, Cyprus dismantled the SX-EW plant and moved it to the Cyprus Sierrita mine south of Tucson. Cyprus maintained the Johnson Camp property until 1989, when it sold its holdings in the district to Arimetco, Inc.

Arimetco began construction of a 35,000-ppd SX-EW plant in June 1990 and rehabilitated the leach systems on the existing Cyprus pads and the collection, raffinate and plant feed ponds. Construction of the new SX/EW plant and related facilities was completed in September 1990, and mining resumed in the Burro pit in 1991 with a fleet of four 120-ton trucks and related loading equipment. The leach pads were expanded in 1993 and 1994 to accommodate additional new ore. A 350-tph two-stage crushing plant was commissioned in late 1995, followed by construction of a reusable leach pad on top of the original Cyprus leach pad in 1996 to accelerate recovery of copper from new ore, and expansion of the plant to 50,000 tpd capacity. Arimetco also began limited open-pit mining from the Copper Chief deposit in 1996. Mining from both deposits continued until 1997, when Arimetco terminated ore production and the mining fleet was transferred to its Yerington property in Nevada.

During the period 1991-1997 Arimetco mined a total of approximately 16 million tons of ore and 12 million tons of waste rock, primarily from the Burro pit. In September 1998, Summo U.S.A. Corporation (Summo), entered into a Sale and Purchase Agreement with Arimetco to acquire the Johnson Camp property, subject to successful completion of due diligence work on the part of Summo. As part of that due diligence effort, Summo commissioned a feasibility study (the Summo study) for the resumption of mining and SX-EW processing at Johnson Camp.

That study, which was completed in April 1999, estimated mineable reserves in the Burro and Copper Chief deposits totaling 29,149,000 tons containing 0.402 percent total copper at a waste to ore ratio of 0.5:1. Although shown to be feasible at $0.85 copper, Summo elected to pursue other projects, and assigned its rights to the Sale and Purchase Agreement to Nord Copper in June 1999.

Nord Copper subsequently completed its purchase of the project from Arimetco, and elected to commission an update to the Summo Feasibility Study.

The mine, under Nord management, produced copper from inventory in the heap on a reduced basis. Up until the transfer of the property to Nord Copper, Arimetco had produced and shipped a total of approximately 50 million pounds of copper cathode from the Johnson Camp property. Between 1999 and 2003, Nord produced and shipped 4.49 million pounds of copper cathode from the property. There remains a significant inventory of copper in the heaps and in the ground in the Burro and Copper Chief deposits.

13.0 DRILLING

BETA received all drill hole data from files provided by Nord, and has reproduced sections from previous reports detailing the drill hole data base. BETA did not independently validate all of the drill hole data, however BETA did verify that the electronic drill hole data received matched the model as presented in previous reports. BETA summarizes the type and extent of drilling at the Johnson Camp, including the procedures followed and a summary and interpretation of results in this section.

The relationship between the sample length and the true thickness of the mineralization was reviewed by BETA. The majority of the database was drilled vertically, with samples taken nominally every 10 feet. The deposit generally strikes NW-SE and dips NE at 45-50 degrees. The thickness of the host domain of the mineralization is sufficient to allow delineation of the inclined ore body with vertical drilling, given that the assays were composited to 20m and then split to conform to geologic boundaries in the modeling process.

13.1 BURRO DRILL HOLE DATA

The initial drill hole database that was provided for the Summo feasibility study for the Burro deposit contained 142 drill holes that totaled 50,535 feet of drilling. These data were obtained from MEDSYSTEM data files that were created and used by Arimetco. Subsequent to the completion of the resource model, Summo drilled four confirmation holes into the Burro deposit. Because assay data were not available at the time, these holes were not used in the Summo feasibility study in the resource estimate and subsequent floating cone studies for their analysis this deposit. However, these four drill holes have since been loaded into the drill hole database and were available for inspection by BETA.

All but nine of the initial 142 drill holes in the Burro deposit were drilled vertically. All four of the Summo confirmation holes were drilled as angle holes. None of the Burro drill holes have down-hole survey data. Nearly all of the initial drill hole assay intervals contained both a total copper (TCu) and acid soluble copper (AsCu) value. Table 13-1 summarizes some of the basic drill hole statistics for the Burro database that were used in the estimation of geologic resources.

Table 13-1

Burro Drill Hole Summary (excluding four Summo confirmation holes)

Parameter	Value
Number of holes drilled	142
Number of holes with down-hole surveys	0
Number of holes without down-hole surveys	142
Number of feet drilled	50,535
Number of total copper assays	4,689
Maximum total copper assay (%)	4.00
Number of soluble copper assays	4,640
Maximum soluble copper assay (%)	3.80

Figure 13-1
Burro Pit Drill Hole Location Map

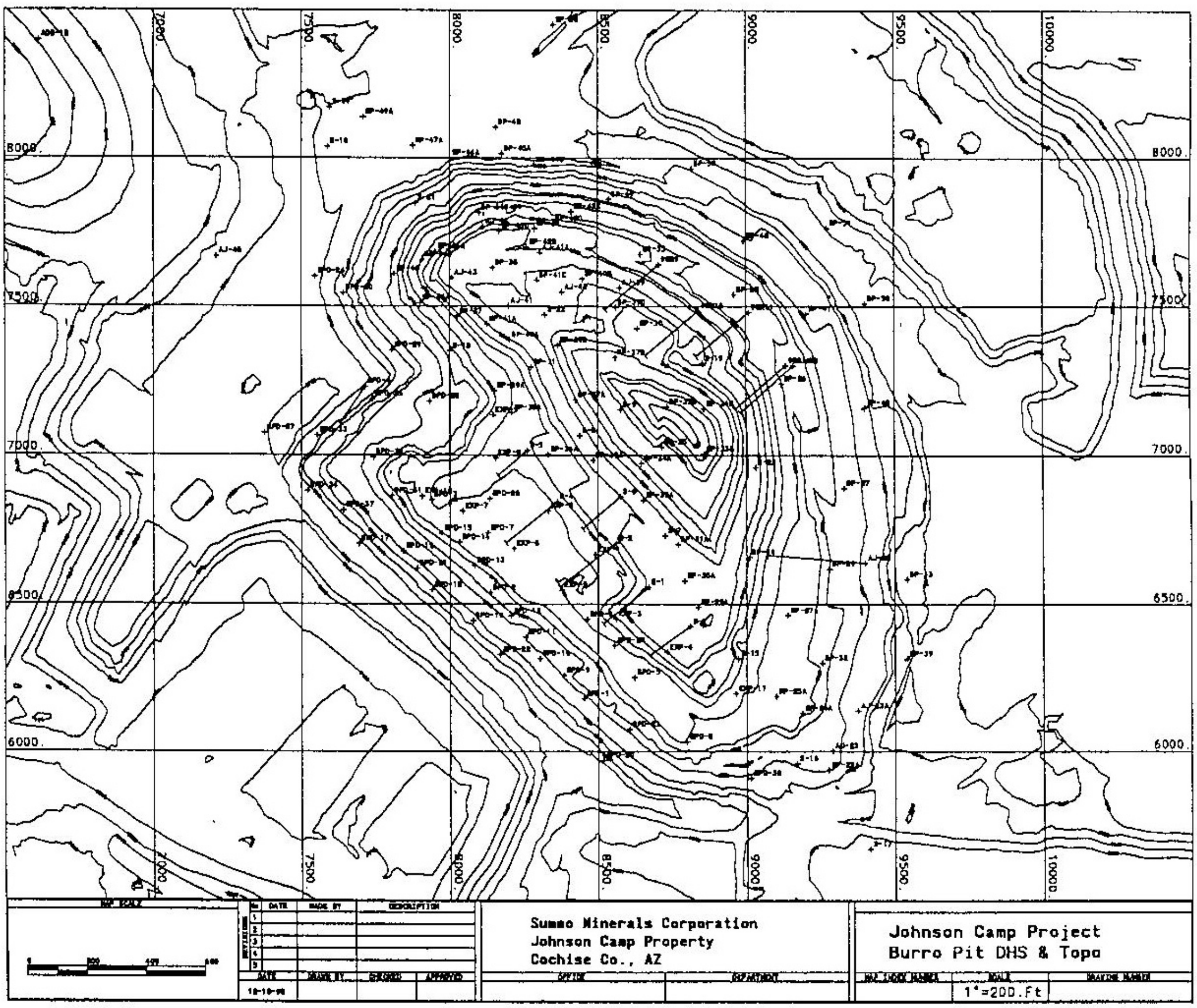


Figure 13-1 is a drill hole collar map for the Burro deposit. This map also shows the extent of previous mining based on an August 1998 aerial survey. The Burro deposit is drilled on approximately 100-foot centers. Cyprus Minerals drilled 94 of these holes in the 1970’s and 80’s. Arimetco drilled the remaining 48 holes in the mid-1990’s.

The total copper and soluble copper assays from the drill hole assay database were grouped into 20-foot long down-hole fixed-length composites. The drill hole compositing routine honored the lithologic codes stored in the drill hole database, which terminated composites at lithologic contacts so that no composites ever straddled rock boundaries. This approach caused some composites near the base of each geologic unit to be less than 20 feet in length.

13.2 COPPER CHIEF DRILL HOLE DATA

The initial drill hole database that was provided for the Summo feasibility study for the Copper Chief deposit contained 151 drill holes that totaled 38,437 feet of drilling. Cyprus drilled 96 of these holes during its tenure in the 1970’s. Arimetco drilled the remaining 55 holes in the mid-1990’s.

These data were obtained from MEDSYSTEM files that were created and used by Arimetco. Subsequent to the completion of the resource model, Summo drilled eight confirmation holes into the Copper Chief deposit. Because assay data were not available at the time, these holes were not used for the Summo feasibility study resource estimate and subsequent mine planning and scheduling, although the holes have since been loaded into the drill hole database.

All but one of the 151 drill holes used to estimate resources for the Copper Chief deposit were drilled vertically, and none have down-hole survey data. Approximately 39 percent of the assay intervals for these holes have both a total copper (TCu) and acid soluble copper value (AsCu). Table 13-2 summarizes some of the basic drill hole statistics for the Copper Chief database that were used in the estimation of geologic resources.

Table 13-2
Copper Chief Drill Hole Statistics (excluding Summo holes)

Parameter	Value
Number of holes drilled	151
Number of holes with down-hole surveys	0
Number of holes without down-hole surveys	151
Number of feet drilled	38,437
Number of total copper assays	5,529
Maximum total copper assay (%)	3.15
Number of soluble copper assays	1,418
Maximum soluble copper assay (%)	2.35

Figure 13-2
Copper Chief Drill Hole Location Map

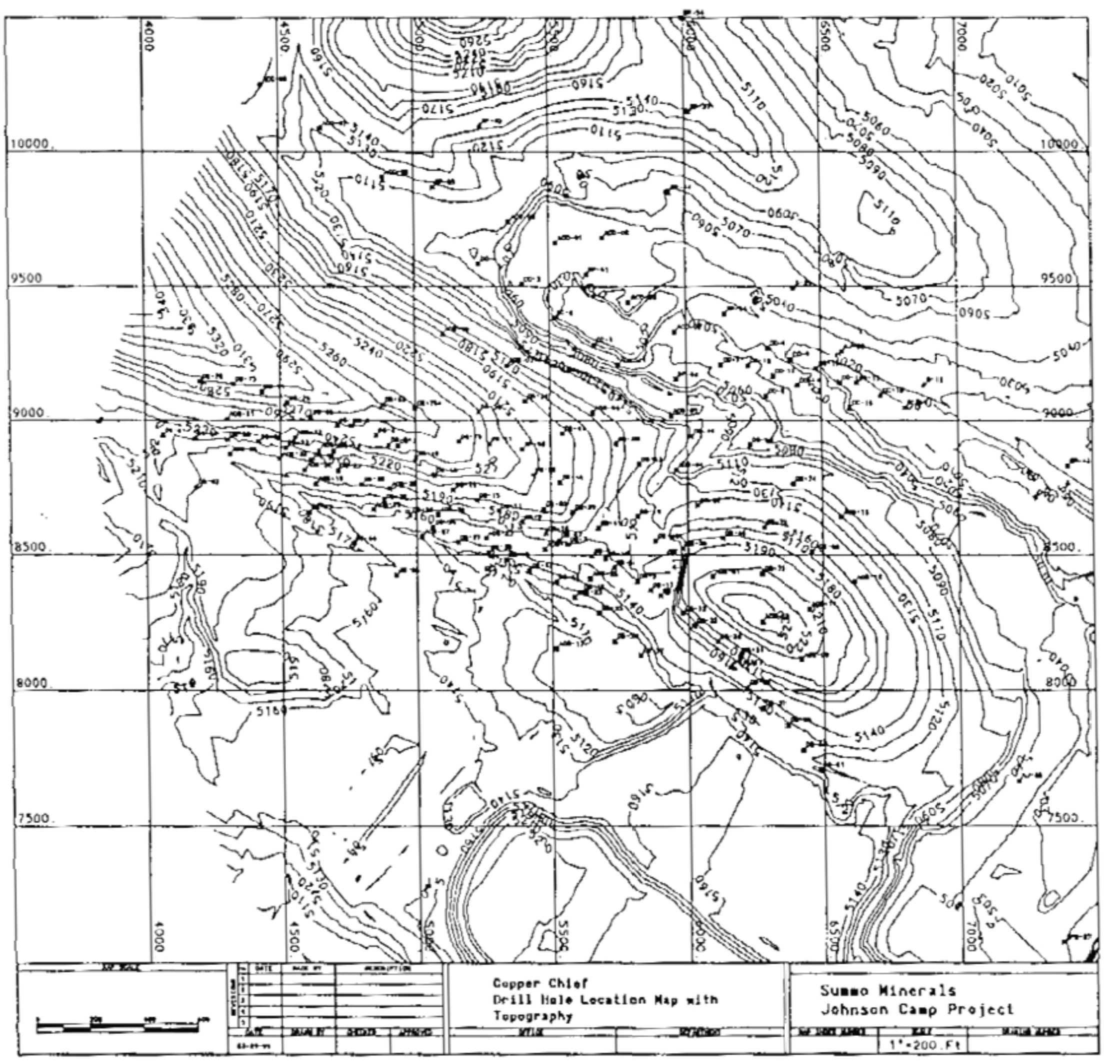


Figure 13-2 is a drill hole collar location map that also shows the topography used for the Copper Chief resource model. Overall, the up-dip portion of the main Copper Chief deposit has been drilled on approximately 100-foot centers, but local areas have closer-spaced drilling on 50-foot centers. The deeper portions of the deposit average approximately 175 feet between drill hole intercepts.

Total copper and soluble copper assays from the drill hole assay database were grouped into 20-foot long down-hole fixed-length composites. These composites honored lithologic codes stored in the drill hole database, such that no composites were allowed to straddle rock boundaries. This approach caused some composites near the base of each geologic unit to be less than 20-feet in length.

14.0 SAMPLING METHOD AND APPROACH

Sample quality, based on review of core on site, discussions with pertinent personnel, and available reports, meet reserve reporting requirements. The drill hole grid and sampling methodology allow for representative results. BETA is not aware of significant factors that may have resulted in sample biases, or of any drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the results provided.

14.1 BURRO DEPOSIT AREA

The statistics of the drill hole assay database for all rock types collectively and by individual rock types are presented herein. Statistical summaries for all rock types combined and for individual rock types are shown in Tables 14-1 and 14-2, respectively. The Lower Abrigo, which is the main mineralized unit in the Burro pit area, is well represented with approximately 38 percent of the total footage that was drilled by the initial 142 holes.

Table 14-1
Burro Pit Drill Hole Copper Assays for all Rock Types

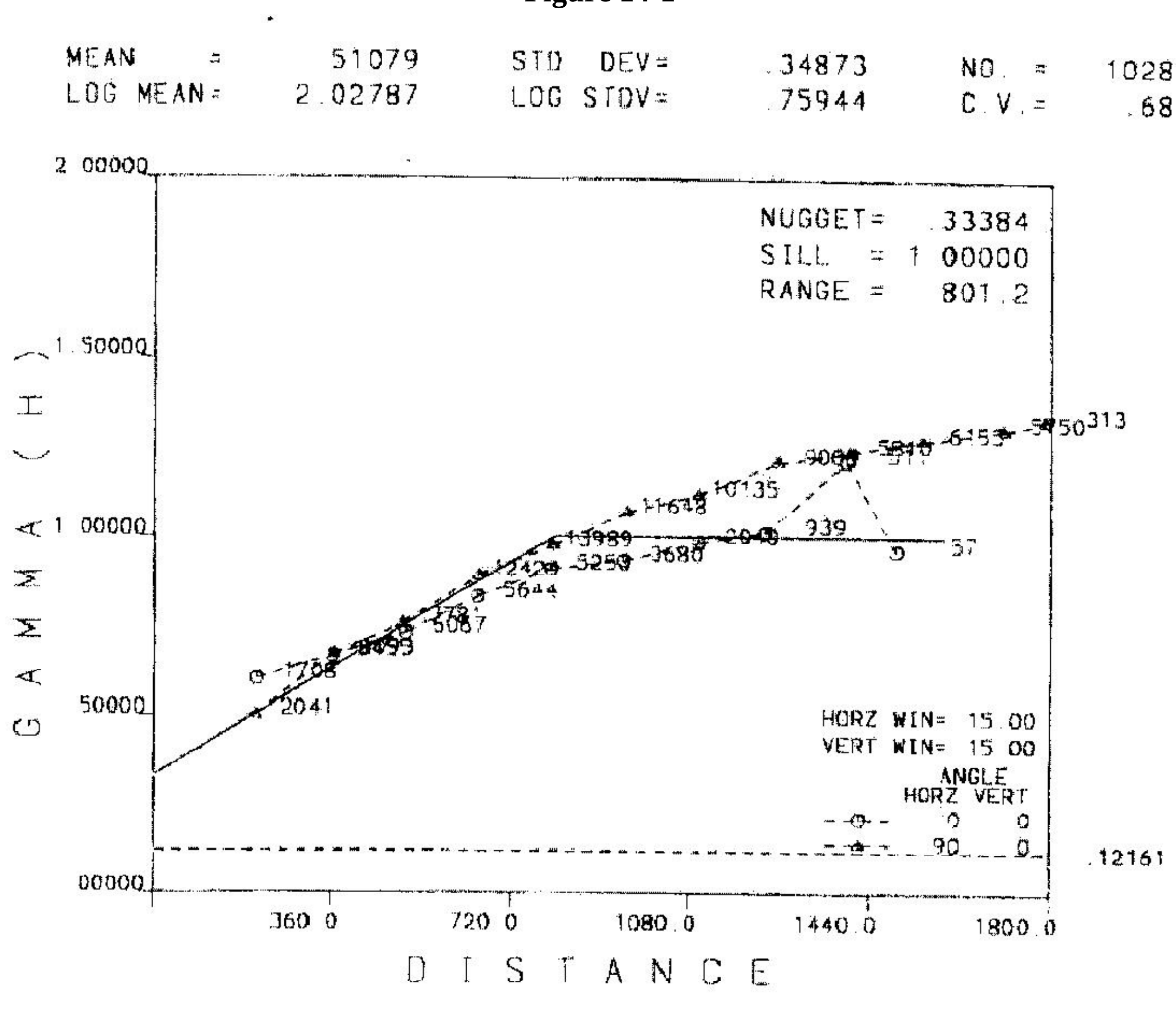
Parameter	Total Copper %	Acid Sol. Copper %
Number of Samples	4,689	4,640
Feet of Drilling	46,673	45,773
Mean	0.332	0.176
Standard Deviation	0.34	0.209
Variance	0.116	0.044
Coefficient of Variation	1	1.2
Minimum value	0	0
Maximum value	4	3.8

Table 14-2
Burro Pit Drill Hole Copper Assays By Rock Type

	Rock 1	Rock 2	Rock 3	Rock 4	Rock 5	Rock 6	Rock 7	Rock 8
Parameter	Upper Abrigo	Middle Abrigo	Lower Abrigo	Bolsa Quartzite	Upper Diabase	Upper Pioneer Shale	Lower Diabase	Lower Pioneer Shale
Number of Samples	19	575	1,959	1,220	264	185	223	204
Feet of Drilling	190	5,731	19,392	11,997	2,574	1,980	2,276	2,035
Mean	0.113	0.201	0.514	0.194	0.325	0.181	0.224	0.083
Standard Deviation	0.087	0.252	0.383	0.217	0.211	0.306	0.207	0.103
Variance	0.008	0.063	0.147	0.047	0.045	0.093	0.043	0.011
Coefficient of variation	0.8	1.3	0.7	1.1	0.6	1.7	0.9	1.2
Minimum	0.03	0	0.02	0	0.01	0.01	0	0
Maximum	0.36	3.35	3.4	2.78	1.36	4	1.77	0.64

Correlograms were calculated and modeled for the five mineralized zones in the Burro deposit. For Zones 1, 4, and 5, global two-dimensional and three-dimensional correlograms were modeled because no anisotropy could be detected in the correlograms for each individual zone. Zones 2 and 3 produced stable structured directional correlograms at orientations nearly parallel to the strike of the lithologic units. Zone 2 (Lower Abrigo) displayed very strong anisotropy along a bearing of N30°W. This lithologic unit also displayed very long ranges throughout a wide range of copper grades, which demonstrates the overall strength and robustness of mineralization within the Lower Abrigo. Figure 14-1 is a directional correlogram for Zone 2 (Lower Abrigo) for sample pairs along a bearing of N30°W. An exponential curve was used to model this correlogram. The nugget and sill values have been normalized so that the total sill equals 1.0. Thus, the primary ore host in the Burro deposit (the Lower Abrigo), has a normalized nugget value of 33 percent.

Figure 14-1



Burro Pit - Zone 2 - TCu Major/Minor Axis Correlogram

14.2 COPPER CHIEF DEPOSIT AREA

Table 14-3
Copper Chief Drill Hole Copper Assays

Parameter	Total Copper	Acid Soluble Copper
Number of Samples	5,529	1,418
Feet of Drilling	38,437	12,926
Mean	0.193	0.161
Standard Deviation	0.232	0.188
Variance	0.482	0.433
Coefficient of Variation	1.200	1.170
Minimum	0.000	0.000
Maximum	3.150	2.350

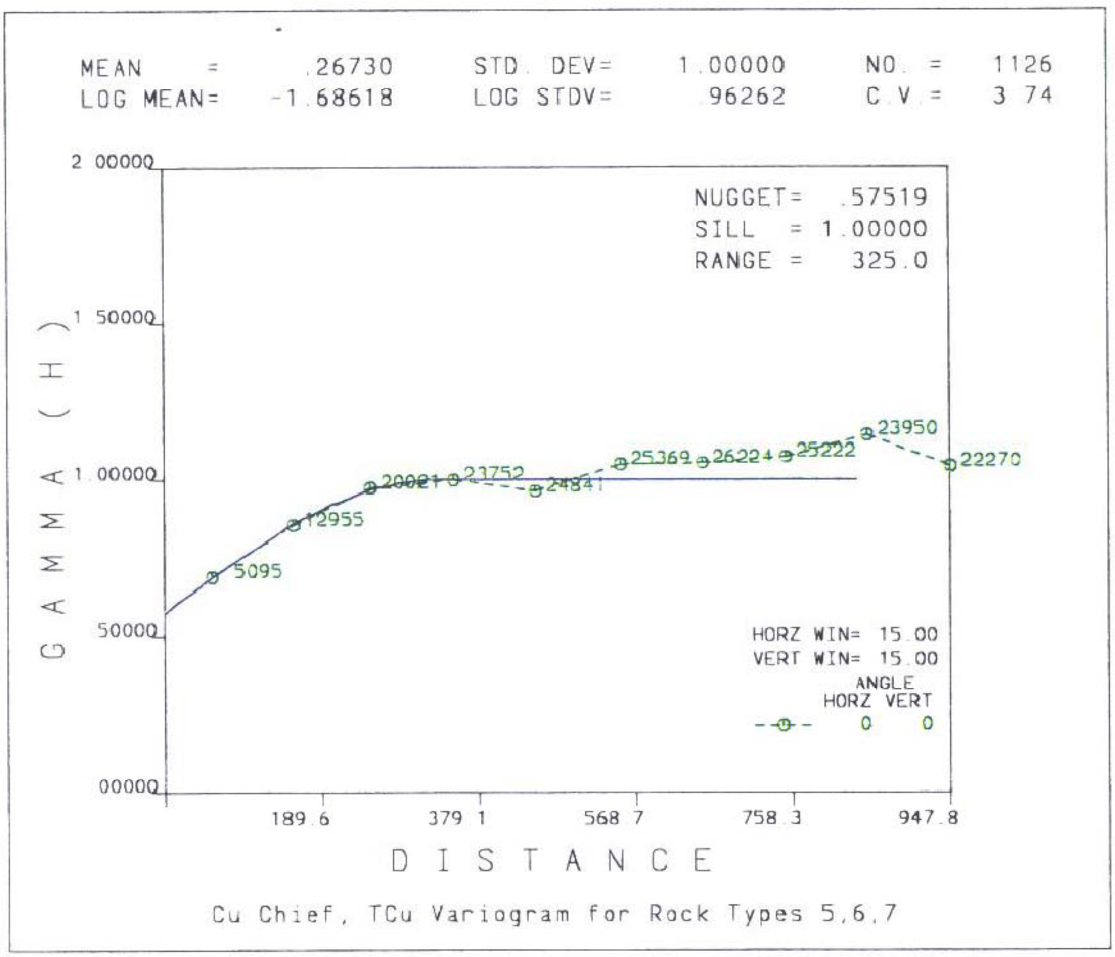
Table 14-4 summarizes the descriptive statistics by rock type for the Copper Chief deposit for assay intervals having total copper assay values greater than or equal to zero. This table shows that rock type six (Rock 6 - Upper Pioneer Shale), which is considered one of three main ore hosts for the Copper Chief deposit, has a mean total copper grade that is quite low relative to the Upper and Lower Diabase rock units. Visually and statistically, the total copper distribution within the Upper Pioneer Shale is erratic. Most of the high-grade copper values within the Upper Pioneer Shale are located at or near the contact with adjacent lithologic units. Because of these issues, and because the Upper Pioneer Shale exhibits localized zones of internal waste, this unit was interpolated independently of the other two ore-bearing rock types.

Table 14-4
Copper Chief Drill Hole Copper Assays By Rock Type

	Rock 2	Rock 3	Rock 4	Rock 5	Rock 6	Rock 7	Rock 8
Parameter	Middle Abrigo	Lower Abrigo	Bolsa Quartzite	Upper Diabase	Upper Pioneer Shale	Lower Diabase	Lower Pioneer Shale
Number of Samples	28	446	1,046	950	501	738	1769
Feet of Drilling	149	4,969	9,663	5,672	2,989	4,227	10,466
Mean	0.23	0.224	0.114	0.319	0.171	0.323	0.138
Standard Deviation	0.289	0.19	0.153	0.244	0.205	0.372	0.179
Variance	0.538	0.436	0.391	0.494	0.453	0.61	0.423
Coefficient of variation	1.26	0.849	1.341	0.764	1.198	1.151	1.295
Minimum	0.04	0.03	0.01	0.01	0.02	0	0
Maximum	1.76	2.16	3.15	2.48	2.7	2.59	2.27

Correlograms were modeled for the Middle Abrigo, Lower Abrigo and the Lower Pioneer Shale with anisotropic directions sub-parallel to the overall geometry of these lithologic units. A single three-dimensional global correlogram was modeled for the Bolsa Quartzite, using an azimuth of 305 degrees and a minus 30-degree dip to the northeast. Initial attempts to model individual correlograms for each of the three main ore hosts (Upper Diabase, Upper Pioneer Shale and Lower Diabase) produced very unstable correlogram results. As a result, a combined correlogram for these units was created (Figure 19-2). The nugget of this correlogram is relatively high at 58 percent, reflecting the short range variability of total copper grades within these three combined units. The majority of this variability is associated with the Pioneer Shale. The Upper Diabase has a much lower nugget to sill ratio of 22 percent, which demonstrates that mineralization is less variable and more robust within this unit.

Figure 14-2



15.0 SAMPLE PREPARTION AND ANALYSES

Sample preparation methods and quality control measures employed by Nord and its predecessors were investigated by BETA to ensure suitability for reportable resource and reserve modeling. BETA reviewed sufficient information so as to accept the adequacy of sample preparation, including the process of sample splitting and reduction, and the measures taken to maintain validity and integrity of analytical sample data.

15.1 CYPRUS DRILLING

Cyprus drilling: a former Cyprus geologist, visited the Nord Tucson Office on May 10 -11, 2006. The geologist was the project geologist for most of the BP Series drilling program in the Burro and Copper Chief areas. The geologist's name appears on most of the Cyprus drilling. He provided detailed, first hand information on the drilling, sampling, geology, and QNQC procedures for the Cyprus drilling programs. BETA received details of this visit from an independent consultant to Nord. BETA has reviewed this information, considers it reasonable, and has incorporated it into this technical report.

Core drilling was NQ size (1.8"). Standardized core logging methodology was utilized. The core as delivered from the drilling rig in wooden boxes, and was marked for split by project geologists.

Core splitting was performed on a clean concrete pad to avoid contamination. The former Cyrpus geologist noted that Cyprus employed a dedicated crew to split the core, with samples shipped to an independent certified lab.

Cyprus reportedly used several independent labs as they were developing only the third SX-EW commercial operation to be used. Cyprus used Bottle Roll tests, spectral, and some multi element assays to better understand the deposit.

Cyprus utilized certified assay labs to perform the work. Certified labs must meet QA/QC standards to maintain their certification.

For internal assaying, Cyprus used a composite check system in their lab. Available data indicate a number of cases where the same drill holes were assayed by more than one lab (check assays).

15.2 ARIMETCO DRILLING

An Arimetco geologist performed the geologic logging for the main body of the Arimetco drilling which provided a consistent suite of geologic logging information. Assays were entered in the logs. The drilling was primarily reverse circulation with some core drilling. Standard handling practice was with a sample cone and Jones splitter. Samples were taken to a certified lab in Tucson for assay.

Arimetco submitted sample pulps for check assays to independent outside labs.

Arimetco utilized certified assay labs to perform the work. Certified labs must meet QA/QC standards to maintain their certification.

A former Arimetco Chief Geologist provided additional QAQC comments to Nord's independent geological consultant. Representative assay certificates from the various laboratories used for Johnson Camp assays can be found on file on site.

Based on a review of Arimetco drill hole assay submittal sheets, it was confirmed that to protect sample integrity, an Arimetco geologist personally delivered the drill hole samples to an independent lab for assay.

16.0 DATA VERIFICATION

This section presents a discussion of quality control measures and data verification procedures applied pursuant to the drill hole assay database for the Johnson Camp project.

BETA has verified the data referred to. BETA has reviewed the data verification done by third parties and concurs with their conclusions. Beta has incorporated and adopted such portions of these reports as BETA deemed warranted into this technical report.

In May, 2006, Nord contracted an independent consulting firm to do a detailed review of the data verification at the Johnson Camp. They noted that at least four different major categories or levels of data verification have been completed at Johnson Camp by Cyprus, Arimetco, Summo Corp., and others in evaluating the geological, drill hole, and assay database. Each major category or level of data verification provides a measure of confidence in the database. Taken in aggregate, all four categories provide corroboration and thus a higher degree of confidence in the data. The categories are individual inter-company verifications, intra-company verifications, third party reviews, and mine-to-deposit model production reconciliations. Each is discussed further.

Inter-Company Verifications

Cyprus Copper conducted their drilling and assaying with both internal and external check assay procedures for data verification. Cyprus had samples assayed at more than one external lab for both total copper and acid-soluble copper. Those external labs were (some still are) reputable commercial analytical labs commonly employed by the mining and exploration industry at the time. A QA/QC procedure was also in place whereby Cyprus composited sample pulps and re-submitted the composite for assay as a comparison with the average of individual assays. In addition Cyprus did bottle roll tests on core samples to provide an additional analysis for comparison. While these procedures were not done for every hole and every sample, they were done in sufficient amount to detect either errors in the analytical process or high variability in assays as a result of the geology – and no significant or consistent variances were noted.

The majority of the drill holes in the resource database are core holes drilled by Cyprus. Arimetco drilled with core and by RC methods. While Arimetco may not have had the same quantity of internal or external check assays as Cyprus, Arimetco used an independent lab extensively, a reputable commercial lab still in business today. In addition, the Arimetco basic data, drill logs and assays sheets were done in sufficient quality typical of industry activity at the time (1990’s).

In summary, both mine operators, Cyprus in the late 1970’s to early 1980’s, and Arimetco in the 1990’s, conducted standard documented copper analyses in-house and with external labs, had some degree of QA/QC procedures in place, and detected no significant problems with repeatability or accuracy of copper assays.

Intra-Company Verifications

The Johnson Camp copper deposit was operated by Cyprus and Arimetco, and evaluated by Summo Corporation prior to Nord's ownership of the property. Arimetco conducted drilling and assaying that confirmed the work of Cyprus, and Summo conducted mapping, drilling, and assaying that confirmed the work of Cyprus and Arimetco. It is a very compelling verification procedure when a second and third company does confirmation drilling and assaying, with different drilling techniques and analytical labs, and the data is correlative.

Summo Corp drilled four holes in the Burro Pit and nine in the Copper Chief Pit, as RC drill holes. I examined the assay sheets and drill hole logs for a randomly selected Summo drill hole in the Burro Pit, 98R9, and for adjacent drill holes by Cyprus, BP 39c, and Arimetco, AJ-39. A visual examination indicates the assay values in all three holes have the same general range of copper values, in the same lithological units. While not intended as true twin-holes, each drill hole generally verifies the others.

Third Party Reviews

While the work of Arimetco and Summo in verification of Cyprus drill data may not be considered a completely independent or un-biased verification, as they had similar interests, the work of third party reviewers is usually conducted to a higher standard. Such is the case for Johnson Camp with the work performed by various third party consultants and engineering firms. In 1999, Summo commissioned an engineering firm to complete a feasibility study for Johnson Camp. Nord commissioned an engineering firm to complete a feasibility study in 2000, and an updated feasibility study in 2005 for Johnson Camp. In the opinion of BETA, these firms are known as reputable consulting/engineering companies providing audits, resource/reserve estimations, and feasibility level evaluations to the mining industry. BETA's review of those documentations identified no serious data verification issues, and BETA considers those documentations reasonable. Database errors and/or omissions were found to be few, and within acceptable limits of error.

As is typical of independent Due Diligence and Feasibility level work, in this case resource and reserve estimation, the Summo study examined the drill hole database, geology, assays, bulk density measurements, QA/QC procedures, and completed various block model-to-drill hole comparisons, and reconciliations of the model with historical productions. The drill hole database used was the Arimetco database. The study was able to verify the block model grades of their resource estimate against the drill hole database. This work verifies that the constructed resource block model is representative of the data base, and the examination by the engineering company and the prior operators (as discussed in 2.1 and 2.2) verifies the database. Working in the other direction, verifying the actual production against the model provides the best level of data verification, and those reconciliations are addressed in the following section.

The engineering company attempted independent sampling of remaining core to compare with historical assays; however, a large portion of the split core from Cyrus drilling is no longer available; and assays for samples that have been archived for over 20 years is not a good comparison with the originally fresh core samples. However, of the limited number of samples collected, individual sample variances occur, but globally the grades do not differ much.

Reconciliations

As the drill hole database is the foundation of the resource and reserve estimates, the most significant verification of the drill hole database is the comparison of its derived block model with the production of mined material. This is accomplished by a reconciliation of the drill hole determined block model tonnage and grade against the blast-hole determined tonnage and grade. Reconciliations are noted in the files on a bench-by-bench basis in 1983 by Cyprus, for multiple benches. The results indicate the model generally replicated or slightly underestimated grade, for similar tonnages.

The Summo study compared total historical production with the block model and found both tonnage and grade to be within 0.8% of the combined Cyprus and Arimetco production; a close correlation between the historical production and the database-derived block model.

ADDITIONAL THIRD PARTY REVIEW

A third party consulting firm observed that the basic information upon which verification relies is available for Johnson camp. BETA believes that a sufficient amount of basic data is present for verification including the following:

- Pre-mine topography at 5-foot contours, as a photogrametrically created map, tied to a ground survey triangulation grid;
- Post Mine topography from photogrammetry;
- Drill hole geological logs. Most all are internally consistent as primarily one geologist name is on the logs for each company's logs;
- Copies of daily drill reports;
- Coordinates for ground control triangulation grid;
- Drill core sampling procedures (Cyprus);
- Original or copies of original assay certificates from commercial analytical labs.; in addition to the Cyprus Johnson Camp Mine lab
- Documented sample preparation and analytical procedures;
- Consistency of analytical procedures for copper from Cyprus to Arimetco;
- Several vintages of geological maps, corporate and USGS;
- Rock density procedures by an independent laboratory;
- Blast hole pattern assay maps;
- Monthly and Year-to-Date production records as truck counts to leach dumps;

- Monthly and Year-to-Date actual production (from blast holes) versus forecast production (from the deposit model);
- Pre-feasibility and Feasibility reports;
- Current availability of geological personnel who actually performed some of the work;
- A limited library, but present nonetheless, of core samples and sample pulps.

Nord commissioned a third party consultant to review the applicability of the drill hole data base. BETA has reviewed the work of the third party and considers it reasonable and has incorporated it into this technical report. The consultant endeavored to locate all relevant drill hole and assay data to make complete data sets for the Copper Chief and Burro Pit deposits.

From April 24-26, 2006, the consultant visited the Johnson Camp Mine and prepared a spreadsheet summary listing all available drill hole data. Eighty-five percent of the drill hole logs were readily available with the remaining logs difficult to find. The missing drillhole logs for the ADB series drill holes, assay certificates for the Summo 1998 drilling and other information necessary to fill in the previously missing data were located in April 2006, thus making the most complete set of data possible. The consultant tabulated the rotary, reverse circulation and core drilling done on the Burro and Copper Chief deposits. Additional detailed assay information was added to the summary spreadsheet, and he coded the various assay labs into three categories: primary assay, secondary (check assays), other, and a third column for bottle roll assays.

From May 5-7, 2006, the consultant visited the Nord offices in Tucson, Arizona for the purpose of completing an exhaustive audit of the Copper Chief and Burro Pit deposit electronic database. The consultant verified geologic drillhole logs for the model and verified assay certificates to the electronic database. BETA considers the results of the verification to be quite positive. For example, the consultant checked, and confirmed approximately 40% of the Copper Chief electronic database and found two typographical errors (Hole AJ-63 intervals 296 ft. to 306 ft.). The consultant checked approximately 20% of the Burro Pit electronic data base with one omission (0 ft. to 90 ft. of drillhole BP-47A was not included in the data base containing 86 ft. of 0.23% Tcu and 0.17% AsCu). The current geologic model denotes this interval as 0, thus leaving a void in the current model. Inclusion of this missing interval into the next run of model will add to the resources/reserves in this area.

With the exception of two shallow (-100 ft. drillholes) drilling programs by Cyprus and Arimetco (EXP- series and the CC series drillholes), all the assay certificates for all the data in the electronic database have been located. Additionally, geologic logs for over 95% of the drilling completed in the resource areas have been located and were reviewed.

BETA has reviewed and considers reasonable the verification work and data reviews performed by various independent consultants and engineering firms in this feasibility study, and has incorporated the verification data in this technical report.

17.0 ADJACENT PROPERTIES

The following provides a superficial insight into the history of prior, existing and possible mining operations within the JCM Project's geological setting. The importance of this information is that it describes a geological setting where additional exploration should define additional mineral resources, within the JCM property.

Geologic Potential of Johnson Camp Area

Although evidence strongly suggests that the Johnson Camp deposits are porphyry-related, no dikes or protrusions of Laramide-aged igneous rocks into Paleozoic meta-sediments have been identified on surface or intersected in drilling or underground workings on the Johnson Camp property. Nevertheless, the style of mineralization and the type of alteration present, along with porphyry-related copper mineralization on the adjacent property to the south, all suggest the possible presence beneath the property of a mineralized apophysis or protrusion of the Laramide quartz monzonite stock that is exposed on the western end of the property.

Recent detailed mapping in the present Burro pit has identified pronounced potassic alteration in a portion of the Bolsa Quartzite, consisting of masses of secondary biotite replacing a matrix of quartz and feldspar accompanied by coarse pink k-feldspar (Thorson, 1998). Petrographic examinations of altered Bolsa have confirmed the presence of potassic alteration products (Hansley and Cookro, 1998). Potassic alteration is not commonly found outside of a narrow halo peripheral to porphyry copper deposits. The occurrence of potassic alteration in the bottom of the present Burro pit suggests the possible presence of a buried porphyry deeper in the section. Intense quartz veining, primarily trending parallel to bedding, is also found in the bottom of the Burro pit, in proximity to the potassic alteration. The veins consist of disseminated sulfides (now oxidized) in a dominant quartz matrix. Vein density reaches a high of more than ten per foot in the northwest corner of the pit, immediately above the area of potassic alteration (Thorson, 1998).

In addition to the alteration evidence, a prominent magnetic low anomaly is present between the Burro pit and the Copper Chief deposit. This anomaly supports the possible presence of an extension or apophysis of the intrusion at depth. Also, according to press releases and other public information issued by JABA, Inc. relative to the adjacent property to the south, quartz monzonite was found in core from a drill hole that intersected a large (+5 billion pound) copper deposit located beneath Interstate 10. (Jaba, 1998). BETA has been unable to verify this information and it is not necessarily indicative of mineralization at Johnson Camp.

Other bulk mineable targets lie along strike from the Copper Chief and Burro deposits. The North target lies mostly within the Middle Abrigo Formation, extending up-dip from the old underground mine workings developed along the manto deposits in the Middle Abrigo Formation. Copper oxides are evident along a sparsely prospected slope below the old workings. Similarly, the Keystone-Walnut target lies along strike to the southeast of the Burro deposit, up-dip from the old underground workings of the Keystone Mine. Drilling at the Keystone Mine by the U.S. Bureau of Mines in the late 1940's (Romslo, 1949) cut bedded replacement and disseminated copper over widths exceeding 50 feet in the Middle Abrigo Formation in a number of holes. These holes did not go deep enough to test the Lower Abrigo Formation, which is the dominant host for bulk-mineable deposits on the Johnson Camp property Surface exposures of Middle Abrigo Formation, carrying copper oxides and chrysocolla suggest the presence of a bulk-mineable target up-dip from the USBM drill intercepts from the KeystoneWalnut target.

Several small lead and zinc mines and prospects are found in the Martin and Escabrosa limestones east of the Burro and Copper Chief deposits. The location and distribution of these lead-zinc prospects could be instrumental in determining a property-wide metal zoning pattern, which may be useful in identifying the locus and source of mineralizing fluids, and possibly targets for other copper deposits on the property.

SECTION 18.0 METALLURGY AND PROCESSING

INTRODUCTION

In compiling this section of this Feasibility Study for the Johnson Camp Project, BETA reviewed the previous work of several third party consultants and engineering companies, considers it reasonable, and has incorporated it into this technical report.

Currently, the existing Johnson Camp leach dumps are being managed to control pond inventories and there is no copper production. Sections of the SX/EW facility are under rehabilitation that must be completed prior to restarting cathode production.

Nord intends to resume mining and leaching to produce approximately 25.0 million pounds of copper per year. In order to resume full operation, Nord has relined an existing solution pond, will construct three new lined ponds, will prepare a new stand-alone lined leach pad facility for approximately 60 percent of the new ore that will be leached and will install a three-stage crushing circuit. The SX/EW plant will be rehabilitated to meet production goals and the EW section expanded.

Copper production will originate from both an active leach program of newly mined ore and the residual leaching of the existing old dumps. Dependent on TCu grade, the newly mined ore will be divided into two sub-categories, i.e., the higher grade HG ore (+0.15% TCu recoverable) will be crushed and subsequently stacked, by conveyors. The low grade LG ore will be direct truck dumped on the existing leach pads. Drawing 8304-C-001 is a plot plan for the overall project. Both crushed HG and LG leach ore will be placed on top of the old heaps during start-up. They will be stacked in separate areas of the existing leach pads. NORD will immediately commence scheduling the ore deposition plan. This plan will define the timing of the construction of the new leach pads and their required size. It is NORD's plan to stack the LG ore exclusively on the existing pads and stack the majority of the HG crushed ore on the new proposed leach pad. It is anticipated that the existing leach pads, with the implementation of this stacking regime, will be continuously under leach. Leaching and subsequently rinsing of these existing leach pads will continue as long as an economic PLS grade is produced.

The operating plan for the LG ore is simply extraction and direct truck haulage from the mine for dumping on the existing leach pads. Recoveries from this LG ore are projected to be in excess of +50%.

The operating plan for the HG ore includes mining, crushing this ore to minus one-inch, acidulating and drum agglomerating the crushed ore with sulfuric acid, and conveying the acidulated ore through a series of movable conveyors to the new leach pad. That ore will be acid–cured with a 144-gram-per-liter raffinate solution before conventional leaching commences. The ore will be stacked in 30-foot lifts on both the old heaps and the new pad. This new ore will be leached with a combination of low-grade leach solution (intermediate leach solution – ILS) and raffinate. The highest grade PLS from the new leach pad system will report to the SX plant. Raffinate from the SX plant will be applied to the existing old leach dumps and LG ore for both new and residual copper recovery. Copper will be recovered from the PLS solution utilizing the existing SX circuit and cathode copper will be produced from the expanded EW circuit using stainless steel blanks. In the past, the EW plant has produced copper of five 9s (99.999 percent copper) quality. The future operation will be at low current densities (22 to 23 amps per square foot) as compared to most operations and this should continue to ensure a high cathode quality.

The Nord updated feasibility study examined the throughput capacity of the proposed crushing and conveying circuit and found the equipment to be adequate to meet the production goals. BETA concurs with these findings. Nord has already purchased the primary crushing station from the Newmont Gold Quarry operation for use at Johnson Camp. The study examined the proposed rehabilitation plan developed by Nord, the required rehabilitation of the SX/EW circuit to meet production goals of 20 million pounds of copper per year and also reviewed the EW expansion plans to reach production of 25 million pounds per year and believes the modifications are adequate to reach this target. The study evaluated the solution pumping system and determined that several of the solution pumps are adequate for the system with minor modifications. Again, BETA concurs with these findings.

The new, lined leach pad will be subdivided into several smaller areas so that solution from the dumps can be segregated by copper grade. This will provide the operation with flexibility in managing PLS solution flows and grades. The stacking/loading plan remains pending. The circulation of solutions through the old heaps and LG ore, to the new heaps, will also aid in producing/controlling the PLS grade.

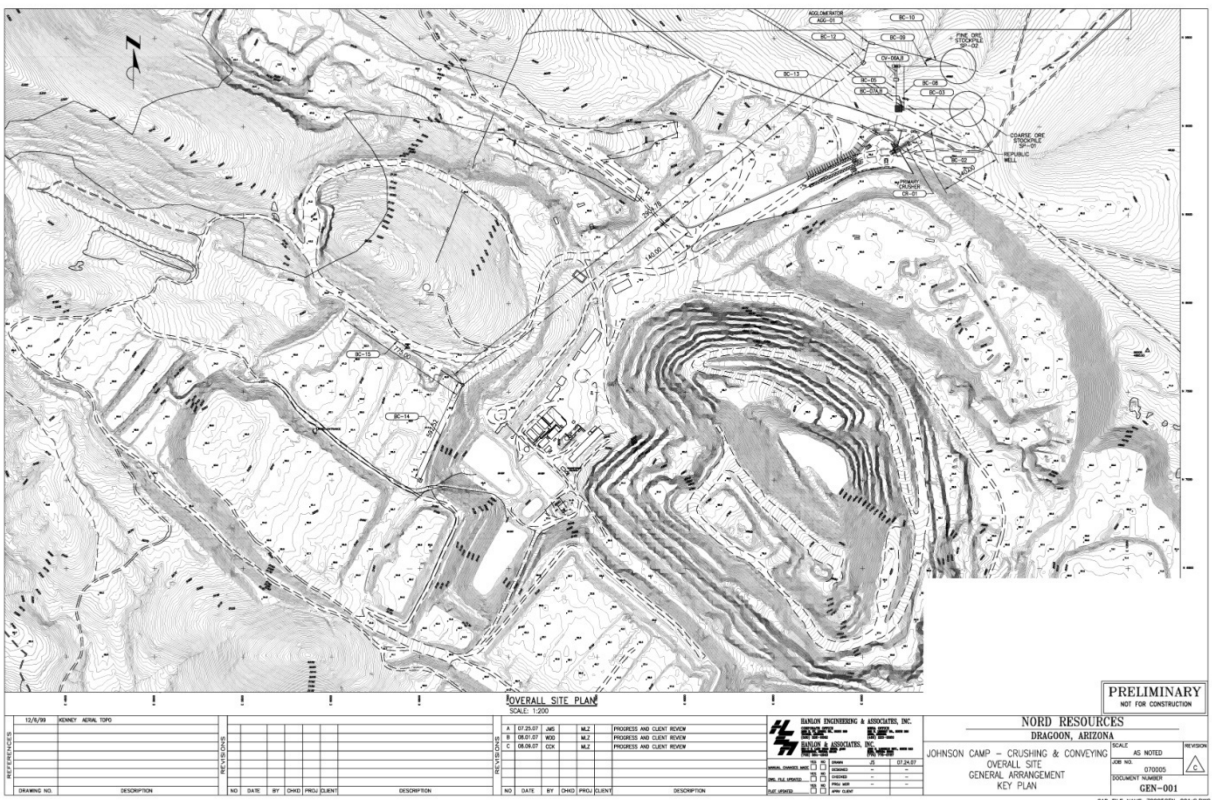
Since 2000 , Nord has upgraded the existing three water wells to a capacity of 600 gpm. For the expansion to 25 million pounds of copper per year, an additional 150 gpm water well is proposed.

18.1 PROCESS DESCRIPTION

The Johnson Camp facility was operated in the 1980s and early 1990s as a dump leach and SX-EW facility producing cathode copper.

Nord intends to resume mining and leaching to produce approximately 25.0 million pounds of copper per year. In order to resume full operation, Nord is currently relining an existing solution pond and will construct three new lined ponds, prepare a new, stand-alone lined leach pad facility for the majority of the 66 million tons of HG ore that will be leached, and plans to install a two-stage crushing circuit. The SX/EW plant will be rehabilitated to meet production goals and the EW section expanded.

Copper production will originate from both an active leach program of newly mined ore and the residual leaching of the existing old dumps. Drawing 18-1 is a plot plan for the overall project. Crushed leach ore will be placed on top of the old heaps initially. The pending stacking/loading plan will define the construction schedule for the new pads. This plan will define the time table for the commencement of stacking the remaining HG leach ore on the new pad, Leaching of direct dump LG ore and the rinsing of the existing stacked ore will continue on the existing pads until the PLS grade is too low for profitable processing.



18.2 PAST PRODUCTION ANALYSIS

Detailed daily or monthly mine production records from both Cyprus' and Arimetco's operations are sparse and incomplete. However, annual summary production data are available and reflect the following open pit mine and heap leach SX/EW copper production from the property to date:

Table 18.2-1: Production Statistics, Cyprus Johnson Copper Company

Year	Ore to Pad (1)	AsCu (%)	Contained AsCu	Pounds Copper Shipped
1975	2,132,260	0.496	21,152,019	6,143,024
1976	1,821,476	0.357	13,005,339	10,059,807
1977	1,563,030	0.399	12,472,979	10,327,424
1978	1,202,500	0.426	10,245,300	10,205,142
1979	1,588,400	0.522	16,582,896	10,032,003
1980	1,499,600	0.411	12,326,712	10,320,407
1981	1,551,500	0.470	14,584,100	10,693,485
1982	1,894,700	0.322	12,201,868	9,702,272
1983	1,962,600	0.504	19,783,008	9,717,616
1984	52,100	0.713	742,946	8,803,361
1985	0	0	0	6,200,836
1986	0	0	0	4,854,796
Sub-total	15,268,166	0.436	133,097,167	107,060,173

(1) Run-of-Mine-Ore

Table 18.2-2: Production Statistics, Arimetco, Inc.

Year	Ore to Pad (1)	TCu (%)	Contained Tcu lbs.	Pounds Copper Shipped
1991	750,100	0.340	5,100,680	5,549,725
1992	2,516,320	0.480	24,156,672	8,156,435
1993	3,259,320	0.340	22,163,376	7,386,504
1994	2,719,690	0.290	15,774,202	5,618,012
1995	2,995,592	0.290	17,374,434	6,345,518
1996*	3,084,254	0.350	21,589,778	9,921,576
1997*	1,254,971	0.379	9,286,785	4,747,995
1998	0	0	0	2,181,304
Sub-total	16,580,247	0.348	115,445,927	49,907,069

(1) Run-of-mine ore except years noted with * where ore was crushed to -3 inches

Recovery of copper by Cyprus totaled 80 percent of the acid soluble copper grade placed on the pad. Cyprus used a variety of analytical techniques to determine acid soluble copper grades during operation, and it is difficult to compare its acid soluble grades with total copper content using a complete digestion. The Summo study estimated the total copper grade of ore mined by Cyprus based upon original drill hole assay data at approximately 0.68 percent TCu (see Section 3.6.7). Using this estimate of the total copper in the run-of-mine ore that Cyprus placed on the heaps renders a recovery of 51 percent of total contained copper. This ore was primarily from the Abrigo Formation, which was the focus of Cyprus' mining activity in the Burro pit. This ore type naturally breaks to a finer screen size when mined than other ore types in the deposit, and also decrepitates readily to predominantly minus six-inch material under acid leach conditions. The finer natural fragment size and decrepitation characteristics may explain the leach recoveries of this ore type.

Recovery of copper by Arimetco totaled only 43 percent of the total copper contained in the mostly run-of-mine ore placed on the heaps. Arimetco mined primarily the Bolsa Quartzite and diabase, two rock types which are significantly harder than the Abrigo Formation and which generate a much coarser natural product than the Abrigo Formation. In addition, the strength of these rock units precludes significant decrepitation under acid leach conditions.

The contrast between the two ore types, (Abrigo ore from Cyprus' operation, and Bolsa Quartzite and diabase ore from Arimetco's operation), are evident on the existing heap by the visible discrepancy in product size. Arimetco realized the difference in metallurgical behavior in 1995, and added a crushing plant in 1996. The initial results from leaching of crushed ore on a new liner system were an increase in solution head grades to the plant and an improvement in recoveries to the point where they matched column leach test work performed on diabase ore at a similar crush size. Unfortunately, crushed ore represented less than 25 percent of the ore that Arimetco had under leach, and by this time the company was in financial trouble and unable to continue operations beyond mid-1997. These operating results, along with the column leach test results, clearly support the need to crush the ore to obtain reasonable recoveries under heap leach conditions.

The Summo study had estimated that the ore placed on the heaps by Cyprus still contained approximately 100 million pounds of copper, and that approximately 26 million pounds of this copper are "soluble", based upon the Cyprus acid soluble data. Factoring that data to allow for the more aggressive procedure used by Arimetco, there could be as much as 50 million pounds of "soluble" copper remaining in the Cyprus ore. The study had estimated that approximately 68 million pounds of copper remain in the Arimetco ore placed on the heaps. Given the slow recovery of copper from this material to date, it is estimated that perhaps as much as half of this copper may be "soluble" under the Arimetco analytical method. Total copper remaining on the heaps totals roughly 168 million pounds, and there may be a total of 70 to 80 million pounds of "recoverable" copper remaining in the existing heaps. However, the current leach kinetics appear to allow for recovery of a maximum of two to three million pounds of copper per year.

In order to verify the dump volume and tonnages for the existing heap leach dumps 1, 2B, 2C, 2D and 3B, the Summo study used a set of AutoCad drawing files of topographic surfaces for the Johnson Camp area. These pre-mining and current topographic surfaces were then gridded to form a wire-frame surface, representing the bottom and top of the dumps, respectively.

A non-orthogonal grid set of cross sections oriented N45°E was constructed on 25-foot centers to aid in the interpretation of the heap leach dumps. Separate solids representing each of the five dumps were modeled using the original and current topographic surfaces, a boundary that was digitized for each dump, and the cross sectional areas interpreted on the non-orthogonal grid set. Polygons were interpreted on most of the cross section planes that bisected each dump. The polygons were interpreted using the original topographic surface as the bottom of the dump, and the August 1998 topographic surface as the top of the dump. These polygons were then linked together in each of the five individual solids. A tonnage factor of 16.25 was used to calculate tons. This factor was derived by applying a 30 percent swell factor to the average tonnage factor of 12.50 for in situ rock. Table 18.2-3 summarizes the volumes and tonnages for the five leach dumps.

Table 18.2-3: Johnson Camp Leach Dump Tonnage Estimates

	Leach Dump Volumes	(Swell Factor - 30%)	
Dump #	Cubic Yards	Cubic Feet	Tons
1	6,419,903	173,337,381	10,666,916
2b	2,089,216	56,408,832	3,471,313
2c	5,327,161	143,833,347	8,851,283
2d	1,773,486	47,884,122	2,946,715
3b	501,372	3,537,044	833,049
Totals	16,111,138	435,000,726	26,629,285

The total reported ore tons mined from the Burro pit by the Cyprus and Arimetco mining efforts was 31,848,413 tons. The 20 percent differences between the Summo study tonnage estimates for the leach dump and the reported ore production by Cyprus and Arimetco can be attributed to several possible factors. First, the 30 percent swell factor may not be uniformly applicable. Decrepitation of dump material over time will cause compaction in the dump, effectively increasing the density of the material contained within a given dump volume. Secondly, slight elevation discrepancies in the aerial surveys used to produce the topographic surfaces can cause significant volume variations. For instance, a one-foot difference in elevation can result in a plus or minus 292,663-ton difference for the leach dumps at Johnson Camp. With a five-foot difference in elevation, the difference in tonnage can be as great as 1,463,315 tons. The opinion of study was that the topographic survey contracted by Summo in 1998 likely has a much greater degree of accuracy than the older original pre-leach pad topographic data. Any errors in elevation probably reside in the older data set. The Nord updated feasibility study also agreed with the findings. BETA has reviewed the analysis and concurred with the conclusions.

18.3 PROJECTED COPPER PRODUCTION FROM EXISTING LEACH PADS

Nord conducted a drilling program to evaluate actual copper content of the existing heaps. The drilling program was conducted to provide an estimate of the copper values in the heaps, but cannot be considered a definitive measure of the copper content of the heaps. A summary of the results is presented in Table 18.3-1. Based on estimated heap tonnages, there are approximately 75 million pounds of acid soluble copper remaining in the heaps.
Actual copper cathode production for Johnson Camp in the last years of operation is as follows;

1999	672,004 lbs
2000	1,632,245 lbs
2001	1,133,914 lbs
2002	495,494 lbs
2003	556,388 lbs
Total	4,490,045 lbs

The above production was achieved by Nord with a significant portion of the heap area not under leach and little or no sulfuric acid makeup to the available leach solution.

Table 18.3-1 Summary of Johnson Camp Heap Drilling

Heap Number	Drill Hole ID	Weighted Copper Total %	AS Cu %	Feet of Sample
1	H1 P-2-2	0.260	0.200	168
H1 P-5-1		0.299	0.265	152
H1 P-5-2		0.321	0.207	177
H1 P-8-1		0.236	0.175	172
H1 P-8-2		0.274	0.190	182
Total				851
Wt. Average		0.278	0.206	
2	H2-1	0.207	0.110	120
H2-2		0.201	0.142	114
H2-3		0.146	0.107	158
H2-4		0.143	0.092	170
H2-5		0.149	0.104	160
H2-6		0.113	0.060	130
H2-7		0.125	0.086	157
H2-8		0.123	0.096	124
Total				1133
Wt. Average		0.149	0.099	
3	H3-1	0.107	0.074	98
H3-2		0.128	0.112	98
H3-3		0.156	0.116	70
H3-4		0.126	0.079	110
Total				376
Wt. Average		0.127	0.093	

Using the time that each dump had been under leach, the estimated feed grade, the estimated recovery to date, and the limit of 80 percent maximum total copper recovery, a shrinking core leaching model was used to predict ongoing copper production as leaching of the existing, old dumps continues. The results of this modeling effort project that the residual copper production from the old heaps would be as follows:

Year	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Lbs Cu (1,000)	2,275	2,600	1,600	1,600	1,600	1,600

The copper production from the dumps was adjusted to reflect the percentage of each dump that will be under leach during the first year. As fresh ore is loaded onto the old heaps, a portion of the available area must be allowed to dry out so that truck access is possible. By year two, all old heaps will be available to accept leach solution.

The Summo study had conducted a detailed examination of the actual copper production during

1998 and 1999. The study compared the copper production estimate using the shrinking core model for 1998 to the actual production achieved at Johnson Camp during 1998. The projected production was 1.96 million pounds of copper, based on the percentage of the total heap actually under leach during this period; the actual production was 2.18 million pounds of copper. This comparison indicated that the shrinking core model projection was a reasonable estimate for future production from the existing, old Johnson Camp dumps. BETA reviewed the analysis and concurred with the conclusions.

18.4 METALLURGICAL TESTWORK

This description of metallurgical testwork is based on third party engineering reports. These reports were reviewed by BETA. BETA concurs with the conclusions arrived at and has incorporated this information into this technical report.

Arimetco commissioned independent consultants to examine the effect of fine crushing of Johnson Camp ore on total copper recovery. The test work indicated that crushing to minus three-inch and minus one-inch increased copper recovery substantially. Summo therefore, as part of their evaluation process in 1999, contracted a metallurgical consultant to design the testing program, initiated a new series of laboratory column tests. The initial test work consisted of eleven eight-inch columns, each containing 135 kilograms of ore. Test work was done at an independent laboratory.

Column tests 11, 12, 13, 14, and 15 were the first laboratory columns to be put under leach. In these columns, the first feed solution was Johnson Camp raffinate. However, the PLS from each column was collected and recycled back to that column. It became apparent after a period of time that the high-grade feed solution to each column was adversely affecting the leaching rate. Raffinate from Johnson Camp was subsequently used as feed solution to the columns to better replicate actual plant condition. In addition, it was discovered on leach day 129 that the free acid determination used by the independent laboratory for PLS and raffinate was not appropriate for the solutions used. The free-acid acid analysis was modified to accurately reflect the free-acid concentration. However, column tests 11, 12, 13, 14, and 15 were run for an extended period of time with insufficient acid. Because of the problems described in this section, the data from these five columns is of limited value for projecting actual recoveries.

Six additional eight-inch columns, tests 16, 17, 18, 19, 20, and 21, were prepared and started when the problems with the first five columns were recognized. Three ore types were tested. For each ore type, one column was charged with ore crushed to minus one-inch while the second column was charged with ore crushed to minus 0.5inch. Plant raffinate was fed to each column. Columns No. 17, 19, and 21 were charged with minus 0.5-inch ore. After 35 days of leaching, it became evident that columns containing minus 0.5-inch ore were leaching at the same rate as the corresponding minus one-inch columns. Therefore, the minus 0.5-inch columns were shut down. These columns were acid washed for ten days after sitting inactive for 50 days. Recoveries were calculated for each of these columns, after the leach period and after the wash period. It is interesting to note that the wash cycle produced approximately six percent more copper, indicating that leaching continued during the 50-day inactive period.

Columns No. 16, 18, and 20 were charged with minus one-inch ore. At day 80 in the leach cycle, sulfuric acid was added to achieve a sulfuric acid content of five grams per liter. The minus one-inch columns were leached for 92 days, then acid and water washed for a total of 10 additional days. After washing, the ore from each column was removed, dried, and screened. A sample of each screen fraction was analyzed to determine the total residue analysis. The calculated feed assay for each column was determined by adding the copper in the residue to the total copper in the pregnant solution. A recovery curve was then developed for each column based on the copper in the PLS and the calculated head assay. Table 18.4-1 lists the pertinent data for each column and this data were used as a basis for projecting actual leach recoveries in the leach dumps.

The data from column tests No. 16, 18, and 20, the three 92-day laboratory leach columns, were extrapolated to 180 days to predict plant recoveries for Burro Pit diabase, Burro Pit Bolsa, and Copper Chief Bolsa ore. Abrigo ore was not tested in the final six columns; therefore, the data from Column No. 11, leached for 182 days was extrapolated to project anticipated recoveries for this ore type. Column 11, as was described previously, was run using recirculating PLS and was deficient in acid so the recoveries were most certainly retarded as compared to a column run exclusively with acidified raffinate.

Actual column data was available for Copper Chief diabase, Burro Pit diabase, Burro Pit Bolsa, and Burro Pit Abrigo ores. From the geological interpretation, it was determined that the Copper Chief Bolsa and Abrigo ores were similar to the corresponding Burro Pit ores and that the same recoveries could be anticipated. These tests also determined that the shale-type ore was similar to Bolsa-type ore, i.e., Bolsa ore recoveries were used for the shale ore.

Table 18.4-1: Summary of Johnson Camp Laboratory Columns

Column No.	Ore Type	Size, inches	Original Head Grades % Cu	Original Head Grades % A/S Cu	Calc. Head % Cu	Residue % Cu	Residue % A/S Cu	Leach Duration Days	Wash Duration Days	Calculated Recovery @ End Leach Cycle %	Calculated Recovery @ End Wash Cycle %
11	Burro Abrigo	minus 1-inch	0.437	0.332	0.426	0.102	0.106	182	10	76.07	
12	Burro Bolsa Mix Ore	minus 1-inch	0.255	0.222							
13	Burro Diabase- si	minus 1-inch	0.194	0.214							
14	Burro Diabase- ox	minus 1-inch	0.751	0.695							
15	Burro Diabase- si	minus 1-inch	0.389	0.356							
16	Copper Chief Diabase	minus 1-inch	0.361		0.375	0.127	0.106	92	10	64.45	66.16
17	Copper Chief Diabase	minus 0.5-inches	0.361		0.341			35	8	58.62	64.47
18	Burro Diabase	minus 1-inch	0.465		0.470	0.112	0.084	92	10	74.57	76.16
19	Burro Diabase	minus 1-inch	0.465		0.432			35	8	70.70	75.22
20	Burro Bolsa	minus 1-inch	0.270		0.263	0.088	0.072	92	10	64.97	66.57
21	Burro Bolsa	minus 1-inch	0.270		0.243			35	8	65.15	71.21

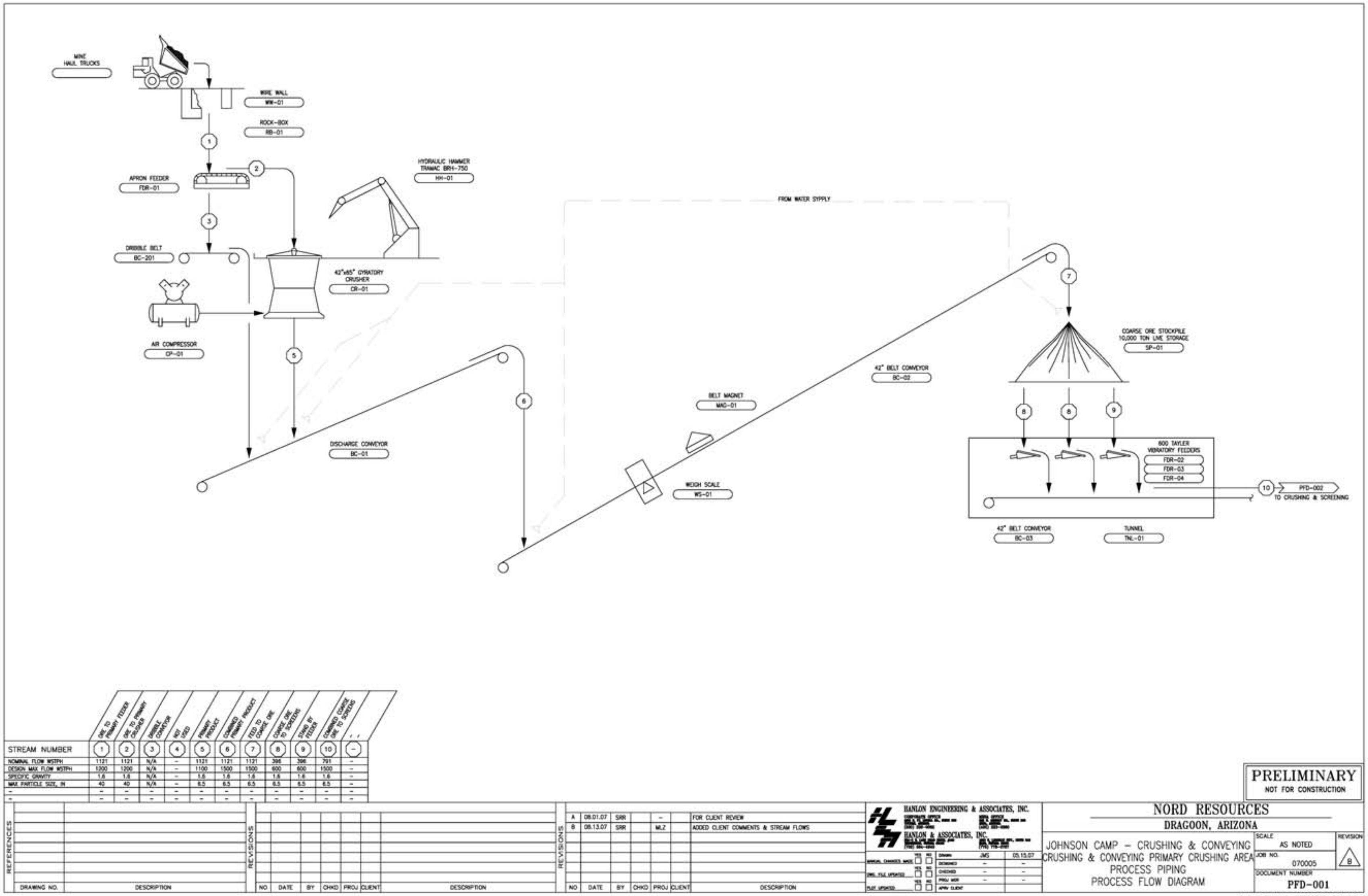
The Summo study, along with an independent metallurgical consultant reviewed all of the laboratory test data that were available. The independent consultant recommended that the laboratory column recoveries be adjusted to reflect the 30-foot actual lift height compared to the 10-foot height used in the laboratory columns. This adjustment resulted in leach cycle times as long as 240 days. He also recommended that the actual recovery curve be adjusted to reflect the effect of soluble copper inventory entrapped in the leach ore that would not be available until final rinse of the pads. This was accomplished by holding back four percent of the extraction copper that would be recovered in the last two years with heap rinsing.

The laboratory column recovery for Bolsa ore was also adjusted to reflect the higher copper grade expected in actual operation (0.516 percent copper) versus the ore grade for the column (0.263 percent copper). A summary of the recovery projections is found in Table 18.4-2. BETA has reviewed this work and concurs with the metallurgical recovery estimates. Test results clearly show the need to crush the ore in order to retain stated recoveries under heap leach conditions.

Table 18.4-2: Recovery Curves

Recovery (Cumulative Percent)

Ore type	Burro Pit Diabase	Cu Chief Diabase	Shale Bolsa	Abrigo
Month				
1	42.0	33.5	34.5	58.0
2	55.0	45.5	47.0	65.0
3	63.0	53.5	55.0	70.5
4	68.0	59.0	61.0	74.0
5	71.0	61.5	64.5	76.0
6	75.0	65.0	67.5	77.8
7	76.0	66.5	69.0	78.5
8	77.0	68.0	70.0	79.0
9	77.5	69.0	71.5	
10	78.0	70.0	72.5	
11	78.5	70.7	73.3	
12	79.0	71.3	74.0	
13	79.5	72.0	74.5	
14	80.0	72.6	75.0	
15	80.5	73.3	75.5	
16	81.0	74.0	76.0	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WIRE WALL
WW-01
ROCK-BOX
RB-01
APRON FEEDER
FDR-01
HYDRAULIC HAMMER
TRAMAC BRH-750
HH-01
DRIBBLE BELT
BC-201
42"x65" GYRATORY CRUSHER
CR-01
AIR COMPRESSOR
CP-01
DISCHARGE CONVEYOR
BC-01
FROM WATER SYPPLY
BELT MAGNET
MAG-01
WEIGH SCALE
WS-01
42" BELT CONVEYOR
BC-02
COARSE ORE STOCKPILE
10,000 TON LIVE STORAGE
SP-01
600 TAYLER VIBRATORY FEEDERS
FDR-02
FDR-03
FDR-04
PFD-002
TO CRUSHING & SCREENING
42" BELT CONVEYOR
BC-03
TUNNEL
TNL-01
STREAM NUMBER
ORE TO PRIMARY FEEDER
ORE TO PRIMARY CRUSHER
DRIBBLE CONVEYOR
NOT USED
PRIMARY PRODUCT
COMBINED PRIMARY PRODUCT
FEED TO COARSE ORE
COARSE ORE TO SCREENS
STAND BY FEEDER
COMBINED COARSE ORE TO SCREENS
NOMINAL FLOW WSTPH: 1121 1121 N/A – 1121 1121 1121 398 398 791 –
DESIGN MAX FLOW WSTPH: 1200 1200 N/A – 1100 1500 1500 600 600 1500 –
SPECIFIC GRAVITY: 1.6 1.6 N/A – 1.6 1.6 1.6 1.6 1.6 1.6 –
MAX PARTICLE SIZE, IN: 40 40 N/A – 6.5 6.5 6.5 6.5 6.5 6.5 –
REFERENCES
DRAWING NO.
DESCRIPTION
REVISIONS
NO DATE BY CHKD PROJ CLIENT DESCRIPTION
A 08.01.07 SRR – FOR CLIENT REVIEW
B 08.13.07 SRR MLZ ADDED CLIENT COMMENTS & STREAM FLOWS
HANLON ENGINEERING & ASSOCIATES, INC.
HANLON & ASSOCIATES, INC.
DRAWN JMS 05.15.07
DESIGNED
CHECKED
PROJ MGR
APRV CLIENT
PRELIMINARY
NOT FOR CONSTRUCTION
NORD RESOURCES
DRAGOON, ARIZONA
JOHNSON CAMP – CRUSHING & CONVEYING
CRUSHING & CONVEYING PRIMARY CRUSHING AREA
PROCESS PIPING
PROCESS FLOW DIAGRAM
SCALE
AS NOTED
JOB NO.
070005
REVISION
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PFD-001
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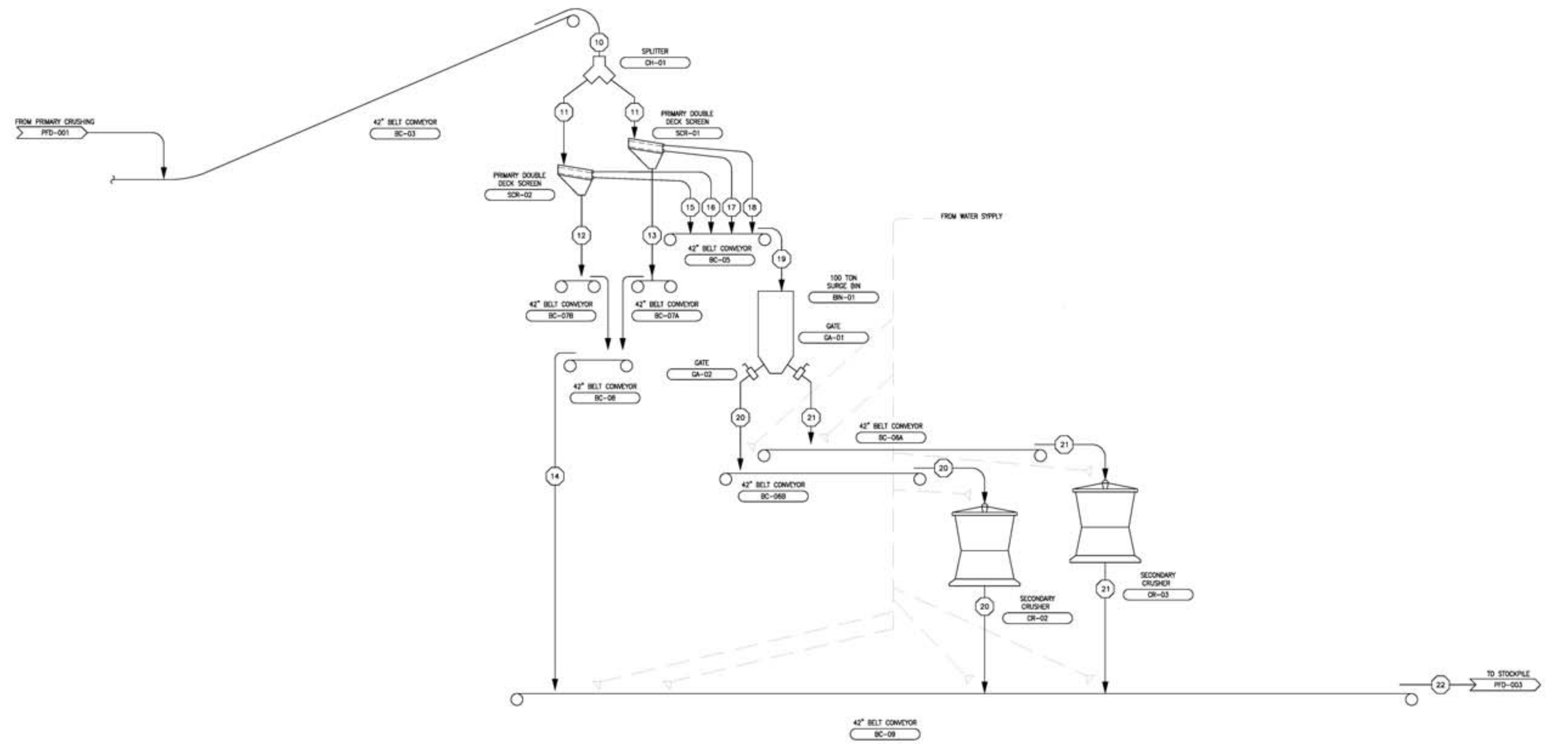


STREAM NUMBER	10	11	12	13	14	15	16	17	18	19	20	21	22
	COMBINED COARSE ORE TO SCREENS	SPLIT FEED TO PRIMARY SCREENS	FINES TO STOCKPILE	FINES TO STOCKPILE	COMBINED FINES TO STOCKPILE	MIDDLE CUT TO SECONDARY	TOP CUT TO SECONDARY	MIDDLE CUT TO SECONDARY	TOP CUT TO SECONDARY	COMBINED TO SECONDARY	FEED TO CR-02	FEED TO CR-03	COMBINED FEED TO STOCKPILE
NOMINAL FLOW WSTPH	791	396	95	95	190	–	–	–	–	601	301	301	791
DESIGN MAX FLOW WSTPH	1500	550	1500	1500	1500	–	–	–	–	1500	1500	1500	1500
SPECIFIC GRAVITY	1.6	1.6	–	–	–	–	–	–	–	1.6	1.6	1.6	1.5
MAX PARTICLE SIZE, IN	6.5	40	1	1	1	–	–	–	–	6.5	6.5	6.5	0.8
–	–	–	–	–	–	–	–	–	–	–	–	–	–
–	–	–	–	–	–	–	–	–	–	–	–	–	–

REFERENCES

REVISIONS

REVISIONS						
A	08.01.07	SRR		–		FOR CLIENT REVIEW
B	08.13.07	SRR		MLZ		ADDED CLIENT COMMENTS & STREAM FLOWS

HANLON ENGINEERING & ASSOCIATES, INC.

HANLON & ASSOCIATES, INC.

MANUAL CHANGES MADE YES NO
DWG. FILE UPDATED YES NO

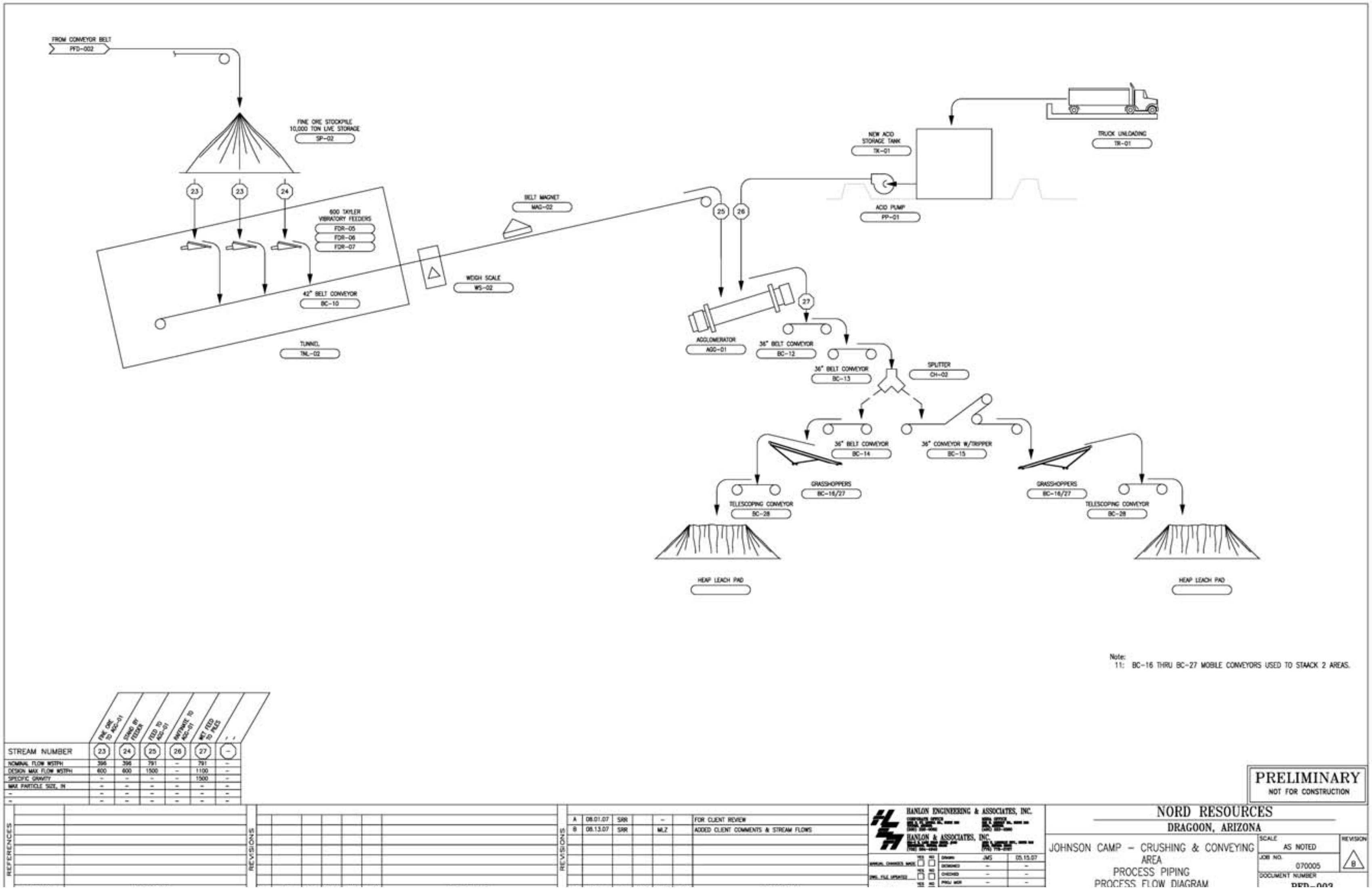
DRAWN	JMS	05.15.07
DESIGNED	–	–
CHECKED	–	–
PROJ MGR	–	–

PRELIMINARY
NOT FOR CONSTRUCTION

NORD RESOURCES
DRAGOON, ARIZONA

JOHNSON CAMP – CRUSHING & CONVEYING
SECONDARY CRUSHING & SCREENING AREA
PROCESS PIPING
PROCESS FLOW DIAGRAM

SCALE AS NOTED
JOB NO. 070005
REVISION B
DOCUMENT NUMBER PFD-002



STREAM NUMBER	23 FINE ORE TO AGG-01	24 STAND BY FEEDER	25 FEED TO AGG-01	26 RAFFINATE TO AGG-01	27 WET FEED TO PILES	–
NOMINAL FLOW WSTPH	396	396	791	–	791	–
DESIGN MAX FLOW WSTPH	600	600	1500	–	1100	–
SPECIFIC GRAVITY	–	–	–	–	1500	–
MAX PARTICLE SIZE, IN	–	–	–	–	–	–
–	–	–	–	–	–	–
–	–	–	–	–	–	–

18.5 PROCESS DESIGN

18.5.1 Plant Layout

The refurbished Johnson Camp facility will be of conventional design and will use as much of the existing Johnson Camp equipment and ancillaries as possible. The existing SX-EW plant will be rehabilitated to meet future production goals. Three existing solution ponds will be relined to meet current environmental standards. A new raffinate pond will be constructed in the plant area. A new leach pad and new combined PLS/ILS pond, plus a storm water pond, will be constructed in an area northeast of the existing plant facilities. A new crushing system will be installed in an area convenient for mine access by truck. The conveyor routing from the crusher to the new leach pad will not interfere with mine traffic.

An independent engineering firm prepared a report for the existing heaps and solution ponds, which includes the design of the various lined ponds, the design of the new heap leach pads and ancillary piping, a water balance, and capital costs for the construction. This report was submitted to the Arizona Department of Environmental Quality, Aquifer Protection Program, pursuant to the Consent Order signed between the Arizona Department of Environmental Quality and Nord Copper Corporation. Additionally, the independent engineering firm prepared a report on the north area leach pad for the new leach pad system to be located northeast of the existing plant facility, includes the design of the various lined ponds, water balance calculations, the design of the new heap leach pads and ancillary piping, and capital costs. The current plan involves placing more ore on both the existing and proposed new heaps than previously anticipated in the report.

The new leach pad area is to be located northeast of the existing plant facility and is to be designed such that leach solutions flow by gravity into the new combined ILS-PLS pond located down slope of the new leach pad. However, the PLS solution must be pumped back to the existing SX plant. A storm water pond is also provided. A new sulfuric acid storage tank will be installed in the plant area. The tank will have a capacity of 30,000 gallons, equivalent to 225 tons of acid.

The process design criteria is presented in Section 27 of this report.

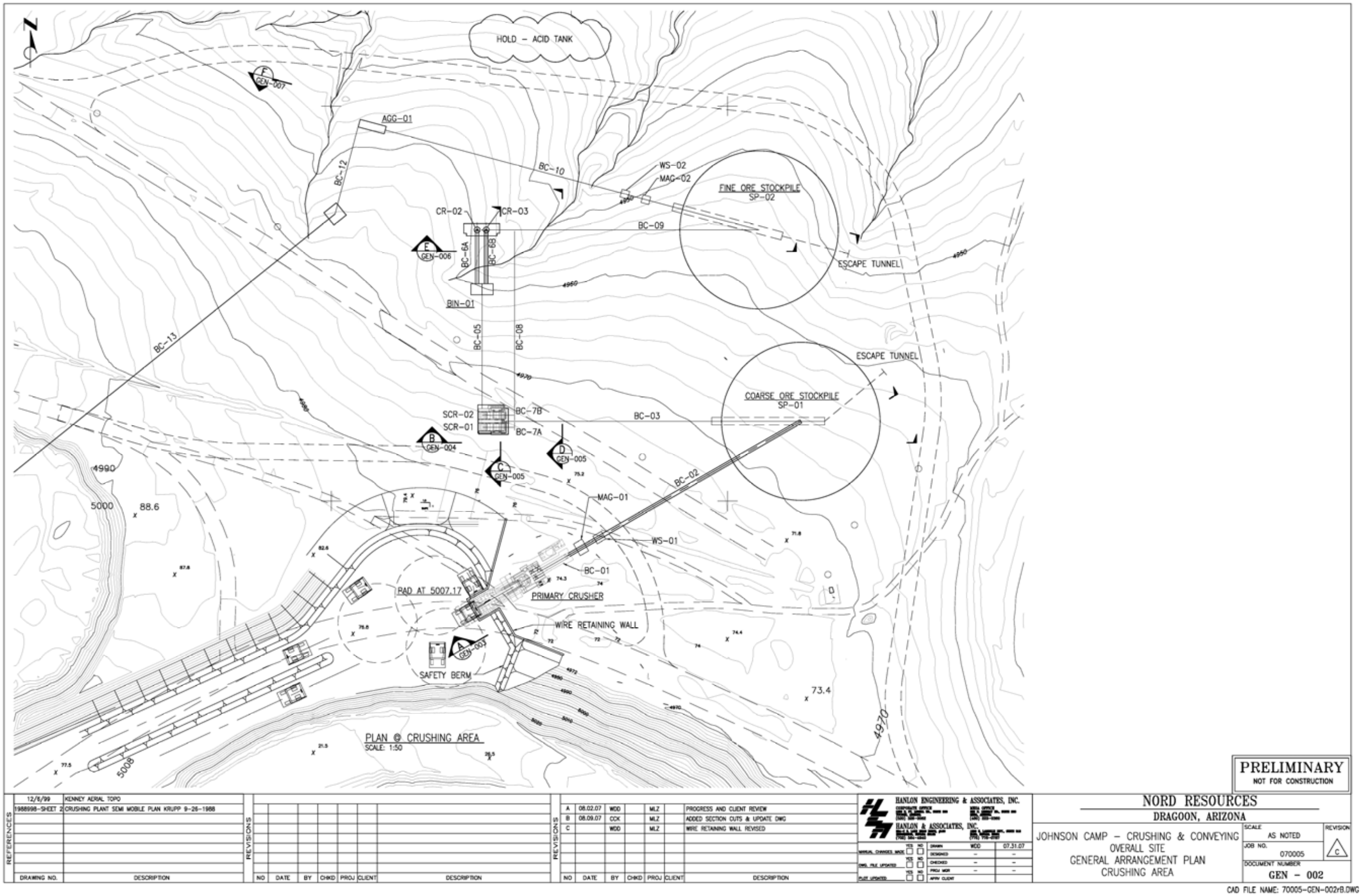
18.4.2 Ore Crushing

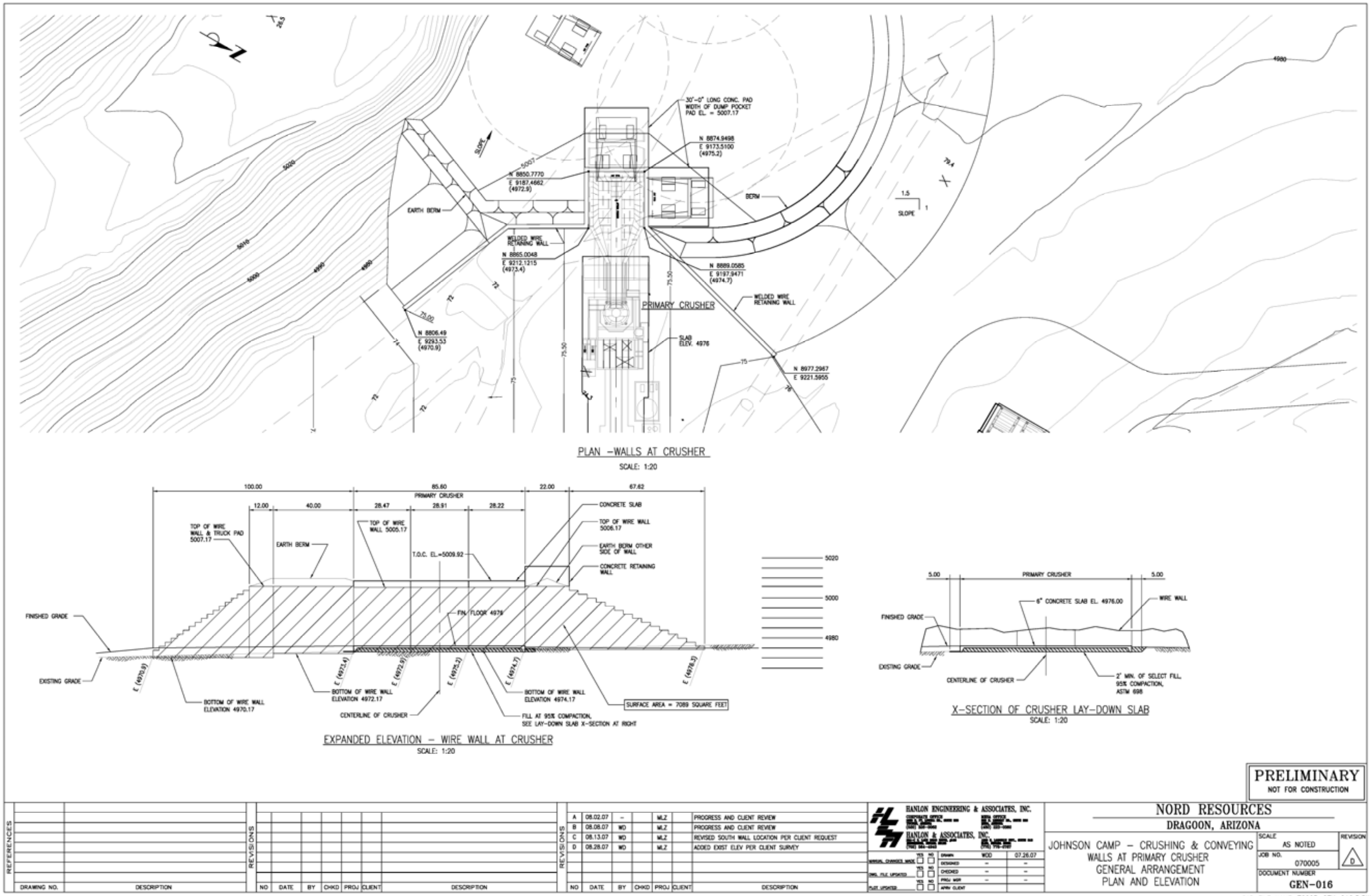
HG Ore from the mine will be directly dumped into the hopper ahead of the gyratory crusher station. Nord has purchased the Newmont Gold Quarry primary station which consists of a 200-ton hopper, rock breaker, 72-in x 58-ft apron feeder and 42 x 65 gyratory crusher. The primary crusher product is then stacked onto a coarse ore stockpile that has a 9,000-ton live storage capacity. The primary crusher typically runs five days per week and two shifts per day, with an average throughput of 900 dstph out of a rated capacity of 1,500 dstph. This schedule will be adjusted to an optimum schedule in concert with the mining contractor after final, detailed mine schedules are estimated.

The ore is withdrawn by three vibrating feeders and is fed to the double-deck scalping screen. Any plus 1.5-inch ore is conveyed into one of two H6800 cone crushers, each which operate with a 0.75-in close side setting. The scalping screen fines and secondary cone crusher discharge are subsequently stacked onto a fines stockpile. Dust collection will be installed throughout the crushing and screening system. An equipment list is presented in Table 18.5.2-1. The crushing plant location and layout is shown in the following general arrangement drawings. The secondary crushing circuit typically runs five days per week, 24 hours per day (19.2 operating hours at 80 percent availability) at an average throughput of 960 dstph out of a rated capacity of 1,050 dstph. Flexibility exists with the stockpiles to operate the secondary plant more shifts if it becomes necessary.

The crushed ore is agglomerated in a rotary agglomerator using acidified raffinate and is then conveyed to the leach dumps using two main overland conveyors and a series of grasshoppers. The ore is placed using a stacker with a telescoping conveyor. The stacking rate will be approximately 1,000 dstph depending on the mine plan.

As noted above, Nord has purchased the Newmont Gold Quarry primary station, which has been relocated from Nevada. Nord has been working with ThyssenKrupp on costs and preliminary designs for the crushing plant. Nord has used ThyssenKrupp for the engineering and procurement of the crushing circuit.


HOLD - ACID TANK
AGG-01
BC-12
BC-10
WS-02
MAG-02
FINE ORE STOCKPILE
SP-02
CR-02
CR-03
BC-09
BC-6A
BC-6B
BIN-01
ESCAPE TUNNEL
BC-13
BC-05
BC-08
ESCAPE TUNNEL
COARSE ORE STOCKPILE
SP-01
SCR-02
SCR-01
BC-7B
BC-7A
BC-03
BC-02
MAG-01
WS-01
BC-01
RAD AT 5007.17
PRIMARY CRUSHER
WIRE RETAINING WALL
SAFETY BERM
PLAN @ CRUSHING AREA
SCALE: 1:50
4990
5000
88.6
73.4
4970
PRELIMINARY
NOT FOR CONSTRUCTION
12/6/99 KENNEY AERIAL TOPO
1988998-SHEET 2 CRUSHING PLANT SEMI MOBILE PLAN KRUPP 9-26-1988
A 08.02.07 WDD MLZ PROGRESS AND CLIENT REVIEW
B 08.09.07 CCK MLZ ADDED SECTION CUTS & UPDATE DWG
C WDD MLZ WIRE RETAINING WALL REVISED
HANLON ENGINEERING & ASSOCIATES, INC.
HANLON & ASSOCIATES, INC.
NORD RESOURCES
DRAGOON, ARIZONA
JOHNSON CAMP - CRUSHING & CONVEYING
OVERALL SITE
GENERAL ARRANGEMENT PLAN
CRUSHING AREA
SCALE AS NOTED
JOB NO. 070005
REVISION C
DOCUMENT NUMBER GEN - 002
CAD FILE NAME: 70005-GEN-002rB.DWG


PLAN -WALLS AT CRUSHER
SCALE: 1:20
PRIMARY CRUSHER
EXPANDED ELEVATION - WIRE WALL AT CRUSHER
SCALE: 1:20
SURFACE AREA = 7089 SQUARE FEET
X-SECTION OF CRUSHER LAY-DOWN SLAB
SCALE: 1:20
PRELIMINARY
NOT FOR CONSTRUCTION
HANLON ENGINEERING & ASSOCIATES, INC.
NORD RESOURCES
DRAGOON, ARIZONA
JOHNSON CAMP - CRUSHING & CONVEYING
WALLS AT PRIMARY CRUSHER
GENERAL ARRANGEMENT
PLAN AND ELEVATION
AS NOTED
070005
GEN-016
CAD FILE NAME: 70005-GEN-016rD.DWG

18.5.3 Ore Handling and Pad Loading

18.5.3.1 First Years of Operation

Subject to finalizing the stacking program, it is planned that during the first two years of operation, crushed HG ore will be conveyed and stacked in 30-foot lifts onto the existing leach pads. Additionally, the LG ore will be direct truck dumped on these existing leach pads. A total of 2.9 million square feet of leach area are available on top of the old heaps. In addition, there is a "slot" available on the pads that can handle three million tons of ore. Placement of the 3.5 years of ore would consume approximately 60 percent of the current pad capacity. There is space available to expand the existing pad. Switching to the new pad is primarily based on providing added space for the HG ore which will improve recovery kinetics. Detailed pad scheduling will define when the switch to the new pad will occur.

18.5.3.2 Remainder of Operation

The remaining HG ore tons scheduled to be mined will be conveyed from the crusher site to the new leach pad using the conveyors relocated from the old heap. This conveyor switchover remains to be programmed. The switchover will require a two-week to three week shutdown, however the crushing and conveying systems will be able to catch up the production by operating seven days and at closer to maximum rated capacity. This schedule will be adjusted to an optimum schedule in concert with the mining contractor after final, detailed mine schedules are estimated.

Table 18.5.2-1: Crushing and Conveying Equipment List

Equip. #	Name	Description	Hp/ each	Source
FDR-01	Feeder	Primary crusher feeder, 72 in x 58 ft	120	ThyssenKrupp Existing
CR-01	Primary crusher	42x65 Superior gyratory	400	ThyssenKrupp Existing
CV-20	Conveyor	Apron feeder dribble conveyor, 78 in x 55 ft	10	ThyssenKrupp Existing
CV-01	Conveyor	Primary crusher discharge conveyor 60 in x 75 ft.	60	ThyssenKrupp Existing
CV-02	Stacker	Coarse ore stacker, 42 in x 300 ft	200	New
FDR-02, 03, 04	Feeder	Coarse ore Reclaim Feeders, vibrating	20	New
CV-03	Conveyor	Coarse ore reclaim conveyor, 42 in x 200 ft	60	New
CV-04	Conveyor	Screen feed conveyor 42 in x 175 ft	100	New
SCR-01	Screen	Double deck screen, 8 ft x 24 ft, top deck 3 inch opening, bottom deck 1.5 inch opening	50	New
SCR-02	Screen	Double deck screen, 8 ft x 24 ft, top deck 3 inch opening, bottom deck 1.5 inch opening	50	New
CR-02	Secondary Crusher	Sandvik H6800 high speed cone	400	New
CR-03	2nd Secondary Crusher	Sandvik H6800 high speed cone	400	New
CV-05	Conveyor	Crusher fines conveyor, 42 in x 350 ft	150	New
CV-06	Conveyor	Crusher fines conveyor, 42 in x 350 ft	150	New
FDR-006, 7,8	Feeder	Fine ore Reclaim Feeders, vibrating	20	New
CV-08	Conveyor	Agglomerator feed conveyor, 42 in x 100 ft	50	New
AGG-001	Agglomerator	10 ft dia x 36 ft, FEECO rotary drum; agglomerator, with Neoprene bonded liner, rubber liners and added corrosion protection in feed chamber, drum and discharge.	200	New (Feeco)
CV-09,10	Conveyor	Overland, 36 in x 4000 ft	400	New / used refurbished
CV-11 thru 18	Conveyor	Grasshopper conveyors, 36 in x 100	40	New
CV-19	Stacker	Heap stacker, 36 in x 150 ft	70	New
LUB-01	Primary System	Lube & Hydraulic motors	20	
LUB-02	Sec. Crusher	Lube & Hydraulic motors	20	
LUB-03	Tert. Crusher	Lube & Hydraulic motors	20	

18.5.4 Leaching

18.5.4.1 Operation on Existing Heaps

Prior to placing the new ore on top of the existing heaps, the Arimetco pad liner on Pad 1 will be removed together with the ore atop the liner. Costs for this are included in the year one capital. Removal of this liner will ensure that the lower ores can be leached. Other items that have changed from the earlier heap leach practice, for ore at Johnson camp are:

1. Agglomeration of the HG ore with acidified raffinate.
2. Placement of the HG ore with conveyors.
3. Solution application with emitters.
4. Solution collection with either rehabilitated or new ponds that meet the requirements of ADEQ.
5. Installation of new pump stations.

All of these will serve to improve the operating efficiencies and performance. The proposed leach solution application schedule is 0.004, 0.002, and 0.001 gallons per minute per square foot for the first four months, the second two months, and the duration of the leach cycle, respectively. Depending on the mining schedule, the leach solution flow rate will vary from 4,500 to 11,000 gpm.

As the crushed HG ore is placed on a new pad, leaching of the existing ore as well as the direct dumped ore will continue on the old heaps. The benefit of having separate leach pads is that the grade of the PLS from the existing heaps will be optimized by pumping this same PLS to the new pad for recirculation through the new ore. The addition of the direct dumped LG ore will ensure that the old heaps will continue to be leached, during much of the mine's life. As the copper gained from the existing ore falls below an economic cut-off, those segments of the existing heap will be taken off leach.

18.5.4.2 Remainder of Operation

The area of the new leach pad requires further engineering to ascertain the required dimensions given the current ore schedule. It is anticipated that the HG ore will be stacked in four 30-foot lifts. The final heap will be 120 feet high. Approximately 10 million square feet of area is available for new leach pad construction. As discussed, the final detailed pad schedule will define the construction timetable for this new leach pad.

The new leach pad pumping system will consist initially of a single new PLS/ILS pond (divided pond) and a new storm water pond will be constructed.

When the new leach pad and solution ponds are completed, crushed and acidulated ore will be conveyed and stacked on the new pad in 30-foot lifts. When sufficient ore is placed on the new pad, leach piping will be installed. Intermediate solution from the old intermediate leach pond will be pumped to the fresh, acidulated ore and applied to the stacked ore using emitters. The proposed application schedule is 0.004, 0.002, and 0.001 gallons per minute per square foot for the first four months, the second two months, and the duration of the leach cycle, respectively. The liner on the pad will be designed to segregate solution flow from discrete areas of the pad. With this design, high-grade and lower-grade PLS solutions can be segregated with the highest grade solution directed to the final PLS pond and on to the SX plant. All remaining lower-grade solution will be directed to the ILS pond. As the leach area expands and solution flow from the new pad exceeds the capacity of the SX plant, the highest-grade solution will be advanced to the SX plant while any excess lower-grade solution will be re-circulated and applied to freshly cured ore. Make-up solution for partially leached ore on the new pad will be intermediate solution generated from residual leaching of the existing and LG ore heaps and pumped from the old ILS pond.

The maximum solution flow to the new heap is estimated to be approximately 7,000 gallons per minute. Of that total, approximately 4,450 gallons per minute will be recycled to the pad and 2,550 gallons per minute will be advanced to the SX plant.

18.5.5 Pad Loading Schedule

18.5.5.1 First Four Years of Operation

Both HG and LG ore will be placed on the existing heap surface during the first two years of operation. Crushed ore will be placed at a rate of 15,000 tons per day or 330,000 tons per month, however the rate and schedule will vary depending on the mine and crushing plant schedules, with the goal of producing the most economical total plan. The plastic liner underlying the final lift on old heap No. 1 will be removed before placing new ore.

The available leach capacity for the old heaps based on three lifts is estimated as follows:

Area Name	Ore Placed, tons x 1,000
1	5,191(1)
2X	1,734(1)
3	2,584(1)
2	1,947(1)
1 & 2 Slot	2,426(1)
Total	13,822

1) The surface of the old heaps is irregular with ore lift surfaces at several different elevations. Therefore, the total tonnage required to level the surface of the first lift is less than might be anticipated from simple area calculations.
2) In addition, approximately 1.3 million sq. ft. of area is available immediately adjacent to the existing heap for the construction of new leach pad #4 as required.

Approximately 13.88 million tons of crushed ore will be placed on the old heaps in a 30-foot lift during the first four years of operation. This compares favorably with the quantity of LG ore projected. The remaining required HG ore tonnage will be placed on the new pads.

HG ore will be placed by overland conveying and stacking after agglomeration. The conveying and stacking will operate five days per week and typically sixteen hours per day with two eight-hour shifts. The two remaining days per week will be available to move conveyors and associated equipment as required.

Solutions from the heaps will be managed from three different collection points. Solution from Heap 1 will report to PLS pond 1. Solutions from Heaps 2, 2X, and 1&2 slot will report to the 1&2 slot sump. Solutions from Heap 3 will report to PLS pond 3. All solutions will have the capability to be sent to the SX facility or to the ILS pond for further stacking to another heap. This flexibility will provide for not only the ability to optimize the PLS grade, but also to manage solution flows to meet the leaching durations for individual ore types.

18.5.5.2 Remainder of Operation

The new leach pad will be compartmentalized in at least three separate segments: the east pad, the north pad, and the south pad. The north and south pads are located due west of the east pad.

Solutions from the east pad will flow to a lined, double-ditch system located on the east side of that pad. In this manner, high-grade and low-grade solutions can be segregated. A central corridor will be constructed between the north and south pads. The corridor will be of sufficient width to contain the conveyor and a lined, double ditch for each pad.

The east leach pad at four million square feet will be constructed before ore loading commences on the new pad. This area is sufficient to place approximately nine million tons of ore in two 30-foot lifts, equivalent to two years of ore production. Ore placed on the first lift will be leached for ten months and then allowed to drain for two months. After the two-month drying period, the second 30-foot lift will be placed on the drained first lift.

Again, subject to detailed scheduling, HG ore will be placed on the north and south pads in two lifts. Ore will be placed on these two pads, first placing a 30-foot lift on one pad, then placing a 30-foot lift on the second pad. A second 30-foot lift will then be placed on the same pads. After reaching a level of two lifts, the scheduling will be optimized with regard to conveyor movements.

The ore-mining schedule is found in Section 4.4. Nord will use this mining schedule to develop the pad loading sequence and also to determine leach solution requirements for both the old heaps and the new pad. Previously, Nord had used these material balances to size pumps and to balance solution flows among the various dumps. No significant deviation from this prior work is anticipated The existing pumping system for the old heaps is adequate to pump required leach solutions to the proper location. The pumping system for new leach pad will require relocation of several pumps and new piping from the SX plant to the new leach pad and new piping for the required circulation around the new pad. The pumping systems are designed to provide significant flexibility with regard to volume pumped and solution routing. BETA believes that the leaching plan and pad design will provide the operator with adequate flexibility to maintain high-grade PLS flow to the SX plant.

18.5.6 SX/EW

PLS solution will be pumped to the SX plant at approximately 2,550 gallons per minute. The solution flow will be split and directed to two SX trains, each consisting of two extraction stages designed to operate in series followed by a single strip stage. New pumper-mixers will be installed in each of the five mix boxes that do not have a new mixer. The testing of the new mixer indicated a significant savings in organic losses to the raffinate over the Arimetco mixers. Rich electrolyte from the strip sections will be filtered, heated, and distributed to the EW cells. The EW plant consists of an old section consisting of 56 cells, each containing 21 cathodes and the new section made up of 16 cells, each containing 36 cathodes. In addition a third set of cells (termed expansion) will be installed that are equal to the new section (16 cells). Cathodes will be stripped from the stainless cathodes using a new automated stripper. Other improvements included in the SXEW modifications are a new cell house crane, a new boiler and associated heat exchanger, a new set of electrolyte filters, a clay filter press and an upgrade to the transformer. The rectifier that is installed on the new section needs only a minor modification, and reinsertion of an existing electrical board, to be able to handle the added voltage from the 16-cell expansion. These additions will augment the many modifications made to the original plant such as replacement of wooden decks in the EW section, new mix box footers in the SX, re-piping the SX, and re-piping in the tank farm.

18.5.7 Water Balance

Since 2000, Nord has expanded the water system to a current capacity of 600 gpm. Included in the expansion was upgrading the Republic well from 100 gpm to 300 gpm and increasing the Moore from 100 gpm to 200 gpm. The Section 19 well has a capacity of 100 gpm. The Burro Pit contains an estimated 32 million gallons of water. A temporary pumping system will be installed with a capacity of 250 gallons per minute. The pit will be emptied in approximately 90 days.

To meet the requirements to produce 25 million pounds per year of cathode, the Nord updated feasibility study called for the installation of a new 150-gpm water well.

Photo 1 - Plant site looking north. Tank house is located on the right while extractors and settlers are housed on the left side. The raffinate pond is in the foreground.

Photo 2 - Plant site looking southeast. Extractors lies in the foreground, tank house in the middle ground and the Burro Pit lies in the background.

19.0 MINERAL RESOURCE AND RESERVE ESTIMATES

19.1 BURRO RESOURCE ESTIMATION

Geologic Framework

The Summo study used the geologic interpretation that was developed by Arimetco as a basis for constraining grade estimation in the feasibility study block model. BETA checked this information and found it to be adequate for resource modeling purposes, and incorporated it into this feasibility study. The interpretation consisted of digital geologic outlines for each of the eight rock types within the Burro deposit. Arimetco generated these outlines on 20-foot-spaced bench plans. This interpretation incorporated a series of northeast trending normal faults that result in minor offsets to the stratigraphic units. In general, the apparent horizontal component of these offsets ranges from 10 to 100 feet. This interpretation is consistent with field observations and geologic mapping that was performed by independent consultants and reviewed by BETA.

Table 19-1 summarizes the rock codes that were used in the Burro block model.

Table 19-1
Lithologic and Mineral Zone Codes in the Burro Pit Model

Lithologic Unit	Lithologic Code	Zone Code	Tonnage Factor
Upper Abrigo	1	1	12.46
Middle Abrigo	2	1	12.46
Lower Abrigo	3	2	12.46
Bolsa Quartzite	4	3	16.61
Upper Diabase	5	4	11.33
Upper Pioneer Shale	6	4	12.00
Lower Diabase	7	4	11.33
Lower Pioneer Shale	8	5	12.00
Default Rock Code	9	N/A	12.50
Dump Material	10	N/A	16.25

Arimetco geologists combined the eight individual rock units within the Burro deposit into five zones having similar mineralogical and statistical characteristics. Zone 1 is comprised of the Upper and Middle Abrigo units. Zones 2 and 3 consist entirely of the Lower Abrigo and Bolsa Quartzite, respectively. The Upper Diabase, Lower Diabase and the Upper Pioneer Shale were combined into Zone 4, and Zone 5 consists of only the Lower Pioneer Shale. These mineral zone codes are summarized in Table 19-1.

Rock Density

As part of its due diligence efforts, Summo collected four bulk samples of the main ore host rocks from within the existing open pits on the Burro and Copper Chief deposits. These samples included three from the Burro pit (Abrigo ore, Bolsa Quartzite ore, and Diabase ore below 150-foot depth beneath original topographic surface) and one from the much smaller Copper Chief pit (Diabase ore above 150-foot depth beneath original topographic surface). Bulk density determinations for these samples were made by Core Laboratories Incorporated, of Bakersfield, California. In BETA's opinion, the results of these determinations are consistent with bulk densities of similar rock types in other known deposits, and the values obtained are consistent with historical tonnage production data assembled by Cyprus and Arimetco.

Bulk sampling did not include a sample of Pioneer shale: a tonnage factor of 12.00 cubic feet per ton was used for this rock type based on density data obtained from published reference sources. For existing waste dumps, a tonnage factor 16.25 cubic feet per ton was assumed. This assumption was based on an average insitu tonnage factor of 12.50 cubic feet per ton for all rock types and a standard swell factor of 30 percent.

3-D Block Model Limits

A MEDSYSTEM three-dimensional block model was created for the portion of the Johnson Camp district that contains both the Burro and Copper Chief deposits. Table 19-2 summarizes the coordinate system and block sizes used in this model.

Table 19-2
Johnson Camp Block Model Limits

Parameter	Minimum	Maximum	Number	Dimension (Ft.)
Easting (Columns)	3,000	11,000	160	50
Northing (Rows)	5,000	11,000	120	50
Elevation (Levels)	3,500	5,400	95	20

Block Model Items

The Johnson Camp block model contains fourteen items that were used for resource modeling and subsequent floating cone studies. The items stored in this model are listed in Table 19-3. This table summarizes all of the items in the model, and includes the minimum, maximum and precision for each item, along with a brief description.

Table 19-3

Item	Minimum	Maximum	Precision	Description
TOPO	0.00	100.00	1.00	Original topography in percent
MINED	0.00	100.00	1.00	Percent of block below current mined surface
TCU	0.00	2.53	0.01	Total copper in %
ROCK	0.00	29.00	1.00	Rock type (codes 0 - 10)
ZONE	0.00	13.00	1.00	Combined rock type assemblages into zones
NCMPS	0.00	13.00	1.00	Number of composites to estimate block TCu
VALUE	0.00	655330.00	10.00	Block net value in dollars
ASCU	0.00	2.53	0.01	Acid soluble copper in %
ORTYP	0.00	13.00	1.00	Ore type code
CLASS	0.00	5.00	1.00	Resource classification code
DIST	0.00	509.00	1.00	Distance to closest drill hole composite
TONF	10.79	21.00	0.01	Tonnage factor in cubic feet/short ton
SOL	0.00	100.00	1.00	Solubility ratio index (AsCu/TCu)
EPASS	0.00	5.00	1.00	Estimation pass number

Burro Copper Grade Estimation

Estimation Strategy/Parameters

Two distinct methods were utilized to estimate block model copper grades for the Burro deposit.

The first was a nearest neighbor grade estimation that used a limited drill hole composite search strategy of 25 feet in plan and 10 feet in the vertical direction. This method estimated block grades for only those blocks that were pierced by drill holes. Blocks that received a nearest neighbor grade estimate were then tagged with a code that excluded that block from receiving a new value from subsequent estimation runs.

The second estimation method consisted of five separate kriging runs for each of the previously described mineral zones. These kriging runs used the parameters listed in Table 14-2 for each of the five zones. In addition, a strict rock type (mineral zone) matching criteria was implemented to prevent mixing grades from adjacent zones.

Burro Model Verification

The Summo study verified the global and local block model copper grades by visual and statistical methods. The Burro portion of the block model was reviewed for potential gross estimation errors by visually examining color-coded copper grades. No obvious problems were detected. Next, a representative set of block model bench maps and cross sections that also contained composite values from the exploration drill holes was plotted. BETA agrees that the total copper grade estimate for the Burro deposit appears reasonable relative to the drill hole composite grades. Total copper assays were used in the resource model because total copper assay values were the only common denominator for all drill hole assays included in the drill hole data base.

Next, grade tonnage curves were generated for the remaining Lower Abrigo block model resources within the Burro deposit. Similar curves were generated for drill hole composites in the same Lower Abrigo material. Figure 14-1 contains these grade tonnage curves and shows that the model has slightly overestimated tons at lower grade cutoffs and slightly under estimated tons at higher-grade cutoffs. The block model grades are consistently lower than the composite grades. The block model grade is biased downward because all estimated Lower Abrigo blocks below the current surface were used to develop the curve, including low-grade inferred blocks situated along the margins of the main mineralized zone.

Additionally, a performance test of the model was made by comparing the results obtained by the four Summo confirmation drill holes with the grades of the blocks in the vicinity of the drill hole intercepts. To complete this test, a number of bench maps were plotted and which compared the 20-foot composite grades in the Summo confirmation holes with the block grades in the model, as well as the composite grades in surrounding drill holes. In general, the Summo confirmation drill hole grades compare reasonably well with the existing block grades. Although individual Summo drill hole composite grades were observed to vary between plus or minus one to 40 percent with local block grades, visually the drill hole composite grades generally compared closely with wider population of adjacent block grades. As an additional check, the mean grades were calculated for the Summo confirmation holes and the original drill hole data by pairing these two data sets according to the closest separation distance between them. The average distance between two data pairs was 82 feet. For the 102 data pairs, mean grades of 0.29 and 0.36 percent total copper were calculated for the Summo confirmation data and initial drill hole data, respectively. Although the Summo confirmation grades are 19 percent lower than the grades from the initial drill holes, the data sets are somewhat disparate. All data from the initial drilling was generated from drill core, while the Summo data is based on reverse circulation drilling. Also, the mean grade comparison was made irrespective of lithology and was based entirely on selecting data pairs based on their proximity to one another.

Burro Geologic Resources

The classification of geologic resources for the Burro deposit was based on the distance to the nearest composite used to interpolate each block grade, and a portion of the variogram range used for each rock type. To establish the classification criteria, the Summo study reviewed the distribution of estimated model tons versus the distance to the drill hole samples as well as the production history of the deposit. After reviewing all of these factors, it was determined that measured resources could be defined as blocks situated within 160 feet of at least one drill hole composite that was used to estimate the grade of each individual block. BETA notes that this distance is a small fraction of the maximum correlogram range that was interpreted for the Lower Abrigo unit and used to estimate block model resources. Indicated resources were defined as blocks that were estimated by at least one drill hole composite within a range of 161 to 260 feet of the block. Blocks that received a total copper grade estimate from composites in excess of 260 feet were classified as inferred resources. These classification parameters based on distance to data were used for all rock types. BETA agrees with the classification parameters as utilized.

Table 19-4 summarizes the parameters that were used for classifying the Burro resources. The total remaining measured and indicated resources in the Burro deposit include over 90 percent of the blocks that received grade estimates. Block model grades above the limits of previous mining were zeroed out and not included in the remaining resource inventory. Table 19-5 summarizes the geologic resources for the Burro deposit by resource class at several total copper cutoff grades.

Table 19-4
Burro Pit Resource Classification
System Based on Distance to the Closest Drill Hole Composite
Used to Estimate Block Grades

Class	From Distance	To Distance
Measured	0’	160’
Indicated	161’	260’
Inferred	261’ +	

Table 19-5

Burro Pit Resource Summary for Total Copper									
TCu	Measured			Indicated			Inferred		
Cutoff	Tons	TCu (%)	lbs. Cu	Tons	TCu (%)	lbs. Cu	Tons	TCu (%)	lbs. Cu
0.00	64,833,000	0.28	366,825,100	37,911,200	0.25	191,982,300	16,881,800	0.24	79,952,200
0.10	53,848,000	0.33	355,289,100	29,860,400	0.31	183,880,300	12,582,400	0.30	76,224,200
0.20	39,128,200	0.40	313,260,400	19,889,400	0.39	155,654,400	8,164,200	0.39	63,925,700
0.30	24,848,200	0.49	243,065,100	12,522,400	0.48	119,739,200	5,035,600	0.48	48,261,200
0.40	14,408,000	0.60	171,973,900	7,458,000	0.57	85,095,800	3,269,400	0.55	36,139,900
0.50	8,625,200	0.70	120,890,800	4,065,800	0.68	55,148,500	1,837,600	0.64	23,495,550
0.60	5,302,800	0.80	84,834,200	2,354,200	0.78	36,603,100	975,200	0.73	14,175,500
0.70	3,307,800	0.90	59,229,450	1,489,200	0.85	25,456,400	491,200	0.81	7,994,800
0.80	2,219,800	0.97	43,139,600	779,000	0.96	14,975,500	194,200	0.92	3,587,262
0.90	1,344,400	1.06	28,436,750	433,600	1.07	9,240,000	44,000	1.23	1,079,232
1.00	802,400	1.14	18,264,200	194,000	1.23	4,774,000	32,000	1.35	863,168

Historical Production/Reconciliation

During a September 1998 site visit to the Johnson Camp operation as part of the Summo feasibility study, the engineering firm was unable to locate complete, comprehensive original production records that tabulated the total Cyprus and/or Arimetco mine production. As a result, the underlying data that was used to review the previous mine production at Johnson Camp came from a previous Summo pre-feasibility study, which had been compiled from Cyprus and Arimetco monthly and annual reports. Table 19-6 is a re-creation of this tabulation, with summaries of ore tons, copper grades, contained copper pounds and pounds of copper produced and shipped by year. The total mine production of 31.8 million tons of ore occurred over a seventeen-year period beginning in 1975 and ending in 1997. Cyprus reported copper grades in percent as soluble copper, whereas Arimetco reported copper grades in percent as total copper.

Table 19-6

Johnson Camp Mine
Historical Reported Mine Production

Cyprus Production

Production Year	Mine Production Ore Tons	Cu Grade AsCu(%)	Contained Cu Pounds	Shipped Cu Pounds
1975	2,132,260	0.496	21,152,019	6,143,024
1976	1,821,476	0.357	13,005,339	10,059,807
1977	1,563,030	0.399	12,472,979	10,327,424
1978	1,202,500	0.426	10,245,300	10,205,142
1979	1,588,400	0.522	16,582,896	10,032,003
1980	1,499,600	0.411	12,326,712	10,320,407
1981	1,551,500	0.470	14,584,100	10,693,485
1982	1,894,700	0.322	12,201,868	9,702,272
1983	1,962,600	0.504	19,783,008	9,717,616
1984	52,100	0.713	742,946	8,803,361
1985	0	0	0	6,200,836
1986	0	0	0	4,854,796
Sub-Total	**15,268,166**	**0.436%**	**133,097,167**	**107,060,173**

Arimetco Production

Production Year	Mine Production Ore Tons	Cu Grade TCu(%)	Contained Cu Pounds	Shipped Cu Pounds
1991	750,100	0.340	5,100,680	5,549,725
1992	2,516,320	0.480	24,156,672	8,156,435
1993	3,259,320	0.340	22,163,376	7,386,504
1994	2,719,690	0.290	15,774,202	5,618,012
1995	2,995,592	0.290	17,374,434	6,345,518
1996	3,084,254	0.350	21,589,778	9,921,576
1997	1,254,971	0.370	9,286,785	4,747,995
1998	0	0	0	2,181,304
Sub-Total	**16,580,247**	**0.348**	**115,445,927**	**49,907,069**

Total Property Production (1975 - 1998)

	Mine Production Ore Tons	Cu Grade TCu(%) (1)	Contained Cu Pounds (1)	Shipped Cu Pounds
Total	**31,848,413**	**?**	**248,543,094**	**156,967,242**

Note: (1) Cyprus Cu grade was annually reported as % soluble copper and Arimetco reported % total copper, so unable to calculate Cu grade in terms of TCu or AsCu. The contained Cu lbs are based on TCu and AsCu grades.

The block model ore tonnage corresponding to historical mining compares very closely with the total production value summarized in Table 19-6. To define the volume in the block model that represents past production, the Summo feasibility study calculated the block model tons and grade for the material volume between the original topographic surface and the August 1998 topographic surface. The difference between the block model ore tonnage of 32.1 million tons and the reported production of 31.8 million tons is only 0.8 percent. It is BETA's opinion that this close comparison suggests that the reported ore tonnage as tabulated is reasonable.

Copper grade reconciliation proved to be more difficult to analyze since the reported Cyprus and Arimetco copper grades were stated in different units (AsCu and TCu, respectively). Because a direct comparison between block model copper grades and actual mining could not be made due to the lack of blast hole assays and other detailed production data from the Cyprus and Arimetco operating tenures, the study back-calculated a total copper grade for the Cyprus mine production. The basis for this calculation was the fact that the block model tonnage estimate was very close to the reported tonnage produced from the mine, leading to the assumption that the model data could be used to provide an approximation of the Cyprus total copper grade. First the Arimetco production data (tons, total copper grade and pounds copper) were subtracted from the study's block model estimate of the total tons and grade representing the entire historical production. The remainder represents the Cyprus production. By dividing the remaining pounds of copper by the remaining ore tons, a total copper grade for the Cyprus production was calculated. Since a Cyprus historic mined tonnage was available, the Cyprus total copper grade estimate was refined by using that tonnage in the grade calculation. The combined total of the Arimetco production data and the back-calculated Cyprus data were compared to the block model estimate. Again, very close agreement was seen between the two data sets, with the study model having slightly more tons (0.79 percent) and slightly less grade (0.80 percent) than the combined Arimetco and back-calculated Cyprus data. BETA agrees with this grade reconciliation. Table 19-7 summarizes all of the data used in the back calculation and reconciliation comparison.

Table 19-7

Johnson Camp Mine
Burro Pit Production vs TWC Block Model

TWC Model vs Reported Production

Production Source	Mine Production Ore Tons	Cu Grade TCu(%) (1)	Contained Cu Pounds (2)
TWC Model Estimate	32,103,185	0.5042	323,728,518
Total Reported Production	31,848,413	?	?

Back Calculated Cyprus Total Copper Grade

Production Source	Mine Production Ore Tons	Cu Grade TCu(%)	Contained Cu Pounds
TWC Model Estimate	32,103,185	0.5042	323,728,518
Less Arimetco's Production	16,580,247	0.3480	115,445,927
Back Calc'd Byprus Production (3)	15,522,938	0.6709	208,282,591
Back Calc'd Byprus Production (4)	15,268,166	0.6821	208,288,321

Total Production Using Back Calculated Cyprus Total Copper Grade

Production Source	Mine Production Ore Tons	Cu Grade TCu(%)	Contained Cu Pounds
Back Calc'd Cyprus Production (4)	15,268,166	0.6821	208,288,321
Arimetco Production	16,580,247	0.3480	115,445,927
Total Production	31,848,413	0.5082	323,734,248

TWC Model vs Calculated Production

Production Source	Mine Production Ore Tons	Cu Grade TCu(%)	Contained Cu Pounds
Back Calc'd Total Production	31,848,413	0.5082	323,734,248
TWC Model Estimate	32,103,185	0.5042	323,728,518
Difference (Production - Model)	-254,772	0.004	5,730
% Difference (Production vs Model)	-79.00%	0.80	0.00

Notes:

(1) Cyprus did not report a total copper value for their production, so can not directly compare with TWC.
(2) Cyprus reported contained pounds of Cu is based on soluble Cu values.
(3) Cyprus tonnage based on difference between TWC model and Artimetco's reported production.
(4) Cyprus tonnage based on their reported historical mined ore tons.

19.2 COPPER CHIEF RESOURCE MODEL

Geologic Framework

BETA reviewed and incorporated a three dimensional geologic model of the Copper Chief deposit prepared in conjunction with the Summo feasibility study into this technical report. Unlike with the Burro deposit, Arimetco did not develop a geologic interpretation for the Copper Chief deposit, which required that the Summo study develop an independent interpretation for the purpose of using rock type to constrain the estimation of copper. The Summo study developed this interpretation by constructing polygonal boundaries for each rock unit in two dimensions on 24 cross sections that were oriented N35°E, approximately normal to the general strike of the lithologic units. The underlying data for the geologic interpretation were the rock codes stored for each drill hole in the Arimetco database. The sectional polygons for each rock type were then linked together to form a series of wire-frame solids. The only rock unit not modeled separately was the Upper Abrigo, which due to very limited assay data was combined with the Middle Abrigo unit for the geologic interpretation.

The three-dimensional solids that were created from the cross sectional interpretation were cut on 20-foot elevation plans and then reconciled in plan and linked together to form new wire-frame solids. The new solids were then used to load lithologic codes to the three-dimensional block model based on a block majority rule (i.e., the lithologic code assigned to a block was based on the lithology solid that the majority of the block fell within). The block model lithologic codes were checked visually, and several minor errors were detected along several rock boundary contacts. These errors were corrected so that the lithologic model codes more precisely reflect the geometry of the wire-frame solids. The rock codes used in the Copper Chief block model are summarized in Table 19-8.

Table 19-8
Lithologic Codes in the Copper Chief Model

Lithologic Unit	Lithologic Code	Tonnage Factor
Upper Abrigo	1	12.46
Middle Abrigo	2	12.46
Lower Abrigo	3	12.46
Bolsa Quartzite	4	12.61
Upper Diabase	5	11.73
Upper Pioneer Shale	6	12.00
Lower Diabase	7	11.33
Lower Pioneer Shale	8	12.00
Default Rock Code	9	12.50
Dump Material	10	16.25

Rock Density

Summo supplied the tonnage factor data for the major ore host rocks based on bulk density test work completed by an independent laboratory. For the less important rock types not included in the bulk density test work, tonnage factors were assigned based on density data values obtained from published reference sources. Table 19-8 summarizes the tonnage factors used for each lithologic unit in the Copper Chief resource model. The tonnage factor used for waste dump material was estimated to be 16.25 cubic feet per ton. This assumption was based on using an average insitu tonnage factor of 12.50 cubic feet per ton for all rock types and then applying a standard swell factor of 30 percent. BETA notes that rock densities used are within normal range for rock type.

Block Model

The Copper Chief block model is a subset of the overall Johnson Camp model that contains both the Burro and Copper Chief deposits. BETA notes that the Summo study used the 7,200 East coordinate as an east-west dividing line between the two deposits. Table 19-2 summarizes the basic block size parameters that were used in the Johnson Camp block model.

The block model items used for the Copper Chief deposit are the same as those used for the Burro model, as described herein. Table 19-3 summarizes all of these items, including the minimum, maximum, and precision for each item, along with a brief description.

Copper Chief Copper Grade Estimation

Estimation Strategy/Parameters

Three separate and distinct estimation passes were used to estimate total and acid soluble copper grades for the Copper Chief deposit.

1. The first grade estimation pass was a nearest neighbor calculation that used a restricted drill hole composite search strategy of 25 feet in the east and north directions and 10 feet in the vertical direction. Blocks that received a grade estimate from the nearest neighbor run were tagged and became unavailable for subsequent estimation runs.

2. The second estimation pass used a series of ordinary kriging runs that utilized specific parameters for each rock type that were developed from variography. Blocks that received a grade estimate from these kriging runs were tagged and became unavailable for subsequent estimation runs.

3. The third and final estimation pass used the same parameters as the first set of kriging runs, but the ranges were doubled. These final extended range runs were executed only to "fill-in" gaps in areas of sparse drilling. Blocks that were estimated in the final estimation pass were classified as inferred resources. These inferred blocks represent viable target areas that could possibly be upgraded to indicated or measured resources with additional drilling.

All of the grade estimation runs used strict rock code matching. That is, the only drill hole composites that could be used to estimate a block grade had to have the same rock code as the block. Excluding the nearest neighbor estimation, all interpolation runs required a minimum of one composite and a maximum of four composites to estimate block grades, with only three composites allowed from a single drill hole.

Table 19-9 summarizes the estimation parameters used for the interpolation of total and acid soluble copper grades for each rock type.

Table 19-9

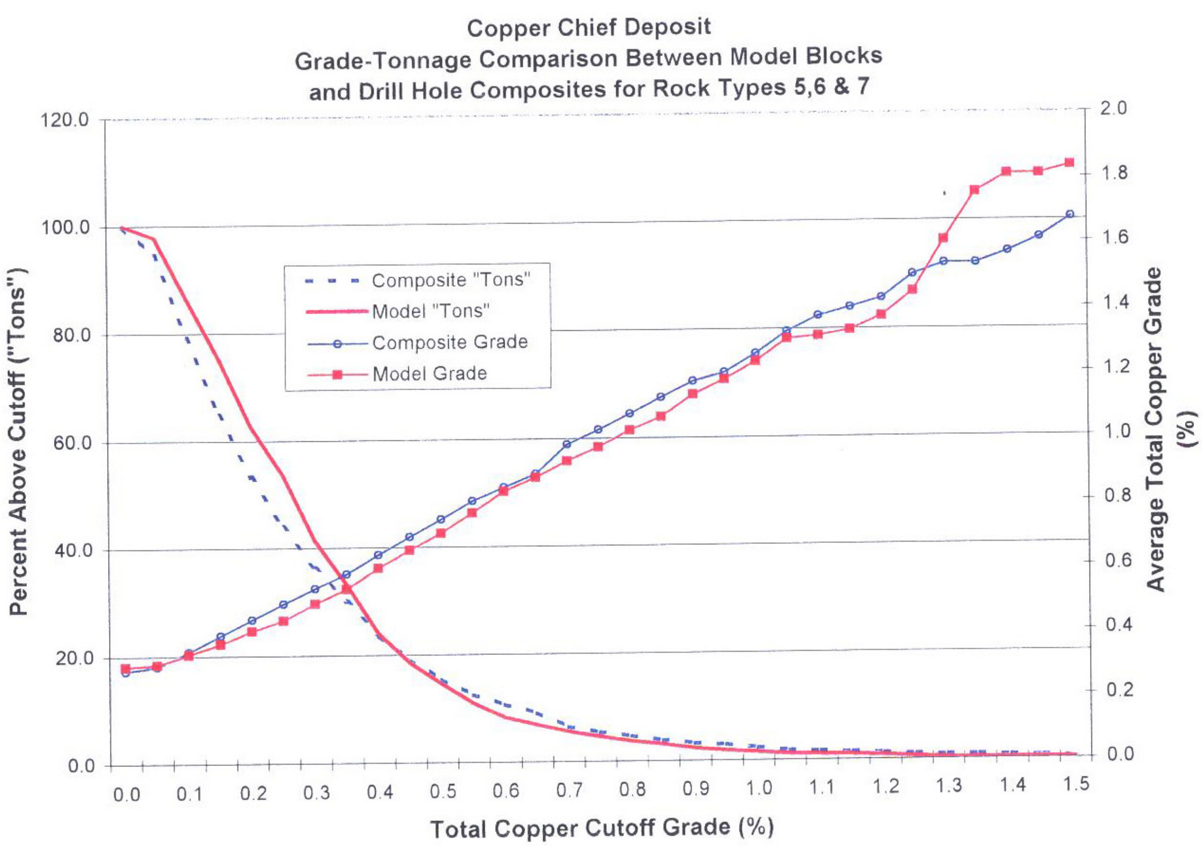
Copper Chief Total Copper Resource Estimation Parameters													
Pass #1													
Est. Type	Model Type	Min. #Cmps	Max. #Cmps	Max/Hole	X	Y	Z	AZ	Plunge	Dip	Rock Types	Nugget	Sill-Nug
NN	n/a	1	1	1	25	25	10	-	-	-	All	-	-
Pass #2													
Est. Type	Model Type	Min. #Cmps	Max. #Cmps	Max/Hole	X	Y	Z	AZ	Plunge	Dip	Rock Types	Nugget	Sill-Nug
Krige	Exponential	1	4	3	160	120	40	120	-45	30	1 and 2	0.600	0.400
Krige	Exponential	1	4	3	360	270	90	100	-15	30	3	0.010	0.990
Krige	Exponential	1	4	3	175	130	40	305	0	-30	4	0.580	0.440
Krige	Spherical	1	4	3	325	240	40	305	0	-30	5	0.575	0.425
Krige	Spherical	1	4	3	325	240	40	305	0	-30	6	0.575	0.425
Krige	Spherical	1	4	3	325	240	40	305	0	-30	7	0.575	0.425
Krige	Exponential	1	4	3	300	225	75	120	0	30	8	0.210	0.780
Pass #3													
Est. Type	Model Type	Min. #Cmps	Max. #Cmps	Max/Hole	X	Y	Z	AZ	Plunge	Dip	Rock Types	Nugget	Sill-Nug
Krige	Exponential	1	4	3	320	240	80	120	-45	30	1 and 2	0.600	0.400
Krige	Exponential	1	4	3	720	540	180	100	-15	30	3	0.010	0.990
Krige	Exponential	1	4	3	350	260	80	305	0	-30	4	0.580	0.440
Krige	Spherical	1	4	3	650	480	80	305	0	-30	5	0.575	0.425
Krige	Spherical	1	4	3	650	480	80	305	0	-30	6	0.575	0.425
Krige	Spherical	1	4	3	650	480	80	305	0	-30	7	0.575	0.425
Krige	Exponential	1	4	3	600	450	150	120	0	30	8	0.210	0.780

Model Verification

The global and local block model copper grades were verified by visual and statistical methods. The entire block model was initially reviewed for potential gross estimation errors by visually examining color-coded copper grades. No obvious discrepancies were detected. Next, a representative set of bench maps and cross-sections were plotted showing color-coded block model grades with drill hole composite grades. In BETA's opinion, the total copper grade estimate for the Copper Chief deposit appears reasonable relative to the drill hole composite grades. Total copper assays were used in the resource model because total copper assay values were the only common denominator for all drill hole assays included in the drill hole data base.

Grade-tonnage curves were generated from the block model and the drill hole data for total copper. In both data sets, the tonnage curves actually represent the percentage of data above the copper cutoff grade, and not actual tons. These curves were then compared with one another to gauge how closely the model values correspond to the underlying drill hole data throughout a range of copper cutoff grades. The data for these curves was restricted to only those blocks and composites within the Upper Diabase, Upper Pioneer Shale and the Lower Diabase. In general, the block model tonnage is higher than the composites below a total copper cutoff grade of 0.40 percent. At total copper cutoff grades above 0.40 percent, the block model tonnage is less than the composites by an average of approximately 18 percent. The block model copper grades are consistently about 4 percent lower than the composite grades throughout most of the cutoff grade ranges that were examined. These relationships are typical of block model estimates derived from ordinary kriging. However, it is noted that the block model grade curve shown in Figure 19-1 is biased downward because low-grade blocks that were estimated along the edges of the deposit were included with data from the core of the deposit to develop the curve.

Figure 19-1



An additional verification test of the model was performed by comparing the eight Copper Chief confirmation holes drilled by Summo with the estimated block grades in the vicinity of the drill hole intercepts as part of the Summo study. Three bench plans were plotted with total copper grades estimated using only the Cyprus and Arimetco drill hole data along with the Summo's confirmation drill hole grades. In general, the Summo drill hole grades correspond reasonably well with the block model total copper grades. It was concluded that the discrepancies observed could be attributed to the proximity of the Summo samples to rock type boundaries, where sharp differences in grade are often seen.

Summo confirmation drilling was compared with the earlier Cyprus and Arimetco drilling results. Each of the Summo drill hole composites were paired with the closest Cyprus or Arimetco composite. The average distance between pairs in the two data sets was 111 feet. For the 139 data pairs, mean total copper grades of 0.26 percent and 0.29 percent were calculated for the Summo confirmation drilling and the earlier Cyprus/Arimetco drilling, respectively. Although the Summo confirmation grade data are 10 percent lower than the previous drilling, as with the Burro deposit most of the earlier drill hole composites were derived from core drilling while the Summo confirmation data are based on reverse circulation drilling. Also, these comparisons were made without regard to rock type and were based solely on selecting data pairs within close proximity to one another.

Geologic Resources

The classification of geologic resources for the Copper Chief deposit was based on the distance from a block to the nearest composite used to interpolate grade for that block and a percentage of the variogram range for each rock type. For the main ore host rocks of the Copper Chief deposit, (Upper Diabase, Upper Pioneer Shale and Lower Diabase), measured geologic resources were confined to blocks that were estimated by at least one drill hole composite having the same rock code within 25 percent of the maximum variogram range. Indicated resources were defined as blocks that were estimated by at least one drill hole composite having the same rock code that was within 25 to 50 percent of the maximum variogram range. All other blocks that received a total copper grade estimate were classified as inferred resources. Table 19-10 summarizes the ranges used for classifying block model resources for each rock type.

Table 19-10

Copper Chief Resource Classification System

Based on Distance to the Closest Drill Hole
Composite Used To Estimate Block Grades

Class	From (ft)	To (ft)	Rock Type	Passes
Measured	0’	88’	2	1,2
Measured	0’	150’	3	1,2
Measured	0’	88’	4	1,2
Measured	0’	150’	5,6,7,8	2
Indicated	89’	131’	2	2
Indicated	151’	245’	3	2
Indicated	89’	131’	4	2
Indicated	151’	245’	5,6,7,8	2
Inferred	132’	∞	2	3
Inferred	246’	∞	3	3
Inferred	132’	∞	4	3
Inferred	246’	∞	5,6,7,8	3

Geologic resources for the Copper Chief deposit are summarized in Table 19-11. These resources do not include the material removed by the small Arimetco starter pit.

Table 19-11

Copper Chief Resource Summary for Total Copper

TCu	Measured			Indicated			Inferred		
Cutoff	Tons	TCu (%)	lbs. Cu	Tons	TCu (%)	lbs. Cu	Tons	TCu (%)	lbs. Cu
0.00	65,135,200	0.19	247,513,760	49,139,400	0.16	152,332,140	149,567,400	0.15	433,745,460
0.10	43,012,400	0.26	223,664,480	28,405,200	0.23	128,959,608	86,432,400	0.21	368,202,024
0.20	23,314,000	0.36	167,394,520	12,403,600	0.34	84,592,552	33,258,200	0.33	222,164,776
0.30	12,073,600	0.47	114,216,256	5,009,600	0.49	49,494,848	14,376,200	0.46	133,123,612
0.40	6,568,000	0.58	76,582,880	2,710,400	0.62	33,663,168	6,885,800	0.58	80,426,144
0.50	3,807,800	0.69	52,395,328	1,662,800	0.74	24,443,160	3,672,000	0.71	51,922,080
0.60	2,023,400	0.82	33,143,292	1,042,600	0.85	17,724,220	2,412,000	0.79	38,206,080
0.70	1,275,000	0.92	23,562,000	760,000	0.93	14,120,800	1,368,000	0.91	24,897,600
0.80	819,000	1.02	16,756,740	524,000	1.01	10,616,240	832,000	1.02	16,889,600
0.90	544,000	1.11	12,076,800	316,000	1.12	7,103,680	432,000	1.18	10,221,120
1.00	320,000	1.23	7,852,800	224,000	1.10	4,910,080	296,000	1.29	7,636,800

Copper mineralogy varies within the deposits. In the Burro Pit, approximately 76% of the total estimated ore reserve tonnage is located above a depth of 4,560 feet in a zone dominated by the copper oxide minerals chrysocolla and malachite. Some native copper has been observed disseminated throughout this range. In addition to copper oxide mineralization, copper sulfide mineralization is evident below an elevation of 4,600 feet “in a mixed zone”. Sulfide minerals, which typically convert to oxides on exposure to oxygen, are not as amenable to heap leach copper recovery techniques as oxides. Accordingly, we believe that approximately 24% of the ore reserve in the Burro Pit could exhibit reduced copper recovery due to the presence of copper sulfide mineralization.

In the Copper Chief Pit, the oxide copper mineralization is similar to that of the Burro Pit. The entire Copper Chief Pit ore reserve is located above the 4,560 elevation in the zone dominated by the copper oxide minerals chrysocolla and malachite. BETA does not expect that the recovery of copper from this deposit will be materially affected by sulfide mineralization.

In summary, for the total project, approximately 85% of the ore reserves are located above the 4,560 elevation in the zone dominated by the copper oxide minerals chrysocolla and malachite. Approximately 15% of the total ore reserves could exhibit reduced copper recovery due to the presence of copper sulfide mineralization.

The bulk samples for the Summo metallurgical testing were taken from several areas of the Burro and Copper Chief Pits, with all sample locations above the 4,560 foot elevation in the zone dominated by the copper oxide minerals chrysocolla and malachite. The assay results for the Abrigo formation sample taken from an elevation of 4,620 feet, however, indicated a sulfide content of 4.49%. This suggests that the leaching of copper from ore mined at this elevation may be less than optimal.

The Summo test work initially consisted of five columns, each containing 135 kilograms (approximately 298 pounds) of ore, taken from five ore samples of approximately 1,000 pounds each. Some problems were encountered with the first five columns, however, so an additional six columns were prepared and tested. All column tests were conducted at a nominal crush size of one inch based on the results from the Arimetco program, except one which was done at a nominal crush size of ½ inch.

The forecasted recoveries of copper that were reviewed by BETA in preparing this technical report are based on the column tests and are dependent on the crushing of the ore to a nominal size of one inch. The Arimetco test program indicated the importance of this parameter. Cyprus operated Johnson Camp Mine was for a run-of-mine operation whereby non-crushed ore was placed on the leach pads. Arimetco also ran the Johnson Camp Mine as a run-of-mine operation until late 1995, when it began crushing the ore to approximately 3 inches. Current copper recovery estimates provide for extracting 74 to 81 percent of the total copper content of the ore mined, depending on ore type and with crushing to a nominal size of one inch.

According to Cyprus' records, it achieved copper extraction of up to 80 percent of the acid soluble copper from uncrushed, run-of-mine material. However, the Arimetco operation, which leached new run-of-mine ore, old Cyprus run-of-mine ore, and 4.3 million tons of ore reported to have been crushed to a nominal size of three inches, achieved copper recovery (from 1991 through 1998) of 43 percent of total copper. Arimetco's records do not distinguish between copper extracted from old Cyprus material, new run-of-mine ore, and new crushed ore.

In preparing this technical report, BETA reviewed the metallurgical test work and concurs with the metallurgical recovery estimates. As indicated above, however, the increase in projected copper recovery rates over the historic copper recovery rates is premised on ensuring that the ore is crushed to a nominal size of one inch prior to being placed on the leach pads. This is consistent with Arimetco's initial results from leaching of crushed ore placed on a new liner system – namely, an increase in leach solution copper grade and an improvement in recoveries to the point where they matched the metallurgical test work performed on certain ore at a similar crush size.

In summary, expectations with respect to copper recovery rates significantly exceed historical experience at the Johnson Camp Mine, as Nord plans to crush the ore to a smaller size with the view to increasing leaching efficiency. BETA believes that these expectations are reasonable, given the view that Cyprus and Arimetco placed uncrushed or improperly crushed ore on the leach pads, which resulted in differing recovery projections and rates. However, there can be no assurance that Nord will be able to meet these expectations and projections at an operational level.

BETA cautions that copper recovery rates for ore anticipated to be mined below the 4,560 foot elevation (approximately 15% of estimated total ore reserves) may be inhibited due to the presence of copper sulfide mineralization. In addition, although the column test on the sample of Abrigo ore which contained 4.49% sulfides exhibited good copper recoveries (as shown in Table 18-4.2), the leaching of copper from ore mined below this elevation may be less than optimal.

20.0 INFRASTRUCTURE

The existing facilities include a truck shop, core storage building, administrative and engineering office and warehouse, laboratory, plant mechanical shop, a 4,000-gpm solvent extraction plant, a tank farm, a 52,000-pounds-per-day capacity electrowinning plant with 74 electrowinning cells, four solution storage ponds with a total capacity of approximately eight million gallons, miscellaneous piping to and from the above facilities, and various used vehicles, pumps, and other equipment.

Power Supply

The Johnson Camp operation receives electrical power from Sulfur Springs Valley Electric Cooperative (SSVEC) by way of a 69,000-volt line owned by SSVEC. Power is received at a single substation owned by Nord. SSVEC currently purchases the power from Arizona Electric Power Company (AEPCO), which operates a gas/coal fired generating plant southwest of Wilcox.

The Johnson Camp substation transformer is rated at 5,000 KVA. For the expansion to 25 million pounds per year, a second 5,000 KVA was purchased to operated in tandem inside a new substation. SSVEC indicates that their lines and equipment can supply at least 10,000 KVA to the substation.

Nord must negotiate a new long term power contract with SSVEC as they currently operate on a short term contract. A rate of $0.055 per kWh was used, which remains typical for industrial power in this area. A long-term contract for the expanded plant may provide for a lower cost.

Water Supply

Production water for the Johnson Camp project is currently supplied from three wells that together provide approximately 600 gpm of water. These wells have been upgraded since the 2000 Feasibility Study. These sources include the Moore shaft (200 gpm), the Republic well (300 gpm), and the Section 19 well (100 gpm).

Water rights for the Johnson Camp property are confirmed and on file with the State of Arizona and include:

Well Name	Registry No.
Moore Mine	36-66376
Republic Mine	36-66377
Black Prince Mine	36-66378
Section 19 Well	36-66379

Additional water may be required to expand the leaching operation. A groundwater hydrology consulting firm, has suggested several well locations around the Burro pit to dewater the pit and provide make-up water. The cost of adding an additional 150-gpm well in project capital has been included. The cost used was based on information from a well driller experienced with the site conditions.

Transportation

Due to its location just one mile north of Interstate 10 (the major east-west transportation route across the southern U.S.), the Johnson Camp site provides excellent access for transportation and delivery of bulk supplies and shipment of copper cathodes. Major bulk consumables are readily available from nearby Tucson. Sulfuric acid is produced at three copper smelters within a 200-mile radius, including Asarco at Hayden (127 miles to the north), Phelps Dodge at Miami (167 miles to the north), and Grupo Mexico at Nacozari (130 miles to the south). Diesel fuel is readily available from Willcox, 17 miles to the east on Interstate 10. The current economic projections provide for bringing acid by rail from Mexico to Benson and truck haulage from Benson to the site.

The mine's close proximity to the Union Pacific Railway mainline through Dragoon gives Nord the option of shipping cathode direct to customers by truck or rail, particularly in light of the high quality of the cathodes produced in the past at Johnson Camp. In the past Nord shipped cathode direct to one or more customers, with Mitsui acting as agent for at least part of the cathode.

Administration Facilities

The administration facilities at the Johnson Camp site consist of a separate office building and a combined warehouse/plant mechanical shop/laboratory building. Although the administrative building is functional in its current condition, minor repairs to specific offices and the restroom facilities will be required. The size of the building is not sufficient to house and support management, accounting/clerical personnel, and technical departments such as mine geology and engineering. A new administration building will be added in close proximity to the project entrance to provide suitable office space.

The warehouse facility is adequate for the size of operation planned by Nord, in light of its close proximity to Benson and Tucson and the availability of same-day or next-day delivery of consumables. Minor repairs will be necessary to provide secure access to the warehouse area. Ample space exists in the vicinity of the warehouse/plant/truck shop area for fenced open air storage of larger inventory items that do not require cover.

21.0 ORE RESERVES AND MINING

21.1 INTRODUCTION

Mining of the Johnson Camp Mine Project is by open-pit methods utilizing mid-size earth moving equipment. Feasible pit shapes complete with haul-road designs have been modeled based on: the disposition of grade values in the resource model; economic parameters such as copper price and mining and operating costs; and technical parameters such as pit slopes and copper recovery.

Total proven and probable minable reserves are 73.4 million tons of ore at an average total copper grade of 0.335 percent total copper (TCU), containing 492 million pounds of copper and over 374 million pounds of recoverable copper, minable at a strip ratio of 0.66 to 1. Proven Reserves total 55.0 million tons grading 0.338% containing over 318 million pounds of copper of which 245 million pounds are recoverable. Probable reserves total 18.4 million tons grading 0.327% copper containing 173 million pounds of copper of which 129 million pounds are recoverable. (Table 21-1). The estimate is based on diluted, proven and probable reserves located within the Burro and the Copper Chief pits using a 0.065 percent total copper internal cutoff. Recovery is based on rock type, as discussed in this section, and presented as average recovery by bench. Production and equipment requirements are based on a mining and processing schedule of approximately 25 million pounds of copper produced per year during the mine's 16 year life. The ore reserve estimates are compliant with CIM and SEC Guide 7 guidelines.

Table 21-1
Proven and Probable Minable Reserves

	Tons	Grade	Pounds Copper Contained	Pounds Copper Recoverable
Proven	54,978,000	0.338	318,540,300	245,279,300
Probable	18,410,400	0.327	173,481,600	128,862,500
Proven + Probable	73,388,500	0.335	492,021,800	374,141,700

A low-grade component of the minable reserves will be separated and processed differently. Material grading between 0.065% and 0.150% total copper will be trucked directly to the leach pads and leached run-of-mine. Ore grading above 0.15% will be crushed and stacked. The minable reserves are stated by process option in the following tables 21-2 and 21-3. A breakdown of proven and probable ore by bench by pit is provided in this report.

Table 21-2
Crusher grade (>0.15%) Proven and Probable Minable Reserves

	Tons	Grade	Pounds Copper Contained	Pounds Copper Recoverable
Burro	38,417,500	0.399	306,818,400	239,418,300
Copper Chief	24,276,000	0.335	162,528,500	123,385,900
Proven + Probable	62,693,400	0.374	469,346,900	362,804,250

Table 21-3
Low Grade (<0.15%) Proven and Probable Minable Reserves

	Tons	Grade	Pounds Copper Contained	Pounds Copper Recoverable
Burro	5,627,700	0.104	11,721,900	5,860,900
Copper Chief	5,065,750	0.108	10,953,100	5,476,500
Proven + Probable	10,693,400	0.106	22,675,000	11,337,500

Both the Burro pit and the Copper Chief pit have starter pits designed in order to maximize production while minimizing total tons moved during the first three years of production, per instruction from Nord. A summary of proven and probable ore by pit is provided in tables 21-4 through 21-7.

Table 21-4
BURRO PIT Crusher grade (>0.15%) Proven and Probable Minable Reserves

	Tons	Grade	Pounds Copper Contained	Pounds Copper Recoverable
Starter Pit	7,464,658	0.491	73,357,637	56,552,505
Final Pit	30,952,798	0.377	233,460,739	182,865,812
Proven + Probable	38,417,456	0.399	306,818,376	239,418,318

Table 21-5
BURRO PIT Low Grade (<0.15%) Proven and Probable Minable Reserves

	Tons	Grade	Pounds Copper Contained	Pounds Copper Recoverable
Starter Pit	604,356	0.100	1,205,160	602,580
Final Pit	5,023,331	0.105	10,516,736	5,258,368
Proven + Probable	5,627,687	0.104	11,721,895	5,860,948

Table 21-6
CHIEF PIT Crusher grade (>0.15%) Proven and Probable Minable Reserves

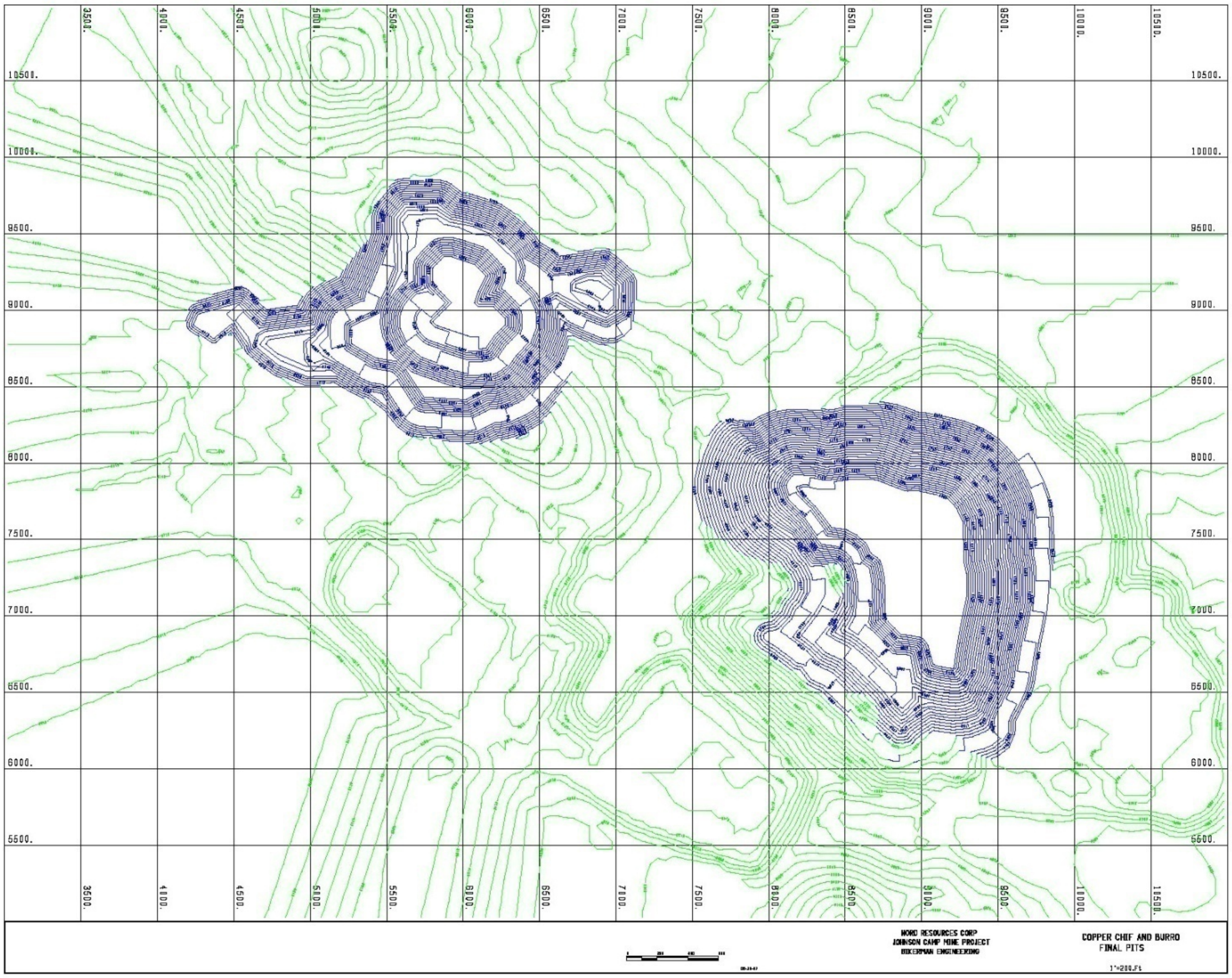
	Tons	Grade	Pounds Copper Contained	Pounds Copper Recoverable
Starter Pit	6,100,869	0.297	36,248,313	28,636,167
Final Pit	18,175,103	0.347	126,280,204	94,749,769
Proven + Probable	24,275,972	0.335	162,528,517	123,385,937

Table 21-7
CHIEF PIT Low Grade (<0.15%) Proven and Probable Minable Reserves

	Tons	Grade	Pounds Copper Contained	Pounds Copper Recoverable
Starter Pit	786,531	0.118	1,860,944	930,472
Final Pit	4,279,219	0.106	9,092,128	4,546,064
Proven + Probable	5,065,750	0.108	10,953,072	5,476,536

Figure 21-1

Final Pit Designs with Topography



Use of Total Copper Values

For reasons discussed below, the ore reserve estimate is based on total copper assays and recoveries rather than soluble copper assays and recoveries.

Total copper values were available for both the Copper Chief and Burro deposits. However, only 39 percent of the Copper Chief assay intervals also had acid soluble copper values, and the available data on acid soluble copper was incomplete for all samples. In addition, the database of acid soluble copper values for the Burro deposit reflects two different analytical techniques: (a) a conventional acid soluble method used by Cyprus for 94 of the holes included in the drill hole database; and (b) a more aggressive methodology used by Arimetco for the other 48 drill holes included in the database for the purpose of estimating the ultimate recoveries that may be experienced in the heaps at the Johnson Camp Mine. In summary, total copper assays were the only common denominator for all drill hole assays included in the drill hole database. As a result, only a total copper grade resource model was constructed for both deposits.

Estimation of total copper recovery for each ore type involved:

- examination of Cyprus drill hole data that contained both acid soluble assays and total copper assays, with the view to determining a correlation (expressed as a percentage) between such acid soluble assays and total copper values for each are type; and
- application of the correlation to the acid soluble copper recovery determined for the particular are type based on column tests and certain other parameters. Four column tests were used to estimate recoveries, one each for the following major rock types at the Johnson Camp Mine: Abrigo, Bolsa Quartzite, Pioneer Shale, and Diabase formations.

Thus, expressed as a formula:

$(A/ B) \times C = D$

Where:
A is the acid soluble assay;
B is total copper assay;
C is the acid soluble recovery for an ore type; and
D is the total copper recovery for that ore type.

BETA notes that a reserve estimate based on total copper is an indirect measurement of the amount of copper that is metallurgically available for recovery. Accordingly, there is a risk that the amount of copper recoverable is over-estimated.

21.2 MINABLE RESERVES

21.2.1 Economic and Design Parameters

Minable reserves for the Johnson Camp project are based upon the measured and indicated resources in the computerized 3-D block model described herein. Minable pit shapes optimize the extraction of the mineral inventory given the economic and technical parameters determined for this feasibility. The pit optimization procedures utilized in definition of the final pit design take the following factors and assumptions into consideration:

- copper price of $US 1.50 per pound;
- process recovery of contained copper values dependent on rock type;
- mining cost of $US 1.51 per ton of ore moved;
- mining cost of $US 1.61 per ton of waste moved;
- crushing cost of $US 0.637 per ton of ore;
- processing and laboratory cost of $US 0.285 per pound of copper produced;
- G & A and social cost of $US 0.35 per ton of ore;
- environmental cost of $US 0.03 per ton of ore;
- overall pit slope of 45 degrees on footwall and 55 degrees on hanging wall;
- minimum pit bottom of 60 feet;
- twenty-foot bench mining heights;
- bench face slope of 63 degrees;
- ultimate haul road grade of no greater than 10%; and
- total haul road width of 80 feet with berms.

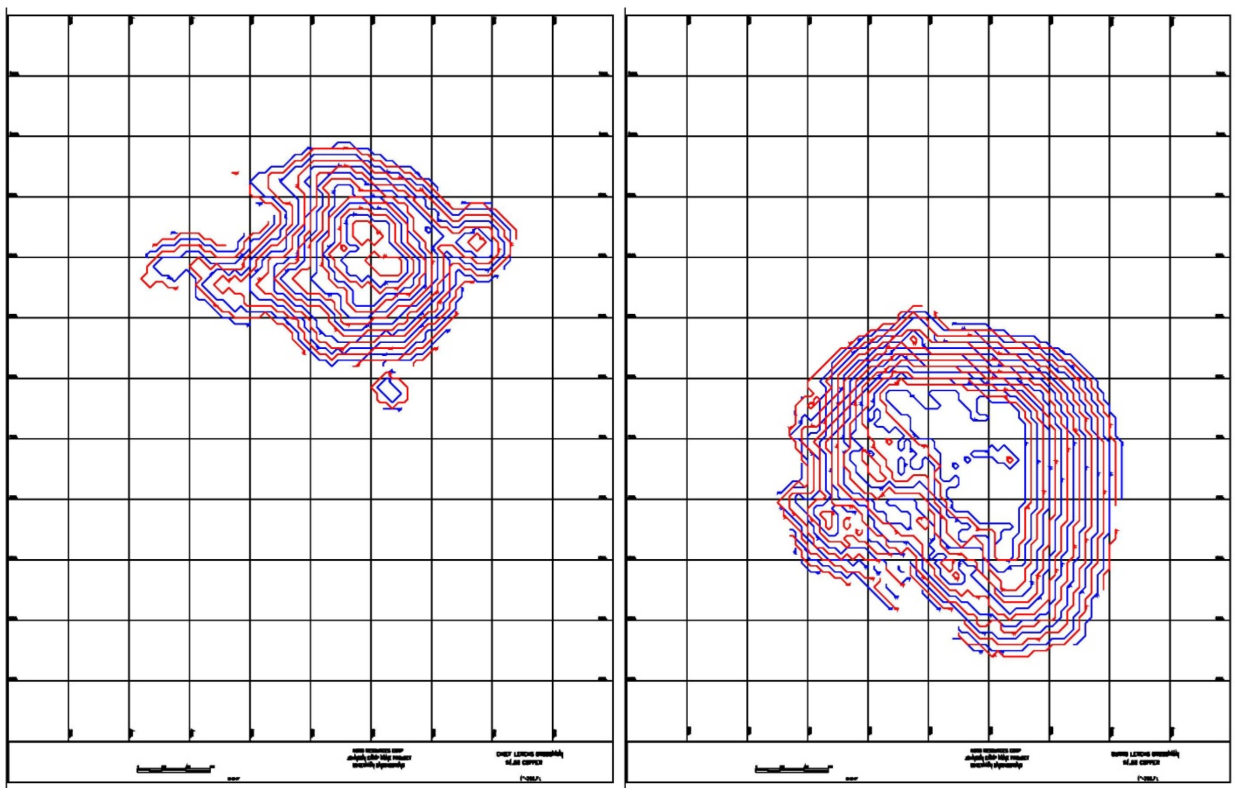
A Lerchs-Grossman algorithm was utilized to optimize the pit. This algorithm provided a basic pit shape outline that served as the basis for final pit design. The routine essentially floats an economic cone over all blocks in the 3-D block model to determine what mineralized material can be mined and processed given the economic parameters input. The Lerchs-Grossman pit shells are shown in Figure 21-2.

BETA ran sensitivity analyses of the pit shell reserves to changes in the copper price. BETA ran pit shell reserves at $1.25, $1.50, and $1.75 per pound. The copper price at the time of this report has remained over $3.00 per pound. Nord selected the base case price of $1.50 per pound. BETA is of the opinion that this is a conservative price that is appropriate for use in the study.

Final pit design is shown in Figure 21-1.
A schematic of the haul road profile is shown in Figure 21-3.

Figure 21-2
Floated Pit Results

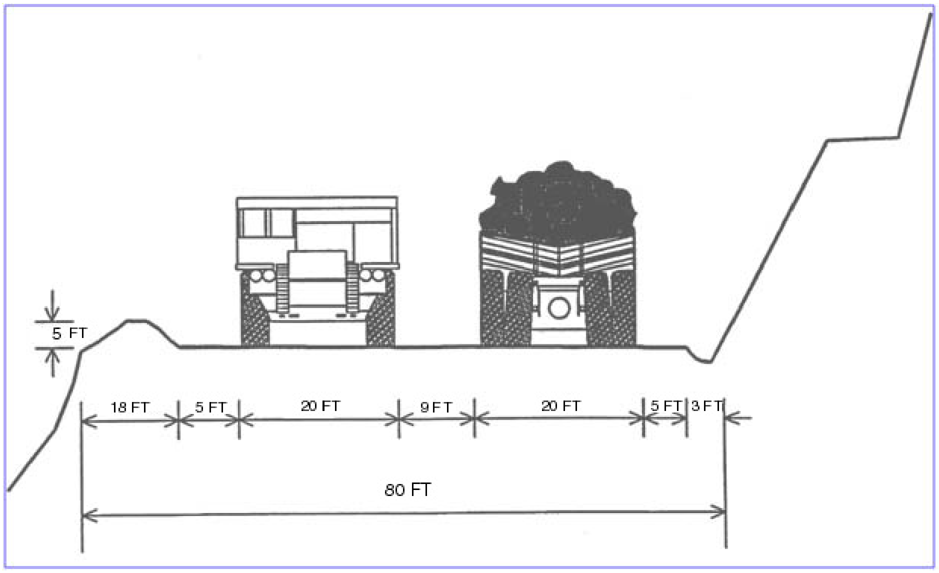
Copper Chief **Burro**



BETA used variable pit slopes to approximate the ramp-equivalent final pit slopes for the optimization runs. The variable interramp slope angles used by BETA to calculate the ramp-equivalent angles were supplied by an independent slope stability consultant. The ramp-equivalent slopes used for the Copper Chief pit are 55 degrees on the east wall, and 46 degrees on the west wall of the pit. For the Burro pit, 50 degrees was used for the northeast, northwest and south walls, and 30 degrees for the southeast and west walls. The pit slopes are discussed in Section 21.2.2.

BETA has redesigned the Burro and Copper Chief pits in preparation of this study. In doing so, BETA updated the 3-D block value-models according to current economic conditions, ran Lerchs-Grossmann (LG) pit shells, designed final pit shapes with variable pit slopes and feasible haul roads, and calculated ore reserves within the pit designs.

Figure 21-3
Haul Road Design



The parameters used for the algorithm are listed in Table 21-10. The costs used in the pit optimization run reflect current cash production cost estimates at an annual production rate of 25 million pounds of copper per year. The mine operating costs are based on a bid proposal estimate and revision prepared by a mining contractor, for Nord, based on the 2005 Feasibility Study mining schedule. BETA believes the cost estimate appropriate for use in estimating mining cost for this study.

BETA has developed a two-phase mine design for the Burro pit. The Burro pit design consists of a preliminary mining phase which deepens the bottom of the existing pit and a final phase that starts at the surface and enlarges and deepens the existing pit to below the starter pit. The starter pit establishes the Burro pit final ramp design on the west side of the pit. The Burro final pit will have to carry temporary internal ramp access in order to have access on the benches where the ramp has already been finalized by the starter pit.

The Copper Chief pit is a new design in an area with limited previous mining activity. When the Burro and Copper Chief pits have been mined to final limits as currently designed, the area will look as shown in Figure 21-1.

Overall ramp widths were set to 80 feet. Ramps were designed using a ten percent gradient for both the Burro and Copper Chief pits.

The Burro starter pit is shown on Figure 21-4. The starter pit takes advantage of the existing pit bottom and ramps to establish early access to ore with limited stripping requirements. Establishing the starter pit in the bottom of the existing Burro pit will require carefully mining out existing ramps to obtain working room.

The Burro final pit design is shown on Figure 21-6. The access ramp starts just south of the primary crusher location and wraps clockwise around the pit, taking out some of the waste dump on the east side of the pit. The access ramp wraps almost a full 360 degrees around the pit before making a switchback to reach the ultimate pit bottom. If the rock in the southwest sector of the pit proves to be very weak along the bedding planes, the slope between the two ramps will have to be redesigned (flattened).

Figure 21-4 Burro Starter Pit Design

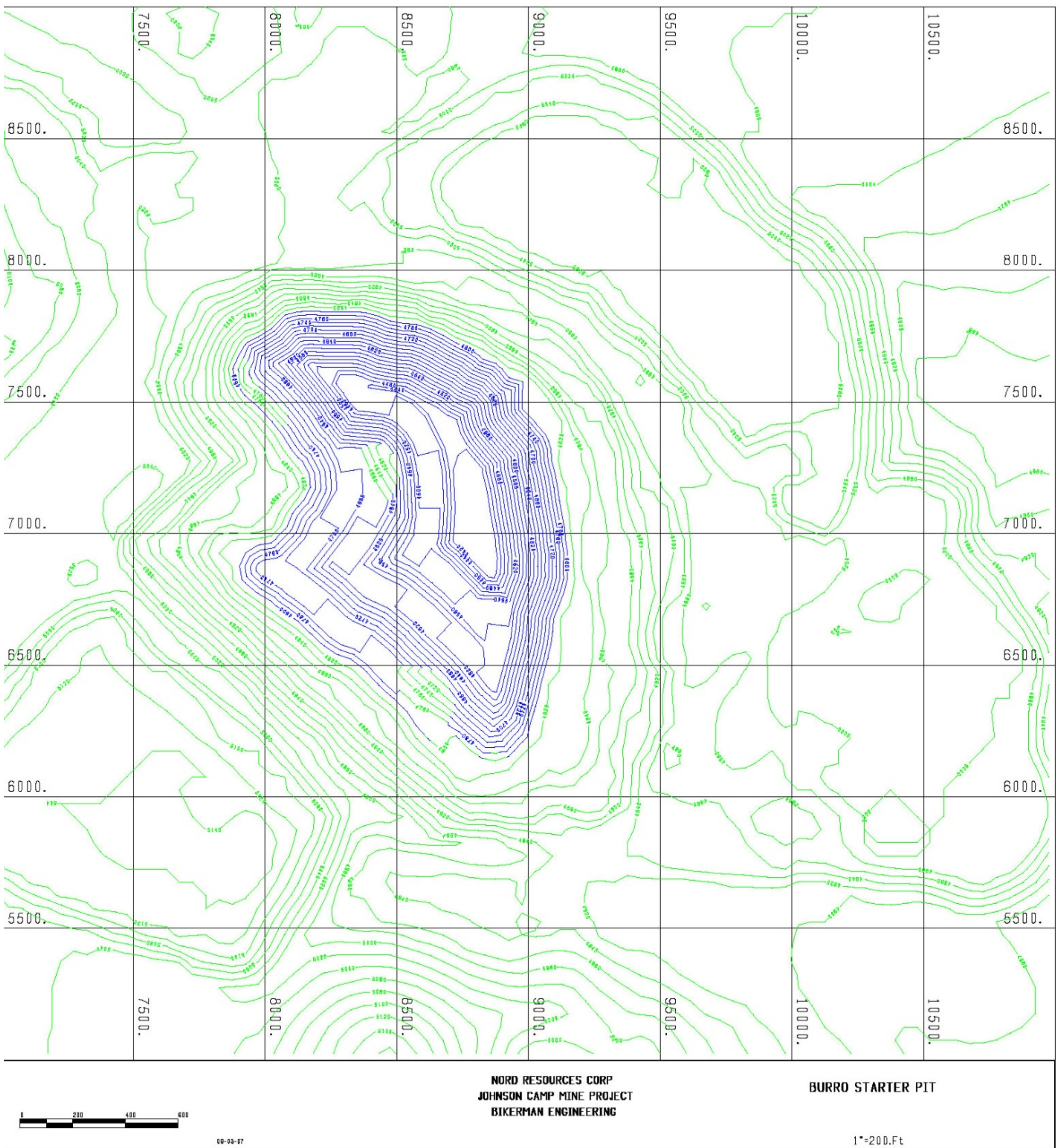


Figure 21-5

Copper Chief Starter Pit Design

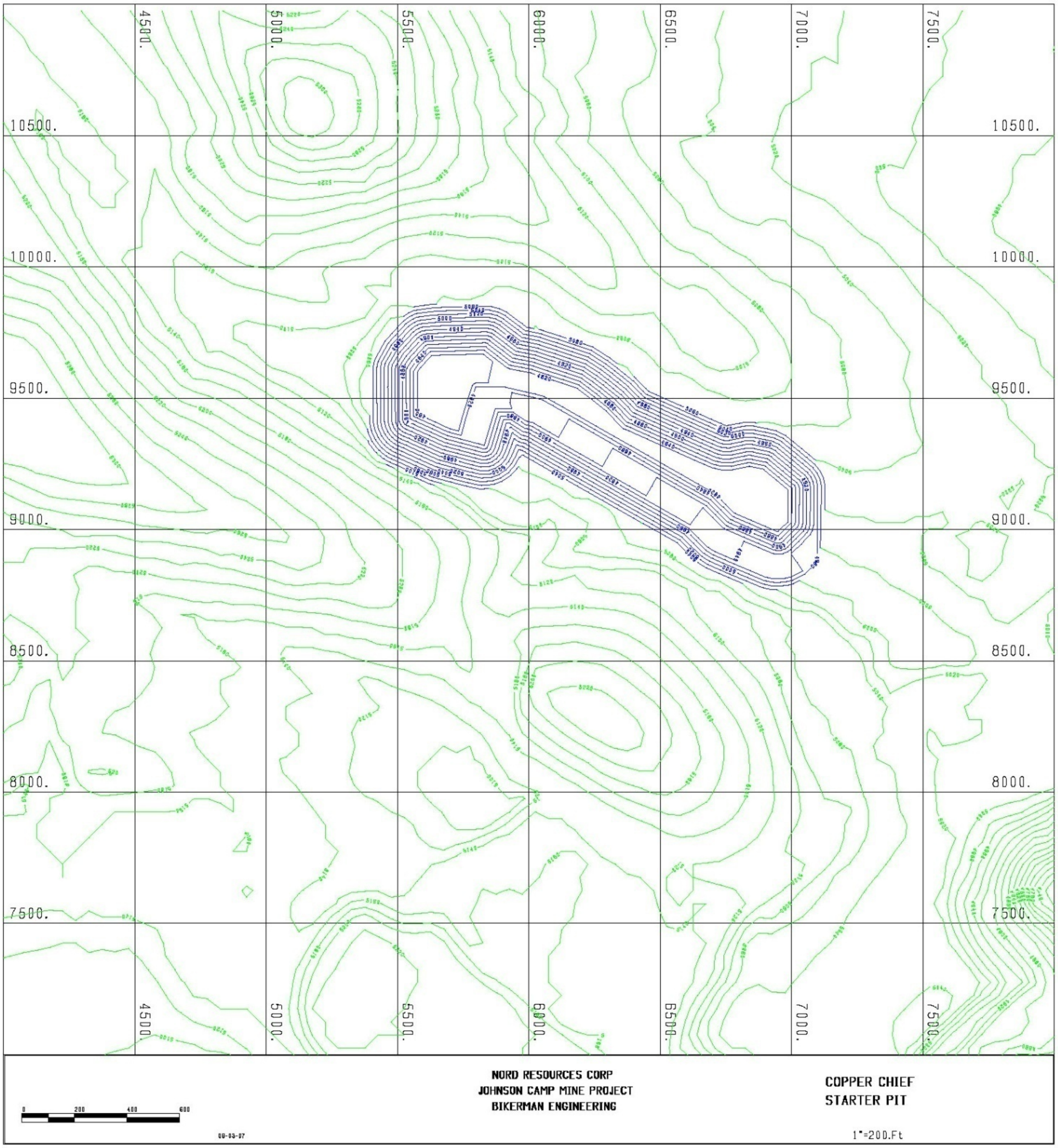


Figure 21-6

Burro Final Pit Design

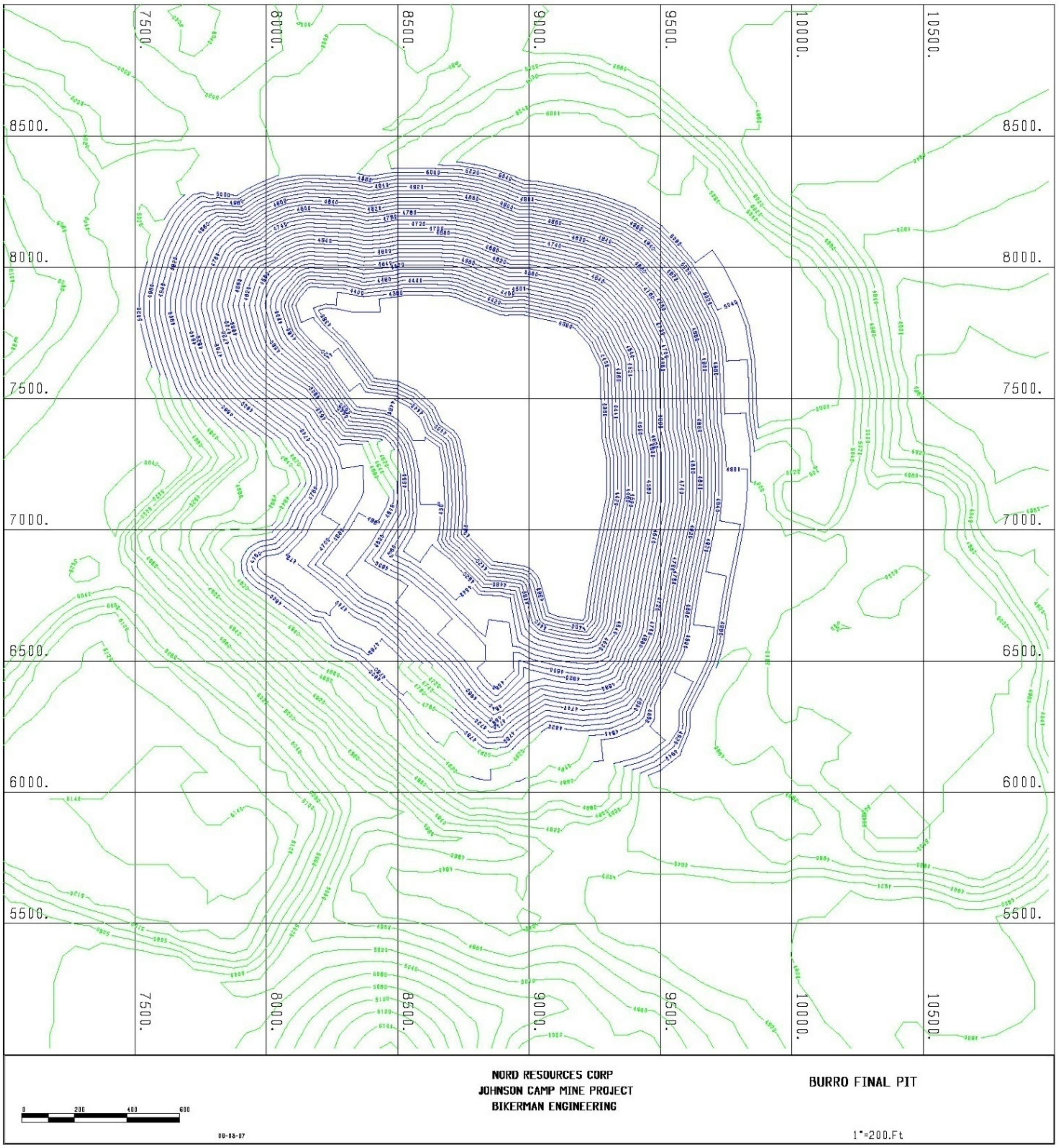
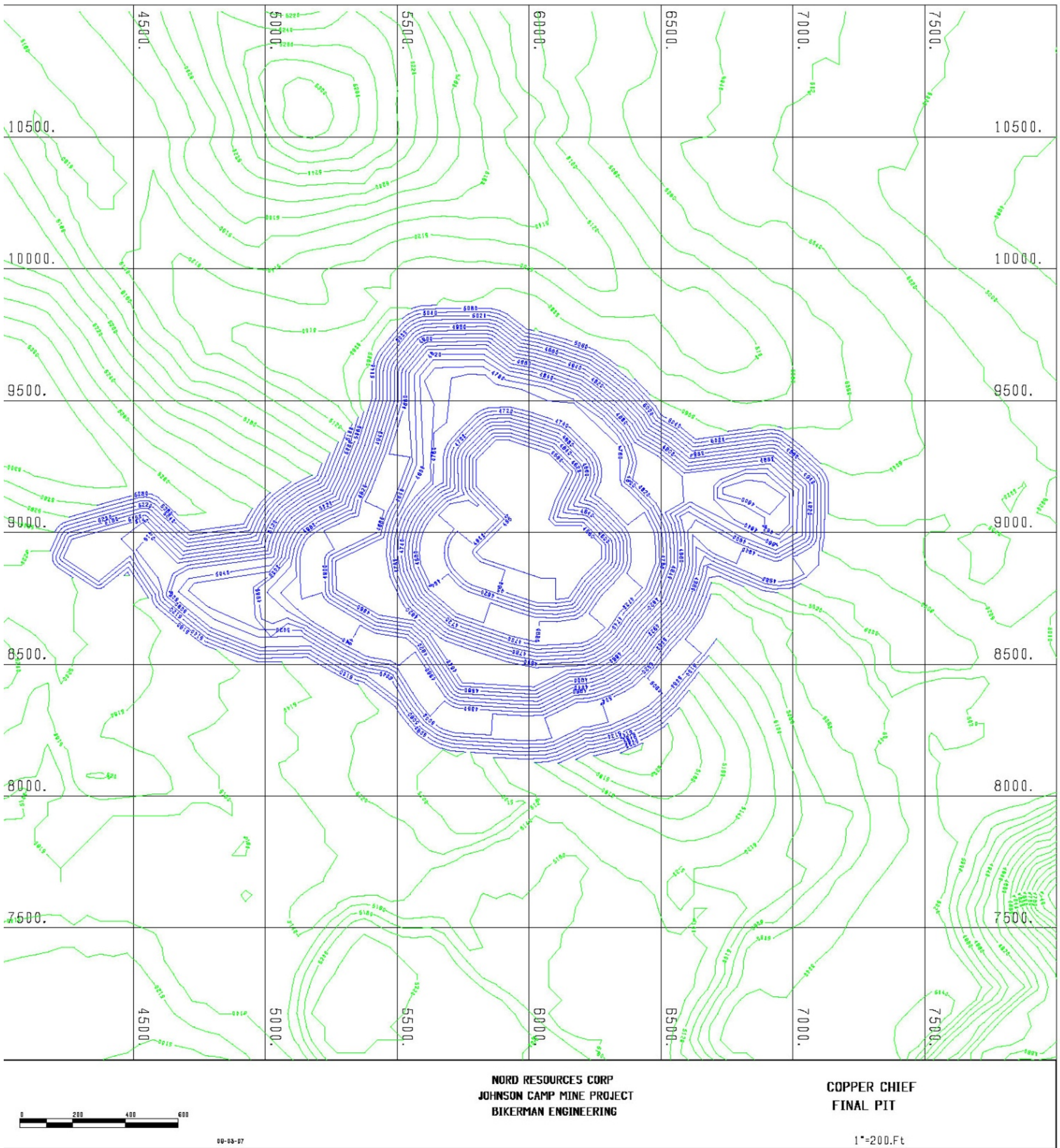


Figure 21-7

Copper Chief Final Pit Design



CUTOFF GRADES

The internal and external cutoff grades for all material types in both the Burro and Copper Chief pits are shown in Table 21-8 and 21-9 respectively.

The internal cutoff grade for distinguishing between ore and waste is calculated as shown in Table 21-10. In order for rock to be above the internal cutoff grade, the net revenue from processing the rock (Item 16 in Table 21-10) must exceed the sum of all cash operating costs excluding the mining cost. In the example in Table 21-10, Upper Abrigo material must have a total copper grade (TCu) above 0.065 percent in order to be categorized as ore for heap leaching.

BETA developed a computer program to calculate the economic value of each block in the resource model by the method illustrated in Table 21-10, taking into account both material type and the particular pit in which the block is located. BETA ran the LG program against the calculated dollar values in the block model using Mintec, Inc.'s MEDSYSTEM software.

The program BETA developed to calculate dollar values considered only blocks in the geologic model categorized as measured or indicated resources. Classification of resources into measured, indicated and inferred categories was performed as part of the Nord updated feasibility study. BETA reviewed the classifications of resources, concluded it was reasonable, and has incorporated the classification of resources into this technical report.

Measured and indicated resource blocks having values that exceed the internal cutoff grade were then classified as proven or probable ore blocks. All inferred resource blocks were treated as waste, regardless of their estimated copper grade.

Table 21-8

Cutoff Grades - Burro Pit

TABLE 21-8 COPPER = $1.50/lb	INTERNAL CUTOFF GRADES	EXTERNAL CUTOFF GRADES
	Burro Pit	Burro Pit
	TCu, %	TCu, %
Upper Abrigo	0.065	0.146
Middle Abrigo	0.065	0.146
Lower Abrigo	0.065	0.146
Bolsa Quartzite	0.067	0.151
Upper Diabase	0.063	0.142
Upper Pioneer Shale	0.067	0.151
Lower Diabase	0.063	0.142
Lower Pioneer Shale	0.067	0.151

Table 21-9

Cutoff Grades – Copper Chief Pit

TABLE 21-9 COPPER = $1.50/lb	INTERNAL CUTOFF GRADES	EXTERNAL CUTOFF GRADES
	Copper Chief Pit	Copper Chief Pit
	TCu, %	TCu, %
Upper Abrigo	0.065	0.146
Middle Abrigo	0.065	0.146
Lower Abrigo	0.065	0.146
Bolsa Quartzite	0.067	0.151
Upper Diabase	0.069	0.155
Upper Pioneer Shale	0.067	0.151
Lower Diabase	0.069	0.155
Lower Pioneer Shale	0.067	0.151

Table 21-10

TABLE 21- CUTOFF CALCS

CUTOFF GRADE CALCULATION FOR UPPER ABRIGO MATERIAL IN BURRO PIT

		2007 BETA	
Material grade % TCU		**0.0648**	%
Model column num. (>83 = Burro)	$	0.02	
Rock Type		1	
Topo % value in block		100	
1 Block Volume			
X-dimension		50	ft
Y-dimension		50	ft
Z-dimension		20	ft
Block volume		50000	cu ft
2 Adjust For Topo			
Topo %		100	
Adjusted block volume		50000	cu ft
3 Tonnage factor		12.46	cu. ft/ton
4 Block tons		4,013	
5 Recovery		% TCu	
1 Upper Abrigo		79	
2 Middle Abrigo		79	
3 Lower Abrigo		79	
4 Bolsa Quartzite		76	
5 Upper Diabase		81	
6 Upper Pioneer Shale		76	
7 Lower Diabase		81	
8 Lower Pioneer Shale		76	
Recovery used for rock type inputted		79	
6 Saleable copper produced		4,108	
7 Gross Revenue			
Copper price	$	1.50	
Marketing & Freight	$	0.01	
Copper sales	$	6,121.20	
Severance Tax			
% of sales 1.25%	$	76.52	
Total gross revenue	$	6,044.69	
8 Ore Rehandling			
Ore rehandling cost / ore ton -		0	
9 Crushing			
Crushing cost / ore ton		0.637	
Crushing cost	$	2,556.18	
10 Crushing & Leaching Sustaining Capital			
Crushing & leaching sustaining capital / ore ton	$	0.025	
Crushing & leaching sustaining capital cost	$	100.32	
11 Processing			
Cost per lb of copper produced		0.285	
Processing cost	$	1,170.83	
12 Processing Sustaining Capital			
Processing sustaining capital cost / lb rec. copper		0.0025	
Processing sustaining capital cost	$	10.27	
13 Pad Construction Cost			
Pad construction cost / ore ton	$	0.20	
Pad construction cost	$	802.57	
14 G&A Operating Cost			
G&A operating cost / ore ton		0.35	
G&A operating cost cost	$	1,404.49	
15 Total Non-mining Operating Cost			
Ore rehandling			
Crushing	$	2,556.18	
Crushing & leaching sustaining capital cost	$	100.32	
Processing cost	$	1,170.83	
Processing sustaining capital cost	$	10.27	
Pad construction cost	$	802.57	
G&A operating cost cost	$	1,404.49	
Total	$	6,044.67	
16 Net Revenue			
Gross revenue	$	6,044.69	
less total non-mining operating costs	$	(6,044.67)	
Net revenue	$	0.02	
17 If Net Revenue is Positive			
Ore Mining Costs			
Fixed ore mining cost / ton of ore		1.51	
Total mining cost		6,059.39	
Block Value			
Net revenue	$	0.02	
less mining cost		6,059.39	
Total Block Value	$	(6,059.37)	
Else if Net Revenue is Negative and not Dump Material			
Waste Mining Costs			
Fixed waste mining cost / ton of waste		1.61	
Total mining cost		6,460.67	
Block Value	$	(6,460.65)	

21.2.2 Slope Stability

BETA has reviewed and incorporated into this technical report the work done by a slope stability consulting firm for this project. The consulting firm visited the property and analyzed the rock structural data, but did not do any rock strength testing or stability modeling.

The Summo study originally used an interramp slope angle of 55 degrees in designing all of the initial pit walls. After the slope stability consulting firm examined the property, they approved the use of 55-degree interramp slopes except in the southwest corner of the Burro pit. For the southwest corner of the Burro pit the consultant recommended that the interramp pit slopes be reduced to a slope angle near the bedding angle and that the slopes not be excavated through the Bolsa Quartzite into the underlying Upper Diabase.

The southwest corner of the Burro pit is in rock that is dipping parallel to the proposed pit wall and appears to be weak along the contact with the Bolsa Quartzite and the Upper Diabase sill. If the contact daylights into the pit wall and the rock is not sufficiently strong along the contact, the southwest corner of the pit wall with two ramps potentially could fail along the structure. The geology and rock strengths in the southwest corner of the Burro pit will need further study prior to being mined.

BETA has reduced the slopes in the area of concern to an interramp slope angle of 45 degrees. All other interramp slopes in both the Burro and Copper Chief pits are based on the initial 55-degree slope design.

21.2.3 Minable Reserves by Bench

The minable reserves for the Johnson Camp project are listed by bench in the following tables. The reserves are detailed by pit, by bench, and by grade (crusher grade/low grade).

The Burro Starter pit reserves are detailed in Tables 21-11 through 21-14.

The Burro Final pit reserves are detailed in Tables 21-15 through 21-19.

The Copper Chief Starter pit reserves are detailed in Tables 21-19 through 21-22.

The Copper Chief Final pit reserves are detailed in Tables 21-23 through 21-26.

Table 21-11
Burro Starter Pit
Proven and Probable Reserves by Bench

BENCH TOE	PROVEN						PROBABLE				
	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERY BENCH AVG %	RECOVERABLE COPPER Lbs	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs
4800	4,985	10,968	0.250	54,840	78.1	42,830	1,103	2,422	0.239	11,574	9,040
4780	49,346	109,407	0.222	485,767	78.2	379,870	3,223	6,971	0.289	40,261	31,484
4760	122,380	274,030	0.263	1,441,398	78.3	1,128,614	12,432	27,191	0.307	167,122	130,857
4740	136,778	300,336	0.309	1,856,076	77.7	1,442,171	26,229	57,709	0.315	363,803	282,675
4720	152,508	337,111	0.277	1,867,595	77.8	1,452,989	39,463	86,501	0.350	606,299	471,701
4700	169,156	368,651	0.359	2,646,914	77.3	2,046,065	44,310	95,871	0.311	595,449	460,282
4680	192,931	421,806	0.411	3,467,245	77.3	2,680,181	78,056	169,382	0.404	1,368,673	1,057,984
4660	215,676	469,306	0.396	3,716,904	77.3	2,873,166	71,620	154,806	0.461	1,425,779	1,102,127
4640	233,626	506,879	0.435	4,409,847	77.1	3,399,992	65,130	141,673	0.499	1,414,150	1,090,309
4620	251,724	544,846	0.485	5,285,006	77.1	4,074,740	57,314	123,093	0.561	1,381,025	1,064,770
4600	237,699	513,747	0.515	5,291,594	77.1	4,079,819	50,555	109,303	0.606	1,325,197	1,021,727
4580	224,870	486,055	0.557	5,414,653	77.1	4,174,697	53,057	113,943	0.641	1,461,324	1,126,681
4560	222,222	480,992	0.584	5,617,987	77.0	4,325,850	55,518	120,663	0.619	1,493,576	1,150,053
4540	199,703	431,919	0.560	4,837,493	77.0	3,724,869	39,963	87,246	0.643	1,122,521	864,341
4520	154,370	333,129	0.532	3,544,493	76.8	2,722,170	43,871	98,223	0.479	941,568	723,124
4500	141,463	304,813	0.500	3,048,130	76.6	2,334,868	23,796	53,141	0.500	531,410	407,060
4480	110,741	238,048	0.512	2,437,612	76.6	1,867,210	21,981	49,027	0.448	438,880	336,182
4460	73,296	158,293	0.538	1,703,233	76.9	1,309,786	25,945	58,077	0.437	508,069	390,705
4440	49,556	106,191	0.535	1,136,244	76.2	865,818	15,351	33,040	0.522	345,174	263,023
4420	27,778	60,059	0.429	515,306	76.3	393,179	12,018	25,734	0.469	241,388	184,179
TOTAL	2,970,808	6,456,586	0.455	58,778,336		45,318,884	740,935	1,614,016	0.489	15,783,243	12,168,305

Table 21-12
Burro Starter Pit
Proven plus Probable Reserves by Bench

PROVEN AND PROBABLE							
BENCH TOE	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs	WASTE TOTAL (TONS)	ROM S/R
4800	6,088	13,390	0.248	66,414	51,870	1,223	0.09
4780	52,569	116,378	0.226	526,029	411,354	5,946	0.05
4760	134,812	301,221	0.267	1,608,520	1,259,471	21,531	0.07
4740	163,007	358,045	0.310	2,219,879	1,724,846	53,013	0.15
4720	191,971	423,612	0.292	2,473,894	1,924,690	46,847	0.11
4700	213,466	464,522	0.349	3,242,364	2,506,347	60,445	0.13
4680	270,987	591,188	0.409	4,835,918	3,738,164	45,562	0.08
4660	287,296	624,112	0.412	5,142,683	3,975,294	71,284	0.11
4640	298,756	648,552	0.449	5,823,997	4,490,302	98,411	0.15
4620	309,038	667,939	0.499	6,666,031	5,139,510	67,604	0.10
4600	288,254	623,050	0.531	6,616,791	5,101,546	67,730	0.11
4580	277,927	599,998	0.573	6,875,977	5,301,378	40,300	0.07
4560	277,740	601,655	0.591	7,111,562	5,475,903	39,483	0.07
4540	239,666	519,165	0.574	5,960,014	4,589,211	51,677	0.10
4520	198,241	431,352	0.520	4,486,061	3,445,295	47,412	0.11
4500	165,259	357,954	0.500	3,579,540	2,741,928	31,434	0.09
4480	132,722	287,075	0.501	2,876,492	2,203,392	16,030	0.06
4460	99,241	216,370	0.511	2,211,301	1,700,491	5,868	0.03
4440	64,907	139,231	0.532	1,481,418	1,128,840	2,815	0.02
4420	39,796	85,793	0.441	756,694	577,358	912	0.01
TOTAL	3,711,743	8,070,602	0.462	74,561,579	57,487,190	775,527	0.10

Table 21-13
Burro Starter Pit
Inferred resources within Pit by Bench

	POSSIBLE				
BENCH TOE	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs @ BENCH AVG
4800	-	-	-	-	-
4780	-	-	-	-	-
4760	2,381	5,250	0.330	34,650	27,131
4740	463	1,103	0.120	2,647	2,057
4720	3,167	7,098	0.327	46,421	36,115
4700	1,685	3,608	0.516	37,235	28,782
4680	481	1,145	0.095	2,176	1,682
4660	-	-	-	-	-
4640	10,241	22,902	0.356	163,062	125,721
4620	7,278	15,817	0.498	157,537	121,461
4600	6,352	14,308	0.334	95,577	73,690
4580	1,056	2,260	0.558	25,222	19,446
4560	5,981	13,358	0.555	148,274	114,171
4540	17,741	38,411	0.742	570,019	438,915
4520	15,389	33,689	0.675	454,802	349,288
4500	12,815	28,433	0.538	305,939	234,349
4480	7,167	16,030	0.442	141,705	108,546
4460	2,741	5,868	0.559	65,604	50,450
4440	1,315	2,815	0.718	40,423	30,803
4420	426	912	0.737	13,443	10,257
TOTAL	96,679	213,007	0.541	2,304,736	1,772,864

Table 21-14
Burro Starter Pit
High Grade and Low Grade Reserves by Bench

BENCH TOE	HIGH GRADE (greater than 0.15%)						LOW GRADE (greater than 0.065% and less than 0.15%)				
	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERY BENCH AVG %	RECOVERABLE COPPER Lbs	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs @ 0.5
4800	4,145	9,178	0.309	56,720	78.1	44,298	1,943	4,212	0.115	9,694	4,847
4780	32,536	72,288	0.298	430,836	78.2	336,914	20,033	44,090	0.108	95,192	47,596
4760	101,588	227,414	0.317	1,441,805	78.3	1,128,933	33,224	73,807	0.113	166,715	83,358
4740	127,533	280,061	0.368	2,061,249	77.7	1,601,590	35,474	77,984	0.102	158,630	79,315
4720	139,067	304,153	0.368	2,238,566	77.8	1,741,604	52,904	119,459	0.098	235,328	117,664
4700	174,999	379,285	0.405	3,072,209	77.3	2,374,817	38,467	85,237	0.100	170,155	85,078
4680	244,246	531,091	0.445	4,726,710	77.3	3,653,747	26,741	60,097	0.091	109,208	54,604
4660	267,888	580,613	0.436	5,062,945	77.3	3,913,657	19,408	43,499	0.092	79,738	39,869
4640	278,738	603,086	0.476	5,741,379	77.1	4,426,603	20,018	45,466	0.091	82,618	41,309
4620	299,851	647,561	0.512	6,631,025	77.1	5,112,520	9,187	20,378	0.086	35,007	17,503
4600	281,569	608,462	0.541	6,583,559	77.1	5,075,924	6,685	14,588	0.114	33,232	16,616
4580	275,742	595,247	0.577	6,869,150	77.1	5,296,115	2,185	4,751	0.072	6,827	3,413
4560	273,111	591,742	0.599	7,089,069	77.0	5,458,583	4,629	9,913	0.113	22,493	11,246
4540	239,666	519,165	0.574	5,960,014	77.0	4,589,211	-	-	-	-	
4520	198,241	431,352	0.520	4,486,061	76.8	3,445,295	-	-	-	-	
4500	165,259	357,954	0.500	3,579,540	76.6	2,741,928	-	-	-	-	
4480	132,055	285,487	0.504	2,877,709	76.6	2,204,325	-	-	-	-	-
4460	98,852	215,495	0.513	2,210,979	76.9	1,700,243	389	875	0.018	323	161
4440	64,907	139,231	0.532	1,481,418	76.2	1,128,840	-	-	-	-	
4420	39,796	85,793	0.441	756,694	76.3	577,358	-	-	-	-	
	3,439,789	7,464,658	0.491	73,357,637		56,552,505	271,287	604,356	0.100	1,205,160	602,580

Table 21-15
Burro Final Pit
Proven and Probable Reserves by Bench

BENCH TOE	PROVEN INSITU ORE (YDS)	PROVEN INSITU ORE (TONS)	PROVEN INSITU GRADE TCU	PROVEN INSITU COPPER Lbs	PROVEN RECOVERY BENCH AVG %	PROVEN RECOVERABLE COPPER Lbs	PROBABLE INSITU ORE (YDS)	PROBABLE INSITU ORE (TONS)	PROBABLE INSITU GRADE TCU	PROBABLE INSITU COPPER Lbs	PROBABLE RECOVERABLE COPPER Lbs
5060	-	-	-	-	-	-	-	-	-	-	-
5040	-	-	-	-	-	-	-	-	-	-	-
5020	481	1,031	0.510	10,516	76.80	8,076	1,094	2,353	0.478	22,512	17,289
5000	26,819	58,086	0.260	302,047	78.80	238,013	24,574	53,194	0.335	356,730	281,104
4980	61,862	134,008	0.256	686,121	78.90	541,349	60,649	131,340	0.309	810,442	639,439
4960	77,548	168,042	0.254	853,653	78.90	673,533	67,419	145,912	0.265	772,628	609,604
4940	129,641	280,871	0.256	1,438,060	78.90	1,134,629	83,351	180,382	0.241	868,205	685,014
4920	164,905	357,181	0.240	1,714,469	78.90	1,352,716	102,607	222,146	0.227	1,008,368	795,602
4900	187,264	405,621	0.243	1,971,318	79.00	1,557,341	110,639	239,617	0.211	1,009,682	797,648
4880	227,178	492,027	0.254	2,499,497	79.00	1,974,603	107,723	233,279	0.207	967,466	764,298
4860	253,572	549,182	0.263	2,888,697	79.00	2,282,071	114,391	247,791	0.221	1,096,168	865,972
4840	269,821	584,172	0.274	3,201,263	78.90	2,525,796	157,842	341,894	0.260	1,780,973	1,405,187
4820	317,470	687,561	0.258	3,547,815	78.90	2,799,226	167,770	363,347	0.220	1,601,634	1,263,690
4800	362,424	784,767	0.255	4,002,312	78.90	3,157,824	221,148	478,872	0.202	1,936,792	1,528,129
4780	394,956	855,092	0.230	3,933,423	78.90	3,103,471	221,910	480,485	0.205	1,969,827	1,554,194
4760	319,109	690,904	0.238	3,288,703	78.70	2,588,209	113,635	245,509	0.211	1,037,525	816,532
4740	324,509	702,400	0.223	3,132,704	78.60	2,462,305	133,633	289,043	0.196	1,130,501	888,574
4720	362,482	784,325	0.254	3,984,371	78.60	3,131,716	178,000	384,659	0.287	2,211,244	1,738,038
4700	385,644	834,351	0.242	4,038,259	78.50	3,170,033	176,338	380,949	0.232	1,770,875	1,390,137
4680	402,608	871,710	0.252	4,393,418	78.40	3,444,440	219,240	473,769	0.283	2,683,801	2,104,100
4660	411,981	891,283	0.264	4,705,974	78.40	3,689,484	209,759	453,250	0.291	2,635,176	2,065,978
4640	498,628	1,079,421	0.281	6,066,346	78.30	4,749,949	207,630	448,543	0.298	2,673,608	2,093,435
4620	501,702	1,085,206	0.300	6,511,236	78.30	5,098,298	208,667	450,985	0.273	2,460,119	1,926,274
4600	509,684	1,104,058	0.335	7,397,189	78.30	5,791,999	199,351	431,573	0.303	2,615,126	2,047,643
4580	508,388	1,101,231	0.356	7,840,765	78.40	6,147,160	184,259	398,558	0.330	2,627,763	2,060,166
4560	522,313	1,132,566	0.374	8,471,594	78.30	6,633,258	178,926	386,875	0.362	2,802,659	2,194,482
4540	503,073	1,091,058	0.419	9,143,066	78.30	7,159,021	184,203	398,039	0.408	3,246,221	2,541,791
4520	480,795	1,046,029	0.433	9,058,611	78.30	7,092,893	201,889	436,309	0.436	3,808,083	2,981,729
4500	472,684	1,028,589	0.455	9,360,160	78.40	7,338,365	191,277	413,174	0.441	3,644,542	2,857,321
4480	454,851	989,540	0.495	9,796,446	78.40	7,680,414	183,185	396,062	0.432	3,422,197	2,683,003
4460	456,203	994,365	0.495	9,844,214	78.10	7,688,331	169,166	367,004	0.417	3,061,565	2,391,082
4440	447,443	975,460	0.493	9,618,036	78.10	7,511,686	167,926	364,371	0.453	3,297,935	2,575,687
4420	400,480	873,388	0.474	8,279,718	77.90	6,449,901	178,871	390,453	0.458	3,575,110	2,785,011
4400	368,055	799,900	0.494	7,903,012	77.70	6,140,640	193,536	422,717	0.456	3,858,564	2,998,104
4380	260,592	570,325	0.453	5,167,145	77.70	4,014,871	208,463	453,626	0.451	4,089,373	3,177,442
4360	216,833	475,297	0.444	4,220,637	77.80	3,283,656	180,018	391,002	0.493	3,853,269	2,997,844
TOTAL	11,281,998	24,479,047	0.346	169,270,794	78.40	132,615,275	5,309,089	11,497,082	0.325	74,706,681	58,521,541

Table 21-16
Burro Final Pit
Proven plus Probable Reserves by Bench

PROVEN AND PROBABLE							
BENCH TOE	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs	WASTE TOTAL (TONS)	ROM S/R
5060	-	-	-	-	-	138,783	-
5040	-	-	-	-	-	488,145	-
5020	1,575	3,384	0.488	33,028	25,365	525,030	155.15
5000	51,393	111,280	0.296	658,778	519,117	623,730	5.61
4980	122,511	265,348	0.282	1,496,563	1,180,788	796,540	3.00
4960	144,967	313,954	0.259	1,626,282	1,283,136	1,096,116	3.49
4940	212,992	461,253	0.250	2,306,265	1,819,643	1,149,797	2.49
4920	267,512	579,327	0.235	2,722,837	2,148,318	1,211,894	2.09
4900	297,903	645,238	0.231	2,981,000	2,354,990	1,233,802	1.91
4880	334,901	725,306	0.239	3,466,963	2,738,901	1,233,865	1.70
4860	367,963	796,973	0.250	3,984,865	3,148,043	1,211,948	1.52
4840	427,663	926,066	0.269	4,982,235	3,930,983	1,200,374	1.30
4820	485,240	1,050,908	0.245	5,149,449	4,062,915	1,139,704	1.08
4800	583,572	1,263,639	0.235	5,939,103	4,685,953	984,074	0.78
4780	616,866	1,335,577	0.221	5,903,250	4,657,665	900,340	0.67
4760	432,744	936,413	0.231	4,326,228	3,404,741	1,273,712	1.36
4740	458,142	991,443	0.215	4,263,205	3,350,879	1,184,433	1.19
4720	540,482	1,168,984	0.265	6,195,615	4,869,754	954,906	0.82
4700	561,982	1,215,300	0.239	5,809,134	4,560,170	856,501	0.70
4680	621,848	1,345,479	0.263	7,077,220	5,548,540	676,637	0.50
4660	621,740	1,344,533	0.273	7,341,150	5,755,462	624,562	0.46
4640	706,258	1,527,964	0.286	8,739,954	6,843,384	383,236	0.25
4620	710,369	1,536,191	0.292	8,971,355	7,024,571	314,818	0.20
4600	709,035	1,535,631	0.326	10,012,314	7,839,642	262,191	0.17
4580	692,647	1,499,789	0.349	10,468,527	8,207,325	239,315	0.16
4560	701,239	1,519,441	0.371	11,274,252	8,827,739	155,702	0.10
4540	687,276	1,489,097	0.416	12,389,287	9,700,812	124,971	0.08
4520	682,684	1,482,338	0.434	12,866,694	10,074,621	73,671	0.05
4500	663,961	1,441,763	0.451	13,004,702	10,195,687	55,207	0.04
4480	638,036	1,385,602	0.477	13,218,643	10,363,416	53,466	0.04
4460	625,369	1,361,369	0.474	12,905,778	10,079,413	44,060	0.03
4440	615,369	1,339,831	0.482	12,915,971	10,087,373	30,658	0.02
4420	579,351	1,263,841	0.469	11,854,829	9,234,911	55,485	0.04
4400	561,591	1,222,617	0.481	11,761,576	9,138,744	71,765	0.06
4380	469,055	1,023,951	0.452	9,256,517	7,192,314	165,703	0.16
4360	396,851	866,299	0.466	8,073,907	6,281,499	220,406	0.25
TOTAL	16,591,087	35,976,129	0.339	243,977,475	191,136,816	21,755,547	0.60

Table 21-17
Burro Final Pit
Inferred resources within Pit by Bench

	POSSIBLE				
BENCH TOE	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs @ 0.73
5060	-	-	-	-	-
5040	-	-	-	-	-
5020	370	803	0.720	11,563	8,441
5000	12,512	27,019	0.337	182,108	132,939
4980	25,778	55,763	0.325	362,460	264,595
4960	15,426	33,353	0.330	220,130	160,695
4940	11,778	25,426	0.298	151,539	110,623
4920	9,870	21,293	0.276	117,537	85,802
4900	5,056	10,868	0.191	41,516	30,307
4880	6,722	14,498	0.286	82,929	60,538
4860	2,889	6,212	0.236	29,321	21,404
4840	14,519	31,229	0.231	144,278	105,323
4820	16,574	35,703	0.124	88,543	64,637
4800	28,093	60,479	0.153	185,066	135,098
4780	17,204	36,999	0.115	85,098	62,121
4760	17,174	37,337	0.168	125,452	91,580
4740	14,667	31,549	0.143	90,230	65,868
4720	32,870	70,957	0.195	276,732	202,015
4700	23,333	50,233	0.182	182,848	133,479
4680	42,667	92,070	0.170	313,038	228,518
4660	34,407	74,308	0.197	292,774	213,725
4640	30,037	64,953	0.169	219,541	160,265
4620	27,352	59,075	0.150	177,225	129,374
4600	22,018	47,498	0.211	200,442	146,322
4580	23,481	50,616	0.223	225,747	164,796
4560	30,204	65,177	0.248	323,278	235,993
4540	20,500	44,072	0.365	321,726	234,860
4520	25,130	53,851	0.288	310,182	226,433
4500	20,500	43,898	0.270	237,049	173,046
4480	20,907	44,766	0.303	271,282	198,036
4460	20,556	44,060	0.409	360,411	263,100
4440	14,296	30,658	0.398	244,038	178,148
4420	23,667	50,722	0.453	459,541	335,465
4400	32,833	70,397	0.419	589,927	430,647
4380	64,852	140,701	0.346	973,651	710,765
4360	68,167	147,250	0.370	1,089,650	795,445
TOTAL	776,409	1,673,793	0.268	8,986,850	6,560,401

Table 21-18
Burro Final Pit
High Grade and Low Grade Reserves by Bench

BENCH TOE	HIGH GRADE (greater than 0.15%) INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	G RECOVER BENCH AVC %	RECOVERABLE COPPER (Lbs)	LOW GRADE (greater than 0.065% and less than 0.15%) INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs @ 50%
5060	-	-	-	-	-	-	-	-	-	-	-
5040	-	-	-	-	-	-	-	-	-	-	-
5020	1,575	3,384	0.488	33,028	76.80	25,365	-	-	-	-	-
5000	51,393	111,280	0.296	658,778	78.80	519,117	-	-		-	-
4980	119,641	259,128	0.286	1,482,212	78.90	1,169,465	2,870	6,220	0.115	14,351	7,175
4960	124,430	269,546	0.285	1,536,412	78.90	1,212,229	20,537	44,408	0.101	89,870	44,935
4940	164,641	356,688	0.293	2,090,192	78.90	1,649,161	48,351	104,565	0.103	216,073	108,037
4920	185,697	402,281	0.289	2,325,184	78.90	1,834,570	81,815	177,046	0.112	397,653	198,826
4900	210,069	455,145	0.281	2,557,915	79.00	2,020,753	87,834	190,093	0.111	423,085	211,542
4880	241,530	523,313	0.289	3,024,749	79.00	2,389,552	93,371	201,993	0.109	442,214	221,107
4860	290,640	629,714	0.288	3,627,153	79.00	2,865,451	77,323	167,259	0.107	357,712	178,856
4840	333,834	723,042	0.317	4,584,086	78.90	3,616,844	93,829	203,024	0.098	398,149	199,074
4820	351,824	762,146	0.303	4,618,605	78.90	3,644,079	133,416	288,762	0.092	530,844	265,422
4800	409,147	886,261	0.296	5,246,665	78.90	4,139,619	174,425	377,378	0.092	692,438	346,219
4780	429,561	930,490	0.276	5,136,305	78.90	4,052,544	187,305	405,087	0.095	766,946	383,473
4760	309,921	671,145	0.283	3,798,681	78.70	2,989,562	122,823	265,268	0.099	527,547	263,774
4740	298,663	646,763	0.274	3,544,261	78.60	2,785,789	159,479	344,680	0.104	718,944	359,472
4720	362,939	785,330	0.342	5,371,657	78.60	4,222,123	177,543	383,654	0.107	823,958	411,979
4700	365,144	789,785	0.312	4,928,258	78.50	3,868,683	196,838	425,515	0.104	880,876	440,438
4680	486,607	1,052,693	0.308	6,484,589	78.40	5,083,918	135,241	292,786	0.101	592,631	296,315
4660	540,111	1,167,877	0.298	6,960,547	78.40	5,457,069	81,629	176,656	0.108	380,603	190,302
4640	624,295	1,350,690	0.308	8,320,250	78.30	6,514,756	81,963	177,274	0.118	419,704	209,852
4620	623,684	1,348,748	0.317	8,551,062	78.30	6,695,482	86,685	187,443	0.112	420,293	210,147
4600	664,721	1,439,922	0.340	9,791,470	78.30	7,666,721	44,314	95,709	0.115	220,845	110,422
4580	647,499	1,402,366	0.364	10,209,224	78.40	8,004,032	45,148	97,423	0.133	259,303	129,651
4560	661,165	1,432,696	0.387	11,089,067	78.30	8,682,739	40,074	86,745	0.107	185,185	92,593
4540	660,350	1,430,836	0.428	12,247,956	78.30	9,590,150	26,926	58,261	0.121	141,331	70,665
4520	646,906	1,403,905	0.452	12,691,301	78.30	9,937,289	35,778	78,433	0.112	175,393	87,696
4500	628,443	1,364,107	0.470	12,822,606	78.40	10,052,923	35,518	77,656	0.117	182,096	91,048
4480	623,295	1,353,264	0.486	13,153,726	78.40	10,312,521	14,741	32,338	0.100	64,917	32,459
4460	619,350	1,348,176	0.478	12,888,563	78.10	10,065,967	6,019	13,193	0.065	17,216	8,608
4440	604,165	1,315,044	0.489	12,861,130	78.10	10,044,543	11,204	24,787	0.111	54,841	27,420
4420	576,258	1,257,183	0.470	11,817,520	77.90	9,205,848	3,093	6,658	0.280	37,308	18,654
4400	557,943	1,214,546	0.483	11,732,514	77.70	9,116,164	3,648	8,071	0.180	29,061	14,531
4380	461,277	1,006,724	0.458	9,221,592	77.70	7,165,177	7,778	17,227	0.101	34,925	17,463
4360	393,333	858,580	0.469	8,053,480	77.80	6,265,608	3,518	7,719	0.132	20,426	10,213
TOTAL	14,270,051	30,952,798	0.377	233,460,739	78.40	182,865,812	2,321,036	5,023,331	0.105	10,516,736	5,258,368

Table 21-19
Copper Chief Starter Pit
Proven and Probable Reserves by Bench

BENCH TOE	PROVEN						PROBABLE				
	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERY BENCH AVG %	RECOVERABLE COPPER Lbs	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs
5140	-	-	-	-	79.0	-	90	195	0.652	2,543	2,009
5120	-	-	-	-	79.0	-	50	108	0.630	1,361	1,075
5100	-	-	-	-	79.0	-	-	-		-	-
5080	1,286	2,787	0.594	33,110	79.0	26,157	462	1,001	0.371	7,422	5,863
5060	8,740	18,939	0.296	112,119	79.0	88,574	6,204	13,443	0.108	29,067	22,963
5040	54,600	118,315	0.231	546,615	79.0	431,826	41,685	90,329	0.132	237,886	187,930
5020	187,467	406,228	0.237	1,925,521	79.0	1,521,161	83,422	180,770	0.169	610,311	482,145
5000	294,796	638,804	0.265	3,385,661	79.0	2,674,672	116,444	252,328	0.184	927,418	732,660
4980	321,629	696,950	0.285	3,972,615	79.0	3,138,366	118,944	257,745	0.196	1,010,893	798,605
4960	288,814	625,843	0.311	3,892,743	79.0	3,075,267	117,889	255,457	0.211	1,077,789	851,453
4940	264,555	573,275	0.308	3,531,374	79.0	2,789,785	99,907	216,493	0.217	938,713	741,583
4920	235,444	510,145	0.314	3,203,711	79.0	2,530,931	84,944	184,069	0.235	864,383	682,863
4900	191,407	414,761	0.326	2,704,242	79.0	2,136,351	73,685	159,671	0.250	799,793	631,837
4880	153,592	332,825	0.345	2,296,493	79.0	1,814,229	64,815	140,450	0.308	864,985	683,338
4860	103,833	225,000	0.323	1,453,500	79.0	1,148,265	63,778	138,202	0.391	1,081,650	854,503
4840	59,778	129,535	0.251	650,266	79.0	513,710	58,703	127,207	0.390	992,883	784,378
4820	25,148	54,494	0.224	244,133	79.0	192,865	55,482	120,225	0.288	692,361	546,965
4800	537	1,164	0.450	10,476	79.0	8,276	296	642	0.563	7,223	5,706
TOTAL	2,191,626	4,749,065	0.294	27,962,578	79.0	22,090,436	986,800	2,138,335	0.237	10,146,679	8,015,876

Table 21-20
Copper Chief Starter Pit
Proven plus Probable Reserves by Bench

PROVEN AND PROBABLE							
BENCH TOE	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs	WASTE TOTAL (TONS)	ROM S/R
5140	90	195	0.652	2,543	2,009	181	0.93
5120	50	108	0.630	1,361	1,075	1,122	10.39
5100	-	-	-	-	-	1,899	
5080	1,748	3,788	0.535	40,532	32,020	5,734	1.51
5060	14,944	32,382	0.218	141,186	111,537	25,715	0.79
5040	96,285	208,644	0.188	784,501	619,756	83,974	0.40
5020	270,889	586,998	0.216	2,535,831	2,003,307	227,677	0.39
5000	411,240	891,132	0.242	4,313,079	3,407,332	257,373	0.29
4980	440,573	954,695	0.261	4,983,508	3,936,971	239,542	0.25
4960	406,703	881,300	0.282	4,970,532	3,926,720	222,601	0.25
4940	364,462	789,768	0.283	4,470,087	3,531,369	190,484	0.24
4920	320,388	694,214	0.293	4,068,094	3,213,794	167,543	0.24
4900	265,092	574,432	0.305	3,504,035	2,768,188	171,936	0.30
4880	218,407	473,275	0.334	3,161,477	2,497,567	160,477	0.34
4860	167,611	363,202	0.349	2,535,150	2,002,768	159,416	0.44
4840	118,481	256,742	0.320	1,643,149	1,298,088	154,707	0.60
4820	80,630	174,719	0.268	936,494	739,830	121,063	0.69
4800	833	1,806	0.490	17,699	13,982	5,868	3.25
TOTAL	3,178,426	6,887,400	0.277	38,109,257	30,106,313	########	0.32

Table 21-21
Copper Chief Starter Pit
Inferred resources within Pit by Bench

	POSSIBLE				
BENCH TOE	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs
5140	83	181	0.424	1,535	1,213
5120	241	515	0.110	1,133	895
5100	444	952	0.120	2,285	1,805
5080	1,875	4,015	0.118	9,475	7,486
5060	8,365	18,021	0.110	39,646	31,320
5040	25,668	55,460	0.123	136,432	107,781
5020	74,330	160,880	0.154	495,510	391,453
5000	77,781	168,290	0.163	548,625	433,414
4980	76,352	165,071	0.167	551,337	435,556
4960	83,389	179,812	0.161	578,995	457,406
4940	69,444	149,500	0.169	505,310	399,195
4920	62,778	134,942	0.172	464,200	366,718
4900	66,963	143,815	0.172	494,724	390,832
4880	65,944	141,564	0.173	489,811	386,951
4860	67,574	144,983	0.166	481,344	380,261
4840	71,463	153,240	0.156	478,109	377,706
4820	55,055	118,048	0.177	417,890	330,133
4800	41,426	88,699	0.135	239,487	189,195
TOTAL	849,175	1,827,988	0.162	5,935,849	4,689,320

Table 21-22
Copper Chief Starter Pit
High Grade and Low Grade Reserves by Bench

BENCH TOE	HIGH GRADE (greater than 0.15%) INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERY BENCH AVG %	RECOVERABLE COPPER Lbs	LOW GRADE (greater than 0.065% and less than 0.15%) INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs @ 0.5
5140	90	195	0.652	2,543	79.0	2,009	-	-	-	-	-
5120	50	108	0.630	1,361	79.0	1,075	-	-	-	-	-
5100	-	-	-	-	79.0	-	-	-	-	-	-
5080	1,748	3,788	0.535	40,532	79.0	32,020	-	-	-	-	-
5060	4,544	9,846	0.473	93,143	79.0	73,583	10,400	22,536	0.107	48,042	24,021
5040	42,972	93,118	0.283	527,048	79.0	416,368	53,313	115,526	0.111	257,454	128,727
5020	192,481	417,095	0.256	2,135,526	79.0	1,687,066	78,408	169,903	0.118	400,305	200,152
5000	352,907	764,727	0.262	4,007,169	79.0	3,165,664	58,333	126,405	0.121	305,909	152,955
4980	401,536	870,104	0.274	4,768,170	79.0	3,766,854	39,037	84,591	0.127	215,338	107,669
4960	375,055	812,721	0.296	4,811,308	79.0	3,800,934	31,648	68,579	0.116	159,224	79,612
4940	344,111	745,666	0.292	4,354,689	79.0	3,440,205	20,351	44,102	0.131	115,397	57,699
4920	297,870	645,466	0.307	3,963,161	79.0	3,130,897	22,518	48,748	0.108	104,933	52,466
4900	249,314	540,242	0.317	3,425,134	79.0	2,705,856	15,778	34,190	0.115	78,901	39,450
4880	209,481	453,933	0.342	3,104,902	79.0	2,452,872	8,926	19,342	0.146	56,575	28,288
4860	162,704	352,568	0.356	2,510,284	79.0	1,983,124	4,907	10,634	0.117	24,866	12,433
4840	111,037	240,610	0.334	1,607,275	79.0	1,269,747	7,444	16,132	0.111	35,874	17,937
4820	68,704	148,876	0.295	878,368	79.0	693,911	11,926	25,843	0.112	58,125	29,063
4800	833	1,806	0.490	17,699	79.0	13,982	-	-	-	-	-
TOTAL	2,815,437	6,100,869	0.297	36,248,313	79.0	28,636,167	362,989	786,531	0.118	1,860,944	930,472

Table 21-23
Copper Chief Final Pit
Proven and Probable Reserves by Bench

	PROVEN						PROBABLE				
BENCH TOE	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERY BENCH AVG %	RECOVERABLE COPPER Lbs	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs
5280	89	205	0.210	861	74.0	637	333	767	0	3,377	2,499
5260	7,396	17,025	0.213	72,527	74.0	53,670	2,500	5,754	(0)	22,234	16,453
5240	27,927	63,769	0.209	266,554	74.2	197,783	1,389	3,125	0.102	6,373	4,729
5220	62,591	142,726	0.193	550,922	74.3	409,335	-	-	-	-	-
5200	94,991	215,711	0.215	927,557	74.6	691,958	6,389	14,471	0.135	39,207	29,248
5180	140,107	318,115	0.251	1,596,937	74.6	1,191,315	12,685	29,127	0.287	167,052	124,621
5160	196,471	446,035	0.275	2,453,193	74.5	1,827,628	10,463	23,988	0.314	150,735	112,298
5140	281,400	639,962	0.290	3,711,780	74.7	2,772,699	8,354	19,080	0.459	175,176	130,856
5120	333,888	759,494	0.292	4,435,445	74.7	3,313,277	12,297	28,112	0.235	131,935	98,556
5100	369,088	839,431	0.297	4,986,220	74.7	3,724,706	16,582	37,936	0.204	155,150	115,897
5080	387,435	881,154	0.311	5,482,682	74.8	4,101,046	11,418	25,882	0.204	105,678	79,047
5060	413,435	938,200	0.302	5,669,001	74.9	4,246,082	21,610	48,735	0.167	163,011	122,096
5040	407,148	923,223	0.296	5,473,474	75.0	4,105,106	20,880	47,104	0.215	202,785	152,089
5020	399,902	906,057	0.288	5,213,310	75.1	3,915,196	26,596	59,958	0.246	295,466	221,895
5000	375,777	851,771	0.284	4,842,313	75.0	3,631,735	21,500	48,479	0.239	231,450	173,588
4980	368,777	836,409	0.280	4,675,530	74.9	3,501,972	19,760	45,025	0.284	256,059	191,788
4960	363,314	824,368	0.286	4,721,510	74.9	3,536,411	19,407	44,103	0.240	211,662	158,535
4940	339,740	772,187	0.301	4,649,035	75.0	3,486,776	36,259	81,960	0.173	284,072	213,054
4920	338,221	768,732	0.306	4,699,749	75.1	3,529,512	26,408	59,500	0.206	245,037	184,023
4900	330,499	749,986	0.306	4,587,075	75.2	3,449,480	33,482	75,542	0.241	364,649	274,216
4880	325,629	738,050	0.309	4,557,108	75.2	3,426,945	30,277	68,446	0.289	395,164	297,163
4860	318,462	722,084	0.294	4,247,946	75.3	3,198,703	43,611	97,962	0.355	695,032	523,359
4840	291,684	663,112	0.313	4,153,175	75.2	3,123,188	50,927	115,757	0.475	1,100,845	827,836
4820	252,074	573,515	0.354	4,064,009	75.1	3,052,070	36,184	83,284	0.612	1,019,688	765,786
4800	260,592	592,315	0.372	4,405,008	75.4	3,321,376	75,019	167,141	0.358	1,195,407	901,337
4780	241,444	549,371	0.363	3,988,433	75.1	2,995,314	53,481	120,415	0.319	767,047	576,052
4760	225,000	511,394	0.325	3,324,061	75.0	2,493,046	64,351	146,011	0.370	1,080,553	810,414
4740	230,277	522,664	0.294	3,073,264	75.0	2,304,948	59,482	135,482	0.362	980,915	735,686
4720	205,833	466,895	0.287	2,679,977	75.0	2,009,983	79,759	181,622	0.369	1,340,828	1,005,621
4700	183,778	416,467	0.307	2,557,107	75.1	1,920,388	84,444	191,844	0.275	1,056,260	793,251
4680	170,667	386,606	0.327	2,528,403	75.2	1,901,359	78,870	178,792	0.248	886,601	666,724
4660	143,167	324,668	0.337	2,188,262	75.2	1,645,573	89,500	202,345	0.256	1,037,057	779,867
4640	129,518	293,506	0.310	1,819,737	75.3	1,370,262	78,704	177,903	0.286	1,018,145	766,663
4620	98,278	222,350	0.272	1,209,584	75.3	910,817	79,444	180,270	0.355	1,278,608	962,792
4600	81,389	183,780	0.220	808,632	75.3	608,900	68,926	156,798	0.402	1,262,082	950,348
4580	63,315	142,985	0.195	557,642	75.4	420,462	57,185	129,874	0.338	877,597	661,708
4560	39,389	88,997	0.175	311,490	75.5	235,175	56,759	128,409	0.265	679,882	513,311
TOTAL	8,498,692	19,293,319	0.299	115,489,512	75.0	86,624,832	1,395,235	3,161,003	0.315	19,882,820	14,943,405

Table 21-24
Copper Chief Final Pit
Proven plus Probable Reserves by Bench

PROVEN AND PROBABLE							
BENCH TOE	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs	WASTE TOTAL (TONS)	ROM S/R
5280	422	972	0.218	4,238	3,136	5,507	5.67
5260	9,896	22,779	0.208	94,761	70,123	16,891	0.74
5240	29,316	66,894	0.204	272,928	202,512	72,736	1.09
5220	62,591	142,726	0.193	550,922	409,335	177,397	1.24
5200	101,380	230,182	0.210	966,764	721,206	376,396	1.64
5180	152,792	347,242	0.254	1,763,989	1,315,936	563,807	1.62
5160	206,934	470,023	0.277	2,603,927	1,939,926	742,328	1.58
5140	289,754	659,042	0.295	3,886,956	2,903,556	922,496	1.40
5120	346,185	787,606	0.290	4,567,380	3,411,833	1,093,910	1.39
5100	385,670	877,367	0.293	5,141,371	3,840,604	1,179,974	1.34
5080	398,853	907,036	0.308	5,588,361	4,180,094	1,223,719	1.34
5060	435,045	986,935	0.295	5,832,012	4,368,177	1,198,096	1.18
5040	428,028	970,327	0.292	5,676,260	4,257,195	1,230,540	1.04
5020	426,498	966,015	0.285	5,508,776	4,137,091	1,308,896	0.84
5000	397,277	900,250	0.282	5,073,763	3,805,322	1,296,586	0.72
4980	388,537	881,434	0.280	4,931,589	3,693,760	1,228,910	0.67
4960	382,721	868,471	0.284	4,933,172	3,694,946	1,149,739	0.66
4940	375,999	854,147	0.289	4,933,107	3,699,830	1,063,105	0.65
4920	364,629	828,232	0.299	4,944,786	3,713,534	1,003,201	0.66
4900	363,981	825,528	0.300	4,951,723	3,723,696	969,750	0.69
4880	355,906	806,496	0.307	4,952,271	3,724,108	938,181	0.73
4860	362,073	820,046	0.301	4,942,977	3,722,062	881,214	0.74
4840	342,611	778,869	0.337	5,254,020	3,951,023	871,595	0.84
4820	288,258	656,799	0.387	5,083,696	3,817,856	780,576	0.94
4800	335,611	759,456	0.369	5,600,415	4,222,713	726,949	0.95
4780	294,925	669,786	0.355	4,755,481	3,571,366	685,536	1.02
4760	289,351	657,405	0.335	4,404,614	3,303,460	587,969	0.89
4740	289,759	658,146	0.308	4,054,179	3,040,635	383,482	0.58
4720	285,592	648,517	0.310	4,020,805	3,015,604	297,770	0.46
4700	268,222	608,311	0.297	3,613,367	2,713,639	247,842	0.41
4680	249,537	565,398	0.302	3,415,004	2,568,083	200,033	0.35
4660	232,667	527,013	0.306	3,225,320	2,425,440	153,947	0.29
4640	208,222	471,409	0.301	2,837,882	2,136,925	124,912	0.26
4620	177,722	402,620	0.309	2,488,192	1,873,608	107,622	0.27
4600	150,315	340,578	0.304	2,070,714	1,559,248	86,744	0.25
4580	120,500	272,859	0.263	1,435,238	1,082,170	70,674	0.26
4560	96,148	217,406	0.228	991,371	748,485	52,173	0.24
TOTAL	9,893,927	22,454,322	0.301	135,372,332	101,568,237	24,021,203	1.07

Table 21-25
Copper Chief Final Pit
Inferred resources within Pit by Bench

	POSSIBLE				
BENCH TOE	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs
5280	2,530	5,507	0.130	14,318	10,595
5260	5,156	11,039	0.109	24,065	17,808
5240	24,122	51,649	0.103	106,397	78,947
5220	61,155	130,942	0.108	282,835	210,146
5200	117,694	252,002	0.107	539,284	402,306
5180	170,352	364,810	0.108	787,990	587,840
5160	214,759	459,833	0.113	1,039,223	774,221
5140	234,269	501,640	0.121	1,212,872	906,015
5120	238,953	511,652	0.123	1,258,798	940,322
5100	280,028	599,824	0.125	1,499,655	1,120,242
5080	313,564	671,630	0.131	1,760,715	1,317,014
5060	355,325	761,026	0.134	2,032,619	1,522,431
5040	360,198	771,361	0.139	2,145,594	1,609,196
5020	338,648	725,031	0.133	1,931,886	1,450,846
5000	314,055	672,351	0.127	1,704,292	1,278,219
4980	303,944	650,667	0.126	1,634,841	1,224,496
4960	278,147	595,443	0.118	1,405,658	1,052,838
4940	268,037	573,881	0.117	1,346,545	1,009,909
4920	262,314	561,800	0.125	1,403,068	1,053,704
4900	269,407	576,931	0.122	1,408,046	1,058,850
4880	277,407	593,996	0.120	1,422,645	1,069,829
4860	265,703	568,918	0.126	1,431,911	1,078,229
4840	268,555	575,071	0.138	1,590,294	1,195,901
4820	273,648	586,163	0.142	1,666,575	1,251,598
4800	274,333	587,743	0.172	2,019,829	1,522,951
4780	292,796	627,251	0.164	2,057,383	1,545,095
4760	246,907	529,262	0.160	1,693,638	1,270,229
4740	159,778	342,336	0.132	903,767	677,825
4720	121,278	259,687	0.141	732,317	549,238
4700	96,278	206,634	0.158	652,963	490,376
4680	82,630	177,908	0.181	644,027	484,308
4660	61,352	132,447	0.181	479,458	360,553
4640	55,222	120,120	0.212	509,309	383,510
4620	48,556	106,539	0.260	554,003	417,164
4600	39,167	86,744	0.313	543,017	408,892
4580	30,685	69,008	0.388	535,502	403,769
4560	20,889	47,423	0.451	427,755	322,955
TOTAL	7,027,841	15,066,269	0.137	41,403,095	31,058,369

Table 21-26
Copper Chief Final Pit
High Grade and Low Grade Reserves by Bench

BENCH	HIGH GRADE (greater than 0.15%)						LOW GRADE (greater than 0.065% and less than 0.15%)				
TOE	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERY BENCH AVG %	RECOVERABLE COPPER Lbs	INSITU ORE (YDS)	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs @ 50%
5280	422	972	0.218	4,238	74.0	3,136	-	-	-	-	-
5260	9,051	20,833	0.215	89,582	74.0	66,291	845	1,946	0.133	5,179	2,589
5240	24,482	56,080	0.220	246,752	74.2	183,090	4,834	10,814	0.121	26,176	13,088
5220	47,352	108,527	0.216	468,837	74.3	348,346	15,239	34,199	0.120	82,086	41,043
5200	70,491	160,865	0.252	810,760	74.6	604,827	30,889	69,317	0.113	156,005	78,002
5180	120,221	274,568	0.292	1,603,477	74.6	1,196,194	32,571	72,674	0.110	160,512	80,256
5160	161,987	369,933	0.322	2,382,369	74.5	1,774,865	44,947	100,090	0.111	221,559	110,779
5140	231,317	527,202	0.342	3,604,853	74.7	2,692,825	58,437	131,840	0.107	282,103	141,051
5120	273,042	623,042	0.339	4,223,596	74.7	3,155,026	73,143	164,564	0.104	343,784	171,892
5100	307,685	701,081	0.341	4,781,372	74.7	3,571,685	77,985	176,286	0.102	359,998	179,999
5080	330,631	753,456	0.349	5,260,176	74.8	3,934,612	68,222	153,580	0.107	328,184	164,092
5060	332,038	756,050	0.353	5,344,718	74.9	4,003,194	103,007	230,885	0.106	487,294	243,647
5040	317,701	722,690	0.359	5,183,608	75.0	3,887,706	110,327	247,637	0.099	492,652	246,326
5020	327,981	745,199	0.339	5,047,451	75.1	3,790,635	98,517	220,816	0.104	461,325	230,663
5000	309,092	702,888	0.333	4,681,111	75.0	3,510,833	88,185	197,362	0.099	392,651	196,326
4980	299,000	680,890	0.331	4,506,774	74.9	3,375,574	89,537	200,544	0.106	424,814	212,407
4960	276,369	629,930	0.349	4,392,805	74.9	3,290,211	106,352	238,541	0.113	540,367	270,183
4940	288,054	656,366	0.345	4,534,193	75.0	3,400,645	87,945	197,781	0.101	398,913	199,457
4920	289,314	659,004	0.349	4,594,162	75.1	3,450,216	75,315	169,228	0.104	350,624	175,312
4900	298,685	679,104	0.342	4,646,936	75.2	3,494,496	65,296	146,424	0.104	304,787	152,393
4880	304,684	692,160	0.342	4,734,374	75.2	3,560,250	51,222	114,336	0.095	217,897	108,948
4860	302,628	687,458	0.339	4,665,895	75.3	3,513,419	59,445	132,588	0.104	277,082	138,541
4840	293,962	669,326	0.375	5,017,059	75.2	3,772,829	48,649	109,543	0.108	236,961	118,481
4820	267,147	608,419	0.411	4,998,241	75.1	3,753,679	21,111	48,380	0.088	85,456	42,728
4800	303,074	686,178	0.397	5,444,894	75.4	4,105,450	32,537	73,278	0.106	155,521	77,760
4780	270,370	614,440	0.377	4,632,878	75.1	3,479,291	24,555	55,346	0.111	122,603	61,302
4760	259,129	589,405	0.361	4,255,504	75.0	3,191,628	30,222	68,000	0.110	149,109	74,555
4740	245,407	558,259	0.344	3,840,822	75.0	2,880,616	44,352	99,887	0.107	213,357	106,679
4720	231,092	525,892	0.357	3,754,869	75.0	2,816,152	54,500	122,625	0.108	265,937	132,968
4700	211,111	479,811	0.347	3,329,888	75.1	2,500,746	57,111	128,500	0.110	283,479	141,740
4680	184,667	419,440	0.369	3,095,467	75.2	2,327,791	64,870	145,958	0.109	319,537	159,768
4660	174,148	395,347	0.370	2,925,568	75.2	2,200,027	58,519	131,666	0.114	299,752	149,876
4640	160,037	362,992	0.359	2,606,283	75.3	1,962,531	48,185	108,417	0.107	231,600	115,800
4620	145,685	330,537	0.351	2,320,370	75.3	1,747,238	32,037	72,083	0.116	167,822	83,911
4600	128,815	292,203	0.334	1,951,916	75.3	1,469,793	21,500	48,375	0.123	118,798	59,399
4580	109,778	248,734	0.276	1,373,012	75.4	1,035,251	10,722	24,125	0.129	62,227	31,113
4560	82,111	185,822	0.249	925,394	75.5	698,672	14,037	31,584	0.104	65,978	32,989
TOTAL	7,988,760	18,175,103	0.347	126,280,204	75.0	94,749,769	1,905,167	4,279,219	0.106	9,092,128	4,546,064

21.3 PRODUCTION PARAMETERS

21.3.1 Selective Mining

The mining activities at the Johnson Camp Mine Project will require the appropriate separation of ore and waste. As mineralization is generally limited to within defined geologic boundaries for mining purposes, the ore body can be described as mineralized zones within a structurally controlled host rock. Additionally, the rock types hosting the mineralization are visually distinguishable from the rock types of the Hanging Wall (HW) and Foot Wall (FW). These physical attributes and visual characteristics will facilitate separation/segregation of the ore from the waste.

The Drilling Operation will also be used to confirm and define the actual zones of high grade mineralization, low grade mineralization and waste within the Johnson Camp mine. Drill cuttings will be subjected to assay to define extractable copper. These assay results would be utilized to confirm the ore block modeling and would be plotted to provide extraction controls for the loading equipment.

Loading equipment selection is generally predicated on the physical characteristics of the orebody, to be extracted. The selected loading unit must minimize ore dilution while achieving operating efficiency. The inherent physical characteristics of the zones of mineralization at Johnson Camp Mine Project indicate that loading units similar to a Caterpillar 992 front-end loader will be used for ore and waste loading. These loading units will allow for appropriate ore, low grade ore and waste separation.

21.3.2 Production Schedule

Production of ore and waste will begin in the first year of operation at the Johnson Camp Mine Project. The mining schedule is based upon all proven and probable reserves located within a pit defined using only measured and indicated resources. The schedule assumes a production of 25 million pounds of copper produced per year throughout mine life. The production schedule is detailed in Table 21-27.

Several mine sequencing alternatives were investigated by BETA in order to identify the sequence that allows full-capacity copper production as rapidly as possible with minimum front-end total material movement, as well as sequences that exploit the higher-grade Burro pit first. BETA notes that the schedule that is presented in this study is a feasible alternative; BETA has not performed an optimization for scheduling of this project. Scheduling was based on producing a minimum 25 million pounds of recoverable copper per year, using the average calculated copper recovery for all rock types by bench.

BETA has scheduled the low grade ore, defined as mine blocks that grade between 0.065 and 0.15% copper, to be direct dumped onto the leach pads. The total recovery for all run of mine leach ore is assumed to be 50% based on the operating experience of Cyprus and Arimetco. High grade ore, which is sent to the crushing plant and stacked via conveyor, is scheduled with appropriate recovery by rock types contained in each bench.

The schedule assumes the Burro Starter pit is mined first, followed by the Copper Chief Starter pit. The remainder of the Burro pit is mined third, and finally the Copper Chief ultimate pit. Each of the pits scheduled (Burro Starter, Chief Starter, Burro Final, and Copper Chief Final is scheduled to be mined to completion prior to beginning the next phase pit. The resulting unsmoothed production schedule contains significant fluctuations in total material movement. BETA recommends that Nord discuss the schedule with the mining contractor to ensure the production schedule optimizes the use of the mining fleet.

Table 21-27
Production Schedule

	ORE TO CRUSHING PLANT				LOW GRADE ORE RUN OF MINE TO PADS				WASTE	STRIP	Total
	TONS	TCU	LBS CU	LBS CU	TONS	TCU	LBS CU	LBS CU	TONS	RATIO	Tons
	(000s)		Contained	Recoverable	(000s)		Contained	Recoverable	(000s)		(000s)
YEAR 1	3,331	0.427	28,435	22,154	574	0.099	1,142	571	472	0.12	4,377
YEAR 2	2,590	0.564	29,209	22,369	29	0.107	63	31	199	0.08	2,818
YEAR 3	3,956	0.366	28,949	22,707	590	0.117	1,386	693	1,170	0.26	5,717
YEAR 4	4,628	0.318	29,439	23,041	531	0.068	718	359	7,162	1.39	12,320
YEAR 5	4,524	0.301	27,220	21,594	1,834	0.098	3,611	1,806	7,911	1.24	14,269
YEAR 6	4,788	0.291	27,847	21,628	1,712	0.104	3,544	1,772	5,174	0.80	11,674
YEAR 7	4,871	0.316	30,770	24,279	637	0.113	1,441	721	1,888	0.34	7,396
YEAR 8	4,100	0.383	31,366	24,707	242	0.121	586	293	651	0.15	4,993
YEAR 9	3,396	0.457	31,009	24,789	188	0.112	422	211	277	0.08	3,862
YEAR 10	3,376	0.481	32,491	24,964	38	0.095	72	36	83	0.02	3,497
YEAR 11	3,355	0.477	32,025	24,949	32	0.158	101	51	164	0.05	3,550
YEAR 12	4,319	0.374	32,275	24,181	769	0.106	1,637	819	5,201	1.02	10,289
YEAR 13	4,411	0.353	31,144	23,436	1,489	0.105	3,127	1,564	8,636	1.46	14,537
YEAR 14	4,682	0.341	31,967	24,064	918	0.102	1,872	936	6,508	1.16	12,108
YEAR 15	4,360	0.372	32,443	24,245	694	0.109	1,510	755	3,130	0.62	8,183
YEAR 16	2,007	0.330	13,256	9,944	416	0.114	946	473	596	0.25	3,019

Table 21-28
Production Schedule By Pit
Years 1-8

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Mine Production								
Burro Starter Pit								
Total Burro Starter pit ore mined, tons (000)	3,905,459	2,619,307	1,545,835					
Total Burro Starter pit grade, %TCu	0.379	0.559	0.508					
Waste mined, tons(000)	471,866	199,190	104,471					
Total material mined, tons(000)	4,377,325	2,818,497	1,650,306					
Strip Ratio, w aste / ore	0.12	0.08	0.07					
Burro Final Pit								
Total Burro Final pit ore mined, tons (000)				1,734,546	6,357,896	6,500,220	5,507,749	4,341,987
Total Burro Final pit grade, %TCu				0.272	0.242	0.241	0.292	0.368
Waste mined, tons(000)				6,030,035	7,911,234	5,173,508	1,887,920	651,359
Total material mined, tons(000)				7,764,581	14,269,130	11,673,728	7,395,669	4,993,346
Strip Ratio, w aste / ore				3.48	1.24	0.80	0.34	0.15
Copper Chief Starter Pit								
Total Copper Chief Starter ore mined, Tons (000)			3,000,536	3,424,215				
Total Copper Chief Starter grade, %TCu			0.244	0.302				
Waste mined, tons(000)			1,065,818	1,131,494				
Total material mined, tons(000)			4,066,354	4,555,709				
Strip Ratio, w aste / ore			0.36	0.33				
Copper Chief Final Pit								
Total Copper Chief Final pit ore mined,tons (000)								
Total Copper Chief Final pit grade, %TCu								
Waste mined, tons(000)								
Total material mined, tons(000)								
Strip Ratio, w aste / ore								
Total Material Mined								
Total ore mined, tons (000)	3,905,459	2,619,307	4,546,371	5,158,761	6,357,896	6,500,220	5,507,749	4,341,987
Total ore grade, %TCu	0.379	0.559	0.334	0.292	0.242	0.241	0.292	0.368
Total w aste mined, tons(000)	471,866	199,190	1,170,289	7,161,529	7,911,234	5,173,508	1,887,920	651,359
Total material mined, tons(000)	4,377,325	2,818,497	5,716,660	12,320,290	14,269,130	11,673,728	7,395,669	4,993,346
Strip Ratio, w aste / ore	0.12	0.08	0.26	1.39	1.24	0.80	0.34	0.15

Table 21-29
Production Schedule By Pit
Years 9-16

	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Mine Production								
Burro Starter Pit								
Total Burro Starter pit ore mined, tons (000)								
Total Burro Starter pit grade, %TCu								
Waste mined, tons(000)								
Total material mined, tons(000)								
Strip Ratio, w aste / ore								
Burro Final Pit								
Total Burro Final pit ore mined, tons (000)	3,584,803	3,414,300	3,386,589	1,610,689				
Total Burro Final pit grade, %TCu	0.438	0.477	0.474	0.460				
Waste mined, tons(000)	277,221	83,085	163,701	49,276				
Total material mined, tons(000)	3,862,024	3,497,385	3,550,290	1,659,965				
Strip Ratio, w aste / ore	0.08	0.02	0.05	0.03				
Copper Chief Starter Pit								
Total Copper Chief Starter ore mined, Tons (000)								
Total Copper Chief Starter grade, %TCu								
Waste mined, tons(000)								
Total material mined, tons(000)								
Strip Ratio, w aste / ore								
Copper Chief Final Pit								
Total Copper Chief Final pit ore mined,tons (000)				3,477,821	5,900,284	5,600,160	5,053,283	2,422,774
Total Copper Chief Final pit grade, %TCu				0.274	0.290	0.302	0.336	0.293
Waste mined, tons(000)				5,151,442	8,636,486	6,507,622	3,129,581	596,072
Total material mined, tons(000)				8,629,263	14,536,770	12,107,782	8,182,864	3,018,846
Strip Ratio, w aste / ore				1.48	1.46	1.16	0.62	0.25
Total Material Mined								
Total ore mined, tons (000)	3,584,803	3,414,300	3,386,589	5,088,510	5,900,284	5,600,160	5,053,283	2,422,774
Total ore grade, %TCu	0.438	0.477	0.474	0.333	0.290	0.302	0.336	0.293
Total w aste mined, tons(000)	277,221	83,085	163,701	5,200,718	8,636,486	6,507,622	3,129,581	596,072
Total material mined, tons(000)	3,862,024	3,497,385	3,550,290	10,289,228	14,536,770	12,107,782	8,182,864	3,018,846
Strip Ratio, w aste / ore	0.08	0.02	0.05	1.02	1.46	1.16	0.62	0.25

Table 21-30
Detail Production Schedule By Bench

YEAR	PIT	BENCH	COPPER LBS CONTAINED	COPPER LBS RECOVERED	ORE TONS	WASTE	STRIP RATIO	TONS MOVED
		4800	66,414	49,146	13,390	1,223	0.09	14,613
		4780	592,443	433,656	129,768	7,169	0.06	136,937
		4760	2,200,963	1,645,947	430,989	28,700	0.07	459,689
		4740	4,420,842	3,326,852	789,034	81,713	0.10	870,747
1	Burro	4720	6,894,736	5,186,120	1,212,646	128,560	0.11	1,341,206
	Starter	4700	10,137,100	7,646,015	1,677,168	189,005	0.11	1,866,173
		4680	14,973,018	11,354,366	2,268,356	234,567	0.10	2,502,923
		4660	20,115,700	15,307,891	2,892,468	305,851	0.11	3,198,319
		4640	25,939,697	19,775,803	3,541,020	404,262	0.11	3,945,282
		4620	29,576,803	22,725,000	3,905,459	471,866	0.12	4,377,325
		4620	3,028,926	2,180,827	303,500			
2	Burro	4600	9,645,717	7,273,367	926,550	67,730	0.07	994,280
	Starter	4580	16,521,694	12,572,895	1,526,548	108,030	0.07	1,634,578
		4560	23,633,256	18,042,725	2,128,203	147,513	0.07	2,275,716
		4540	29,271,138	22,400,000	2,619,307	199,190	0.08	2,818,497
		4540	322,133	231,936	28,060			
		4520	4,808,194	3,677,230	459,412	47,412	0.10	506,824
3	Burro	4500	8,387,734	6,419,158	817,366	78,846	0.10	896,212
	Starter	4480	11,264,225	8,623,483	1,104,441	94,876	0.09	1,199,317
		4460	13,475,527	10,323,887	1,320,811	100,744	0.08	1,421,555
		4440	14,956,944	11,452,727	1,460,042	103,559	0.07	1,563,601
		4420	15,713,639	12,030,085	1,545,835	104,471	0.07	1,650,306
		5140	15,716,181	12,032,094	1,546,030	104,652	0.07	1,650,682
		5120	15,717,542	12,033,169	1,546,138	105,774	0.07	1,651,912
	Copper	5100	15,717,542	12,033,169	1,546,138	107,673	0.07	1,653,811
3	Chief	5080	15,758,074	12,065,189	1,549,926	113,407	0.07	1,663,333
	Starter	5060	15,899,259	12,162,793	1,582,308	139,122	0.09	1,721,430
		5040	16,683,761	12,707,888	1,790,952	223,096	0.12	2,014,048
		5020	19,219,592	14,595,106	2,377,950	450,773	0.19	2,828,723
		5000	23,532,671	17,913,725	3,269,082	708,146	0.22	3,977,228
		4980	28,516,179	21,788,248	4,223,777	947,688	0.22	5,171,465
		4960	30,335,609	23,400,000	4,546,371	1,170,289	0.26	5,716,660
		4960	3,151,102	2,268,793	558,706			
		4940	7,621,189	5,766,697	1,348,474	190,484	0.14	1,538,958
	Copper	4920	11,689,283	8,950,061	2,042,688	358,027	0.18	2,400,715
4	Chief	4900	15,193,318	11,695,367	2,617,120	529,963	0.20	3,147,083
	Starter	4880	18,354,795	14,176,527	3,090,395	690,440	0.22	3,780,835
		4860	20,889,945	16,172,084	3,453,597	849,856	0.25	4,303,453
		4840	22,533,094	17,459,769	3,710,339	1,004,563	0.27	4,714,902
		4820	23,469,588	18,182,742	3,885,058	1,125,626	0.29	5,010,684
		4800	23,487,286	18,196,724	3,886,864	1,131,494	0.29	5,018,358
		5060	23,487,286	18,196,724	3,886,864	1,270,277	0.33	5,157,141
		5040	23,487,286	18,196,724	3,886,864	1,758,422	0.45	5,645,286
		5020	23,520,314	18,222,090	3,890,248	2,283,452	0.59	6,173,700
4	Burro	5000	24,179,092	18,741,206	4,001,528	2,907,182	0.73	6,908,710
	Final	4980	25,675,655	19,917,847	4,266,876	3,703,722	0.87	7,970,598
		4960	27,301,936	21,175,011	4,580,830	4,799,838	1.05	9,380,668
		4940	29,608,201	22,932,209	5,042,083	5,949,635	1.18	10,991,718
		4920	30,156,586	23,400,000	5,158,761	7,161,529	1.39	12,320,290
		4920	2,174,452	1,565,606	462,649			
		4900	5,155,452	3,797,901	1,107,887	1,233,802	1.11	2,341,689
		4880	8,622,415	6,408,559	1,833,193	2,467,667	1.35	4,300,860
5	Burro	4860	12,607,280	9,452,866	2,630,166	3,679,615	1.40	6,309,781
	Final	4840	17,589,515	13,268,785	3,556,232	4,879,989	1.37	8,436,221
		4820	22,738,964	17,178,286	4,607,140	6,019,693	1.31	10,626,833
		4800	28,678,067	21,664,124	5,870,779	7,004,948	1.19	12,875,727
		4780	30,831,122	23,400,000	6,357,896	7,911,234	1.24	14,269,130
		4780	3,750,196	2,700,141	848,461			
		4760	8,076,424	5,953,477	1,784,874	1,295,204	0.73	3,080,078
6	Burro	4740	12,339,629	9,098,738	2,776,317	2,532,694	0.91	5,309,011
	Final	4720	18,535,244	13,732,839	3,945,301	3,534,447	0.90	7,479,748
		4700	24,344,378	18,041,960	5,160,601	4,451,309	0.86	9,611,910
		4680	31,390,774	23,400,000	6,500,220	5,173,508	0.80	11,673,728

Table 21-30
Detail Production Schedule By Bench (continued)

YEAR	PIT	BENCH	COPPER LBS CONTAINED	COPPER LBS RECOVERED	ORE TONS	WASTE	STRIP RATIO	TONS MOVED
		4680	30,824	22,193	5,860			
7	Burro	4660	7,371,974	5,669,563	1,350,393	695,891	0.52	2,046,284
	Final	4640	16,111,928	12,394,171	2,878,357	1,175,621	0.41	4,053,978
		4620	25,083,283	19,299,800	4,414,548	1,558,043	0.35	5,972,591
		4600	32,210,955	25,000,000	5,507,749	1,887,920	0.34	7,395,669
		4600	2,884,642	2,076,943	442,430			
8	Burro	4580	13,353,170	10,210,626	1,942,219	279,526	0.14	2,221,745
	Final	4560	24,627,422	18,985,958	3,461,660	474,711	0.14	3,936,371
		4540	31,951,746	25,000,000	4,341,987	651,359	0.15	4,993,346
		4540	5,064,963	3,646,773	608,770			
9	Burro	4520	17,931,656	13,671,758	2,091,108	121,083	0.06	2,212,191
	Final	4500	30,936,359	23,815,729	3,532,871	207,724	0.06	3,740,595
		4480	31,431,795	25,000,000	3,584,803	277,221	0.08	3,862,024
	Burro	4480	12,723,207	9,160,709	1,333,670			
10	Final	4460	25,628,985	19,235,284	2,695,039	49,929	0.02	2,744,968
		4440	32,562,668	25,000,000	3,414,300	83,085	0.02	3,497,385
		4440	5,982,288	4,307,247	620,569			
11	Burro	4420	17,837,117	13,531,750	1,884,410	52,738	0.03	1,937,148
	Final	4400	29,598,692	22,662,444	3,107,027	114,425	0.04	3,221,452
		4380	32,125,927	25,000,000	3,386,589	163,701	0.05	3,550,290
	Burro	4380	6,729,283	4,845,084	744,390			
12	Final	4360	14,803,189	11,120,904	1,610,689	49,276	0.03	1,659,965
		5280	14,807,427	11,124,040	1,611,661	54,783	0.03	1,666,444
		5260	14,902,188	11,192,920	1,634,440	71,674	0.04	1,706,114
		5240	15,175,115	11,389,098	1,701,334	144,410	0.08	1,845,744
12	Copper	5220	15,726,038	11,778,487	1,844,060	321,807	0.17	2,165,867
	Chief	5200	16,692,802	12,461,316	2,074,242	698,203	0.34	2,772,445
	Final	5180	18,456,792	13,737,766	2,421,484	1,262,010	0.52	3,683,494
		5160	21,060,719	15,623,410	2,891,507	2,004,338	0.69	4,895,845
		5140	24,947,674	18,457,286	3,550,549	2,926,834	0.82	6,477,383
		5120	29,515,055	21,784,205	4,338,155	4,020,744	0.93	8,358,899
		5100	33,912,135	25,000,000	5,088,510	5,200,718	1.02	10,289,228
		5100	744,290	535,889	127,012			
		5080	6,332,651	4,634,593	1,034,048	1,223,719	1.18	2,257,767
		5060	12,164,663	8,881,434	2,020,983	2,421,815	1.20	4,442,798
	Copper	5040	17,840,923	13,015,466	2,991,310	3,652,355	1.22	6,643,665
13	Chief	5020	23,349,699	17,036,764	3,957,325	4,961,251	1.25	8,918,576
	Final	5000	28,423,461	20,743,923	4,857,575	6,257,837	1.29	11,115,412
		4980	33,355,050	24,331,904	5,739,009	7,486,747	1.30	13,225,756
		4960	34,271,141	25,000,000	5,900,284	8,636,486	1.46	14,536,770
		4960	4,017,081	2,892,299	707,196			
		4940	8,950,188	6,492,400	1,561,343	1,063,105	0.68	2,624,448
		4920	13,894,975	10,117,928	2,389,575	2,066,306	0.86	4,455,881
	Copper	4900	18,846,698	13,764,818	3,215,103	3,036,056	0.94	6,251,159
14	Chief	4880	23,798,969	17,434,016	4,021,599	3,974,237	0.99	7,995,836
	Final	4860	28,741,946	21,085,976	4,841,645	4,855,451	1.00	9,697,096
		4840	33,995,967	24,977,285	5,620,514	5,727,046	1.02	11,347,560
		4820	33,838,425	25,000,000	5,600,160	6,507,622	1.16	12,107,782
		4820	5,241,238	3,773,692	677,153			
		4800	10,841,653	7,956,902	1,436,609	726,949	0.51	2,163,558
		4780	15,597,134	11,497,495	2,106,395	1,412,485	0.67	3,518,880
	Copper	4760	20,001,747	14,763,678	2,763,800	2,000,454	0.72	4,764,254
15	Chief	4740	24,055,927	17,750,973	3,421,946	2,383,936	0.70	5,805,882
	Final	4720	28,076,732	20,700,093	4,070,463	2,681,706	0.66	6,752,169
		4700	31,690,099	23,342,578	4,678,774	2,929,548	0.63	7,608,322
		4680	33,952,134	25,000,000	5,053,283	3,129,581	0.62	8,182,864
		4680	1,152,969	830,138	190,889			
		4660	4,378,289	3,180,041	717,902	153,947	0.21	871,849
	Copper	4640	7,216,171	5,258,371	1,189,311	278,859	0.23	1,468,170
16	Chief	4620	9,704,363	7,089,521	1,591,931	386,481	0.24	1,978,412
	Final	4600	11,775,077	8,618,713	1,932,509	473,225	0.24	2,405,734
		4580	13,210,315	9,685,077	2,205,368	543,899	0.25	2,749,267
		4560	14,201,687	10,416,738	2,422,774	596,072	0.25	3,018,846

21.4 INFERRED RESOURCE WITHIN THE ECONOMIC PIT

Proven and probable mineable reserves include only measured and indicated mineral resources. There are some inferred mineral resources with grades below economic cutoff in the designed pits.

In the Burro starter pit, 27% of the waste, or 213 ktons of the 775 ktons classified as waste, is inferred mineral resources with a grade of 0.541 % TCu. In the Burro final pit, 8% of the waste, or 1,674 ktons of the 21,756 ktons classified as waste is inferred mineral resources with a grade of 0.268 % TCu.

In the Copper Chief Starter pit, 83% of the waste, or 1,828 ktons of the 2,197 ktons classified as waste is inferred mineral resources with a grade of 0.162 % TCu. In the Copper Chief final pit, 63% of the waste, or 15,066 ktons of the 24,021 ktons classified as waste is inferred mineral resources with a grade of 0.162 % TCu.

Table 21-31

Inferred Resources within the Pit

	Possible (currently counted as waste)			
	INSITU ORE (TONS)	INSITU GRADE TCU	INSITU COPPER Lbs	RECOVERABLE COPPER Lbs @ 0.73
Burro Starter Pit	213,007	0.541	2,304,736	1,772,864
Chief Starter Pit	1,827,988	0.162	5,935,849	4,689,320
Burro Final Pit	1,673,793	0.268	8,986,850	6,560,401
Chief Final Pit	15,066,269	0.137	41,403,095	31,058,369
	18,781,057	0.156	58,630,530	44,080,954

22.0 OPERATING COST

INTRODUCTION

Operating and maintenance costs are estimated for a 25 million pound per year copper production operation.

Operating cost is presented for each of the following categories:

- general and mine administration;
- mining of ore and waste;
- crushing and stacking;
- processing (including assay laboratory); and
- environmental.

The operating cost is itemized into the following cost components:

- labor (supervisory, operating & maintenance);
- operating consumable supplies;
- maintenance supplies; and
- power generation.

Derivation of the individual components of these costs is detailed in the following sections.

SUMMARY

Operating costs per ton of ore for each of the cost categories is projected for the project. Table 22-1 summarizes the operating costs for the proposed production level.

Table 22-1
Operating Cost Summary

Production	Operating Costs					
	G & A	Ore	Waste	Crush & Stack	Process	Environment
(lb/year)	$US/ton Ore	$US/ton Ore	$US/ton Waste	$US/ton Ore	$US/lb Cu	$US/ton Ore
25,000,000	0.350	1.509	1.603	0.637	0.285	0.035

The basis for the estimated operating cost is as follows:

- costs are expressed in U.S. Dollars without escalation;
- consumption levels of major reagents are based on review of metallurgical testwork
- labor costs and social burdens are expected costs of skilled laborers, and meet current legal requirements;
- power costs are based on detailed estimates of power draw for all electrical installations and assume 100% grid-supplied power;
- where available and applicable, actual costs from current and nearby operations are utilized. Other sources for costs include estimates provided by Caterpillar Equipment, and Western Mine Engineering's cost estimating guide.

LABOR COST

The operation will employ a total of 71 persons as detailed in Table 22-2.

Table 22-2
Labor Distribution Summary

Facility	Number of Persons
Administration – Tucson	1
Administration - Mine Site	11
Warehouse / Purchasing	1
Mining	7
Mobile Equipment Maintenance	2
Crushing & Stacking	14
Leaching & SX/EW Plant	24
Assay Laboratory	3
Environmental	2
Process Maintenance	6
TOTAL	**71**

Personnel work eight hour shifts, working five days on, and two days off. Additional personnel are included to maintain sufficient work force. Drillers will work two shifts per day, six days per week.

Local salaried staff costs are expected costs of skilled labor. Personnel costs for this study are based upon the base wages including overtime plus a salary burden (Table 22-3).

Personnel requirements and monthly salaries or hourly wages are shown in the individual sections that follow. Salaries include all expected overtime and shift premium costs.

Manning tables were prepared for each year, with typical plant staffing. Wages used are typical for the area. A burden of 32.5 percent was used for both hourly and salaried personnel and is based on current experience at Johnson Camp. An over-time allowance of eight percent was used for hourly employees to cover normal shift schedule requirements, plus an allowance for covering training overtime and similar operating expenses.

Table 22-3
Salary Burdens

Burden Percent of Base Salary	
Payroll Burden	32.5%

Diesel

Diesel fuel costs of $US 2.50 per gallon are utilized, including delivery to site.

POWER COSTS

The price for power is based on Sulphur Springs Valley Electric Cooperative (SSVEC) supplying the power. Nord does not currently have a long term contract with SSVEC and therefore a power cost of $0.055 per kWh was used and deemed appropriate at this time. Nord is working with the utility to provide quarterly load data for obtaining a more accurate price.

The power consumption was estimated based on the Nord study estimates for the crushing plant connected power, connected power for the conveying system, the estimate of solution pumping power based on hydraulic modeling, the estimate for DC power for electrowinning, and the estimate of power consumption for SX, tank farm, and ancillaries. The power consumption was varied by year to reflect changes in equipment power draw for such items as the crushing plant and solution pumps.

Power requirements and costs for the project are detailed in Table 22-4.

Table 22-4

Power Requirements

Area	Equipment/Basis	Installed Hp	Draw at full Capacity (%)
			0.7457
Crushing			
	Primary crusher feeder	120	85
	Dripple conveyor	10	85
	Superior crusher 42x65	400	85
	Primary discharge conveyor	60	85
	Coarse ore stacker	200	85
	Coarse ore feeders, (3)	60	85
	Coarse ore reclaim conveyor	60	85
	Screen feed conveyor	100	85
	Dust collectors (4)	100	85
	Cone Crusher	800	85
	Screen fines stacker	150	85
	Screen (2)	100	85
	Fines feeder	60	85
	Agglomerator feed conveyor	100	85
	Agglomerator	200	85
	Pri & sec tube, hyd	40	85
	Misc. Allowance 5%		
	Total	790	
Conveying			
	Conveyors (11)	1190	85
Leaching			
	New raffinate pump A	200	85
	New raffinate pump B	200	85
	New PLS No. 2 pump	200	85
	PLS pump Pond # 1	200	85
	Exist intermed pump A	200	85
	Exist intermed pump A	200	85
	PlS pump pond # 3	200	85
	Total	1400	

Table 22-5
Power Costs

Production rate, stpy						4,470,000
Cathode, lbs/year						25,000,000
Months per year						12
Total leach solution flow, gpm						6,794
PLS flow, gpm						2,550
Days/week						5
Crushing Shifts/day						2
Hours/shift						10
Crusher Availability %						
	Primary					90%
	Secondary					80%

Area	Equipment/Basis	Installed Hp	Draw at full Capacity (%)	Utilization (% of capacity)	Operating Hours/day	kWh/day
			0.7457			
Crushing						
	Primary crusher feeder	120	85	60	18	821.46
	Dripple conveyor	10	85	60	18	68.46
	Superior crusher 42x65	400	85	60	18	2,738.21
	Primary discharge conveyor	60	85	60	18	410.73
	Coarse ore stacker	200	85	60	18	1,369.11
	Coarse ore feeders, (3)	60	85	98	16	596.32
	Coarse ore reclaim conveyor	60	85	98	16	596.32
	Screen feed conveyor	100	85	98	16	993.87
	Dust collectors (4)	100	85	98	16	993.87
	Cone Crusher, 7ft HD SH (2	800	85	98	16	7,950.95
	Screen fines stacker	150	85	98	16	1,490.80
	Screen (2)	100	85	98	16	993.87
	Fines feeder	60	85	98	16	596.32
	Agglomerator feed conveyor	100	85	98	16	993.87
	Agglomerator	200	85	98	16	1,987.74
	Pri & sec tube, hyd	40	85	80	18	365.09
	Misc. Allowance 5%					1,148.35
	Total	790				24,115.34
Conveying						
	Conveyors (11)	1190	85	98	16	11,827.04
Leaching						
	New raffinate pump A	200	85	50	24	1,521.23
	New raffinate pump B	200	85	50	24	1,521.23
	New PLS No. 2 pump	200	85	33	24	1,004.01
	PLS pump Pond # 1	200	85	33	24	1,004.01
	Exist intermed pump A	200	85	100	24	3,042.46
	Exist intermed pump A	200	85	100	24	3,042.46
	PlS pump pond # 3	200	85	33	24	1,004.01
	Total	1400				12,139.40
SX & Tank Farm (kWh/1,000 gal-PLS)		0.5				2,533.14
EW						
	Non-rectifier (kWh/lb Coppe	0.03				1,532.26
	Rectifier (kWh/lb Copper)	0.97				49,543.08
Ancillaries (kWh/lb Copper)		0.04				2,043.01
					Total kWh/day	91,906.24
Total					Days/yr	kWh/yr
	Crushing and conveying				260.71	9,370,693
	Other				365	24,743,678
					Total	34,114,371
kWh/pound of copper						1.36
kWh/ton of ore						7.63
Cost of power, $/kWh						0.055
Annual cost, $						1,876,290
Cost for year based on months of operation						1,876,290

Table 22-6
Power Requirements

Category	kWh	$US/kWh	Annual Cost
Crushing & Stacking	8,445,537	0.055	464,505
Leaching & SXEW	24,624,026	0.055	1,354,321
TOTAL	**33,069,563**	**0.055**	**1,818,826**

Power generating cost per kWh is based on a contract cost of US$0.055 /KWH.

GENERAL AND ADMINISTRATIVE COSTS

General and administrative costs are summarized in Table 22-Table 22-7.

Table 22-7
General and Administrative Costs Summary

Category	Total Cost	$US/ton
Nord Resources HQ-Tucson	889,750	0.025
Johnson Camp Mine Site	11,420,929	0.325
TOTAL	**12,310,679**	**0.350**

Corporate Office

The cost of maintaining the Corporate office is detailed in Table 22-8. These costs are broken into administrative labor and expenses and detailed in Table 22-9 and Table 22-10, respectively.

The Corporate office is essential for governmental, purchasing and general administrative duties that must be performed in the city. Should the mine site be determined to be a more suitable office site after operations begin, costs are expected to be similar. The personnel are all salaried. The staff will not change in number during mine life.

Table 22-8
Corporate Administrative Cost Summary

Category	Tucson Total Annual Cost	$US/ton
Labor	79,500	0.024
Expenses	5,000	0.002
TOTAL	**119,500**	**0.025**

Table 22-9
Corporate Administrative Labor Costs

Position	Number of Persons	Direct Cost	Loaded Cost
Purchasing Agent	1	60,000	79,500
Annual Cost		60,000	79,500

Table 22-10
Corporate Administrative Expenses

Category	Annual Cost
Postage	2,000
Telephone	2,000
Office supplies	1,000
Annual Cost	5,000

The cost of maintaining the mine office is detailed in Table 22-11. Mine site administrative costs are divided into administrative labor (Table 22 12), administrative expenses (Table 22-13) and warehousing and purchasing (Table 22-14)

Functions included in administrative manpower are management, purchasing, accounting, security, safety and support.

Staff are all salaried and the number of personnel will remain constant throughout mine life.

Table 22-11
Mine Site Administrative Cost Summary

Category	$US/ton
Admin. labor	0.209
Expenses	0.107
Warehouse/purchasing	0.010
TOTAL	**0.325**

Table 22-12
Mine Site Administrative Labor Costs

Position	Number of Persons	Direct Cost	Loaded Cost
General Manager	1	105,000	139,125
Operations Manager	1	90,000	119,250
Secretary	1	24,960	33,072
Mine Accountant	1	42,000	55,650
Payroll	1	24,960	33,072
Health & Safety	1	50,000	66,250
Janitors	1	24,960	33,072
Security Guards	4	99,840	132,288
Annual Cost		526,720	697,904

Table 22-13
Mine Site Administrative Expenses

Category	Annual Cost
Insurance - light vehicles	20,000
Insurance - equipment	10,000
Vehicle operating costs	120,000
Medical Costs	2,000
Delivery truck	12,000
Office equipment	6,000
Communications	15,000
Office supplies	15,000
Engineering supplies	25,000
Postage	2,000
Potable water	5,000
Fees and dues	5,000
Subscriptions	5,000
Outside consultants	50,000
Contingency	58,400
Annual Cost	350,400

Table 22-14
Mine Warehouse and Purchasing Labor

Position	Number of Persons	Direct Cost	Loaded Cost
Warehouse clerks	1	24,960	33,072
Annual Cost		24,960	33,072

MINING COST

Mining costs are based on production estimates as shown in the production schedule, and are separated into manpower, explosives, mining contractor, and fuel premium (Table 22-16).

Table 22-15
Mining Cost Summary

	Unit Cost	Annual Cost
Ore Mining per ton of ore	1.508	5,051,632
Waste Mining per ton of waste	1.603	2,804,301
Totals		7,855,933

Table 22-16
Mining Cost Detail

Category	Ore	$US/ton Ore	Waste	$US/ton Waste
Mine labor	234,774	0.070	121,979	0.070
Explosives Contract	1,266,113	0.378	657,076	0.376
Mining Contractor	3,438,875	1.027	1,966,296	1.124
Fuel Premium	111,870	0.033	58,949	0.034
TOTAL	**5,051,632**	**1.508**	**2,804,301**	**1.603**

Mining Manpower

Manpower requirements are listed in Table 22-17.

Table 22-17
Mine Labor Summary

Position	Pay Rate $/m, $/hr		Number of Persons	Direct Cost	Loaded Cost
Mine Superintendent	$	6,667.00	1	80,004	106,005
Mine Engineer	$	5,833.00	1	69,996	92,745
Mine Geologist	$	5,416.67	1	65,000	86,125
Surveyor	$	3,500.00	1	42,000	55,650
Ore Control Technician	$	2,100.00	3	75,600	100,170
TOTAL			**7**	**332,600**	**440,695**

CRUSHING COSTS

Crushing and stacking costs are summarized in Table 22-18. The costs have been itemized into plant labor, consumable costs (which include spare parts), auxiliary equipment, mobile equipment maintenance labor, and power costs.

Table 22-18
Crushing and Stacking Cost Summary

Category	Annual Cost	$US/ton
Crushing & Stacking Labor	712,051	0.172
Consumables	1,001,782	0.242
Auxiliary Equipment	294,099	0.071
Mobile Equip. Maint Labor	165,295	0.075
Power	464,505	0.113
TOTAL	**2,635,523**	**0.673**

Crushing and Stacking Manpower

Manpower costs are itemized in Table 22-19.

Table 22-19
Crushing and Stacking Labor Costs

Position	Direct Cost	Loaded Cost
Crush & Convey Supt.	74,760	99,057
Conveyor Operators	77,486	102,669
Conveyor Helper	68,370	90,590
Crusher Operators	77,486	102,669
Crusher Laborer	54,696	72,472
Crusher Helper	68,370	90,590
Dozer/Loader Operators	116,229	154,003
TOTAL	**537,397**	**712,051**

Consumable costs are based upon operating experience as well as operating cost guides. The cost figures in Table 22-20 are considered to be conservative. These costs include all maintenance parts and materials.

Power costs in Table 22-22 have been calculated for crushing and stacking based on usage.

Table 22-20
Crushing and Stacking Consumable Costs

Component	Consumption (lb/ton)		Unit Cost ($/lb)	
Primary crusher liners		0.013	$	1.20
Cone crusher liners		0.060	$	1.30
Agglomerator		0.010	$	0.33
Screen Decking		4 replacements/yr	$	0.0206
Chute liners			$	0.0079
Conveyors			$	0.0223
Stacker			$	0.0446
Miscellaneous			$	0.0500
TOTAL	$	0.242	$	2.98

The itemized cost for the auxiliary mobile equipment is included as Table 22-21.

Table 22-21
Process and Auxiliary Equipment Operating Costs

	Cost per Hour ($US)									
	Parts	Fuel	GET	Tires	Lube	Own	Total	Hours	$US	$US/t
Dozer D6LGP	5.71	8.28	22.73	0.00	2.27	75.00	113.99	700	79,790	0.019
Front end loader 992	8.86	57.50	22.73	22.73	3.00	240.00	354.82	600	214,309	0.052
TOTAL	**14.57**	**65.78**	**45.46**	**22.73**	**5.27**	**315.00**	**468.81**		**294,099**	**0.071**

Table 22-22
Crushing and Stacking Power Costs

Category	kWh	$US/kWh	Annual Cost
Crushing & Stacking	8,445,537	0.055	464,505
TOTAL	**8,445,537**		**464,505**

PROCESSING COSTS

Table 22-23
Leaching and SX/EW Cost Summary

Category	Annual Cost	$US/lb
Process labor	1,269,151	0.051
Maintenance Labor	392,283	0.016
Assay lab labor	122,080	0.005
Consumables	2,475,006	0.099
Maintenance Materials	1,249,805	0.050
Power	1,354,321	0.054
Annual Cost	6,862,647	0.275

Table 22-24
SX/EW Maintenance Labor Cost

Position	Direct Cost	Loaded Cost
Maintenance Supervisor	75,000	99,375
Mech/pipe/welder	129,903	172,121
Elect/Instr.	91,160	120,787
Annual Cost	296,063	392,283

Table 22-25
Assay Laboratory Manpower Requirements

Position	Pay Rate $/mo, $/hr	Number of Persons	Direct Cost	Loaded Cost
Assayer	$ 3,120.00	1	37,440	49,608
Lab Technicians	$ 12.00	2	54,696	72,472
Annual Cost			92,136	122,080

Process and plant consumables costs are included in Table 22-28.

The major consumable materials are sulfuric acid, LIX and diluent for the SX section, and heap piping materials. Other items included in consumable costs were cobalt sulfate for EW, analytical supplies, and miscellaneous operating supplies.

Maintenance material costs are shown in Table 22-26. Process leaching and SX/EW power costs are shown in Table 22-27.

Table 22-26
Maintenance Materials Costs

	Annual Cost	$US/lb Cu
Crushing Plant	$ 326,562	$ 0.013
Conveying	$ 93,362	$ 0.004
Lubrication		
Crushing	$ 123,386	$ 0.005
Conveying	$ 41,129	$ 0.002
Maintenance		
Supplies	$ 19,742	$ 0.001
Miscellaneous	$ 205,644	$ 0.008
Leaching (pumps)	$ 27,980	$ 0.001
SX/EW	$ 412,000	$ 0.016
Totals	$ 1,249,805	$ 0.050

Table 22-27
Process and SX/EW Power Costs

Category	kWh	$US/kWh	Annual Cost
Process and SX/EW	24,624,026	0.055	1,354,321

Table 22-28
Processing Consumable Costs

		Cost / lb Cu
Lix Concentration in organic, %	7.60	
Loss in raffinate, ppm (1)	2.28	
Loss in raffinate, gallons/yr	3,024	
Loss in rich electrolyte, ppm(2)	2.28	
Loss in rich electrolyte, gallons/yr	450	
Total loss, gallons/yr	3,474	
Unit cost, $/gallon	36.80	
Annual Cost, $	$ 127,834	$ 0.005
Diluent	-	
Concentration in organic, %	92.39	
Loss in raffinate, ppm (1)	27.72	
Loss in raffinate, gallons/yr	36,782	
Loss in rich electrolyte, ppm(2)	27.72	
Loss in rich electrolyte, gallons/yr	5,474	
Total loss, gallons/yr	42,256	
Unit cost, $/gallon	2.45	
Annual Cost, $	$ 103,527	$ 0.004
Allowance for organic treatment	-	
Annual Cost, $	19,000	$ 0.001
Cobalt Sulfate	-	
Consumption, lb/ton copper	0.380	
Consumption, lb	4,739	
Unit cost, $/lb	17.5	
Annual Cost, $	$ 82,930	$ 0.003
Cathodes and annodes		
Acid	-	
Ore acid requirement, tons residual leach	-	
Consumption, lb gross/ton	29.77	
Lb Cu recovered/ton ore	6.16	
Acid from EW, tons	-	
Lb EW acid/lb copper	1.54	
Net acid consumption, lb/ton	20.29	
Acid price, $/ton delivered	46.13	
Acid cost, $/ton ore	0.48	
Acid cost, $/lb copper	0.08	
Acid cost, $/year	#########	$ 0.078
Heap piping, $/year	$ 130,000	$ 0.005
Analytical supplies (plant), shipments assays	$ 50,000	$ 0.002
Average Cost/Ton of Ore LOM	#########	$ 0.099

ENVIRONMENTAL COSTS

Environmental costs have been estimated for the project (Table 22-29). Labor costs and consumable costs are identified and estimated separately. These costs are in addition to reclamation and closure costs, and represent the average annual operating expense, which may vary from year to year.

Table 22-29
Environmental Costs

Category	Total Cost	$US/ton
Labor	72,472	0.020
Consumables	50,000	0.015
TOTAL	**122,472**	**0.035**

Table 22-30
Environmental Manpower Costs

Position	Pay Rate $/m, $/hr	Number of Persons	Direct Cost	Loaded Cost
Laborers	$ 12.00	2	54,696	72,472
TOTAL		**2**	**54,696**	**72,472**

Table 22-31
Environmental Consumables Costs

Category	Annual Total Cost
Water Sampling	43,000
Other	7,000
TOTAL	**50,000**

23.0 ENVIRONMENTAL PERMITTING

23.1 INTRODUCTION

The Johnson Camp Mine property has been the site of mining for more than 100 years, and evidence of this historic activity is abundant. The Republic mine was the first major operation on the property. The mine consisted of a network of underground workings that extended to 1,000 feet below the surface, associated mine waste dumps, a flotation mill, and an adjacent mill tailings pile. These facilities were largely removed by the open pit mining initiated by Cyprus in 1975, although remnants of the mill foundation and tailings remain in the east wall of the Burro pit. To the north, the Black Prince and Moore mines were developed contemporaneously with the Republic mine, with attendant mine waste rock piles and shaft facilities. These shafts are currently utilized as water wells by Nord. Small piles of waste rock and low-grade material dot the landscape across the property.

Disturbances created by Cyprus' open pit operations dominate the current landscape. Cyprus was largely responsible for the current footprint of the mining operation, having developed the Burro pit and adjacent waste rock piles, and the existing leach pads, solution ponds, processing facility foundations, and service area. Cyprus operated these facilities until 1986, when it shut the operation down. The property and remaining facilities were sold to Arimetco in 1989.

Arimetco expanded the footprint of the disturbance by about 15 percent between 1990 and 1997, commencing limited mining of the Copper Chief deposit north of the Burro pit, adding a new small waste rock pile north of the access road, and new leach pad extensions to the north and east of the Cyprus leach pad area. Except for these expansions, Arimetco operated almost exclusively within the footprint of disturbance created by Cyprus.

Groundwater Quality

Groundwater quality in and around the project area has been sampled for quality for over ten years. Monitoring results indicate the presence of two aquifers (alluvial and bedrock) that reflect the impact of mineralization and historic mining activities. Consent Order #P-130-99 signed with Arizona Department of Environmental Quality (“ADEQ”) determined that there was evidence of groundwater contamination at Johnson Camp, and obligated Nord to characterize the hydrogeology of the site and to obtain an aquifer protection permit. The subsequent Consent Order #P-4-01 reiterated these requirements.

A hydrological study was prepared and submitted with site Aquifer Protection Permit ("APP") application dated July 2003. ADEQ responded to the APP application by letter on 9/2/03, with the following concerns pertaining to the hydrological study:

> *5) In accordance with AAC R18-9-A202(A)(9), please provide a detailed proposal indicating the alert levels, discharge limitations, monitoring requirements, compliance schedules, … that Nord Resources Corporation will use to satisfy the requirements of Arizona Revised Statutes, Title 49, Chapter 2, Article 3, and Articles 1 and 2 of AAC Title 18, Chapter 9.*
> *The information presented in Volume 2, Table 1: Proposed Groundwater Monitoring Plan of the application only addresses alert levels proposed for groundwater monitoring, and does not meet the minimum requirements for consideration as being administratively complete. Nord must also provide detailed proposals for alert levels, discharge limitations and monitoring requirements of the various discharging facilities."*

Nord acquired the services of a consulting hydrogeologist, to address ADEQ's concerns. The consultant's 4/19/05 report was prepared following a 3/2/05 meeting with ADEQ and is based upon review of similar copper mining facilities within Arizona. Four monitoring wells are proposed for the Johnson Camp Mine facility, located down-gradient of current and proposed future mine facilities, specifically east and down-gradient of the proposed expanded waste rock pile and the proposed future leach pad area and PLS/raffinate ponds. The two most northerly compliance wells will be located in areas with minimal if any degraded groundwater resulting from past mining activities. The two southern compliance wells will be located in an area expected to exhibit slightly elevated sulfate concentrations, as a result of their proximity to past mining activities. Groundwater west and up-gradient of the mine pit has been documented in the APP hydrological report as having been degraded by past mining activities (specifically the historic-existing leach pad and PLS and raffinate ponds).

Groundwater level data, however, indicates that the current pit acts as a hydrologic sink that effectively captures the degraded groundwater. Further pit expansion and deepening will expand the hydraulic capture area. Therefore, no compliance wells have been recommended for placement adjacent to and immediately down-gradient of the current active leach pads and PLS/raffinate ponds. Nord contends that several existing and proposed "characterization" monitor wells placed in the vicinity of these facilities will provide data necessary to continue documentation of groundwater flow directions and chemical characterization of groundwater in the historic active mine area.

The consulting hydrologist's letter further addressed proposed compliance monitoring frequencies and testing parameters. The details of the monitoring program will be used in responding to ADEQ's requests regarding the APP application.

23.2 REGULATORY ACTIONS SUMMARY

23.2.1 Consent Order #P-139-99 (6/7/99)

Nord Copper Corporation, its parent company, Nord Resources Corporation (collectively “Nord”), and the Arizona Department of Environmental Quality (“ADEQ”) signed Consent Order #P-139-99 (effective 6/7/99) for the Johnson Camp Mine. The Consent Order has allowed Nord to continue to operate the Johnson Camp facility, and make improvements to the facility that would bring the facility into compliance with current Arizona statutes. The Consent Decree provided findings of fact and conclusions of law, as well as a work plan, compliance schedule, stipulated penalties, progress reporting requirements, dispute resolution methods and several other items of agreement. A summary of the work plan and schedule is presented in Table 23.2.1-1.

One of the conditions of the Consent Decree is that a written status report to ADEQ is required every quarter until full compliance is achieved. The report is to detail progress on the compliance schedule and certify when compliance on each item has been achieved. Consent Order #P-139-99 was superseded by Consent Order #P-4-01.

23.2.2 Consent Order P-4-01 (01/03/01)

Nord and ADEQ entered into Consent Order #P-4-01 on 1/03/01 to replace the preceding Consent Order #P-139-99. The second Consent Order allowed Nord to continue to operate the Johnson Camp facility, and make improvements to the facility that would bring the facility into compliance with current Arizona statutes. The Consent Decree provided findings of fact and conclusions of law, as well as a work plan, compliance schedule, stipulated penalties, progress reporting requirements, dispute resolution methods and several other items of agreement. It was to remain in effect until such time as ADEQ takes action upon Nord’s Aquifer Protection Permit (APP) application and issues an APP for the Johnson Camp Mine.

One of the conditions of the Consent Decree is that a written status report to ADEQ is required every quarter until full compliance is achieved. The report is to detail progress on the compliance schedule and certify when compliance on each item has been achieved. Consent Order #P-4-01 was superseded by Compliance Order #APP-114- 02.

23.2.3 Compliance Order #APP-114-02 (9/17/02)

ADEQ found Nord to be in violation of the state aquifer protection laws (ARS Title 49, Chapter 2, Article 3 and AAC Title 18, Chapter 9, Articles 1 & 2) and Consent Order #P-4-01 and issued Compliance Order #APP-114-02 as a result. The Order required the following:

- Nord shall bring the Johnson Camp Mine into compliance with Arizona’s aquifer protection laws;

- Nord and ADEQ shall enter into a Stipulated Judgment and Stipulated Judgment Entry Agreement for civil penalties in the amount of $4.325 million as a consequence of Nord's violation of Consent Order #P-4-01 and the aquifer protection laws, with the agreement that ADEQ may not file for entry of the judgment unless and until Nord were to violate the Compliance Order and fail to timely cure the violation, or if Nord were to become the subject of a bankruptcy, insolvency or receivership proceeding prior to achieving compliance with the Order; and
- Nord and ADEQ shall enter into the Johnson Camp Mine Escrow Agreement to create an escrow account requiring a $1.5 million deposit by Nord to be used solely to pay for the direct costs of bringing the Johnson Camp Mine into compliance with the Order and the aquifer protection laws.
- Nord is permitted to produce copper.

Upon becoming effective, the Order replaced and superseded Consent Order #P-4-01. This provides Nord the opportunity to renegotiate certain construction elements previously required under the previous Consent Orders and thereby allows more flexibility in component designs. Specifically, it allows Nord to construct the process ponds and solution conveyance ditches and leach pads away from Prescriptive BADCT standards and instead towards Individual BADCT standards. Specifically, Nord may utilize clay liners on leach pads instead of HDPE liners, and utilize clay or in-place soil to replace HDPE liners on solution conveyance ditches, provided that Nord can demonstrate compliance with groundwater quality standards at the selected point of compliance.

Nord utilized the $1.5 million escrowed funds towards environmental remediation activities at the Johnson Camp Mine and to prepare and file the APP application with ADEQ. A summary of the Compliance Order is presented in Table 23.2.3-1. To date, Nord remains in compliance with the Compliance Order.

23.3 PERMITTING FOR OPERATIONS

23.3.1 Water Management

The State of Arizona has confirmed that the Johnson Camp project is not located in an AMA (Arizona Active Management Area) and that additional appropriations are indeed available, and there are no state-imposed limitations as to how much water may be pumped from these wells.

As part of Nord's projected improvements, the solution management system will be upgraded. This will include an increase in the solution storage capacity and the construction of double-lined process ponds capable of meeting Arizona Best Available Demonstrated Control Technology (BADCT). Nord is committed to complying with the requirements addressed in the past Consent Orders #P-130-99 and #P-4-01 and Compliance Order #APP-114-02 which address solution management issues. Negotiations as to facility mitigation, upgrades and future facilities design will proceed as Nord completes the APP permitting process.

23.3.2 Air Quality

Arimetco operated a crushing facility at Johnson Camp from November 1995 to May 1997, and had an Air Quality Permit for the facility. That facility has since been removed from the property, and the Air Quality Permit is no longer valid. During modeling of emissions in 1997, the Air Quality Division determined that Arimetco needed to either control emissions or install a PM-10 Air Monitoring System downwind (to the east) that would insure the ability to monitor compliance.

Reinstalling a crushing plant at the same location as the previous crushing facility will require a new Air Quality Permit. Discussions with ADEQ indicate that a new Air Quality Permit can be processed in three to six months, once the crushing system has been designed and specified, and modeling of emissions for this system are completed. Potential air emissions will be mitigated at the site by watering road surfaces and the use of spray bars and/or scrubbers, or bag houses, in the crushing plant.

23.3.3 Hazardous Materials and Explosives

All use and storage of explosives will be under the direct control of the mining contractor. Material Safety Data Sheets (MSDS) will accompany all chemicals received on site. Hazardous materials (chemicals and fuels) will be managed and handled to minimize exposure and potential release to the environment.

Nord has prepared an Emergency Response Plan that documents hazardous materials utilized at the project, quantities stored, and procedures to be practiced by employees in handling these materials and procedures to be taken in the event of a release thereof.

23.3.4 Weights and Measures Permit

The Johnson Camp project is licensed by the Arizona Department of Weights and Measures for the weighing of cathode copper for sale and shipment.

23.3.5 Aquifer Protection Permit (APP)

The Johnson Camp Mine project currently operates the Compliance Order. ADEQ has determined that the operation must obtain an Aquifer Protection Permit (APP), under the Arizona Environmental Quality Act, passed in 1986. Consent Order #P-130-99 required submittal of an APP application by 9/7/00. Consent Order #P-4-01 required submittal of an APP application by 9/7/01.

The current Compliance Order #APP-114-02 required submittal of an APP application by 6/7/03, which action was completed on time by Nord.

Arimetco submitted an initial APP application in May 1990. In October 1993, a completeness review letter was received from ADEQ regarding the application. The letter indicated the application was incomplete and requested Arimetco to submit additional information. In April 1994, Arimetco submitted additional information to ADEQ to address its request. In April 1996, a letter was sent by ADEQ to Arimetco requesting the installation of a series of groundwater monitor wells and development of a waste rock characterization program on the property. In response, Arimetco completed installation of eight additional groundwater monitoring wells and several piezometers in the pond/plant area. These wells supplement the four previous groundwater monitoring wells and zone wells that are down gradient from the pond/plant area.

Nord has since negotiated the location of the remaining monitor wells locations with ADEQ. In January 1997, Arimetco filed for protection from creditors under Chapter 11 of the U.S. Bankruptcy Code, and no further work was done towards completion of the APP application by it. Arimetco failed to submit a complete application for the APP. ADEQ gave Arimetco several deadlines for submission of a complete application, as well as several extensions to those deadlines. Finally, the State of Arizona notified Arimetco that it was out of compliance with respect to the requirement to apply for and obtain an APP.

Nord recognized the failure of Arimetco to obtain an APP in a timely manner. To address this situation, Nord negotiated Consent Orders #P-130-99 and #P-4-01 and Compliance Order #APP-114-02 with ADEQ. Under the terms of each order, Nord is to proceed with a timely application for an APP while it continues to operate the Johnson Camp facility and to make improvements to the facility that will bring it into compliance with current Arizona statutes.

Nord filed an APP application on 7/21/03. ADEQ responded to the application by letter on 9/2/03, stating certain deficiencies needed to be corrected to allow for APP issuance. Nord met with ADEQ on 3/2/05 to discuss deficiency items, and has since received input from the consulting hydrologist to address certain concerns, as discussed above. Nord is currently in action in addressing remaining APP application deficiencies, and plans to revise its APP application submittal within the near future.

23.3.6 Storm Water NPDES Permit

The Johnson Camp operation was issued NPDES Permit # AZRO5B377 by the U.S. Environmental Protection Agency on 3/7/01. This permit authorizes the operation to discharge storm water associated with industrial activities under the terms and conditions imposed by EPA's storm water baseline industrial general permit issued for use in the State of Arizona. A Storm Water Pollution Prevention Plan (SWPPP) was developed and submitted in August 1999 as part of the submittal requirements of Consent Order #P-130-99. The SWPPP identifies exposed materials with the potential for interaction with storm water run-on and run-off from the facility, and describes Best Management Plans (BMPs) necessary to control against their release.

Consent Order #P-4-01 required Nord to contact ADEQ regarding possible revisions to the SWPPP by 3/4/01. The SWPPP requirements were changed by Federal Register 65 64801 et. seq. on 10/30/00. Nord will update the existing SWPPP to comply with these requirements as appropriate.

23.3.7 U.S. Army Corps of Engineers (ACOE) Section 404 Dredge & Fill Permit

The proposed activities (new heap leach pad and ponds, and waste dump facilities) may require the need for a U.S. Army Corps of Engineers (ACOE) Section 404, Dredge & Fill Permit since proposed activities may impact jurisdictional water(s) of the United States. The permit becomes necessary for dredge and fills activities located within these waters. It is likely that Nord's activities are not located within such waters (typically defined as 100-year floodplain areas). Nord will consult with the ACOE once new facilities and their locations are set.

23.4 Annual Environmental Expenditures

Costs of annual environmental monitoring and remediation were provided by Nord through their consultant, Dale Deming, P.E. The cost associated with rinsing the pads is based on past experience of Nord with the old heaps. In the Nord commissioned feasibility study the engineering firm reviewed the data and believed it to be adequate to support their feasibility study; BETA has reviewed the data, believes it is reasonable, and has incorporated it into this technical report.

Table 23.4-1:
Environmental Expenses

Item	Year 1 2006	Ongoing Annual After 2006
Hydrologic study plan (APP) Note 1	$10,000	
Monitoring well water analysis Note 1	$0	$38,000
Additional monitoring wells	$100,000	
Reclamation plan	$10,000	
Air quality permit	$40,000	
Environmental consultant	$30,000	
Toxic release inventory	$4,000	
Misc. professional services	$15,000	
Contingency	$31,000	$5,000
Total	$240,000	$43,000

23.5 Reclamation and Mine Closure

Arimetco had no reclamation or closure plans for the Johnson Camp property, nor is there a bond outstanding to perform reclamation and closure activities. Reclamation and closure plans are required under the APP program and the Arizona Mined Land Reclamation Law. An APP closure plan must present measures to be taken to prevent discharges of pollutants from the facility after operations cease, the methods that will be used to secure the facility, and any other measures needed to protect groundwater resources, including post-closure monitoring and maintenance as needed.

The reclamation plan requires that all mining disturbances occurring after 1987 be reclaimed to a level that will support the designated post-mining land use. Open pit mines are excluded from reclamation requirements, however, waste dumps, tailing piles, leach facilities, process water ponds and site buildings and roadways will require closure and reclamation. A portion of the reclamation costs would be offset by the salvage and sale of processing and other equipment at the end of mine life. Additionally, Nord plans to continue the sale of landscape rock, aggregates and railroad ballast after mine closure to offset the costs of closure.

Nord submitted a Reclamation and Mine Closure Plan to the Arizona State Mine Inspector office in July 2007. Components of the Reclamation and Mine Closure Plan, as identified by areas and types of disturbance components, are summarized in Table 23-5-1 below.

23.5.1 Post Mining Land Use Objectives

Four separate Post Mining Land Use (PMLU) objectives were envisioned for the site, considering public safety, existing and historic land uses, climate, soil quantity and quality, and economic feasibility. The selected PMLU objectives are as follows.

23.5.1.1 Rangeland

Rangeland habitat is suitable for large portions of the Johnson Camp Mine following closure and will be met by development of post-mining vegetative cover suitable for supporting seasonal livestock grazing at a density consistent with historic and current practices of a maximum of 10 head of cattle per section (640 acres).

23.5.1.2 Future Mineral Exploration and Development

Several areas of the Johnson Camp Mine provide opportunity for future mineral exploration and development. Considerations for future economic feasibility include: the amount of retained ore in and near existing pits; unrecovered ores; economic cutoff grades necessary for development of prospective ore bodies; the prospect of new technologies for economic recovery of metals from materials not currently considered economically feasible; and long term metal prices.

23.5.1.3 Storm Water Management

Certain elements of the leachate collection system, storm water diversion system and retention basins currently manage storm water from contacting disturbed areas and control impacted storm water flows through retention and evaporation. The Burro and Copper Chief Pits will act as hydrologic sinks to capture storm water drainage and drainage from decommissioned leach pads. These facilities are expected to continue to operate and are therefore considered to be a PMLU.

23.5.1.4 Landscape Rock and Borrow Material

A contractor is currently operating a landscape rock business to process waste rock materials for sale as landscape material, riprap and railroad ballast to contractors and the public. Nord has identified a significant amount of available material for this purpose currently stored in the Johnson Camp Mine waste dumps; therefore, this identifies a significant PMLU for the property. Nord intends to assume this landscape rock business and release the current contractor.

23.5.2 Facility Closure Plans

23.5.2.1 Open Pits

Two open pits exist at the Johnson Camp Mine, namely the Burro Pit and the Copper Chief Pit. The Burro Pit is currently 2,600 feet in length by 1,850 feet in width by 520 feet deep, and covers 110 acres. Bench heights are 20 feet with 45^{o} and 55^{o} pit backslopes. Post-1986 to 2003 disturbance is estimated at 76.66 acres. The current base mined elevation is 4,540 feet above mean sea level (AMSL). At closure, the Burro Pit will encompass approximately 91 acres with a perimeter of approximately 7,931 feet. The Copper Chief Pit post-1986 to 2003 disturbance is estimated at 17.66 acres. The current base mined elevation is 5,025 feet AMSL. At closure, the Copper Chief Pit will encompass approximately 54 acres with a perimeter of approximately 11,186 feet.

Pit closure will be accomplished by construction of barriers to prevent access, plus the construction of standard four-strand barbed wire safety fences to surround the pit perimeter. Warning signs constructed of weather resistant material will be placed at 300-foot intervals along the fences. Fences will be monitored on a routine basis until all other reclamation activities have been completed.

23.5.2.2 Heap Leach Pads

Stormwater diversions surrounding leach pad areas will remain in place to prevent stormwater run-on. Surfaces of leach pads will be re-graded during final operations, capped with one foot of growth medium soils, and compacted using haulage equipment to minimize infiltration. The surface will then be scarified and re-vegetated.

23.5.2.3 Waste Rock Disposal Areas

Two waste rock disposal areas are currently in place, namely the North Lobe (003A) and the South Lobe (003B). The North Lobe areas are primarily upper and middle Abrigo calc-silicate, limestone, hornfels and skarn rocks. This dump is covered with volunteer vegetation and no erosional washout or circular or irregular surface failures are noted.

The South Lobe is composed of internal pit material of Bolsa Formation Quartzite (90%), with diabase, hornfels and calcsilicate. This material will continue to be reprocessed as landscape material, and is extremely stable to the point that significant undercutting is necessary to cause slope failure during current landscape material processing operations. No erosional washout or circular or irregular surface failures are noted. Waste rock piles will be re-graded to maintain slope stability, to minimize erosion and to provide surface slopes suitable for revegetation as required by the Rangeland PMLU. The North Lobe waste dump will not be expanded along its side slopes. The top surfaces will be re-vegetated by addition of growth medium, scarification and reseeding. The South Lobe waste dump will continue to be processed for landscape rock, riprap and railroad ballast material. Areas not reprocessed accordingly will have the surface groomed by crown-chaining to remove loose boulders and cobbles and then re-vegetated by addition of growth medium, scarification and reseeding. Future waste rock piles will be built in two tiers with free faces less than 50 feet in height to increase stability and reduce slope failures. At closure, interfaces of old and new waste rock piles will be assessed and high-wall free slope faces crown-chained to remove loose boulders and cobbles, and the surface scarified and re-seeded.

23.5.2.4 Process and Storm Water Ponds

Process ponds to be retained for the Storm Water Management PMLU will be retained following closure. Ponds not designated for this use shall be reclaimed by recirculating retained and rinse solutions to the heap leach pads for consumption/evaporation, and the pond liners cut and folded in place then buried. The area would then be regraded to provide for positive drainage and growth medium placed over the area to an approximate two-foot depth, and the area re-vegetated.

23.5.2.5 Buildings

Buildings and ancillary facilities (post-1986) not designated for a specific PMLU will be demolished, with salvaging of contained materials and equipment to the extent practicable. Excess reagents will be returned to suppliers, sold or otherwise transferred to other mining facilities. Non-salvageable items (HDPE liner, concrete and scrap building materials) shall be disposed of in the on-site landfill or transported offsite to an acceptable disposal facility. Suspected contaminated and/or hazardous materials shall be sampled to determine acceptable disposal alternatives. Concrete foundations will be broken and/or buried by approximately two feet of cover soils.

23.5.2.6 Pipelines and Electrical Systems

Concrete pipelines will be drained, then crushed and buried in place. Culverts will be crushed in place and buried in conjunction with access and haul road reclamation, with the exception of two culverts to remain post closure for maintenance access roads and storm water controls. HDPE pipelines will be drained and disposed of at the onsite landfill. Electrical lines will be gathered for salvage. Electrical poles will be cut at ground level and the poles salvaged and/or disposed of at the onsite landfill. Areas disturbed will be re-vegetated.

23.5.2.7 Roads

Seven access and haul road areas encompassing 10.63 acres will exist at mine closure, which will be reclaimed by regrading to minimize cut banks and to promote drainage, then ripped or scarified to a depth of six inches to two feet and subsequently re-vegetated. Certain roads will need to be maintained to access storm water control diversions, slope stability monitoring points, monitoring wells and to allow for maintenance of public safety controls (berms and fencing) surrounding open pits. Other selected access roads may be left intact to accommodate potential PMLUs such as future industrial/commercial activities or future mining. Haul road widths will be reduced to that necessary to allow for site access. Public access during establishment of vegetation will be discouraged through placement of dirt berms and temporary fencing, accompanied by warning signs.

23.5.3 Soil Placement and Re-vegetation

Growth Medium soils totaling 270,000 tons are currently available within six piles on the property. As future areas are developed (specifically the new Leach Pad Area), additional growth medium soils will be stripped and stockpiled for closure activities. Due to the availability of growth medium soils on site, no import of these materials is anticipated. Upon closure, the growth medium soils will be redistributed and scarified to provide a base for vegetative growth.

Revegetative plant species have been selected to meet the Rangeland PMLU objective and have been matched to soil types, climatic and topographic conditions. Natural plant colonization is further expected to supplement Nord’s revegetative efforts. A seed mix recommended by the Bureau of Land Management, Safford District was utilized as a basis for the Reclamation Plan; however, final plant selection will occur at closure based upon availability, anticipated success rates and costs.

Revegetation will be accomplished using either hydroseeding or broadcast seeding methods, depending upon underlying soils and location. Broadcast seeding is anticipated to occur in the office, plant shop, scrap yard, building areas and the haul and access roads. On reclaimed heap leach pads and waste rock dumps, pure live seed will be dispersed by a mechanical broadcast seeder followed by a drag to bury the seed in flat areas. Steeper sloped areas would be hydroseeded. Mulch may be incorporated as needed in certain areas. Re-vegetated areas will not be irrigated; instead, the planting season will be scheduled to correspond with seasonal rainfall patterns to ensure plant germination with site preparation planned for the Spring season, i.e., seeding in late June (pre-monsoon rainfall) and plant growth occurring during the Summer months when temperature and moisture conditions are optimal.

Table 23-5-1

Johnson Camp Closure Cost Estimate

MINE CLOSURE ELEMENT	ACRES	TASK (units)	UNITS	UNIT COST	TOTAL	REFERENCE
Open Pits:						
Burro Pit	91.1	Fencing (lineal feet)	7,968	$1.88	14,980	ASMI
Copper Chief Pit	54.2	Fencing (lineal feet)	6,146	$1.88	11,554	ASMI
Open Pit Subtotal	**145.3**				**$ 26,534**	
Leach Pads:						
Leach Pads #1-5	355.7	Rinsing		Included within Operations Budget	-	Internal
Pre 1986 Cyprus	50.74	Material Haulage for Backfill; 2000' one way (cu. yd.)	40,927	$1.03	42,155	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	40,927	$0.44	18,008	ASMI
		Ripping & Scarification (acre)	50.74	$120.00	6,089	Internal
		Revegetation Cost--Hydroseed (acre)--20% of Area	10.15	$1,175.00	11,924	ASMI
		Revegetation Cost--Disc (acre)--80% of Area	40.59	$607.00	24,639	ASMI
Leach Pad #1 (Post Arimetco)	17.42	Material Haulage for Backfill; 2000' one way (cu. yd.)	14,051	$1.03	14,473	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	14,051	$0.44	6,182	ASMI
		Ripping & Scarification (acre)	17.42	$120.00	2,090	Internal
		Revegetation Cost--Hydroseed (acre)--20% of Area	3.48	$1,175.00	4,094	ASMI
		Revegetation Cost--Disc (acre)--80% of Area	13.94	$607.00	8,459	ASMI

MINE CLOSURE ELEMENT	ACRES	TASK (units)	UNITS	UNIT COST	TOTAL	REFERENCE
Leach Pad #2 (Post Arimetco)	9.94	Material Haulage for Backfill; 2000' one way (cu. yd.)	8,018	$1.03	8,259	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	8,018	$0.44	3,528	ASMI
		Ripping & Scarification (acre)	9.94	$120.00	1,193	Internal
		Revegetation Cost--Hydroseed (acre)—20% of Area	1.99	$1,175.00	2,336	ASMI
		Revegetation Cost--Disc (acre)--80% of Area	7.95	$607.00	4,827	ASMI
Leach Pad #3	19.8	Material Haulage for Backfill; 2000' one way (cu. yd.)	15,971	$1.03	16,450	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	15,971	$0.44	7,027	ASMI
		Ripping & Scarification (acre)	19.80	$120.00	2,376	Internal
		Revegetation Cost--Hydroseed (acre)—20% of Area	3.96	$1,175.00	4,653	ASMI
		Revegetation Cost--Disc (acre)--80% of Area	15.84	$607.00	9,615	ASMI
Leach Pad #4	25.9	Material Haulage for Backfill; 2000' one way (cu. yd.)	20,891	$1.03	21,518	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	20,891	$0.44	9,192	ASMI
		Ripping & Scarification (acre)	25.90	$120.00	3,108	Internal
		Revegetation Cost--Hydroseed (acre)—20% of Area	5.18	$1,175.00	6,087	ASMI
		Revegetation Cost--Disc (acre)--80% of Area	20.72	$607.00	12,577	ASMI

MINE CLOSURE ELEMENT	ACRES	TASK (units)	UNITS	UNIT COST	TOTAL	REFERENCE
Leach Pad #5	231.9	Material Haulage for Backfill; 2000' one way (cu. yd.)	187,053	$1.03	192,665	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	187,053	$0.44	82,303	ASMI
		Ripping & Scarification (acre)	231.90	$120.00	27,828	Internal
		Revegetation Cost--Hydroseed (acre)—20% of Area	46.38	$1,175.00	54,497	ASMI
		Revegetation Cost--Disc (acre)--80% of Area	185.52	$607.00	112,611	ASMI
Leach Pad Subtotal	**355.7**				**$ 720,763**	
Waste Rock Dumps:						
Waste Rock Dumps	131.23	Crown Chaining (lineal foot)	4,600	$1.76	8,096	Internal
		Material Haulage for Backfill; 2000' one way (cu. yd.)	105,851	$1.03	109,027	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	105,851	$0.44	46,574	ASMI
		Ripping & Scarification (acre)	131.23	$120.00	15,748	Internal
		Revegetation Cost--Hydroseed (acre)--20% of Area	26.25	$1,175.00	30,839	ASMI
		Revegetation Cost--Disc (acre)--80% of Area	104.98	$607.00	63,725	ASMI
Landscape Waste Rock Areas--Post Recovery Reclamation	37.24	Ripping & Scarification (acre)	37.24	$120.00	4,469	Internal
		Revegetation Cost--Disc (acre)	37.24	$607.00	22,605	ASMI
Waste Rock Dumps Subtotal	**168.47**				**$ 301,083**	
Process and Storm Water Ponds:						
PLS Pond #1 + 25%	5.26	Material Haulage for Backfill; 2000' one way (cu. yd.)	4,243	$1.03	4,370	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	4,243	$0.44	1,867	ASMI
		Ripping & Scarification (acre)	5.26	$120.00	631	Internal
		Revegetation Cost--Disc (acre)	5.26	$607.00	3,193	ASMI

MINE CLOSURE ELEMENT	ACRES	TASK (units)	UNITS	UNIT COST	TOTAL	REFERENCE
PLS Pond #3 + 25%	3.99	Material Haulage for Backfill; 2000' one way (cu. yd.)	3,218	$1.03	3,315	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	3,218	$0.44	1,416	ASMI
		Ripping & Scarification (acre)	3.99	$120.00	479	Internal
		Revegetation Cost--Disc (acre)	3.99	$607.00	2,422	ASMI
Raffinate Pond #1 + 25%	1.02	Material Haulage for Backfill; 2000' one way (cu. yd.)	823	$1.03	848	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	823	$0.44	362	ASMI
		Ripping & Scarification (acre)	1.02	$120.00	122	Internal
		Revegetation Cost--Disc (acre)	1.02	$607.00	619	ASMI
New Raffinate Pond #2 + 25%	1.15	Material Haulage for Backfill; 2000' one way (cu. yd.)	928	$1.03	956	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	928	$0.44	408	ASMI
		Ripping & Scarification (acre)	1.15	$120.00	138	Internal
		Revegetation Cost--Disc (acre)	1.15	$607.00	698	ASMI
PLS Ponds #5A, #5B and #5 Emergency Ponds	5.4	Material Haulage for Backfill; 2000' one way (cu. yd.)	4,356	$1.03	4,487	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	4,356	$0.44	1,917	ASMI
		Ripping & Scarification (acre)	5.40	$120.00	648	Internal
		Revegetation Cost--Disc (acre)	5.40	$607.00	3,278	ASMI
Secondary Containment Pond #1 (1.86 acres) and Secondary Containment Pond #2 (1.06 acres)--both below PLS Pond #1	2.92	None; PMLU Use			-	
Storm Water Containment Ponds (8)	0.6	None; PMLU Use			-	
Process and Storm Water Ponds Subtotal	**20.34**				**$ 32,174**	
Buildings and Ancillary Facilities:						
Buildings: Office/Warehouse, Lab/Change House, Mechanic Shop, SX Plant, EW Plant, Truck Shop & Core Shed	17	Demolition & Removal--Metal Building (sq. ft.)	33,426	$3.40	113,648	ASMI
Buildings: EW Rectifier Room, EW Office, SX Rectifier Room & SX Office	0.03	Demolition & Removal--Masonry Block Building (sq. ft.)	1,902	$3.50	6,657	ASMI

MINE CLOSURE ELEMENT	ACRES	TASK (units)	UNITS	UNIT COST	TOTAL	REFERENCE
Buildings (all)		Drill & Blast Concrete Pads (sq. ft.)	35,328	$1.00	35,328	Internal
Building Foundation Soil Cover (0.811 acres)		Material Haulage for Backfill; 2000' one way (cu. yd.)	2,617	$1.03	2,696	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	2,617	$0.44	1,151	ASMI
		Ripping & Scarification (acre)	17.03	$120.00	2,044	Internal
		Revegetation Cost--Disc (acre)	17.03	$607.00	10,337	ASMI
Old Equipment Lube Area		Soil Evacuation & Treatment (cu. yd.)	140	$65.00	9,100	Internal
Buildings and Ancillary Facilities Subtotal	**17.03**				**$ 180,961**	
Pipelines and Electrical Systems:						
HDPE Pipelines		Burial within Heaps (Lump Sum Estimate)			10,000	
Electrical Removal		Powerline Removal	5	$10,000.00	50,000	ASMI
		Transformer Oil Removal (gal.)	200	$0.15	30	ASMI
Pipelines and Electrical Systems Subtotal					**$ 60,030**	
Roads:						
Pre-1986 Disturbance Roads		Use for PMLU			-	
Exploration Road RD-1 (0.50 acres), Exploration Road RD-2 (0.33 acres), Exploration Road RD-3 (0.20 acres), Exploration Road RD-4 (1.04 acres), Access Road RD-5 east of Primary Water Tank (3.49 acres), Access Road RD-6 north of Primary Water Tank (1.35 acres) and Access Road RD-7 northeast of Crusher Area (3.73 acres)	10.63	Ripping & Scarification (acre)	10.63	$120.00	1,276	Internal
		Regrading & Topsoiling Cost (lineal feet); Roads with side slopes < 30%; assumed at 12' width	38,587	$1.70	65,598	ASMI
		Revegetation Cost--Disc (acre)	10.63	$607.00	6,452	ASMI
Roads Subtotal	**10.63**				**$ 73,326**	
Miscellaneous Areas:						

MINE CLOSURE ELEMENT	ACRES	TASK (units)	UNITS	UNIT COST	TOTAL	REFERENCE
Material Storage Area ("Bone Yard")	10.08	Ripping & Scarification (acre)	10.08	$120.00	1,210	Internal
		Revegetation Cost--Disc (acre)	10.08	$607.00	6,119	ASMI
		Cleanup (Lump Sum)			5,000	Internal
Crusher Area	8.7	Material Haulage for Backfill; 2000' one way (cu. yd.)	7,017	$1.03	7,228	ASMI
		Regrading & Topsoiling Cost (cu. yd.)	7,017	$0.44	3,087	ASMI
		Ripping & Scarification (acre)	8.70	$120.00	1,044	Internal
		Revegetation Cost--Disc (acre)	8.70	$607.00	5,281	ASMI
Primary Water Tank		Use for PMLU			-	Internal
Acid Tank (20,000 gal.)		Salvage @ $0 net			-	Internal
Fuel Tanks (2 @ 20,000 gal. each)		Salvage @ $0 net			-	Internal
Vadose Zone & Monitor Well Closure (20)		Plug to ADWR Standards (wells)	20	$500.00	10,000	Internal
Shaft Closure (6)		Dozer push plugging to ASMI Standards (shafts)	6	$2,000.00	12,000	Internal
Waste Oil Tank		Oil Separation & Removal (Lump Sum)			3,000	
Growth Media Stockpiles (GM-A, GM-B, GM-C, GM-D, GM-E & GM-F)	12.01	Ripping & Scarification (acre)	12.01	$120.00	1,441	Internal
		Revegetation Cost--Disc (acre)	12.01	$607.00	7,290	ASMI
Miscellaneous Areas Subtotal	**30.79**				**$ 62,700**	
Subtotal Closure Cost for All Areas	**748.26**				**$ 1,457,301**	
Administrative Costs:						
		Contingency		10%	145,730	ASMI
		General Mobilization/Demobilization		1%	14,573	ASMI
		Indirect Costs		2%	29,146	ASMI
		Contractor Profit		10%	145,730	ASMI
		Contract Administration		4%	58,292	ASMI
Administrative Costs Subtotal					**$393,471**	

MINE CLOSURE ELEMENT	ACRES	TASK (units)	UNITS	UNIT COST	TOTAL	REFERENCE
Total Closure Costs (rounded)					**$ 1,850,000**	
Salvage:						
Electrowinning Plant Cathodes		2500 Stainless Steel Cathodes @ 150 lb. each = 375,000 lb.		($0.86)	(322,500)	Internal
Electrowinning Plant Anodes		2500 Lead Anodes @ 145 lb. each = 362,500 lb.		($1.01)	(366,125)	Internal
Stainless Steel Scrap (pumps, valves & piping)		33,380 lb.		($0.86)	(28,700)	Internal
Copper Hanger Bars		5000 @ 7 lb. each = 35,000 lb.		($2.50)	(87,500)	Internal
Copper Bus Bars		6,000 lb.		($2.50)	(15,000)	Internal
Transformers, net of transportation		Transformer Salvage--non-PCB Units (KVA)	10,000	($3.00)	(30,000)	Internal
Undisturbed Land		350 acres		($4,750.00)	(1,662,500)	Internal
Salvage Subtotal (rounded)					**($2,512,000)**	
Net Closure Cost (rounded)					**($662,000)**	

24.0 HUMAN RESOURCES AND ADMINISTRATION

Human Resource Objectives

The objective of the Human Resource (HR) policies to be implemented by Nord will be to attract a highly skilled, stable and motivated workforce. Nord recognizes that, to meet these objectives, HR policies must be implemented, during the start-up phase.

The objectives of these HR policies are:

- Attract a stable workforce comprised of skilled workers.
- Provide for a highly flexible workforce to maximize employee utilization and control the number of employees.
- Motivate the employees to learn additional skills to increase their personal contribution and worth to the project.

Nord will achieve these goals by implementing the following:

- Will endeavour to hire operational personnel from the area. There is a sufficiently large population base, in the communities near the mine, to meet personnel requirements.
- Nord will exploit all available technology that facilitates employees controlling and monitoring process systems and stationary equipment remotely. Reducing onsite manpower requirements is a priority for Nord.
- Nord will consider the implemention of a “Gain Share Plan” with employee participation, i.e., a policy whereby workers share economically in improvements in actual costs versus budgeted costs, with cost saving shared between the employee and the employer.
- Nord will review the training programs necessary to implement a policy denoted as “Lines of Progression”. This policy provides for economic recognition of employees who assimilate and demonstrate their ability to successfully perform additional job functions. Compensation is based on an employee’s ability to perform; this permits and motivates employees to assimilate the knowledge to perform additional tasks. Wage scales/ranges are defined by demonstrated skill levels; job descriptions are defined by all activities required for a given function. For example, a SX-EW Operator could have demonstrated competency in all facets of the following process sections; Crushing, Stacking, Leaching, Stripping, Refining, etc. Remuneration is based on ability to perform the various tasks associated with these process sections and concomitantly, the employee can be directed to perform any function covered by her/his current level of compensation including laboring if required.

- Shift Rotation: All employees, with positions not requiring shift rotations, will work a 5x2 rotation, with on-call status during weekends. All employees, with positions requiring shift rotations will work a 7 day on/off rotation of 12 hour shifts.

25.0 CAPITAL COST

25.1 INTRODUCTION

The capital cost estimate is based on quotations received by BETA from NORD from manufacturers, as well as from trade publications, Machinery Trader, and historical data from current operations. The cost estimate is considered accurate to within +/- 15% at the summary level and is expressed in US dollars.

25.2 CAPITAL COST ESTIMATE

The plant capital costs are comprised of estimates from several different sources including the following:

1) Engineering firms
2) Independent consultants
3) Equipment Vendors
4) General Contractors
5) Summary data completed by Nord

A summary of capital costs is presented in Table 25.2-1.

Table 25.2-1: Nord - Johnson Camp, Capital Cost Schedule

Cost (000)	Total
Initial plant capital cost, $	$ 26,684
Mine software, hardware & surveying equipment, $	$ 125
Environmental monitoring, $	$ 240
Removal of Pad 1 liner, $	$ 620
New leach pads, $	$ 11,540
Infrastructure for conveyor relocation, $	$ 100
Mine contractor demob, $	$ 375
Plant sustaining capital, $	$ 450
Total Capital, $	$ 40,134

Cost (000)	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Initial plant capital cost, $	$26,684																
Mine software, hardware & surveying equipment, $	$ 125																
Environmental monitoring, $	$ 240																
Removal of Pad 1 liner, $	$ 620																
New leach pads, $			$1,500	$1,500				$8,540									
Infrastructure for conveyor relocation, $		$ 100															
Mine contractor demob, $																	$ 375
Plant sustaining capital, $	-	$ 10	$ 15	$ 20	$ 25	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 20	$ 10
Total Capital, $	$27,669	$ 110	$1,515	$1,520	$ 25	$ 35	$ 35	$8,575	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 20	$ 385

Table 25.2-2: Initial Plant Capital Cost Summary

Cost $	
Direct costs	
Existing SXEW	
SX	$1,162,000
Tank farm	$1,188,000
EW	$1,585,000
EW expansion	$2,021,000
Construction SXEW	$1,118,000
Leach pond, pumps	$2,330,000
Crushing, conveying	$11,935,000
Total Direct	$21,342,000
Indirect	
Engineering, requisition preparation	$440,000
Construction indirects	$354,000
Spare parts	$302,000
Professional services	$14,000
Start-up, commissioning	$156,000
Other costs	$1,049,000
Mine contractor mobilization	$375,000
First fill	$546,000
Sales/gross receipts tax allowance	$202,000
Total Indirect	$3,438,000
Total direct and indirect	$24,780,000
Contingency	$1,904,000
TOTAL DIRECT, INDIRECT & CONTINGENCY	$26,684,000

25.2.1 Solution Ponds and Leach Pads

Capital has been included to complete the last pond remediation and to install the associated pumps. The pond remediation will be partially contracted out.

25.2.2 Crushing, Conveying and Agglomeration

Nord has purchased the primary crusher station from Newmont and has obtained estimates for the remainder of the circuit from pertinent vendors. The main items to be purchased are two new Sandvik H6800 cone crushers and the new conveyors. The capital cost also includes a new drum agglomerator.

25.2.3 SX/EW Plant Modifications

An independent engineering firm, the original designers of the Johnson Camp facility, had prepared an estimate in November 1998 that has served as the basis for the refurbishment and remediation costs in the SX/EW area. The Summo feasibility study had subsequently supplemented this estimate with an additional work scope developed in concert with Nord Copper. For the Nord updated feasibility study, the engineering firm had reviewed the previous work and compared it to the list of items included in the Nord capital budget. BETA believes that the necessary items are included in the capital budget and that the budget is realistic. Recent quotations have been obtained for the major new equipment items, such as the electrolyte filters and the new Lightning mixers. It is the intent of Nord to perform most of the modification work using their own employees, as they have done with the work completed to date.

25.2.4 New EW Plant

The scope of work for the new EW expansion had been previously reviewed and BETA believes that the Nord budget is realistic. Recent budgetary quotations have been received for the major new equipment items such as polymer cells, cathodes, new EW cell house crane, boiler and heat exchanger.

25.2.5 Water Supply

Nord has already increased the capacity of the water system to 600 gpm. Nord obtained a budgetary quotation from a well drilling contractor for a new 150-gpm well for $150,000, however as this is not deemed to be necessary for the project, this has not been included in the capital cost.

25.2.6 Environmental

Table 25.2.3 lists the environmental items to be completed in year zero.

Table 25.2.3: Environmental Expenses

Item	Year 0
Hydrologic study plan (APP) Note 1	$10,000
Additional monitoring wells	$100,000
Reclamation plan	$10,000
Air quality permit	$40,000
Environmental consultant	$30,000
Toxic release inventory	$4,000
Misc. professional services	$15,000
Contingency	$31,000
Total	$240,000

25.2.7 Staffing

The staffing costs are included in operating cost.

25.2.8 Pad 1 Liner

Based on the current mining contract bid and past experience on removing liners on the old heaps, Nord has estimated the cost of removing the Pad 1 liner and associated covering ore at $ 620,000. This estimate is believed to be valid.

25.3 Sustaining Capital

25.3.1 New Leach Pad and Ponds

The capital costs for the solution pads and initial leach pads were estimated by an independent engineering firm in 2000. These costs were reviewed against contractor quotations for earthwork and liner installation values and determined to be in line with industry values. BETA concurs with these determinations. Details of the cost estimate are available for inspection at Nord Resources Corporation's head office.. The independent engineering firm has also provided the estimated cost for quality assurance and certification of the solution ponds, which is included in construction management costs. Nord obtained new budgetary quotations for the installation of 60-mil and 80-mil HDPE liners. Appropriate escalators were then applied to the contractor costs in the engineering estimate, plus the new liner prices, to arrive at current costs for the new pad, as presented in Table 25.3-1.

Table 25.3.1: New Pad and Pond Capital Cost Estimate

Year		2	3	7
Pads				
Earthworks		$ 375,000	$ 375,000	$1,730,000
Synthetic liner		$ 825,000	$ 825,000	$4,040,000
Subtotal		$1,200,000	$1,200,000	$5,770,000
Construction management and indirects		$ 75,000	$ 75,000	$290,000
Contingency	15%	$ 225,000	$ 225,000	$910,000
Total pad		$1,500,000	$1,500,000	$6,970,000
Ponds				
Earthworks				$700,000
Synthetic liner				$300,000
Subtotal				$1,000,000
Pipe lines to and from plant				$230,000
Construction management and indirects				$50,000
Contingency	15%			$190,000
Subtotal				$1,470,000
Total		$1,500,000	$1,500,000	$8,440,000

25.3.2 Environmental

Costs of annual environmental monitoring and remediation were provided by Nord through their consultant, Dale Deming, P.E. The data has been reviewed by BETA and is incorporated in this feasibility study.

26.0 ECONOMIC ANALYSIS

This section provides an economic analysis of the Johnson Camp Mine project, incorporating data developed elsewhere in this feasibility. For purposes of this analysis, a Base Case cash flow model was prepared against which the effects of changes in the key input variables have been evaluated.

All monetary values are expressed in U.S. dollars.

26.1 REVENUE PARAMETERS

26.1.1 Metal Prices and Sales Requirements

For the economic analysis, a three-year trailing average (COMEX) base copper price of $2.45 per pound is used. BETA believes this is a reasonable approach for this technical report. Copper prices have remained above $3.00 throughout the writing of this study.

26.1.2 Production Schedule and Ore Grades

A detailed analysis of the production schedule and mined grade by bench by pit and by year is provided in Section 21. The production schedule is summarized in Table 26-1.

Table 26-1
Production Schedule

	ORE TO CRUSHING PLANT				LOW GRADE ORE RUN OF MINE TO PADS				WASTE	STRIP	Total
	TONS	TCU	LBS CU	LBS CU	TONS	TCU	LBS CU	LBS CU	TONS	RATIO	Tons
	(000s)		Contained	Recoverable	(000s)		Contained	Recoverable	(000s)		(000s)
YEAR 1	3,331	0.427	28,435	22,154	574	0.099	1,142	571	472	0.12	4,377
YEAR 2	2,590	0.564	29,209	22,369	29	0.107	63	31	199	0.08	2,818
YEAR 3	3,956	0.366	28,949	22,707	590	0.117	1,386	693	1,170	0.26	5,717
YEAR 4	4,628	0.318	29,439	23,041	531	0.068	718	359	7,162	1.39	12,320
YEAR 5	4,524	0.301	27,220	21,594	1,834	0.098	3,611	1,806	7,911	1.24	14,269
YEAR 6	4,788	0.291	27,847	21,628	1,712	0.104	3,544	1,772	5,174	0.80	11,674
YEAR 7	4,871	0.316	30,770	24,279	637	0.113	1,441	721	1,888	0.34	7,396
YEAR 8	4,100	0.383	31,366	24,707	242	0.121	586	293	651	0.15	4,993
YEAR 9	3,396	0.457	31,009	24,789	188	0.112	422	211	277	0.08	3,862
YEAR 10	3,376	0.481	32,491	24,964	38	0.095	72	36	83	0.02	3,497
YEAR 11	3,355	0.477	32,025	24,949	32	0.158	101	51	164	0.05	3,550
YEAR 12	4,319	0.374	32,275	24,181	769	0.106	1,637	819	5,201	1.02	10,289
YEAR 13	4,411	0.353	31,144	23,436	1,489	0.105	3,127	1,564	8,636	1.46	14,537
YEAR 14	4,682	0.341	31,967	24,064	918	0.102	1,872	936	6,508	1.16	12,108
YEAR 15	4,360	0.372	32,443	24,245	694	0.109	1,510	755	3,130	0.62	8,183
YEAR 16	2,007	0.330	13,256	9,944	416	0.114	946	473	596	0.25	3,019

26.1.3 Metal Recoveries

Recoveries for high grade ore have been calculated by bench in the block model. Recovery varies by rock type, as described in detail in Section 18.

Table 26-2
Recovery (Cumulative Percent)

Ore type	Burro Pit Diabase	Cu Chief Diabase	Shale Bolsa	Abrigo
	81.0%	74.0%	76.0%	79.0%

For economic analysis, it is assumed that 90% of the recoverable copper placed on the leach pad in the first six months of a year will be recovered in the same year as stacked. It is additionally assumed that 75% of the recoverable value of material stacked in months 6-9 of a year will be recovered in the year stacked. This results in 6,818,000 lbs of recoverable copper that are left in inventory from newly stacked ore at the end of year 1. BETA assumes that this inventory does not fluctuate over the life of the project, and is recovered in year 16.

26.2 INVESTMENT PARAMETERS

26.2.1 Initial Capital Cost

This report provides details of the initial capital cost required to commence production. Table 26-2 summarizes these costs inclusive of freight.

Table 26-2

Initial Capital Cost Summary ($US)

Total Direct	$21,342,000
Total Indirect	$3,438,000
Total direct and indirect	$24,780,000
Contingency	$1,904,000
TOTAL DIRECT, INDIRECT & CONTINGENCY	$26,684,000

26.2.2 Working Capital Requirements

Working capital is estimated at $683,000, to be repaid at the end of mine life.

26.2.3 Salvage Value and Closure Costs

The financial projection assumes a salvage value of the mining, process and service equipment of $2,512,000.

Mine closure costs have been estimated to total $1,850,000.

These costs are detailed in section 23-5 of this report.

26.3 OPERATING COSTS

26.3.1 Marketing Cost

A one percent (1%) marketing and delivery charge as applied to all copper sales.

26.3.2 Royalties and Concession Fees

The Arimetco royalty cost is calculated as $0.02 per pound produced when the copper price is equal to or greater than $1.00 per pound, subject to a cap of $1 million in aggregate.

26.3.3 Production Costs

Operating cost per ton of ore is presented in Table 26-3.

Process cost is determined per pound of copper produced.

Table 26-3
Operating Cost Summary

Production	Operating Costs					
	G & A	Ore	Waste	Crush & Stack	Process	Environment
(lb/year)	$US/ton Ore	$US/ton Ore	$US/ton Waste	$US/ton Ore	$US/lb Cu	$US/ton Ore
25,000,000	0.350	1.509	1.603	0.637	0.285	0.035

26.4 TAX PARAMETERS

The Johnson Camp Project is located in Cochise County, Arizona. As a result the operation is subject to the taxes of Cochise County, the State of Arizona, and the United States of America. Tax issues in the US are often complex and require legal and accounting advice. The paragraphs below briefly describe the taxes levied on the operation.

Property Taxes

The State of Arizona provides for a central assessment of value for mining operations. The Arizona Department of Revenue performs an annual determination of valuation. The valuation can be based on established market value, the value of tangible assets, or on discounted cash flow. The Arizona Department of Revenue then reports the cash value of the property to the county assessor. Assessed value is set at 25% of full cash value and current property tax rate for Johnson Camp mine's tax jurisdiction is $10.71 per $100 of assessed value.

Severance Taxes

Metal mines in Arizona pay a severance tax based on the value of mineral extracted. The severance tax is represented by 2.5% of the net severance base. The severance base is 50% of the difference between the gross value of production and the cost of production.

Income Taxes

Income taxes are payable to both the federal and state governments. Current federal tax rate is at 35% of taxable income and the current Arizona income tax rate is at 6.968% of taxable income. The income taxes calculated for the financial analysis do not reflect the impact, if any, of Nord Resources Corporations federal and state net operating losses.

26.5 FINANCING PARAMETERS

BETA prepared the cash flow model for the Johnson Camp project on an all-equity basis.

26.6 BASE CASE FINANCIAL RESULTS

The summarized base case project cash flow schedule is shown in Table 26-4. Net present values of the cash flows are shown using discount rates from zero to twenty percent. The internal rate of return (IRR), also referred to as the discounted cash flow rate of return (dcfroi) is also shown.

Table 26-4
Cash Flow - Base Case
Johnson Camp Mine

Summary	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Production - Recoverable lbs Cu on heaps		22,725	22,400	23,400	23,400	23,400	23,400	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	10,417
High Grade Ore mined, tons (000)		3,331	2,590	3,956	4,628	4,524	4,788	4,871	4,100	3,396	3,376	3,355	4,319	4,411	4,682	4,360	2,007
Low Grade Ore mined, tons (000)		574	29	590	531	1,834	1,712	637	242	188	38	32	769	1,489	918	694	416
Total Ore mined, tons(000)		3,905	2,619	4,546	5,159	6,358	6,500	5,508	4,342	3,585	3,414	3,387	5,089	5,900	5,600	5,053	2,423
Ore grade, %Total copper		0.379	0.559	0.334	0.292	0.242	0.241	0.292	0.368	0.438	0.477	0.474	0.333	0.290	0.302	0.336	0.293
Waste mined, tons(000)		472	199	1,170	7,162	7,911	5,174	1,888	651	277	83	164	5,201	8,636	6,508	3,130	596
Strip Ratio, Waste/Ore		0.12	0.08	0.26	1.39	1.24	0.80	0.34	0.15	0.08	0.02	0.05	1.02	1.46	1.16	0.62	0.25
Copper produced including from existing dumps, lbs(000)		18,183	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	17,235
Revenue																	
Copper Price, $/lb		$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45
Gross Revenue, $(000)		$ 44,547	$ 61,250	$ 61,250	$ 61,250	$ 61,250	$ 61,250	$ 61,250	$ 61,250	$ 61,250	$ 61,250	$ 61,250	$ 61,250	$ 61,250	$ 61,250	$ 61,250	$ 42,226
Net Revenue after delivery costs, $(000)		$ 44,102	$ 60,638	$ 60,638	$ 60,638	$ 60,638	$ 60,638	$ 60,638	$ 60,638	$ 60,638	$ 60,638	$ 60,638	$ 60,638	$ 60,638	$ 60,638	$ 60,638	$ 41,803
Operating Costs		$ 16,197	$ 14,730	$ 20,522	$ 31,611	$ 35,148	$ 31,185	$ 23,999	$ 19,279	$ 16,791	$ 16,129	$ 16,192	$ 28,178	$ 35,353	$ 31,488	$ 24,809	$ 11,600
Operating costs,excluding delivery, $(000)		$ 16,035	$ 14,583	$ 20,317	$ 31,295	$ 34,796	$ 30,873	$ 23,759	$ 19,086	$ 16,623	$ 15,968	$ 16,030	$ 27,896	$ 35,000	$ 31,173	$ 24,561	$ 11,484
Operating cost $/lb saleable copper		$ 0.88	$ 0.58	$ 0.81	$ 1.25	$ 1.39	$ 1.23	$ 0.95	$ 0.76	$ 0.66	$ 0.64	$ 0.64	$ 1.12	$ 1.40	$ 1.25	$ 0.98	$ 0.67
Property and Severance Tax		$ 804	$ 1,031	$ 962	$ 825	$ 780	$ 828	$ 917	$ 976	$ 1,008	$ 1,018	$ 1,019	$ 872	$ 783	$ 830	$ 913	$ 841
Operating Cash Flow, $(000)		$ 27,264	$ 45,024	$ 39,358	$ 28,517	$ 25,061	$ 28,936	$ 35,962	$ 40,576	$ 43,006	$ 43,652	$ 43,589	$ 31,870	$ 24,855	$ 28,635	$ 35,164	$ 29,478
$/lb saleable copper		$ 1.50	$ 1.80	$ 1.57	$ 1.14	$ 1.00	$ 1.16	$ 1.44	$ 1.62	$ 1.72	$ 1.75	$ 1.74	$ 1.27	$ 0.99	$ 1.15	$ 1.41	$ 1.71
Closure Costs, $(000)																500	1,350
Salvage Value																	2,512
Capital Costs																	
Capital costs, $(000)	27,669	110	1515	1520	25	35	35	8575	35	35	35	35	35	35	35	20	385
Changes in working capital, $(000)		683															(683)
Total Capital & Working Capital cost, $(000)	27,669	793	1,515	1,520	25	35	35	8,575	35	35	35	35	35	35	35	20	(298)
Pre-income tax cash flow from copper operations, $(000)	(27,669)	27,274	44,540	38,801	29,318	25,806	29,729	28,303	41,517	43,979	44,634	44,573	32,707	25,603	29,430	35,556	31,780
Cumulative pre-income tax cash flow from copper operations, $	(27,669)	(395)	44,144	82,945	112,263	138,069	167,798	196,102	237,618	281,598	326,232	370,805	403,511	429,114	458,544	494,100	525,880
Total property pre-income-tax cash flow , $(000)		$ 525,880															
Pre-Income Tax Net Present Value																	
using a 0% discount rate, $ (000)		$ 525,880															
using a 5% discount rate, $ (000)		$ 330,561															
using a 8% discount rate, $ (000)		$ 257,465															
using a 10% discount rate, $ (000)		$ 220,278															
using a 15% discount rate, $ (000)		$ 154,096															
using a 20% discount rate, $ (000)		$ 112,146															
Pre-Income-Tax Internal Rate of Return		119%															
Pre-income tax cash flow from copper operations, $(000)	$ (27,669)	$ 27,274	$ 44,540	$ 38,801	$ 29,318	$ 25,806	$ 29,729	$ 28,303	$ 41,517	$ 43,979	$ 44,634	$ 44,573	$ 32,707	$ 25,603	$ 29,430	$ 35,556	$ 31,780
Income tax	$ -	$ 15,524	$ 11,341	$ 9,893	$ 6,058	$ 5,033	$ 6,581	$ 9,369	$ 11,218	$ 12,353	$ 12,929	$ 13,195	$ 8,788	$ 6,027	$ 7,486	$ 10,207	$ 9,270
After tax cashflow	$ (27,669)	$ 11,749	$ 33,199	$ 28,908	$ 23,259	$ 20,773	$ 23,148	$ 18,934	$ 30,299	$ 31,626	$ 31,705	$ 31,377	$ 23,919	$ 19,576	$ 21,944	$ 25,350	$ 22,510
Cummulative after-tax cashflow	(27,669)	(15,920)	17,279	46,187	69,446	90,219	113,367	132,302	162,601	194,227	225,932	257,309	281,228	300,804	322,748	348,098	370,607
Total property after-tax cash flow , $(000)		$ 370,607															
After-Tax Net Present Value																	
using a 0% discount rate, $ (000)		$ 370,607															
using a 5% discount rate, $ (000)		$ 229,275															
using a 8% discount rate, $ (000)		$ 176,437															
using a 10% discount rate, $ (000)		$ 149,583															
using a 15% discount rate, $ (000)		$ 101,879															
using a 20% discount rate, $ (000)		$ 71,759															
After-Tax Internal Rate of Return		77%															

Table 26-5
Net Revenue Schedule
Johnson Camp Mine

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Copper produced from new mined ore, (000) lbs	15,908	22,400	23,400	23,400	23,400	23,400	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	10,417
Copper produced from existing dumps, (000) lbs	2,275	2,600	1,600	1,600	1,600	1,600										
Copper produced from new inventory, (000) lbs																6,818
Total copper produced, (000) lbs	18,183	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	17,235
Commodity Price																
Copper, $/lb	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45	$ 2.45
Gross Revenue, $	$44,547,125	$61,250,000	$61,250,000	$61,250,000	$61,250,000	$61,250,000	$61,250,000	$61,250,000	$61,250,000	$61,250,000	$61,250,000	$61,250,000	$61,250,000	$61,250,000	$61,250,000	$42,225,750
Delivery cost, $	$ 445,471	$ 612,500	$ 612,500	$ 612,500	$ 612,500	$ 612,500	$ 612,500	$ 612,500	$ 612,500	$ 612,500	$ 612,500	$ 612,500	$ 612,500	$ 612,500	$ 612,500	$ 422,258
Net Revenue, $	$44,101,654	$60,637,500	$60,637,500	$60,637,500	$60,637,500	$60,637,500	$60,637,500	$60,637,500	$60,637,500	$60,637,500	$60,637,500	$60,637,500	$60,637,500	$60,637,500	$60,637,500	$41,803,493

Table 26-6
Capital Cost Schedule
Johnson Camp Mine

Cost (000)	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16		Total
Initial plant capital cost, $	$26,684																		$ 26,684
Mine software, hardware & surveying equipment, $	$ 125																		$ 125
Environmental monitoring, $	$ 240																		$ 240
Removal of Pad 1 liner, $	$ 620																		$ 620
New leach pads, $			$1,500	$1,500				$8,540											$ 11,540
New ponds and pipe, $																			
Infrastructure for conveyor relocation, $		$ 100																	$ 100
Mine contractor demob, $																	$ 375		$ 375
Plant sustaining capital, $	-	$ 10	$ 15	$ 20	$ 25	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 20	$ 10		$ 450
Total Capital, $	$27,669	$ 110	$1,515	$1,520	$ 25	$ 35	$ 35	$8,575	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 35	$ 20	$ 385		$ 40,134

Table 26-7
Operating Costs ($,000)
Johnson Camp Mine

Operating Costs	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Mining Ore,	5,893	3,953	6,860	7,785	9,594	9,809	8,311	6,552	5,409	5,152	5,110	7,679	8,904	8,451	7,625	3,656
Mining Waste,	756	319	1,876	11,480	12,682	8,293	3,026	1,044	444	133	262	8,337	13,844	10,432	5,017	956
Crushing and conveying, $	2,122	1,650	2,520	2,948	2,882	3,050	3,103	2,611	2,163	2,151	2,137	2,751	2,810	2,982	2,777	1,278
Low Grade Ore Placement, $	57	3	59	53	183	171	64	24	19	4	3	77	149	92	69	42
Leaching, $	5,182	7,125	7,125	7,125	7,125	7,125	7,125	7,125	7,125	7,125	7,125	7,125	7,125	7,125	7,125	4,912
Environmental	137	92	159	181	223	228	193	152	125	120	119	178	207	196	177	85
General and Administrative (including mine admin.), $	1,367	917	1,591	1,806	2,225	2,275	1,928	1,520	1,255	1,195	1,185	1,781	2,065	1,960	1,769	500
Delivery, $	227	224	234	234	234	234	250	250	250	250	250	250	250	250	250	172
Arimetco Royalty, $	455	448	98													
Total Operating Cost, $	16,197	14,730	20,522	31,611	35,148	31,185	23,999	19,279	16,791	16,129	16,192	28,178	35,353	31,488	24,809	11,600

Table 26-8
Unit Operating Costs
Johnson Camp Mine

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Copper Operations Operating Unit Costs																
Mining Ore, $/ton	$1.509	$1.509	$1.509	$1.509	$1.509	$1.509	$1.509	$1.509	$1.509	$1.509	$1.509	$1.509	$1.509	$1.509	$1.509	$1.509
Mining Waste, $/ton	$1.603	$1.603	$1.603	$1.603	$1.603	$1.603	$1.603	$1.603	$1.603	$1.603	$1.603	$1.603	$1.603	$1.603	$1.603	$1.603
Crushing and conveying, $/ ore ton	$0.637	$0.637	$0.637	$0.637	$0.637	$0.637	$0.637	$0.637	$0.637	$0.637	$0.637	$0.637	$0.637	$0.637	$0.637	$0.637
Low Grade Ore Placement, $	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100
Leaching, $ / lb produced	$0.285	$0.285	$0.285	$0.285	$0.285	$0.285	$0.285	$0.285	$0.285	$0.285	$0.285	$0.285	$0.285	$0.285	$0.285	$0.285
Environmental	$0.035	$0.035	$0.035	$0.035	$0.035	$0.035	$0.035	$0.035	$0.035	$0.035	$0.035	$0.035	$0.035	$0.035	$0.035	$0.035
General and Admin (including mine), $/ton	$0.350	$0.350	$0.350	$0.350	$0.350	$0.350	$0.350	$0.350	$0.350	$0.350	$0.350	$0.350	$0.350	$0.350	$0.350	$0.350
Delivery, $/lb	$0.010	$0.010	$0.010	$0.010	$0.010	$0.010	$0.010	$0.010	$0.010	$0.010	$0.010	$0.010	$0.010	$0.010	$0.010	$0.010
Arimetco Royalty, $/lb	$0.020	$0.020														

26.7 SENSITIVITY ANALYSIS – AFTER TAX

BETA ran several after-tax cash flow sensitivity analyses to determine the project's net present value and internal rate of return for changes of plus and minus fifteen percent to:

- Revenue (copper price),
- Capital costs, and
- Operating cost.

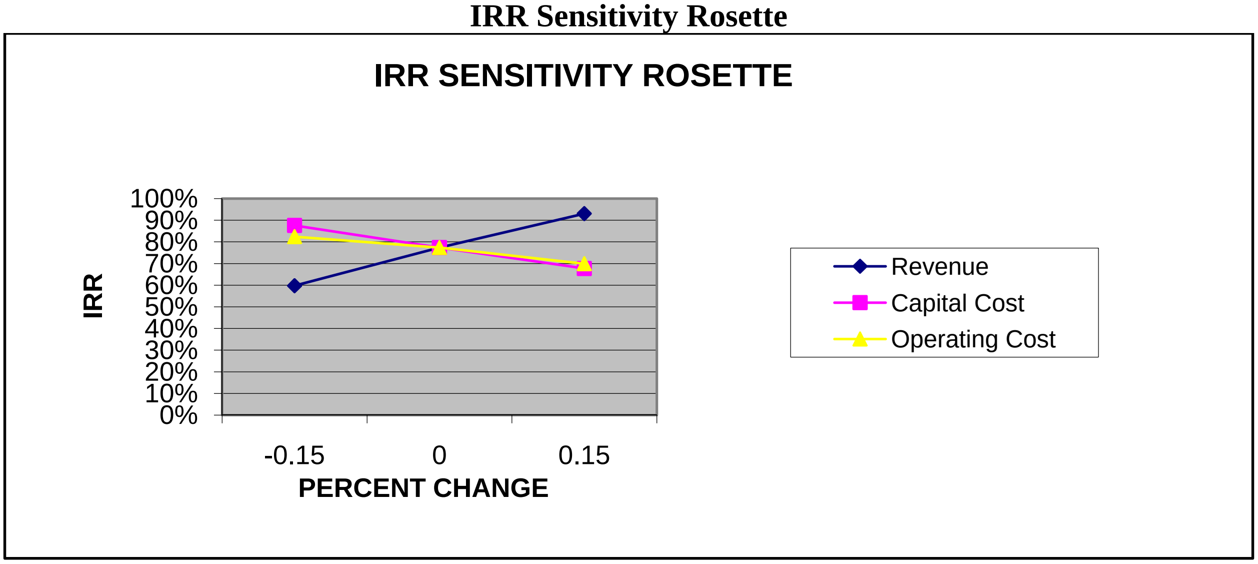
Table 26-9 shows the impact of changes of plus and minus 15% to revenue, capital and operating cost to the internal rate of return. . This table is summarized as follows: The IRR of the base case is 77%. A 15% decrease in capital cost results in an IRR of 88%, whereas an increase of 15% in capital results in an IRR of 68%. A 15% decrease in operating cost results in an IRR of 82%, whereas an increase of 15% in operating cost results in an IRR of 70%. A decrease in copper price of 15% from \$2.45 to \$2.13 results in an IRR of 60%, whereas an increase of 15% to \$2.82 results in an IRR of 93%.

These results are shown graphically in Figure 26-1.

Table 26-9
Sensitivity Analysis – IRR

IRR			
Percent Change	-15%	0%	15%
REVENUE	60%	77%	93%
CAPITAL	88%	77%	68%
OPERATING COST	82%	77%	70%

Figure 26-1
IRR Sensitivity Rosette



BETA performed a further sensitivity analysis reflecting project economics at higher copper prices approaching current price levels. Results of runs including copper prices of $3.19 and $3.55 per pound (+30% and +45%) are shown below in Table 26-10 and Figure 26-2.

Table 26-10
Sensitivity Analysis – IRR versus Copper Price

	IRR				
Cu Price	$ 2.13	$ 2.45	$ 2.82	$ 3.19	$ 3.55
REVENUE	60%	77%	93%	108%	122%

Figure 26-2
IRR Sensitivity to Copper Price

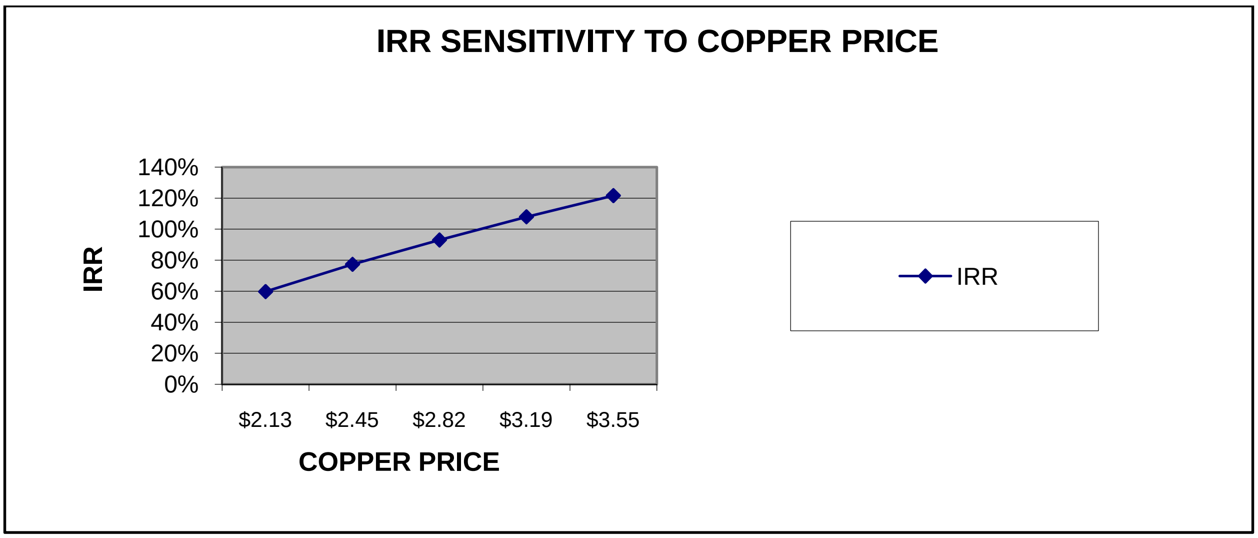


Table 26-11 and Figure 26-3 show the impact of changes of plus and minus 15% to revenue, capital and operating cost to the net present value at an 8% discount rate. The table is summarized as follows: The NPV of the base case at 8% discount rate is $176.4 million. A 15% decrease in capital cost results in an NPV of $180.3 million, whereas an increase of 15% in capital results in an NPV of $171.3 million. A 15% decrease in operating cost results in an NPV of $202.2 million, whereas an increase of 15% in operating cost results in an NPV of $149.5 million. A decrease in copper price from $2.45 to $2.13 results in an NPV of $117.7 million, whereas an increase to $2.82 results in an NPV of $242.7 million.

Table 26-11
Sensitivity Analysis – NPV

NPV @ 8%			
Percent Change	-15%	0%	15%
REVENUE	$ 117.7	$ 176.4	$ 242.7
CAPITAL	$ 180.3	$ 176.4	$ 171.3
OPERATING COST	$ 202.2	$ 176.4	$ 149.5

Figure 26-3
NPV Sensitivity Rosette

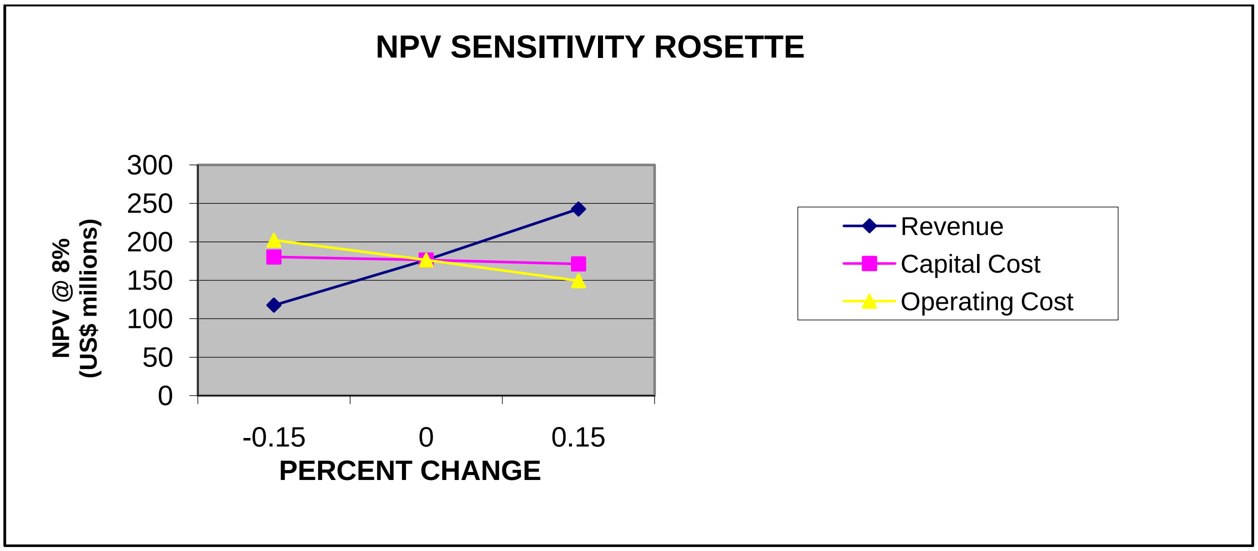


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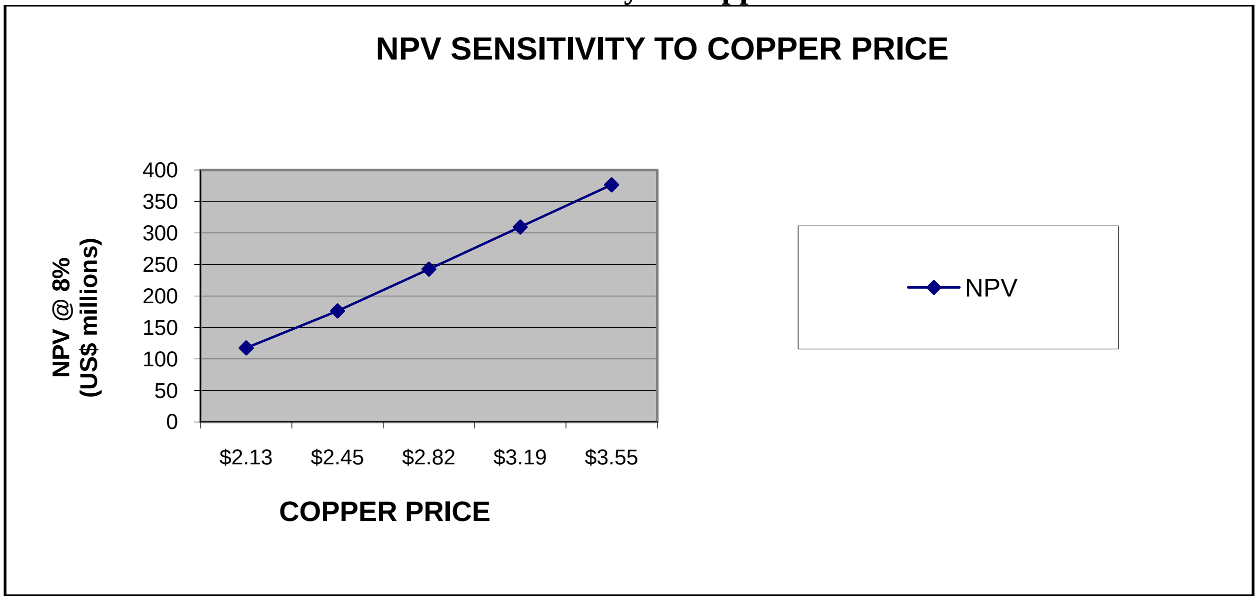
Table 26-12
Sensitivity Analysis – NPV versus Copper Price
($ millions)

NPV @ 8%					
Cu Price	$ 2.13	$ 2.45	$ 2.82	$ 3.19	$ 3.55
REVENUE	$ 117.7	$ 176.4	$ 242.7	$ 309.6	$ 376.6

Figure 26-4
NPV Sensitivity to Copper Price



26.8 PAYBACK

The payback period for this project is less than two years in the base case, after-tax analysis. None of the sensitivity runs had a payback period of over two years. Payback of capital was obtained in one year when a higher copper price was utilized.

Table 26-12
Sensitivity Analysis – Payback Period
(years)

Payback Period (Years)			
Percent Change	-15%	Base Case	15%
REVENUE	1.8	1.6	1.4
CAPITAL	1.4	1.6	1.8
OPERATING COST	1.5	1.6	1.7

27.0 DESIGN SPECIFICATIONS

Process Design Criteria, Nord Copper Johnson Camp Project

1.0 General			Nominal	Design	Code
	1.1 Description				
		The following criteria are for a plant capable of crushing 4,800,000 tons of copper ore to 80 percent minus 1-inch. The crushed ore will be agglomerated, then placed on existing heaps for the initial years and on a new pad for the remainder of the project. Typically, 25 million tons per year of cathode copper will be produced using the existing solvent extraction and electrowinning plant.			
	1.2 Source Codes				
	Code	Source			
	A	Client provided			
	B	Standard industrial practice			
	C	Vendor data			
	D	Assumption based on similar projects			
	E	Process calculations			
	F	ThyssenKrupp crushing plant proposal			
	G	BETA mine plan			
2.0 Process unit operations					
	The process will be comprised of the following unit processes:				
	1	Primary and secondary crushing			
	2	Acid agglomeration			
	3	Ore conveying and stacking			
	4	Heap leaching			
	5	Solvent extraction			
	6	Electrowinning			
3.0 Site data					
	3.1 Location				
		The project is located approximately 15 miles east of Benson, Arizona and less than two miles north of Interstate Highway 10 in Cochise County, Arizona			
4.0 Production					
	4.1 Ore				
	4.1.1 Total ore, tons (x ,000,000)			73.39	G
	4.1.1.1 Average grade, % Total Cu			0.335	G
	4.1.1.2 Nominal ore placed per month, tons			394	G
	4.1.2 Total High Grade ore, tons (x ,000,000)			62.7	G
	4.1.2.1 Average grade of High Grade, % Total Cu			0.374	G
	4.1.3.2 Nominal HG ore placed per month, tons			337	G
	4.1.3 Total Low Grade ore, tons (x ,000,000)			10.7	G
	4.1.3.1 Average grade of Low Grade, % Total Cu			0.106	G
	4.1.3.2 Nominal LG ore placed per month, tons			57	G
	4.1.4 Annual mine tonnage, dst x ,000,000				
	Year		LG	HG	
	1		0.57	3.33	G
	2		0.03	2.59	G

	3		0.59	3.96	G
	4		0.53	4.63	G
	5		1.83	4.52	G
	6		1.71	4.79	G
	7		0.64	4.87	G
	8		0.24	4.10	G
	9		0.19	3.40	G
	10		0.04	3.38	G
	11		0.03	3.35	G
	12		0.77	4.32	G
	13		1.49	4.41	G
	14		0.92	4.68	G
	15		0.69	4.36	G
	16		0.42	2.01	G
	4.2 Copper production		Nominal	Design	Code
	4.2.1 Total copper production (inc residual), lbs (x 000,000)			374,142	E
	4.2.2 Nominal copper production, lbs/month			2,083,333	E
	4.2.3 Nominal copper production, lbs/day			69,444	E
	4.3 Duration of ore placement on heaps, years			15	G
5.0 Crushing, conveying, agglomeration and stacking					
	The crushing circuit has the following unit operations or stages:				
	1	Dump pocket to be fed with mine trucks			
	2	Feeder for feeding primary crusher			
	3	Primary crusher			
	4	Coarse ore stockpile			
	5	Scalping screen			
	6	Secondary crusher, open circuit			
	7	Tertiary crusher, open circuit			
	8	Fines stockpile			
	9	Drum agglomerator			
	10	Conveying system to heap leach			
	11	Stacker at heap leach			
5.1 Crusher operating schedule (Varies annually)					
	5.1.1 Weeks per year		52	52	A
	5.1.2 Days per week		5	4 to 5	A
	5.1.3 Shifts per day		2	2 to 3	A
	5.1.4 Hours per shift		10	8 to 10	A
	5.1.5 Availability of scheduled hours				
		5.1.5.1Primary	90	85 to 90	D
		5.1.5.2 Secondary	80	80	D
		5.1.5.3 Tertiary	80	80	D
	5.1.6 Hours available per year				
		5.1.6.1Primary	4,680	3,744 to 5,304	E
		5.1.6.2 Secondary	4,160	3,328 to 4,992	E
		5.1.6.3 Tertiary	4,160	3,328 to 4,992	E
	5.2 Mine production				
	5.2.1 Mine ore, dst		4,100,000	4,800,000	A
	5.2.2 Plant feed rate, dstph		875	962	E
	5.2.3 Mine trucks, short tons			100	A

	5.2.4 Moisture, percent		3%	D
5.3 Primary crusher				
	5.3.1 Feed hopper			
	5.3.1.1 Method of feeding	Direct Dump by Trucks		A
	5.3.1.3 Oversize handling	Rock Breaker		F
	5.3.2 Primary feeder (recommendation)			
	5.3.2.1 Type	Apron		F
	5.3.2.2 Size	72 in x 58 ft.		F
	5.3.2.3 Feed rate, dstph	876	962	E
		Nominal	Design	Code
	5.3.3 Primary crusher			
	5.3.3.1 Type	Gyratory		F
	5.3.3.2 Size	42 in x 65 in		F
	5.3.3.3 Model	Superior		F
	5.3.3.4 Feed rate, dstph	657	721	E
	5.3.3.5 Open Side Setting (inches)	5.5	5.5	C
5.4 Coarse Ore Stockpile				
	5.4.1 Total storage, tons		40,000	A
	5.4.2 Live storage, tons		9,000	A
	5.4.3 Discharge		Feeders	
5.5 Secondary scalping screen				
	5.5.1 Type	Double Deck Vibrating		
	5.5.2 Size	8 ft. x 24 ft.		
	5.5.3 Deck opening, inches			
	5.5.3.1 Top		3	
	5.5.3.2 Bottom		1.5	
	5.5.4 Feed Rate, dstph	986	962	
	5.5.5 Percent of feed to fines	35	35	
5.6 Secondary Crusher (2 of)				
	5.6.1 Type	Heavy Duty Short Head		A
	5.6.2 Size/Model	7 ft		A
	5.6.3 Feed			E
	5.6.3.1 Source	Scalping Screen Oversize		A
	5.6.3.2 Feed Rate	641	800	E
	5.6.4 Closed Side Setting (inches)	0.75	0.75	A
5.7 Fine Ore Storage				
	5.9.1 Total storage, tons		40,000	A
	5.9.2 Live storage, tons		9,000	A
	5.9.3 Discharge		Feeders	A
5.8 Agglomeration		Nominal	Design	Code
	5.8.1 Type	Drum, acid resistant		A
	5.8.2 Model	FEECO, typical		A
	5.8.3 Feed, dstph	986	962	E
	5.8.4 Diameter, ft.		10	C
	5.8.5 Length, ft.		35	C
5.9 Conveying System				
	5.9.1 Average feed rate, dstph	986	962	E
	5.9.2 Acid addition point	Drum Agglomerator		A
	5.9.3 Acid addition, lb/ton	20	50	A
	5.9.4 Conveyor list	Width (in)	Length (ft)	

		5.9.4.1 Conveyor 1	36	4,000	A
		5.9.4.2 Conveyor 2	36	4,000	A
		5.9.4.3 Conveyor 3	36	100	A
			Nominal	Design	Code
5.10 Stacker					
	5.10.1 Average feed rate, dstph		986	962	E
	5.10.2 Heap height, feet				
	5.10.2.1 Maximum			36	A
	5.10.2.2 Minimum			25	A
	5.10.3 Size				
		5.10.3.1 Stacker	36 in x 130 ft		D
		5.10.3.2 Telescopic Conveyor	36 in x 20 ft		D
6.0 Heap Leach, Existing Pads					
	6.1 General pad information				
		6.1.1 Total tonnage to pad		13,897	G
		6.1.2 Available leach area, ft^2		2,900,000	A
		6.1.3 Lift height, ft		30	A
		6.1.4 Ore bulk density, lbs/ft^3		95	A
		6.1.5 Number of lifts		3	A
	6.2 Acid Cure				
		6.2.1 Acid addition, lbs. H_2SO_4/ton ore	18.7		A
		6.2.2 Application rate, gpm/ft^2	0.004		A
		6.2.3 H_2SO_4 concentration in cure solution, gpl	144		E
		6.2.4 Cure duration, days	3.39		E
		6.2.5 Cure solution, gpm	86		E
		6.2.6 Cure system	separate cure line to heaps		
		6.2.7 Method of acid injection	direct injection in cure PLS line		
	6.3 Leach solutions				
		6.3.1 Application rate			
		6.3.1.1 1st four months	0.004		
		6.3.1.2 2nd two months	0.002		
		6.3.1.3 Remaining ten months	0.001		
		6.3.2 Estimated leach flow, gpm	Nominal	Design	Code
		6.3.2.1 Year 1	4,500		E
		6.3.2.2 Year 2	6,700		E
		6.3.2.3 Year 3	4,800		E
		6.3.2.4 Year 4	6,200		E
		6.3.2.5 Rinse	As req'd		E
	6.4 Makeup water to saturate old heaps				
		6.4.1 Old heap height, ft		100	E
		6.4.2 Moisture increase from rest to leach, %		5	E
		6.4.3 Makeup water, gallons/day		263	E
		6.4.4 Time for one pass of old heaps, months		6	E
			Nominal	Design	Code
7.0 Heap Leach, New Pad					

	7.1 General pad information			
	7.1.1 Total tonnage to pad	21,244		G
	7.1.2 Available leach area, ft^2			A
	7.1.2.1 Year 4 area		4,000,000	E
	7.1.2.2 Year 6 area		1,200,000	E
	7.1.2.3 Total		5,200,000	
	7.1.3 Lift height, ft		30	A
	7.1.4 Ore bulk density, lbs/ft^3		95	A
	7.1.5 Number of lifts		4	E
	7.1.6 Plastic liner		80 mil HDPE	A
	7.1.7Ore placement		Conveyor	A
	7.2 Acid cure			
	7.2.1 Acid addition, lbs. H_2SO_4/ton ore		18.7	A
	7.2.2 Application rate, gpm/ft^2		0.004	A
	7.2.3 H_2SO_4 concentration in cure solution, gpl		144	E
	7.2.4 Cure duration, days		3.39	E
	7.2.5 Cure solution, gpm		86	E
	7.2.6 Cure system	Separate cure lines to heaps		
	7.2.7 Method of acid injection	direct injection in cure PLS line		
	7.3 Leach solution			
	7.3.1 Application Rate	0.001		A
	7.3.2. Estimated leach flow rate			
	7.3.2.1 Year 1	7,727		A
	7.3.2.2 Year 2	7,304		A
	7.3.2.3 Year 3	10,234		
	7.3.2.4Year 4	11,475		E
	7.3.2.5 Year 5	8,989		E
	7.3.2.6 Year 6	8,952		E
	7.3.2.7 Year 7	6,545		E
	7.3.2.8 Year 8	6,166		E
	7.3.2.9 Year 9	5,650		E
	7.3.2.10 Year 10	8,323		E
	7.3.2.11 Year 10	7,784		E
	7.3.2.12 Year 10	6,954		E
	7.3.2.13 Year 10	7,580		E
	7.3.2.14 Year 10	6,943		E
	7.3.2.15 Year 10	8,172		E
	7.3.3 PLS flow, gpm	2550		E
	7.4 Leach ponds			
	7.4.1 New divided PLS/ILS pond	9,000,000		E
	7.4.2 New ILS pond, maximum volume, gal	Hold for future reserve increase confirm with final engineering		
	7.4.3 New storm water pond, maximum volume, gal	6,000,000		E

8.0 Solvent extraction		Nominal	Design	Code
	8.1 Operating schedule			
	8.1.1 Days per year	365		E
	8.1.2 Shifts per day	2		E
	8.1.3 Hours per shift	12		E

		8.1.4 Plant availability, %	98		E
	8.2 Extraction efficiency, %		92		E
	8.3 PLS flowrate, gpm		2,550	2,930	E
	8.4 Solvent extraction layout				
		8.4.1 Number of trains	2		A
		8.4.2 Extraction stages per train	2		A
		8.4.3 Strip stages per train	1		A
		8.4.4 Extraction configuration	Series operation		A
	8.5 Extraction Stages				
		8.5.1 Pump mixers per stage	1		A
		8.5.2 Auxiliary mixers per stage	1		A
		8.5.3 Size, ft			
		Length	59.73		A
		Width	29.10		A
		Depth			
		8.5.4 O/A ratio, overall	1:1		E
		8.5.5 Specific settling rate, gal/min/ft^2	1.47	1.69	E
		8.5.6 Covered settlers	yes		A
	8.6 Strip Stages				
		8.6.1 Pump mixers per stage	1		A
		8.6.2 Auxiliary mixers per stage	1		A
		8.6.3 Size, ft			
		Length	58.25		A
		Width	32.00		A
		Depth			
		8.6.4 O/A ratio, overall	1.3:1		E
		8.6.5 Specific settling rate, gal/min/ft^2	1.37	1.57	E
		8.6.6 Covered settlers	yes		A
	8.7 Strip solution				
		8.7.1 Total strip solution to strip sections, gal/min	2,550	2,930	E
		8.7.2 Total rich electrolyte advance, gal/min	406		E
		8.7.3 Delta across EW circuit, gpl Cu	11		E
		8.7.4 Lean electrolyte composition			
		copper, minimum, gpl	32.0		E
		acid, maximum, gpl	175		E
	8.8 Organic specification		Nominal	Design	Code
		8.8.1 Reagent (typical)	ACORGA M5774		A
		8.8.2 Reagent concentration, percent	7.5		E
		8.8.3 Diluent (typical)	Conoco 170		A
	8.9 Entrainment, ppm		30		E
9.0 Electrowinning					
	9.1 Operating schedule				
		9.1.1 Days per year	365		E
		9.1.2 Shifts per day	3		E
		9.1.3 Hours per day	24		E

		9.1.4 Plant availability, %	99		E
	9.2 Copper production				
		9.2.1 Daily, lbs/day including availability			
		9.2.1.1 Old section	35,220	38,200	E
		9.2.1.2 New section	16,460	17,860	E
		9.2.1.3 Expansion	16,460	17,860	E
		9.2.1.4 Total	68,140	73,920	E
		9.2.2 Annual	24,900,000	27,000,000	E
	9.3 Cathode cycle				
		9.3.1 Growth period, days	7		A
		9.3.2 Harvesting schedule, days per week	5		A
		9.3.3 Weight of cathode/ per side, lbs	100		E
	9.4 Electrowinning criteria				
		9.4.1 Method of production	Permanent SS cathodes		A
		9.4.2 Current density, amps/ft^2			
		9.4.2.1 Old section	22	23.9	E
		9.4.2.2 New section	22	23.9	E
		9.4.2.3 Expansion	22	23.9	A
		9.4.3 Current efficiency, %			
		9.4.3.1 Old section	92		E
		9.4.3.2 New section	92		E
		9.4.3.3 Expansion	92		A
		9.4.4 Old section			
		9.4.4.1 Number of sections	1		A
		9.4.4.2 Number of cells	56		A
		9.4.4.3 Cathodes per cell	22		A
		9.4.4.4 Cathode height, ft	3.75		A
		9.4.4.5 Cathode width, ft	3.00		A
		9.4.4.6 Anodes per cell	23		A
		9.4.5 New section			
		9.4.5.1 Number of sections	1		A
		9.4.5.2 Number of cells	16		A
		9.4.5.3 Cathodes per cell	36		A
		9.4.5.4 Cathode height, ft	3.75		A
	9.4 Electrowinning criteria cont'd		Nominal	Design	Code
		9.4.5.5 Cathode width, ft	3.00		A
		9.4.5.6 Anodes per cell	37		A
		9.4.6 Voltage per cell, V			
		9.4.6.1 Old section	2.1		E
		9.4.6.2 New section	2.1		E
		9.4.6.3 Expansion	2.1		A
	9.5 Electrolyte solution criteria				
		9.5.1 Lean electrolyte			
		9.5.1.1 Flow, gallons per minute	518		E
		9.5.1.2 Cu concentration, g/l	32		E
		9.5.1.3 H_2SO_4 concentration, g/l	165		E
		9.5.1.4 Fe concentration, g/l	2		E
		9.5.1.5 Co++ concentration, g/l	100		A

		9.5.1.6 Specific gravity, gm/cc	1.27		E
		9.5.2 Rich electrolyte			
		9.5.2.1 Flow, gallons per minute	518		E
		9.5.2.2 Cu concentration, g/l	43		E
		9.5.2.3 H_2SO_4 concentration, g/l	150		E
		9.5.2.4 Specific gravity, gm/cc	1.27		E
		9.5.2.5 Operating temperature, ^oF	110		E
		9.5.3 Rich electrolyte organic removal			
		9.5.3.1 Number of filters	2 x 400 gpm each		A
		9.5.3.2 Type of filters	garnet and anthracite		A
	9.6 Electrolyte flow to EW circuit				
		9.6.1 Total electrolyte flow, gpm	1,865		E
		9.6.2 Electrolyte flow/cell, gpm			
		9.6.2.1 Old section	17.5		E
		9.6.2.2 New section	27.7		E
		9.6.2.3 Expansion	27.7		E
		9.6.3 Electrolyte flow per cathode, gpm/ft^2			
		9.6.3.1 Old section	0.035		E
		9.6.3.2 New section	0.034		E
		9.6.3.3 Expansion	0.034		
	9.7 Rectifier				
		9.7.1 Old section, operating amps	11,815	12,385	E
		9.7.2 New section, operating amps	19,336	20,266	E
		9.7.3 Efficiency for operating cost estimate, percent	90		A
10.0 Tank Farm					
	10.1 Loaded organic tank capacity, gal		30,000		A
	10.2 Rich electrolyte storage tank, gal		6,000		A
	10.3 Lean electrolyte storage tank, gal		8,000		A
	10.4 Electrolyte circulation tank capacity, gal		8,000		A
	10.5 Kerosene storage tank, gal		13,500		A
10.0 Tank Farm cont'd			Nominal	Design	Code
	10.6 Organic recovery tank/ divided in half, gal		25,000		A
	10.7 Clay treatment storage tank, gal		6,000		A
	10.8 Crud treatment storage tank, gal		2,500		A
11.0 Sludge Treatment Facility			Mix with Organic, settle, clay treat		A

28.0 INTERPRETATION AND CONCLUSIONS

Overall, the feasibility study report addresses all of the topics that need to be addressed for a full feasibility study and be compliant with NI 43-101.

BETA's conclusion, based on the work performed and presented herein, is that the Johnson Camp Mine Project is both economically and technically feasible. BETA has reviewed the underlying assumptions to the geologic, resource, reserve and economic models and is satisfied with their suitability for use.

BETA agrees that the Johnson Camp database of geological, assay, and survey information is substantially well documented to provide confidence in the digital drill hole database and the resource block model derived there from. The various data have been verified on multiple levels, and no significant quantity of database errors or omissions were noted by BETA.

BETA is of the opinion that the Johnson Camp resource database at this point in time appears to have been demonstrated as both credible and verifiable, and that the mineral resource and mineral reserve statements included in this report are accurate and within normal limits required by a feasibility study.

29.0 RECOMMENDATIONS

Database auditing and verification should be an ongoing process with active operations at Johnson Camp;

Exploration drilling is recommended between the Burro and Copper Chief pits and along trend to the northwest of the Copper Chief pit and to the southeast of the Burro pit.

The feasibility of increasing annual ore and copper production through the expansion of site facilities should be investigated.

Additional metallurgical testing of certain ores found in the lower reaches of the deposit, below the 4,560 elevation, is recommended prior to mining those areas.

Additional geotechnical investigation may be desired to refine pit wall slopes.

30.0 CERTIFICATES

Certificate of the Author

JOHNSON CAMP MINE PROJECT FEASIBILITY STUDY, Cochise County, Arizona
TECHNICAL REPORT DATED SEPTEMBER 2007

I, Michael Bikerman, Ph.D., residing at 1057 Lindendale Lane, Pittsburgh, PA, 15243, hereby certify that:

1. I have been gainfully employed since 1956 in the field of geology and, since 1995, as a Registered Geologist
2. I am currently a consultant in the firm of Bikerman Engineering & Technology Associates, Inc. located at 76 Lyme Street, Old Lyme, Connecticut, 06371.
3. I am a graduate of the University of Arizona, Tucson, AZ, with the degree of Ph.D. in Geology in 1956.
4. I am a registered professional geologist, PG -001740-G, in the Commonwealth of Pennsylvania.
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 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. The work submitted is based upon my personal examination of the project data.
7. I personally conducted a site visit of the Johnson Camp Mine and inspected related documentation in Nord's Tucson offices from June 12-15, 2007, and have reviewed in detail all supporting data on which the results are based.
8. 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.
9. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
10. I have not received interest, direct or indirect, in the property nor do I have any beneficial interest, direct or indirect, in the securities of NORD Resources, Corp, or any of its parents or subsidiaries, and I am independent of the issuer under NI43-101.
11. I have had no prior involvement with the properties discussed in this technical report.
12. As of the date of this Certificate and 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.

Michael Bikerman

Michael Bikerman, Ph.D., PG
Pittsburgh, PA USA

CERTIFICATES

JOHNSON CAMP MINE PROJECT FEASIBILITY STUDY, Cochise County, Arizona
TECHNICAL REPORT DATED SEPTEMBER 2007

I, David Bikerman, M.S., Engineer of Mines, residing at 76 Lyme Street, Old Lyme, Connecticut, 06371, hereby certify that:

1. I have been gainfully employed since 1978 in the field of mining and, since 1981, as a Mining Engineer
2. I am currently a principal in the firm of Bikerman Engineering & Technology Associates, Inc. located at 76 Lyme Street, Old Lyme, Connecticut, 06371.
3. I am a graduate of the Henry Krumb School of Mines of Columbia University in the City of New York, NY, with the degree of Professional Engineer of Mines in 1995, and the degree of Master of Science in Mining Engineering in 1985. I am a graduate of the University of Pittsburgh, Pittsburgh, PA, with a degree of Bachelors of Science in Mining Engineering in 1981.
4. I am a member of the Society of Mining Engineers (SME), and member of the Australian Institute of Geoscientists.
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 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. The work submitted is based upon my personal examination of the project data.
7. I personally conducted a site visit of the Johnson Camp Mine and inspected related documentation in Nord's Tucson offices from June 12-15, 2007, and have reviewed in detail all supporting data on which the results are based.
8. 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.
9. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
10. I have not received interest, direct or indirect, in the property nor do I have any beneficial interest, direct or indirect, in the securities of NORD Resources, Corp., or any of its parents or subsidiaries, and I am independent of the issuer under NI43-101.
11. I have had no prior involvement with the properties discussed in this technical report.
12. As of the date of this Certificate and 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.

David Bikerman, M.S., E.M.
Old Lyme, Connecticut, USA

CERTIFICATES

JOHNSON CAMP MINE PROJECT FEASIBILITY STUDY, Cochise County, Arizona
TECHNICAL REPORT DATED SEPTEMBER 2007

I, Thomas McGrail, Engineer of Mines, residing at A.P. 47, San Juan del Sur, Rivas, Nicaragua, hereby certify that:

1. I have been gainfully employed since 1972 in the field of mining and, since 1981, as a Mining Engineer.
2. I am currently a consultant to the firm of Bikerman Engineering and Technology Associates, Inc.
3. I am a graduate of the Technical University of Nova Scotia with a Bachelor's Degree in Mining (Distinction) granted in 1981.
4. I am a member of the Australian Institute of Geoscientists.
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 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. The work submitted is based upon my personal examination of the project data.
7. I personally conducted a site visit of the Johnson Camp Mine and inspected related documentation in Nord's Tucson offices from June 12-15, 2007, and have reviewed in detail all supporting data on which the results are based.
8. 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.
9. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
10. I have not received interest, direct or indirect, in the property nor do I have any beneficial interest, direct or indirect, in the securities of NORD Resources, Corp., or any of its parents or subsidiaries, and I am independent of the issuer under NI43-101.
11. I have had no prior involvement with the properties discussed in this technical report.
12. As of the date of this Certificate and 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.

Thomas McGrail

San Juan del Sur, Rivas, Nicaragua

CERTIFICATES

I, Dale A. Deming, PE, Bachelor of Science Mining Engineering and Masters of Business Administration, residing at 7860 N. John Hancock Avenue, Tucson, Arizona 85741, hereby certifies that:

1. I have been gainfully employed since 1978 in the field of mining as a Mining Engineer, Environmental Engineer, Environmental Manager, and as a sole proprietor Mining Consultant since 1996.
2. I am currently a sole proprietor Mining Consultant with the business named as Dale A. Deming, PE located at 7860 N. John Hancock Avenue, Tucson, Arizona 85741.
3. I am a graduate of the Mackay School of Mines of the University of Nevada, Reno in Reno, Nevada, with the degree of Bachelor of Science in Mining Engineering in 1977. I am also a graduate of the Karl Eller Center for the Private Market Economy, University of Arizona, Tucson, Arizona, with a degree of Masters of Business Administration, Entrepreneurship/Finance Concentration in 1989.
4. I have been a registered Arizona Professional Mining Engineer (#18195) since 1985 and am in good standing.
5. I am a member of the Society of Mining Engineers (SME) in good standing.
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 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. The work submitted is based upon my personal examination of the project data.
8. I have personally visited the property on which the work is done, and have reviewed in detail all supporting data on which the results are based. My last visit to the property was on 9/26/07. I have previously provided independent consulting services regarding the property.
9. I have read National Instrument 43-101 and form 43-101F1, and the section of the Technical Report containing my work product has been prepared in compliance with that instrument and form.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
11. I am independent of NORD Resources, Corp., and I have not received interest, direct or indirect, in the property nor do I have any beneficial interest, direct or indirect, in the securities of NORD Resources, Corp., or any of its parents or subsidiaries.
12. As of the date of this Certificate and 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.

Dale A. Deming, PE
Tucson, Arizona, USA
October, 2007

APPENDIX 1 - REFERENCES

Title	Author	Date
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Johnson Camp Mine Data Compilation and Due Diligence Confirmation Review of Issues Related to the Burro and Copper Chief Copper Deposits to Confirm the Electronic Drillhole Database Used in the Winters, Dorsey & Company, LLC 2005 Johnson Camp Feasibility Study	Clive R. G. Bailey, RS, CPG EDGE CONSULTING	May 15, 2006
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APPENDIX 2

CIM DEFINITION STANDARDS

FOR MINERAL RESOURCES

AND MINERAL RESERVES

CIM DEFINITION STANDARDS - For Mineral Resources and Mineral Reserves

Prepared by the CIM Standing Committee on Reserve Definitions
Adopted by CIM Council on December 11, 2005

FOREWORD

CIM Council, on August 20, 2000, approved the "CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines," developed by the CIM Standing Committee on Reserve Definitions. The CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards) establish definitions and guidelines for the reporting of exploration information, mineral resources and mineral reserves in Canada. The Mineral Resource and Mineral Reserve definitions were incorporated, by reference, in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101), which became effective February 1, 2001.

At the August 20, 2000 Council meeting a new CIM Standing Committee on Reserve Definitions was established consisting of the following: John Postle, Bernie Haystead, Larry Cochrane, Normand Champigny, Mike Hoffman, Colin McKenny, Jack Mullins, Phil Olson, Fred Payne, Jody Todd and Joe Ringwald.

Subsequent to the publishing of the August 20, 2000 CIM Standards on Mineral Resources and Reserves, various CIM committees have compiled and published more extensive documentation on mining industry standard practices for estimating Mineral Resources and Mineral Reserves. These standard practices provide more detailed guidance than that contained in the August 20, 2000 CIM Standards on Mineral Resources and Reserves. On November 14, 2004 CIM Council adopted an update to the CIM Definition Standards to reflect the more detailed guidance available and effect certain editorial changes required to maintain consistency with current regulations. This version of the CIM Definition Standards includes further editorial changes required to maintain compatibility with the new version of National Instrument 43-101 which is expected to become law at the end of 2005. The CIM Definition Standards can be viewed on the CIM website at www.cim.org.

Readers should be aware that reports written by persons issuing technical reports that disclose information about exploration or other mining properties to the public are governed by a number of regulations in Canada. The most important of these are NI 43-101 for mineral properties and National Instrument 51-101 for oil and gas properties.

CIM DEFINITION STANDARDS

The CIM Definition Standards presented herein provide standards for the classification of Mineral Resource and Mineral Reserve estimates into various categories. The category to which a resource or reserve estimate is assigned depends on the level of confidence in the geological information available on the mineral deposit; the quality and quantity of data available on the deposit; the level of detail of the technical and economic information which has been generated about the deposit, and the interpretation of the data and information. In the document the definitions are in bold type and the guidance is in italics.

DEFINITIONS

Throughout the CIM Definition Standards, where appropriate, ‘quality’ may be substituted for ‘grade’ and ‘volume’ may be substituted for ‘tonnage’. Technical Reports dealing with estimates of Mineral Resources and Mineral Reserves must use only the terms and definitions contained herein.

Qualified Person

Mineral Resource and Mineral Reserve estimates and resulting Technical Reports must be prepared by or under the direction of, and dated and signed by, a Qualified Person.

A “Qualified Person” means an individual who is an engineer or geoscientist with at least five years of experience in mineral exploration, mine development or operation or mineral project assessment, or any combination of these; has experience relevant to the subject matter of the mineral project and the technical report; and is a member or licensee in good standing of a professional association.

The Qualified Person(s) should be clearly satisfied that they could face their peers and demonstrate competence and relevant experience in the commodity, type of deposit and situation under consideration. If doubt exists, the person must either seek or obtain opinions from other colleagues or demonstrate that he or she has obtained assistance from experts in areas where he or she lacked the necessary expertise.

Determination of what constitutes relevant experience can be a difficult area and common sense has to be exercised. For example, in estimating Mineral Resources for vein gold mineralization, experience in a high-nugget, vein-type mineralization such as tin, uranium etc. should be relevant whereas experience in massive base metal deposits may not be. As a second example, for a person to qualify as a Qualified Person in the estimation of Mineral Reserves for alluvial gold deposits, he or she would need to have relevant experience in the evaluation and extraction of such deposits. Experience with placer deposits containing minerals other than gold, may not necessarily provide appropriate relevant experience for gold.

In addition to experience in the style of mineralization, a Qualified Person preparing or taking responsibility for Mineral Resource estimates must have sufficient experience in the sampling, assaying, or other property testing techniques that are relevant to the deposit under consideration in order to be aware of problems that could affect the reliability of the data. Some appreciation of extraction and processing techniques applicable to that deposit type might also be important.

Estimation of Mineral Resources is often a team effort, for example, involving one person or team collecting the data and another person or team preparing the Mineral Resource estimate. Within this team, geologists usually occupy the pivotal role. Estimation of Mineral Reserves is almost always a team effort involving a number of technical disciplines, and within this team mining engineers have an important role. Documentation for a Mineral Resource and Mineral

Reserve estimate must be compiled by, or under the supervision of, a Qualified Person(s), whether a geologist, mining engineer or member of another discipline. It is recommended that, where there is a clear division of responsibilities within a team, each Qualified Person should accept responsibility for his or her particular contribution. For example, one Qualified Person could accept responsibility for the collection of Mineral Resource data, another for the Mineral Reserve estimation process, another for the mining study, and the project leader could accept responsibility for the overall document. It is important that the Qualified Person accepting overall responsibility for a Mineral Resource and/or Mineral Reserve estimate and supporting documentation, which has been prepared in whole or in part by others, is satisfied that the other contributors are Qualified Persons with respect to the work for which they are taking responsibility and that such persons are provided adequate documentation.

Preliminary Feasibility Study

The CIM Definition Standards requires the completion of a Preliminary Feasibility Study as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves.

A Preliminary Feasibility Study is a comprehensive study of the viability of a mineral project that has advanced to a stage where the mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, has been established and an effective method of mineral processing has been determined, and includes a financial analysis based on reasonable assumptions of technical, engineering, legal, operating, economic, social, and environmental factors and the evaluation of other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be classified as a Mineral Reserve.

Exploration Information

Exploration information means geological, geophysical, geochemical, sampling, drilling, trenching, analytical testing, assaying, mineralogical, metallurgical and other similar information concerning a particular property that is derived from activities undertaken to locate, investigate, define or delineate a mineral prospect or mineral deposit.

*It is recognised that in the review and compilation of data on a project or property, previous or historical estimates of tonnage and grade, not meeting the minimum requirement for classification as Mineral Resource, may be encountered. If a Qualified Person reports Exploration Information in the form of tonnage and grade, it must be clearly stated that these estimates are conceptual or order of magnitude and that they do not meet the criteria of a Mineral Resource***.**

Mineral Resource

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

A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

Inferred Mineral Resource

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Indicated Mineral Resource

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

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

Measured Mineral Resource

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

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

Mineral Reserve

Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant processing, metallurgical, economic, marketing, legal, environment, socio-economic and government factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

Probable Mineral Reserve

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

Proven Mineral Reserve

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

RESOURCE AND RESERVE CLASSIFICATION

Technical Reports dealing with estimates of Mineral Resources and Mineral Reserves must use only the terms and the definitions contained herein. Figure 1, displays the relationship between the Mineral Resource and Mineral Reserve categories.

The CIM Definition Standards provide for a direct relationship between Indicated Mineral Resources and Probable Mineral Reserves and between Measured Mineral Resources and Proven Mineral Reserves. In other words, the level of geoscientific confidence for Probable Mineral Reserves is the same as that required for the in situ determination of Indicated Mineral Resources and for Proven Mineral Reserves is the same as that required for the in situ determination of Measured Mineral Resources.

Figure 1
Relationship between Mineral Resources and Mineral Reserves

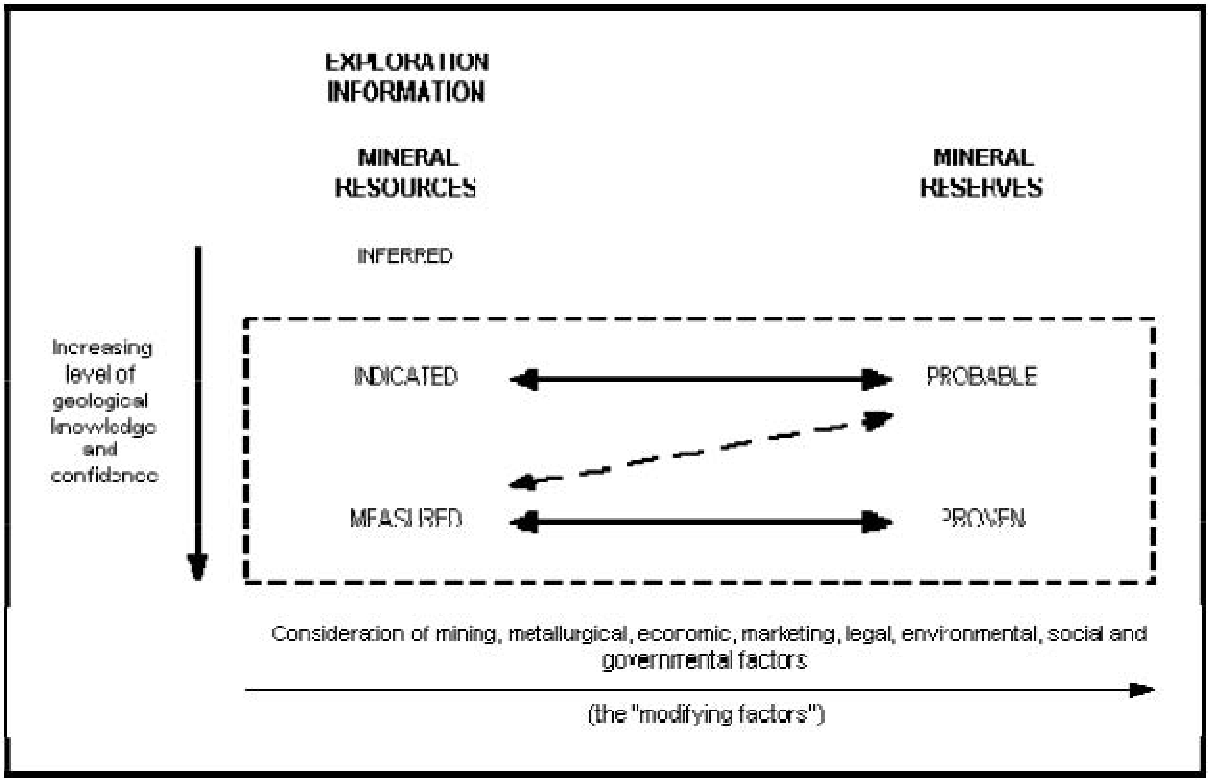

Figure 1 sets out the framework for classifying tonnage and grade estimates so as to reflect different levels of geological confidence and different degrees of technical and economic evaluation. Mineral Resources can be estimated by a Qualified Person, with input from persons in other disciplines, as necessary, on the basis of geoscientific information and reasonable assumptions of technical and economic factors likely to influence the prospect of economic extraction. Mineral Reserves, which are a modified sub-set of the Indicated and Measured Mineral Resources (shown within the dashed outline in Figure 1), require consideration of factors affecting profitable extraction, including mining, processing, metallurgical, economic, marketing, legal, environmental, socio-economic and governmental factors, and should be estimated with input from a range of disciplines. Additional test work, e.g. metallurgy, mining, environmental is required to reclassify a resource as a reserve.

In certain situations, Measured Mineral Resources could convert to Probable Mineral Reserves because of uncertainties associated with the modifying factors that are taken into account in the conversion from Mineral Resources to Mineral Reserves. This relationship is shown by the dashed arrow in Figure 1 (although the trend of the dashed arrow includes a vertical component, it does not, in this instance, imply a reduction in the level of geological knowledge or confidence). In such a situation these modifying factors should be fully explained. Under no circumstances can Indicated Resources convert directly to Proven Reserves.

In certain situations previously reported Mineral Reserves could revert to Mineral Resources. It is not intended that re-classification from Mineral Reserves to Mineral Resources should be applied as a result of changes expected to be of a short term or temporary nature, or where company management has made a deliberate decision to operate in the short term on a non-economic basis. Examples of such situations might be a commodity price drop expected to be of short duration, mine emergency of a non-permanent nature, transport strike etc.

GUIDANCE FOR REPORTING MINERAL RESOURCE AND MINERAL RESERVE INFORMATION

Qualified Persons preparing public Mineral Resource and Mineral Reserve reports in Canada must follow the requirements in Form 43-101F1 of National Instrument 43-101, available on the following websites: www.osc.gov.ca; www.bcsc.bc.ca; www.albertasecurities.com and www.cvmq.com.

The following discussion is included for additional guidance when preparing a Technical Report. For the CIM Definition Standards a Technical Report is defined as a report that contains the relevant supporting documentation, estimation procedures and description of the Exploration Information, or the Mineral Resource and Mineral Reserve estimate.

Technical Reports of a Mineral Resource must specify one or more of the categories of 'Inferred', 'Indicated' and 'Measured' and Technical Reports of Mineral Reserves must specify one or both of the categories of 'Proven' and 'Probable'. Categories must not be reported in a combined form unless details for the individual categories are also provided. Inferred Mineral Resources cannot be combined with other categories and must always be reported separately. Mineral Resources must never be added to Mineral Reserves and reported as total Resources and Reserves. Mineral Resources and Mineral Reserves must not be reported in terms of contained metal or mineral content unless corresponding tonnages, grades and mining, mineral processing and metallurgical recoveries are also presented

Qualified Persons are encouraged to provide information that is as comprehensive as possible in their Technical Reports on Exploration Information, Mineral Resources and Mineral Reserves. The Mineral Exploration Best Practices Guidelines, the Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines and the Guidelines for the Reporting of Diamond Exploration Results provide, in a summary form, a list of the main criteria which should be considered when reporting Exploration Information, Mineral Resources and Mineral Reserve estimates. These guidelines are available on the CIM website, www.cim.org.

These Guidelines are not prescriptive and it may not be necessary to comment on each item in the guidelines, however, the need for comment on each item should be considered. It is essential to discuss any matters that might materially affect the reader's understanding of the estimates being reported. Problems encountered in the collection of data or with the sufficiency of data must be clearly disclosed at all times, particularly when they affect directly the reliability of, or confidence in, a statement of Exploration Information or an estimate of Mineral Resources and Mineral Reserves; for example, poor sample recovery, poor reproducibility of assay or laboratory results, limited information on tonnage factors etc.

Mineral Resources and Mineral Reserves must be reported on a site by site basis.

Where estimates for both Mineral Resources and Mineral Reserves are reported, for consistency, it is recommended that Mineral Resources be reported exclusive of Mineral Reserves. Notwithstanding, it is recognized that there are legitimate reasons, in some situations, for reporting Mineral Resources inclusive of Mineral Reserves (the Australian approach) and, in other situations, for reporting Mineral Resources additional to Mineral Reserves (the South African and United States approach). When reporting both Mineral Resources and Mineral Reserves, a clarifying statement must be included that clearly indicates whether Mineral Reserves are part of the Mineral Resource or that they have been removed from the Mineral Resource. A single form of reporting should be used in a report. Appropriate forms of clarifying statements may be:

- *'The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves,' or*
- *'The Measured and Indicated Mineral Resources are additional to the Mineral Reserves.'*

Inferred Mineral Resources are, by definition, always additional to Mineral Reserves.

REPORTING OF COAL RESERVES

For consistency in public reporting of coal resources and reserves, it is recommended that all issuers use the Mineral Resource and Mineral Reserve categories set out in the CIM Definition Standards. Qualified Person(s) should be guided by the Estimation of Mineral Resources and Mineral Reserve Best Practices Guidelines for Coal and by GSC Paper 88-21: A Standardized coal Resource/Reserve Reporting System for Canada. It is acceptable to use the GSC Paper 88-21 as a framework for the development and categorization of coal estimates, but the GSC 88-21 categories should be converted to the equivalent CIM Definition categories for public reporting. When using GSC 88-21 as a framework, in the classification of coal by A.S.T.M. ranking, the "Group" designation is preferred over the less descriptive "Class" designation.

REPORTING OF INDUSTRIAL MINERALS

When reporting Mineral Resource and Mineral Reserve estimates relating to an industrial mineral site, the Qualified Person(s) should be guided by the Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines for Industrial Minerals.

APPENDIX 3 – SEC INDUSTRY GUIDE 7

Description of Property by Issuers Engaged or to Be Engaged in Significant Mining Operations
Guide 7.

(a) *Definitions.* The following definitions apply to registrants engaged or to be engaged in significant mining operations:

(1) Reserve. That part of a mineral deposit which could be economically and legally extracted or produced at the time of the reserve determination.

Note: Reserves are customarily stated in terms of “ore” when dealing with metalliferous minerals; when other materials such as coal, oil, shale, tar, sands, limestone, etc. are involved, an appropriate term such as “recoverable coal” may be substituted.

(2) *Proven (Measured) Reserves.* Reserves for which (a) quantity is computed from dimensions revealed in outcrops, trenches, workings or drill holes; grade and/or quality are computed from the results of detailed sampling and (b) the sites for inspection, sampling and measurement are spaced so closely and the geologic character is so well defined that size, shape, depth and

(3) *Probable (Indicated) Reserves.* Reserves for which quantity and grade and/or quality are computed form information similar to that used for proven (measure) reserves, but the sites for inspection, sampling, and measurement are farther apart or are otherwise less adequately spaced. The degree of assurance, although lower than that for proven (measured) reserves, is high enough to assume continuity between points of observation.

(4) (i) *Exploration State* — includes all issuers engaged in the search for mineral deposits (reserves) which are not in either the development or production stage.

(ii) *Development Stage* — includes all issuers engaged in the preparation of an established commercially minable deposit (reserves) for its extraction which are not in the production stage.

(iii) *Production Stage* — includes all issuers engaged in the exploitation of a mineral deposit (reserve).

Instruction to paragraph
(a) Mining companies in the exploration stage should not refer to themselves as development stage companies in the financial statements, even though such companies should comply with FASB Statement No. 7, if applicable.

(b) *Mining Operation Disclosure.* Furnish the following information as to each of the mines, plants and other significant properties owned or operated, or presently intended to be owned or operated, by the registrant:

(1) The location and means of access to the property.

(2) A brief description of the tile, claim, lease or option under which the registrant and its subsidiaries have or will have the right to hold or operate the property, indicating any conditions which the registrant must meet in order to obtain or retain the property. If held by leases or options, the expiration dates of such leases or options should be stated. Appropriate maps may be used to portray the locations of significant properties;

(3) A brief history of previous operations, including the names of previous operators, insofar as known;

(4) (i) A brief description of the present condition of the property, the work completed by the registrant on the property, the registrant's proposed program of exploration and development, and the current state of exploration and/or development of the property. Mines should be identified as either open-pit or underground. If the property is without known reserves and the proposed program is exploratory in nature, a statement to that effect shall be made; (ii) The age, details as to modernization and physical condition of the plant and equipment, including subsurface improvements and equipment. Further, the total cost for each property and its associated plant and equipment should be stated. The source of power utilized with respect to each property should also be disclosed.

(5) A brief description of the rock formations and mineralization of existing or potential economic significance on the property, including the identity of the principal metallic or other constituents insofar as known. If proven (measured) or probable (indicated) reserves have been established, state (i) the estimated tonnages and grades (or quality, where appropriate) of such classes of reserves, and (ii) the name of the person making the estimates and the nature of his relationship to the registrant.

Instructions to paragraph (b)(5):
1. It should be stated whether the reserve estimate is of in-place material or of recoverable material. Any inplace estimate should be qualified to show the anticipated losses resulting from mining methods and beneficiation or preparation.
2. The summation of proven (measured) and probable (indicated) ore reserves is acceptable if the difference in degree of assurance between the two classes of reserves cannot be readily defined.
3. Estimates other than proved (measured) or probable (indicated) reserves, and any estimated values of such reserves shall not be disclosed unless such information is required to be disclosed by foreign or state law; provided, however, that where such estimates previously have been provided to a person (or any of its affiliates) that is offering to acquire, merge, or consolidate with, the registrant or otherwise to acquire the registrant's securities, such estimates may be included.

6) If technical terms relating to geology, mining or related matters whose definition cannot readily be found in conventional dictionaries (as opposed to technical dictionaries or glossaries) are used, an appropriate glossary should be included in this report.

(7) Detailed geographic maps and reports, feasibility studies and other highly technical data should not be included in the report but should be, to the degree appropriate and necessary for the

Commission's understanding of the registrant's presentation of business and property matters, furnished as supplemental information.

(c) *Supplemental Information.*

(1) If an estimate of proven (measured) or probable (indicated) reserves is set forth in the report, furnish: (i) maps drawn to scale showing any mine workings and the outlines of the reserve blocks involved together with the pertinent sample-assay thereon. (ii) all pertinent drill data and related maps. (iii) the calculations whereby the basic sample-assay or drill data were translated into the estimates made the grade and tonnage of reserves in each block and in the complete reserve estimate.

Instructions to paragraph (c)(1):
Maps and drawings submitted to the staff should include: (a) A legend or explanation showing, by means of pattern or symbol, every pattern or symbol used on the map or drawing; the use of the symbols used by the U.S. Geological Survey is encouraged; (b) A graphical bar scale should be included; additional representations of scale such as "one inch equals one mile" may be utilized provided the original scale of the map has not been altered; (c) A north arrow on the maps; (d) An index map showing where the property is situated in relationship to the state or province, etc., in which it was located; (e) A title of the map or drawing and the date on which it was drawn; (f) In the even interpretive data is submitted in conjunction with any map, the identity of the geologist or engineer that prepared such data; and (g) Any drawing should be simple enough or of sufficiently large scale to clearly show all features on the drawing.

(2) Furnish a complete copy of every material engineering, geological or metallurgical report concerning the registrant's property, including governmental reports, which are known and available to the registrant. Every such report should include the name of its author and the date of its preparation, if known to the registrant.

Instruction to paragraph (c)(2)
Any of the above-required reports as to which the staff has access need not be submitted. In this regard, issuers should consult with the staff prior to filing the report. Any reports not submitted should be identified in a list furnished to the staff. This list should also identify any known governmental reports concerning the registrant's property.

(3) Furnish copies of all documents such as title documents, operating permits and easements needed to support representations made in the report.