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THE SUPERIOR OIL COMPANY
Minerals Division

February 3, 1976

Memorandum

To: P. Malozemoff
J. E. Thompson
W. H. Burt

From: J. C. Keenan

Re: Vekol Hills - Project Re-evaluation

Respectfully submitted is a report which summarizes a re-evaluation of the Vekol Project based on a July 1, 1978 start of production.

The re-evaluation is the result of a complete updating done entirely by Newmont personnel of the capital and operating cost information contained in the April, 1972 evaluation. All estimates were based on the second half 1975 costs and were in turn, escalated exclusive of the treatment charges to the time of expenditure at a 10% per annum factor for labor and an 8% per annum for materials.

Contributing to the completion of this report were the following:

P. J. Crescenzo	-	Engineering
V. Haldane	-	Re-estimate of Plant & Equipment
D. J. Christie	-	Re-estimate of Mill Operation
L. A. Cassara	-	Estimate of Treatment Costs
W. E. Baker	-	Financial
J. R. Denny	-	Financial Computer Runs
N. Gritzuk	-	Wage and Salary Estimates
J. Johnston	-	Wage and Salary Estimates

The following summary compares the 1972 estimate with the January 1976 estimate:

	1972 ($000)	1976 ($000)	
Start of Production	Jan. 1975	July 1978	
Capital Cost:			
Preproduction Stripping	10,075	15,395	+53%
Mine Equipment etc.	11,773	20,834	+71%
Total Mine	21,848	36,229	
Plant Construction	28,316	44,393	57%
Owners Cost	3,833	5,921	54%
Total	53,997	86,543	
Contingency (10%)	5,400	8,600	
Land Rental & Adr. Royalty	116	1,099	
Operating Inventory	1,000	1,500	50%
Working Capital	4,000	8,266	107%
Total Capital	$64,513	$106,008	64.3%

Operating Costs	¢ per pound payable Cu				
Concentrate					
Freight	1.1		1.462		
Smelting	8.5	(incl.Pol SG)	8.113		
Refining	4.0		6.000		
Total Treatment Charges	13.6		15.575		14.5%
Royalty	1.8	(@50¢/cu)	6.4	(@80¢/cu)	255%
Mining	5.9		9.81		66%
Milling	10.5		18.65		77%
Overheads	5.4		8.04		49%
Total Operating Costs	21.8		36.50		67%
Total Costs	37.2¢		58.475¢		57%

An approximate copper price of 51¢ per pound in the 1972 estimate yielded the same 7% return on investment as a price of 80¢ per pound yields in the 1976 estimate.

VEKOL HILLS

PROJECT EVALUATION

Prepared by:

Newmont Mining Corporation

January 1976

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APPENDIX

SECTION 1

INTRODUCTION

This report revises and updates the Vekol Hills Project Evaluation Report of April 1972. All mine and plant equipment, labor and material costs in both the capital and operating areas have been completely re-estimated using second half 1975 quotes and wage schedules and subsequently escalated forward on the basis of a July 1978 start of production.

1.1 Location

The Vekol Hills ore deposit is located on the Papago Indian Reservation in Pinal County, 26.5 miles southwest of Casa Grande in Arizona. See General Location Map, Appendix 1.

1.2 Ownership

The Vekol Hills mineral deposit was covered by Mining Lease Contracts #14-20-0450-5193 dated January 14, 1965 and modified April 6, 1967 and #14-20-0450-6112 of April 6, 1967, between the Papago Tribe, Lessors and Vekol Copper Mining Company, Lessee, a subsidiary of Newmont Mining Corporation. The leases covered 2760 acres of land. However, on January 14, 1975 these leases expired.

On January 28, 1974, prior to the expiration of the original leases, Vekol and the Papago Tribe agreed to a new mineral lease covering 2560 acres and a new water well lease. The new leases will run for five years from the date that the leases are approved by the Secretary of Interior, said approval subject to an environmental impact action.

The new agreement requires the payment of nonrecoverable cash payments on the date the lease was signed and when it is given final approval by the Secretary of Interior in addition to production royalties. (See Section 9-Royalties).

Beneficial interests in the leases of 48.96362% and 0.03638% are currently retained by Superior Oil Company and Joseph A. Mann, respectively.

The Vekol Copper Mining Company, as lessee, is the operator of the property.

The leases are renewable as long thereafter as the minerals specified are produced in paying quantities.

1.3 Investigations and Reports

A number of studies have been made in analyzing the Vekol Hills orebody and its potential. Some were preliminary, requiring further evaluation as additional information became available.

Other studies were for the purpose of evaluating variations in approach to mining and/or milling.

The delineation and evaluation of the orebody, its geology and ore reserves, was carried out by Newmont Exploration Limited (N.E.L.). See Section 3 for a summary description of the work and findings by the Geological group.

The primary assay work for development of copper and molybdenum ore reserves was performed at the San Manuel laboratories. Process development was conducted by the Metallurgical Department of Newmont Exploration Limited in their laboratories at Danbury, Connecticut.

Pilot plant investigations of grinding were carried out at the Institute of Mineral Research, Houghton, Michigan, as a part of the N.E.L. metallurgical studies.

Metallurgical results and basic data for process design were established. See Section 4 for a summary of the findings of the metallurgical group. Details of the various studies and their findings were presented in four progress reports, three special subject reports and various memoranda and letters during 1970 to 1972. These details have been consolidated into a single report which will be printed shortly.

Mr. W. K. Pincock. P. E., was retained as a consultant in the development of a mining plan. Work was coordinated between mine planning, N.E.L. geologists and the metallurgical group in developing mining limits and the mining reserve.

Five reports were issued by Mr. Pincock. The first report was based on incomplete geological data and a preliminary cutoff of 0.28% sulfide copper. Subsequent studies were based on the latest geological and metallurgical findings, as well as refinements indicated by computerized studies of several mining schedules and up-to-date cost data. These evaluations also considered different sizes of mining equipment and, in the December 13, 1970 study, included the use of an in-pit crusher.

As the early approaches did not yield a viable project, other possibilities were considered. The study of September 1971, considered two variations using scrapers for the removal of alluvial overburden so as to minimize capital outlay for mining equipment and reduce operating costs. (See Section 5.2).

The mining plan which is evaluated in this study (see Section 5) is based, essentially, on the same pit outlines as used in the December 1970 and September 1971 studies. However, ore reserves have been reduced to reflect latest available information on metallurgical recoveries. 4,221,000 tons were changed from "ore" to "oxide" category. The grade of the remaining ore was unchanged, but the overall metallurgical recovery increased from an average of 85% to 87.6%.

Mill design and cost estimates have been developed by the staff of Newmont Services Ltd., utilizing basic data and design criteria established by the Danbury laboratories and other Newmont personnel. The estimate presented here (see Section 6), is the seventh estimate. The first six estimates were based on a conventional mill design, with differences in the several estimates being primarily the result of updating and refinement of cost data, with relatively minor refinements in the projected facilities. Exceptions to this occurred in the March 1971 estimate when evaluating the project with a crusher in the pit and with the addition of a third water well.

The plant evaluated in this report represents a departure from conventional southwestern U. S. concentrator design. Included are such features as minimum building structures, substitution of a mobile crane for overhead cranes, the use of covered fine ore ground storage instead of multiple steel ore bins, installation of grinding units in open air, the use of the largest type of proven equipment and the elimination of a separate sump and pump floor in flotation. This plant is sized to handle the required tonnages within normal design margins and is believed to be a practical plant.

The Parsons-Jurden Corporation of New York was contracted to review and evaluate the feasibility studies completed by Newmont Exploration Limited through December 1970. A report of their findings was issued March 15, 1971. In general, as of that date, Parsons-Jurden agreed with Newmont Exploration's work and findings in the development of ore reserves, found the mining plan completely feasible and the concentrating facilities adequate for the required tonnages and metallurgy. Parsons-Jurden made several suggestions for improvement and disagreed with some cost data used in developing N.E.L's estimates. The suggestions and questioned cost data were given full consideration in subsequent estimates.

Dames and Moore, consulting engineers in the applied earth sciences, were retained to review the tailings disposal system, specifically to evaluate the potential water recovery considering several methods of operating with and without tailings thickeners. Considering comparative capital costs of thickeners versus no thickeners and the probability that there would be less water recovery without thickeners, the proposed plant includes thickeners.

The Empire Machinery Company conducted seismic studies and on-site ripping/scraping tests of the alluvium at Vekol Hills. It was concluded that a substantial part of the 63 million tons of alluvium could be moved with scrapers.

Mr. J. W. Cooksley, Jr., Geophysicist, was retained to conduct refraction seismic studies to evaluate rippability of the alluvium cover. His down-the-hole testing verified and enlarged upon the previous seismic work done by Empire Machinery.

Dames & Moore consultants were retained by the Vekol Copper Mining Company during 1974 to prepare an Environmental Assessment Study of the proposed project for the purpose of assisting the Bureau of Indian Affairs in their preparation of an Environmental Impact Statement as required by Section 102 of the National Environmental Policy Act. Concurrent with the Dames & Moore Study, the Cultural Resources Management Section of the Arizona State Museum was retained to conduct an ethnoarchaeological study of the project in further compliance with Section 102.

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APPenDIX

SECTION 2

SUMMARY

2.1 General

A number of studies of the feasibility of putting the Vekol Hills orebody into production have been made. Each study was based on information and cost data available at the time, as well as mine and plant designs believed feasible and meriting evaluation. Each subsequent study included refinements and corrections over previous studies.

The evaluation reported here is complete and is an updating and revision of the April 1972 Vekol Hills evaluation.

2.2 Capital Requirements and Return

An investment of $106,008,000 would be required to put Vekol into production if the project were initiated in 1976 and production were commenced by July 1978.

The following estimated results could be expected with 100% equity capital.

	Copper Price	
	80¢/lb.	$1.15/lb.
Operating Profits, before taxes	$242,093,736	$510,881,944
Ratio of oper.prof.to total capital	2.28	4.82
Profits after taxes	94,399,000	263,293,014
Ratio of prof. after taxes to capital	0.89	2.48
Payout period on total capital (urs.)	8.47	4.54
Cash Flow after taxes	198,907,006	369,301,014
True rate of return on total capital	7.01%	16.33%
Average yearly total cash available as a percent of total capital	12.50%	23.22%

A one cent increase in average copper price over the life of the operation would result in a change of approximately $4.8 million in net cash flow after taxes.

2.3 Capital Cost

The estimated total capital requirement, at 100% equity, with project initiation in 1976 and completion by July 1, 1978 would be:

	$000
Pre-production stripping	15,395
Mine equipment, shop and drainage control	20,834
Total Mine	36,229
Plant construction	44,393
Owners cost	5,921
Total	86,543
Contingency (10%)	8,600
Land rental, advanced royalty, and other payments	1,099
Operating inventory	1,500
Working capital	8,266
Total Capital	106,008

2.4 Mining

Plans were developed for mining the Vekol orebody at a rate sufficient to supply the mill with 20,000 tons of ore per day. Efforts were made, and alternatives were evaluated, to minimize stripping requirements in the pre-production and early production years. The selected mining plan using a conventional approach with shovels and trucks would result in the following:

Ore:	104,651,000 tons	
Grade:	0.543% copper	0.014% molybdenum
Pre-production mining:	62,000,000 tons	
Working waste/ore ratio:		
1st 3 years of production:		3.7 to 1
Average for orebody:		2.07 to 1

The reserve was based on a copper cutoff grade of 0.30% and pit slopes of 40 to 50 degrees.

An alternative plan, using scrapers for pre-production stripping, was evaluated and shows merit. Tests of the alluvial cover have indicated feasibility. This approach will require further evaluation prior to final commitment to a mining plan and equipment.

In designing the pit full consideration was given to the geology of the orebody, available waste dump areas and haulage requirements.

Equipment selection was on the basis of units proven capable of meeting the pertinent operating requirements. Final selection would be based on competitive bids where more than one supplier can meet requirements.

Basic equipment complement in this evaluation consists of two 11 yard and two 12 yard shovels, three BE 60R drills, twenty 120 ton trucks, three D9G tractors, one 15 yard front end loader and one 75 ton mobile crane.

2.5 Concentrating

The projected plant is designed to process 20,000 tons of ore per day, producing about 326 dry short tons per day of concentrate containing 28% copper and 3 dry short tons per day of molybdenite containing 54% molybdenum.

The plant design used in this study departs from the conventional plant in that building structure, overhead cranes and building enclosures have been reduced to a minimum, fine ore is put into covered ground storage, rather than bins, and the largest available, proven, processing units are used. Process design criteria were based on metallurgical testing.

Pit run ore will be delivered to a 54" gyratory crusher, crushed to -9", stored in a 30,000 ton, live coarse ore pile. Coarse ore will be delivered by conveyors to a closed circuit crushing, screening plant, containing two 7 foot standard crushers, two 7 foot shorthead crushers and five 6' x 16' screens, for reduction to minus ½ inch. The minus ½" fine ore will be conveyed to a 9000 ton, live, covered storage area, from which it will be drawn by conveyor to a single stage grinding plant. Grinding will be done by three 18' x 25'3" ball mills, in parallel, closed circuit with cyclones. Rougher, scavenger flotation will also be in three parallel lines, using eleven 365 cubic feet flotation cells. After cleaning and recleaning in a conventional copper flotation circuit, the copper moly concentrate will be separated in a molybdenum plant. Each product will be dewatered. The copper concentrate will be stored on a covered pad for truck shipment and the molybdenite will be packed in 55 gallon drums.

Tailings at 30% solids will flow by gravity to two 350 feet thickeners and after thickening to 50% solids will be pumped to tailings ponds. The tailings ponds will be formed with starter dams of mine waste. A specially prepared ground cover consisting of a fine grained tailing fraction will be placed to minimize subsurface ground contamination from the tailing storage.

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Make-up process and potable water will be from 3 wells, already drilled. Process water will be from a two million gallon reservoir which will receive decanted water from the tailings ponds, overflow water from the tailings thickeners and fresh, makeup water.

Power will be from the Papago Tribal Utility Authority, who will purchase same from the Arizona Public Service Company.

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SECTION 3

GEOLOGY AND ORE RESERVES

3.1 Geology of the Deposit

The Vekol Hills deposit occurs in a sequence of quartzite, limestones, diabase sills, and sandstones that are Precambrian to Devonian in age. This sequence is intruded by Laramide(?) feldspar porphyry stocks and dikes, and a few small sills of Laramide (?) hornblend porphyry. A surficial deposit of alluvium covers the deposit.

In the immediate vicinity of the deposit the Precambrian-Devonian sequence strikes northeast and maintains a uniform dip of about 30 to 40 degress to the northwest. A small stock of feldspar porphyry intrudes the sequence on the immediate south flank of the deposit, and northeast-striking feldspar porphyry dikes, with southeasterly dips of 40 to 70 degrees, extend into the deposit.

The lowest stratigraphic unit in the mineralized sequence is the Precambrian Dripping Springs Quartzite. This formation is light grey to brown, porous, and is generally thin bedded, well sorted and fine to medium grained where observed. The formation may contain appreciable quantities of potash feldspar and it fractures readily creating an important aquifer. Several diabase sills occur in this formation.

The Precambrian Mescal Limestone conformably overlies the Dripping Springs Quartzite. It is a light grey to reddish brown or white, fine-grained dolomitic limestone commonly containing close-spaced chert bands. Contact metamorphism of the limestone, particularly at the diabase contact, has formed considerable amounts of very fine garnet, epidote, tremolite, chlorite, serpentine and talc. The limestone is well fractured and these fractures are commonly filled with calcite veinlets about 0.1 inch thick. Sulfide mineralization is largely confined to these very fine fractures, and calcite veinlets. This is mainly in the form of very fine-grained pyrite, chalcopyrite and minor prrhotite, commonly accompanied by magnetite. Molybdenite-bearing quartz veinlets occur in the limestone as in the Dripping Springs Quartzite. Moderately oxidized sections exhibit cuprite and green copper staining.

Diabase sills, presumably Precambrian in age, occur in both the Dripping Springs Quartzite and the Mescal Limestone. The diabase is greenish grey, ophitic when unaltered, to weakly foliated when altered. Alteration minerals include biotite, chlorite, epidote, quartz, minor potash feldspar and varying amounts of clay. Primary sulfide mineralization consists mainly of pyrite and chalcopyrite which is essentially confined to fractures. However, quartz veinlets with these sulfides and molybdenite occur as in the Dripping Springs Quartzite. Oxidized portions of the diabase are slightly limonite stained.

The Cambrian Bolsa Quartzite overlies the Mescal Limestone, perhaps with slight angular unconformity, and consists mainly of quartzite interbedded with cross-bedded sandstone in the immediate vicinity of the deposit. This formation is largely barren of mineralization.

Conformably overlying the Bolsa formation is a unit identified as the Santa Catalina formation which is Cambrian in age. The Santa Catalina is dark green to black and consists of limy shale and siltstone. The principal constituents are quartz, montmorillonite, potash feldspar and calcite. Some garnet, epidote and chlorite occur as alteration products. The principal primary sulfides are chalcopyrite and pyrite which occur as disseminated mineralization along fractures and in quartz veinlets. The montmorillonite in the Santa Catalina contains copper, and one analysis, performed on selectively-separated montmorillonite, shows a content of 1.74 percent copper. The sulfides in the Santa Catalina are, in general, much less oxidized relative to the sulfides in the other rock units.

The feldspar porphyry contains pale green to white plagioclase phenocrysts, about 0.1 inches in diameter, in a pink, medium to fine grained matrix of quartz and potash feldspar with accessory amounts of mica and chlorite. Alteration of feldspar to montmorillonite, strong silicification and introduced calcite are common results of hydrothermal alteration. Moderate to strong argillization occurs within the proximity of strong fractures and is possibly due to late hydrothermal alteration. The principal primary sulfides are pyrite and chalcopyrite which mainly occur as disseminations along fractures. Quartz veinlets may be more abundant in the porphyry than in other host rocks. Sulfides in the veinlets are commonly oxidized to malachite, chrysocolla and ferruginous oxides.

Pyrite and chalcopyrite are the main primary sulfides and occur in disseminated form and as fracture-fillings. Pyrite represents a relatively high proportion of the disseminated primary sulfides. The quartz veinlets are generally about 0.2 inches thick and their frequency increases in the proximity of feldspar porphyry intrusive.

Molybdenite tends to be concentrated along the quartz veinlets and is less abundant than the accompanying pyrite and chalcopyrite. Bornite is not uncommon along the fractures and veinlets. Mineralization in the Dripping Springs tends to be more strongly oxidized relative to mineralization in the other rock units. This condition is apparently due to the permeability of the unit.

Further information on the mineralogical composition of the important host rocks, from Progress Report No. 4 is as follows:

Mineralogical Analyses of Rock Type

Mineral	Mescal Limestone	Dripping Springs Quartzite	Diabase	Santa Catalina Formation	Porphyry
% Quartz	2 - 5	45 - 55	5 - 10	30 - 35	45 - 50
% K-Feldspar	-	35 - 45	(2.	15 - 20	20 - 25
% Plagioclase	-	-	10 - 20	1 - 2	5 - 10
% Micas	5 - 10	2 - 5	25 - 35	5 - 10	2 - 5
% Calcite	10 - 15	(1.	(1.	10 - 15	2 - 5
% Amphibole	2 - 5	2 - 5	5 - 15	2 - 5	-
% Pyroxene	5 - 15	-	2 - 5	-	-
% Chlorite	2 - 5	2 - 5	1 - 2	2 - 5	2 - 5
% Montmorillonite	10 - 20	-	?	10 - 20	10 - 20
% Iron Oxides*	2 - 5	2 - 5	2 - 5	1.	2 - 5
% Sulfide	2 - 5	1.	1 - 2	-	-
% Garnet	10 - 20	-	-	-	-
Unidentified Layered Silicate	-	-	Moderate	-	-

* Iron oxides include magnetite, hematite, ilmenite, small amounts of goethite.

There is no distinct pattern of alteration or primary mineralization zoning. This condition is probably due to the contrasting susceptibilities of the various host rocks to hydrothermal alteration and mineralization.

Some generalizations can be made concerning the oxidation of primary mineralization. The oxidation of sulfides and associated gangue decreases with depth, but this vertical decrease is strongly modified by the varying ease with which oxidation takes place in each host rock due to the contrasting lithologies. Irregular and gradational zones of progressive oxidation are recognizable.

Primary sulfides at depth are essentially pyrite and chalcopyrite with minor molybdenite. The quantity of molybdenite present is largely dependent upon the frequency of quartz veinlets. These veinlets occur in all rock types and tend to occur with greater frequency in the proximity of the feldspar porphyry contacts. Increased pyrite content is not necessarily paralleled by an increase in chalcopyrite content; this condition is particularly observed in the diabase and the Dripping Springs Quartzite.

The amount of secondary sulfide mineralization increases upward upon approaching the oxide zone from the primary sulfide zone. The secondary sulfide minerals, mainly bornite and chalcocite, represent over half the copper content within several feet of the oxide zone. Bornite-chalcocite zones also extend downward, along fractures, into the pyrite-chalcopyrite zone.

Mineralization, inside the oxide zone but near the contact of the sulfide zone, consists mainly of native copper, cuprite and usually chalcocite or bornite. These minerals occur in veins and disseminated. The disseminated primary sulfides in the Dripping Springs are commonly altered to chalcocite.

Aside from the usual copper-bearing minerals mentioned above, there are some mixtures of ferruginous oxides from highly altered diabase and from the Dripping Springs Quartzite that have been determined to be mixtures of hematite and goethite, containing about 3.00 percent copper. The proportion of copper-bearing to barren ferrous oxides in the deposit is not known but is expected to be small.

3.2 Surface Exploration Drilling Program

Surface drilling was initiated in 1966 and was completed in early 1970. The surface drill hole sampling program consisted of vertical holes drilled at 200 foot centers along a square, surveyed grid. Rotary drill holes, referred to as "R" series holes, were generally drilled to the water table. The holes were subsequently deepened by core drill holes, referred to as "V" series holes. Rotary drill hole sampling practices and procedures were carried on as follows:

Holes were drilled with tri-cone rotary, tungsten carbide insert percussion and finger rotary bits, depending on ground hardness and other drilling characteristics. Bit sizes ranged from 5-½ to 4-3/4 inches in diameter. Large diameter bits were used to penetrate the alluvium section following which 5-¼ inch inner diameter thin-wall butt-welded casing was set. Once in rock all cuttings were collected from each 5-foot advance, either in a sample box or covered tub, by means of a close-fitting flexible hose connected to an air cyclone. All cuttings were dry or nearly so, and sampling was stopped when a significant amount of water was encountered in the hole. Following each 5-foot run, the hole was thoroughly blown, the hose and cyclone cleaned by shaking and tapping, and the cuttings weighing 60-80 pounds were successively reduced by a Jones Splitter to two or three samples of 5 to 6 pounds each. These were bagged and marked; one sample was assayed, and the remaining splits stored at the Casa Grande warehouse.

Core holes were Nx size, reduced to Bx size where necessary. The split core samples, averaging about 5 feet in length and the rotary samples were assayed for total copper at the San Manuel assay office of Magma Copper Company. Approximately 61% of the sulfide ore intercepts within the pit have been composited and assayed for Mo. The average length of each composite was about 20 feet or 4 individual sample intervals. Gold and silver assays were limited in number, inasmuch as meaningful precious metal content in this type deposit can best be determined from recoverable quantities in copper or molybdenum concentrates. Acid soluble copper analyses have been performed on samples composited for bench elevation intervals. Split core not used for sample preparation, rotary sample rejects and duplicate samples in cloth bags, each containing 5 to 6 pounds of sample, and all pulps and rejects of core samples are stored in Newmont Exploration Limited's Casa Grande warehouse. Certain of this stored material has subsequently been used for check assay and metallurgical test work.

A total of 192 holes were drilled within the maximum pit perimeter. Of these 51 were rotary holes, 16 were cored from bedrock, and 125 were rotary holes deepened by coring.

Approximately half of the holes were surveyed by Tro-Pari. Readings were recorded at 200-foot intervals for the majority of holes and some at 100-foot and 50-foot intervals. About 25 percent of the holes were estimated to have a horizontal deflection of about 50 feet. The maximum horizontal deflection was estimated to be on the order of 100 feet per 1000 feet of depth. Most deflections were in the same direction as might be expected from the interpreted prevalent northwestward dip of lithologic units. The magnetite content of the rocks is also expected to have influenced the deflection measurements.

3.3 Underground Exploration Program

Underground exploration on the 1400 elevation was designed to check sampling and assay data compiled from the surface drilling, test continuity of ore grade material between drill holes as well as rock type and structural projections, and provide bulk samples for metallurgical test work. Starting March 10, 1970 a two compartment timbered shaft was sunk 441 feet, from which 2809 feet of crosscut and drift was driven and five vertical raises totaling 955 feet were driven. Following is a summary showing position and effective footage of the headings, disregarding distance on curves, stations and turnouts.

Heading	Drill Grid Position	Footage
Shaft	N200 E200	411' (incl. sump)
E 2 XC N & S	From 570 to N600	670'
N 6 Dr. E	From E200 to #1095	895'
N 6 Dr. W	From E200 to W240	440'
N 2 Dr. W	From E200 to W10	210'
Zero Dr. W	From E200 to W290	490'
Raise R59	N400 E200	210' above sill
R62	N600 W200	200' above sill
R67	N600 E400	221' above sill
R78	N Zero E200	137' above sill
R79	N Zero W200	202' above sill

Core drilling from underground totaled 12,603 feet. Part of the drilling consisted of fan patterns in vertical planes paralleling specified grid lines to substantiate continuity of ore grade material between drill holes, both above and below the level. The second objective was a series of horizontal to sub-horizontal holes aimed at specific targets relating to structure and/or rock competency affecting pit design. All drill core was split and assayed for total and acid soluble copper.

A 405-ton bulk sample was mined and shipped to Houghton, Michigan for grindability and related metallurgical test work, supervised by Newmont Exploration Limited personnel at the pilot plant facilities of the Institute of Mineral Research, Michigan Technological University. The sample was selectively mined to conform to estimated proportional quantities of rock types contained in the deposit.

Sampling of all rock excavated was on the basis of individual blasts. Each round in shaft, drifts and raises was transported from the heading and hoisted separately. At the surface each round was separately passed through a crushing plant consisting of a primary crusher and two sets of roll crushers. Following the first set of rolls the 3/4 inch product passed through a Vezin type mechanical sampler where 5% of the total tonnage was cut as a sample to be passed to the secondary rolls. This product, reduced to 3/8 inch passed through a similar mechanical sampler where 5% of the product was cut to be further split through riffles and prepared for assay pulps. Rejects from the first set of rolls were stockpiled and rejects from the second rolls were sealed in metal drums for storage at the Casa Grande warehouse.

Survey control for all underground work was done by a registered professional engineer. Careful attention to detail of openings including floor and back elevations and raise detail was recorded to provide all necessary information for future pit operation. Geologic mapping of underground openings and core logging was done by Newmont Exploration Limited personnel.

The results of the underground work confirmed the ore-waste boundaries and rock type and structural interpretations extrapolated from surface drilling. Analysis of assay data both in the drifts and raises indicate a generally higher copper content than shown by averaging drill hole samples in this test area.

3.4 Reserve Estimate

The estimate of tons and grade within the currently proposed limits of the Vekol Hills pit is based on both computer and manual calculations. These calculations have been performed and updated during February to December 1970 to correspond with refinements in the pit plan. On February 24, 1970 a manual calculation of reserves for a preliminary pit plan was completed. New preliminary pit limits were established by Mr. W. K. Pincock, Consultant for Newmont, on March 10, 1970. Reserves were updated during March 10 to May 23, 1970, to agree with the new plan. On October 29, 1970, a computer check calculation was completed on the manually calculated reserves of February 1970. Pit limits were further revised by Mr. Pincock during November 1970 on the basis of estimated mining costs, the value of the contained copper and waste stripping ratios. Mr. Pincock's November, 1970 pit plan increased the size of the proposed pit and the May 23, 1970, reserve estimate was manually updated for tonnage by Mr. Pincock;

however, no corresponding grade calculations were performed. On December 8, 1970 a computer calculation was performed to check the tonnage, obtained manually for Mr. Pincock's pit plan of November 1970, and to obtain grade corresponding to Mr. Pincock's tonnages. The current reserves consist of Mr. Pincock's tonnages measured in November 1970 and the corresponding grades obtained by computer methods on December 8, 1970.

The procedures used to calculate the reserves in February 1970 were as follows:

1. Northwest-trending and northeast-trending vertical sections, on 200-foot spacing (corresponding with the drill grid) were constructed at a scale of 1" = 50', and the surveyed positions of the drill holes plotted on them.

2. Geological and assay data from the drill logs and data sheets were transferred to the plotteddrill holes. This included rock type, structure, mineralization type (oxide or sulfide), and total copper assays.

3. Hole to hole correlations of rock type, structure and ore intercepts were made on both sets of sections, and the geology and ore block outlines rectified between the two sets. The criteria used to outline the ore types were:

   Sulfide ore - 0.30 percent or more total copper and 0.25 percent sulfide copper.

   Oxide ore - 0.30 percent oxide copper and less than 0.25 percent sulfide copper.

   Waste rock - less than 0.30 percent total copper.

   Areas of sulfide or oxide ore were delineated on visual estimates of the copper minerals noted in the drill logs, pending analysis for acid soluble copper from bench interval sample-pulp composites.

4. Geologic plans were prepared for the median elevation of each bench to scale 1" = 200 feet, utilizing data from the vertical cross sections. The outlines of the geologic ore blocks were finalized on these plans utilizing all geologic data available.

5. A set of bench median plans were prepared to scale 1" = 200 feet, showing only bench composite assays at drill hole pierce points, the geologic ore block outlines and the pit outline.

6. Blocks of mineable ore were established for each bench on each vertical section, utilizing the geologic ore block outlines as a guide, and vertical cutoff lines were drawn delimiting the intervals of sulfide ore, oxide ore, waste and alluvium within the pit. The horizontal intervals of the reserve types between the vertical cutoff lines were measured, and their locations transferred to the 1' = 200 feet bench plans. Each of these measurements was also tabulated by section number, bench elevation and reserve type.

7. On each bench plan, the vertical cutoff lines were joined to delimit the boundaries of polygonal areas of alluvium, waste and the ore reserve types, and the areas of the polygons were calculated. The corresponding volumes of the reserve types were determined utilizing the 50-foot bench height factor, and converted to tons using the following tonnage factors for dry, in-place, short tons:

Type	Volume (cu ft./ton)
Sulfide ore	12.0
Oxide ore	12.5
Waste rock	12.5
Alluvium	14.9

These tonnage factors are based on the results of specific gravity measurements taken on composite of drill core.

8. Grades were determined from the northwest-trending sections for each horizontal interval of each bench between the vertical cutoff lines by first cumulating, for each drill hole, a set of sample assays corresponding to the bench intercept (usually 8 to 10 samples). These drill hole bench intercept grades were next used to obtain the grade of each horizontal bench reserve type, usually by calculating an area-weighted average of the bench intercept grades between the respective vertical cutoff lines. However because of the strong influence of bedding upon grade, geological interpretation was occasionally used to project grade values from pertinent drill holes above or below the bench and weight the grade of small, stratigraphically controlled reserve blocks. Grades were considered to extend 100 feet northeast and 100 feet southwest of the section line.

9. Tonnage and corresponding grade for each ore reserve block on each bench along the northwest-trending sections and weighted averages of the ore-blocks on each bench have been calculated and totaled on the reserve record sheets.

Total copper, soluble copper and molybdenum analyses were done by Magma Copper Company at San Manuel, check assays were by Union Assay office, Salt Lake City.

Acid soluble copper determinations were run on 738 bench interval composite samples representative of the deposit and confirmed the distribution of oxidized and partially oxidized mineralization defined in the ore reserve estimate.

3.5 Molybdenum Reserve

A statistical study of the molybdenum grade for mineable sulfide ore blocks within the pit design of January 30, 1970 was made in February, 1970. The basic information used for this study was as follows:

1. Drill hole samples collected and assayed for copper during the 1966-1969 period and which form the basis for the present copper ore reserve estimate were, in part, (941 samples, 61% of total) composited and analyzed by Magma for Mo. The samples which were not assayed for Mo. (39% of total samples) came from random locations within the deposit and were omitted primarily because of an excessive work load at the Magma laboratory during the period the samples were collected and delivered for copper analysis. Each of the composited samples assayed for Mo. represents about 20 feet of drilled ore intercept.

2. The weighted average Mo. content was computed for the total footage, distinguishing between the northeast or Santa Catalina ore, with a markedly lower Mo. content and the larger central orebody.

3. The average Mo. content of these two areas was applied to the then current ore reserve tonnages and combined to provide an average grade for the undiluted sulfide ore tonnage.

4. The average grade reported at 0.016% Mo. and considering dilution, a working figure of 0.014% Mo. was recommended.

Subsequent metallurgical test work conducted on a master composite sample made up of a substantial quantity of drill core and lesser amount of rotary cuttings reported 0.012% Mo. Details regarding the makeup of this sample are in the Danbury laboratory Progress Report No. 3, May 25, 1970.

A review of all factual data was undertaken in March, 1972, to reconcile the independently arrived at differences in apparent molybdenum content. The problem resolved itself to a study of sample density relative to the pit parameters now established.

In order to do this the following procedures were followed:

1. All Mo. composite assays were plotted on a set of 100 scale cross sections.

2. Bench averages were calculated and plotted on both sections and bench plans to better illustrate the lateral as well as vertical distribution of sample density relative to copper sample density.

3. Drill core and rotary samples sent to Danbury in December, 1969, and the first quarter of 1970 for the metallurgical and master composite sample were plotted on the same sections and plans to show the spatial relationship of this sampling to the pit outline and to the composite sampling.

4. The total number of bench interval molybdenum averages and the total number of bench intervals represented in the master composite sample were compared with the sulfide copper intercepts used in tonnage estimates. Sixty-one per cent of the total copper intercepts have been assayed for molybdenum and 35% are represented in the master composite. Additional core from less than .30% Cu zones within the pit and some core, both ore grade and waste, outside of pit limits was also included in the master composite.

5. Molybdenum assay intercepts were averaged arithmetically for each bench and the bench sulfide ore tonnage applied to calculate an average grade for the deposit. The average grade by this method is 0.0166% Mo. compared to 0.0161% reported in 1970.

Inasmuch as the master composite sample included material not within the present sulfide ore boundaries and was selected for specific metallurgical research purposes regarding rock type and mineralogical properties, the more complete coverage of the composited molybdenum samples is considered more representative of the sulfide ore within the pit limits.

SECTION 4

METALLURGY

4.1 INTRODUCTION

[illegible] metallurgical testing of the Vekol Hills deposit [illegible] in November 1969 and [illegible] continued [illegible] [illegible] were tested individually [illegible] [illegible] of the drill core [illegible] were [illegible] of the rotary drilling.

In addition three bulk samples from the underground explor-ation were subjected to laboratory flotation and [illegible] [illegible] pilot plant grinding and flotation test-ing. The pilot plant copper concentrate was employed in labor-atory testing of the copper/molybdenum separation process.

4.2 SUMMARY

The Vekol Hills orebody is quite complex [illegible] with [illegible] different rock types [illegible] which contain significant [illegible] of the [illegible] [illegible] [illegible] [illegible] [illegible] of importance are the [illegible] [illegible] [illegible] and [illegible]. A significant [illegible] [illegible] [illegible] [illegible] of the [illegible] [illegible] [illegible].

Copper oxidation is severe in the upper levels of the orebody but decreases with depth [illegible] [illegible]. At and below [illegible] [illegible] [illegible] of [illegible] [illegible] [illegible] [illegible]. However, [illegible] amounts of highly oxidized material, mainly in the [illegible] and limestone formations, [illegible] on and below the [illegible] [illegible].

Copper sulfide mineralization is mainly in the form of [illegible] with minor amounts of [illegible] and [illegible]. [illegible] amounts of pyrite have been detected.

[illegible] [illegible] [illegible] [illegible] [illegible] [illegible] [illegible] in the form [illegible] [illegible] [illegible] [illegible] [illegible] also been [illegible] [illegible] and [illegible] [illegible] [illegible] [illegible] [illegible] [illegible] [illegible].

SECTION 4

METALLURGY

4.1 Introduction

Intensive metallurgical testing of the Vekol Hills deposit was started by the Danbury laboratory in November 1969 and has continued intermittently up to the present. During this period samples from 74 rotary drill holes and 52 core holes were tested, individually and/ore in composites by laboratory procedures. Many of the drill core holes were extensions of the rotary drilling.

In addition three bulk samples from the underground exploration were subjected to laboratory flotation and grindability testing and to pilot plant pebble grinding and flotation testing. The pilot plant copper concentrate was employed in laboratory testing of the copper/molybdenum separation process.

4.2 Summary

The Vekol Hills orebody is quite complex geologically with thirteen different rock types identified, at least five of which contain significant tonnages of ore grade material. Within the main pit, the ore bearing formations in decreasing order of importance are the diabase, quartzite, limestone, shale and porphyry. A significant tonnage of ore grade material in the northeast extension of the pit lies mainly in the shale formation.

Copper oxidation is severe in the upper levels of the orebody but decreases with depth down to approximately 1400 ft. elevation. At and below this elevation an average of approximately 7% of the total copper reports as oxide. However, significant tonnages of highly oxidized material, mainly in the quartzite and limestone formations, exist on and below the 1400 ft. elevation.

Copper sulfide mineralization is mainly in the form of chalcopyrite with minor amounts of chalcocite and covellite. Only minor amounts of pyrite have been detected.

Oxide copper mineralization occurs generally in the form of malachite and chrysocolla. Oxide copper has also been detected in montmorillonite and associated with hematite and goethite. Oxide copper associated with the iron oxides is undetected by the oxide copper analytical technique but reports in the total copper assay.

As the "sulfide" copper content of the Vekol Hills ores is obtained by the difference between the total and oxide copper assays, the presence of the copper-bearing iron oxides results in high "sulfide" copper contents of the heads and apparently low sulfide copper recoveries in flotation.

Four classifications have been provided for material in the Vekol Hills orebody, namely "sulfide" ore, "mixed oxide-sulfide" ore, "oxide" ore and waste. "Sulfide" ore is considered suitable as feed to the sulfide copper flotation plant and the specifications are "not less than 0.30% total copper and not less than 0.25% sulfide copper."

As this can include highly oxidized material that gives poor recoveries of the sulfide copper in flotation and as the "sulfide" copper content can be misleading, due to copper containing iron oxides, it is recommended that all concentrator feed be tested by laboratory flotation techniques on blast-hole samples.

A "Master Composite Sample" representing all "sulfide" ore within the main pit had head values of:

0.53% total copper. .051% oxide copper. .012% molybdenum.

The total copper content is in reasonable agreement with the ore reserve calculations.

4.3 Copper Metallurgy

Laboratory, bench scale, closed-circuit flotation of the "Master Composite Sample" recovered approximately 85% of the total copper and 75% of the molybdenum in a concentrate assaying 31.32% copper and 0.62% molybdenum and containing 0.02 ounces of gold and 2.50 ounces of silver per ton. On the basis of this and other flotation testwork, the overall copper metallurgy for the Vekol Hills sulfide ores is estimated at:

	Ratio of Concentration	Assays Cu.%	Au.oz.	Ag.oz.	Recovery Cu.%
Heads. (Ore reserves plus dilution)		0.543			
Copper Concentrate	61/1	28.00	0.02	2.50	85

The above estimate applies to all sulfide ores within the main pit and the northeast extension. However, copper recovery will vary with the type of ore and the pit elevation. The laboratory testing indicates that copper recovery will average approximately 75% on the quartzite and limestone ores and close to 93% on the diabase and shale.

Estimates of copper recovery by bench level were made based on very limited flotation test results.

Bench Level	Estimated % Copper Recovery
1650 & 1600	45
1550	62
1500	73
1450	85
1400	89
Below 1400	89

Spectrographic analyses of copper concentrates produced in laboratory closed-circuit tests suggest that molybdenum, gold and silver are the only by-product credits available. However, no smelter penalties should be incurred in the smelting of the Vekol Hills copper concentrates.

"Sulfide" ores in the northeast extension of the main pit lie mainly in the Santa Catalina shale formation and locally high concentrations of zinc were reported. Detailed testwork on all drill samples from this area indicate that while copper concentrates containing up to 12% zinc may be produced on occasions, overall the zinc will not be a major problem. Blending of the northeast extension ores with ores from the main pit will limit the occurrence of high zinc assays in the copper concentrate.

Copper metallurgy on ores from the northeast extension is generally slightly better than that obtained on ores from the same elevation from the main pit.

4.4 Molybdenum Metallurgy

The molybdenum content of ores tested at Danbury has averaged .011%, with ores from the upper levels reporting slightly lower and ores from the lower levels slightly higher than this figure. This compares to a head of .015% Mo. before dilution and .014% after dilution which was derived from the early drilling at Vekol Hills. (See Section 3.5 Molybdenum Reserves.)

Molybdenum recovery in the copper concentrates is variable but tends to increase with increasing depth. Batch flotation tests on sulfide ores within the main pit indicated recoveries of approximately 40% on the 1550 bench and above, 65% on the 1500 bench and 85 to 90% below this.

Laboratory studies on copper-molybdenum separations were carried out on copper concentrate produced in the pilot plant operations. These studies suggest that the Vekol Hills copper-moly concentrate responds well to a slightly modified San Manuel separation technique. This technique allowed for the recovery of 60% of the molybdenum contained in a copper concentrate assaying 0.64% Mo., in a molybdenum product assaying:

56.01% Mo.; 0.85% Cu.; 0.13% Pb.; .071% Re.

It is confidently predicted that plant operations on similar material would allow for the recovery of 75% of the molybdenum in the copper concentrate. However, molybdenum recovery in the final molybdenum product will be a function of the molybdenum content of the separation feed and this in turn will be a function of the molybdenum heads to the copper circuits and the recovery obtained therein.

Considering a reserve grade of .014% Mo., the following is the estimated molybdenum metallurgy.

Mo.% in plant feed	.014
Mo. recovery in copper circuits	80%
Mo.% in copper concentrate	.68
Mo. recovery in separation circuits	75%
Overall Mo. recovery	60%
Mo.%, in concentrate	50.0
Cu.% in concentrate	Less than 1.0
Pb.%, in concentrate	Less than 0.25
Re.%, in concentrate	.07
Ratio of concentration	5950/1

A spectrographic analysis of the molybdenum concentrate indicates that rhenium is the only possible by-product credit for this concentrate.

4.5 Grinding Tests

A pilot plant investigation of three bulk samples and a composite of the three bulk samples indicated that the Vekol Hills ores are amenable to pebble grinding techniques. Conventional grinding tests in the pilot plant indicated however that the Vekol Hills ores will have a low steel media consumption and that the use of pebble grinding would not be economically justifiable.

4.6 Design Criteria

The following criteria based on metallurgical findings and accepted practices were established as a basis for plant design.

Metallurgy

	Product Weight DST/D	Assays % Cu.	Assays % Mo.	Metal Content Tons/Day Cu.	Metal Content Tons/Day Mo.	Distribution % Cu.	Distribution % Mo.
Heads	20,000.0	0.543	0.014	108.60	2.80	100.0	100.0
Cu. Conc.	326.6	28.3	0.172	92.28	0.56	84.98	20.0
Mo. Conc.	3.1	0.85	54.0	0.03	1.68	0.02	60.0
Tails	19,670.3	0.083	0.0028	16.29	0.56	15.00	20.0
Combined Cu-Mo Concentrate	329.7	28.0	0.679	92.31	2.24	85.00	80.0
Rougher-Scav. Concentrate	1,608.0	6.0	-	96.44	-	88.8	-

Overall Plant

Feed Rate S.T.P.D. 20,000 tons

Mining Schedule - Ore Delivery

Two shifts/day, seven days/week, 20,000 tons/day or 10,000 tons/shift.

Primary Crushing

Crusher Size	54" Gyratory
Feed rate S.T. per week	140,000
Shifts/week	14
Effective hours/shift	6
Design rate S.T.P.H.	3,000 tons

Flotation Sections

Flotation Stage	Retention Time Minutes	Feed % Solids
Roughers	5	32
Rougher Scavenger	5	32
Cleaners	3	16
Cleaner Tails Scavengers	10	17
Recleaners	3	13

Reagents

Reagent	Solution % Strength	Consumption #/ton Plant Fd.	Solution Storage (Hours)
Aeropromotor No. 3302 (Liquid use w/o dilution)	100%	0.009	24
Aeropromotor No. 3501 (reactive caustic white solid, water soluble)	10%	0.030	24
Dow Froth (liquid use w/o dilution)	100%	0.025	24
Pine oil (liquid use w/o dilution)	100%	0.025	24
Lime	10%	4.4	12

Bulk storage for collectors and frothers - days - 30

Bulk storage for lime tons - 400, use solution tanks each 6 hours

Concentrate Regrind

Percent Availability	97
Feed rate (rougher & scavenger conc.) S.T.P.H.	67
Mill design power (based on total rougher & scavenger conc.) KwH/ton	4.82
Approx. mill motor hp.	500

Classifying Cyclones Regrind Circuit

Size separation percent - 325 mesh	95
Overflow density	16
Underflow density	70

Concentrate Dewatering

Thickener size factor ft.^{2}/ S.T.P.H.	9
Underflow density percent solids	60

Filtering

Feed percent solids	60
Loading lbs./hr./ft.^{2}	48
Cake moisture percent	14

Vacuum Pumps

Vacuum inches Hg	23
Capacity CFM/ft.^{2}	3

Dryer

Feed rate S.T.P.H.	14
Discharge Moisture percent	7
Size factor lbs. water/ft.^{3}/hr.	2.5

Concentrate Storage

Capacity hrs.	72
Concentrate bulk density lbs./ft.^{3} loose	100
compacted	135
Angle of repose	
Minimum for area	40
Maximum for volume	45

Tailings Dewatering

Thickener size factor ft.^{2}/S.T.P.D.	10
Underflow density percent solids	50
Tailings pond	
Number required	2
Size factor 20 acres per 1000 daily tons min.	400 acres
Tailings area available - acres	700 approx.

SECTION 5

[illegible]

General

A [illegible] mining plan has been worked out for [illegible] years. After ten years the stripping would be [illegible] [illegible] the [illegible] for the [illegible] would be estimated.

This mining plan is based on ore reserves which have been [illegible] to [illegible] certain material which has been found through flotation testing to yield [illegible] production. This revision has had the effect of [illegible] some 4,711,000 tons (with indicated [illegible] of less than [illegible]) from the [illegible] and requiring some five million tons additional pre-production stripping.

It is recognized that, as pre-production stripping progresses, metallurgical testing of blast-hole samples [illegible] sufficient [illegible] ore as to reduce the [illegible] required. [illegible] the [illegible] million [illegible] based on [illegible] flotation testing. However, there is no doubt that the present plan will provide [illegible] at the rate of [illegible] tons per day [illegible].

[illegible] selection has been made so as to meet the [illegible] of this mining plan, based on a mining [illegible] of a seven day work week consisting of [illegible] production [illegible] and one shift down for equipment maintenance. The work schedule was dropped back to a five-day week as [illegible].

Alternative Mining Methods

As described in Section 3, a number of alternative [illegible] to [illegible] conventional methods have been considered. [illegible]

[illegible]

SECTION 5

PROJECT DESCRIPTION - MINE

5.1 General

A detailed mining plan has been worked out for ten years. After ten years the stripping would be essentially complete and the mining operation is rather straightforward so that the tonnages for the remaining years could be estimated.

This mining plan is based on ore reserves which have been revised downward to remove certain material which has been found through flotation testing to yield uneconomically low recoveries. This revision has had the effect of removing some 4,221,000 tons (with indicated recoveries of less than 55%) from the ore reserve and requiring some five million tons additional pre-production stripping.

It is recognized that, as pre-production stripping progresses, metallurgical testing of blast-hole samples might prove-up sufficient amenable ore as to reduce the amount of pre-production stripping required, since the recoveries used in removing the four million tons were based on limited flotation testing. However, there is no doubt that the present plan will provide timely access to metallurgically acceptable ore at the rate of 20,000 tons per day beginning with year one.

Equipment selection has been made so as to meet the tonnage requirements of this mining plan, based on a mining schedule of a seven day work week consisting of 20 production shifts per week and one shift down for equipment maintenance. The work schedule was dropped back to a five-day week as the tonnage requirements decreased.

5.2 Alternative Mining Schemes

As discussed in Section 1, a number of alternative approaches to mining the Vekol orebody have been considered, several of which were evaluated in considerable detail, according to their indicated merits. Several pit designs were evolved in attempts to minimize pre-production stripping and improve cash flow.

Removal of alluvial materials with scrapers was also studied with the objective of minimizing stripping costs and pre-developing the orebody such that operating equipment and costs could be minimized.

Seismic testing and ripping-scraping tests were made which indicated that a large part of the alluvial overburden can be removed with scrapers.

The April 1972 Project Evaluation made a comparison of the capital requirements and cash flow between the conventional shovel/truck approach and the two cases utilizing scrapers in combination with shovels and trucks based on 100% equity and 50¢/lb. copper. While the capital and operating costs used are no longer valid, the comparative relationship is still relevant and is as follows:

	Shovel/truck	Scraper Case I[1.]	Scraper Case II[2.]
Capital required($000)	64,513	68,898	65,484
15 yr. cash flow($000)	120,118	126,408	123,195
Net	55,605	57,408	57,711
Payback to equity (yrs)	8.90	8.75	8.70
True rate of return	6.48%	6.51%	6.81%

As can be seen, the use of scrapers, based on preliminary estimates and cost data provided by equipment suppliers, apparently has merit. Therefore, it is intended that, prior to a final commitment on mining plan and equipment, further evaluation will be made by securing firm bids for waste removal from several interested construction companies.

However, this study, as mentioned previously, has been based on the conventional shovel/truck system as the basic estimating data, cost and capabilities, are available and believed more reliable.

5.3 Pit Design

The rock types encountered in the pit area include both diabase and porphyry intrusives as well as quartzites and limestones. The

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1. Scraper Case I considered removal of 47 million tons of pre-production material with scrapers and 47 million tons with shovels and trucks.

2. Scraper Case II would remove 47 million tons with scraper and 28 million tons with shovels and trucks. Both cases sized the shovel/truck fleet so as to continue with the same equipment complement during the first several years of production.

Dip of the sedimentary rocks is slightly toward the northwest and is favorable to a steep slope particularly on the northern portion of the pit as well as both ends.

The geological group started with the basic drill hole data and developed cross sections and longitudinal sections at 200 foot intervals as well as level maps on 50 foot intervals. Ore outlines, lithology and their best interpretation of the faulting pattern were shown on all these maps.

Using a cutoff grade for ore of 0.30% Cu., a graph was constructed showing the breakeven stripping ratio for the grade ranges encountered at Vekol. The data from this curve was used to determine the breakeven limit on each section. The level intercepts of this breakeven limit were transferred to a plan map and formed the basis for a preliminary pit design.

Modifications were made to this preliminary design based upon the results of computer output on drill hole locations and bench composite grades as well as an economic analysis of reserves.

A maximum slope of 40° in the alluvial cover was used. In the southwestern portion of the pit a maximum slope of 48° was used. The maximum slope used in the ends and northern area is 50°. The overall slopes from top to bottom in the various areas including haul roads are marked on the pit composite map and vary from 49° 38' to 35° 26'.

There are several faults including two major ones crossing the pit to the northwest. The faulting crosses the pit at a favorable angle and the crushed zones do not appear to be extensive. Depending upon the width of these zones, some small and limited failures can be expected, but these can be coped with.

The possibility exists that as future cost studies are made, it may be advantageous to install in-pit crushing. The present ramp system is designed with this possibility in mind and could accommodate a crusher-conveyor installation with no further modifications.

With the information currently available this design is feasible. The most critical area in the pit wall is on the southwest side in the porphyry area. Some additional drilling was done in this area to get more detailed data on the ground conditions and indicates that some slight modifications may be made, however, this will not affect the feasibility of the project. Design modifications should not be finalized until some detail geology in this area becomes available from mining operations in the upper benches.

5.4 Reserves

A. Specific Gravity

Material Weights used for this study were (as discussed in Section 3):

Sulfide Ore	-	12 cu. ft. per ton
Oxide and Waste	-	12.5 cu. ft. per ton
Alluvium	-	14.9 cu. ft. per ton

B. Ore Outlines

The ore outlines were established by the geological group on both sets of sections as well as the level maps. The effect of various rock types, dip of formations, faulting, as well as assay information, have all been given consideration in the location of the ore outlines on the level maps.

The definition of sulfide milling ore as used in establishing these reserves is material that contains 0.30 percent or more total copper and 0.25 percent or more sulfide copper.

Soluble copper assays were made to establish the amount of oxide present in the total copper assay. The difference between the total copper and soluble copper being taken as sulfide copper. Below the oxide zone this is essentially true in that dis-crepancies are negligible. However, in the upper oxidized area the difference between the two assays does not necessarily signify sulfide. It may contain some nonsoluble oxides which still leaves the amount of sulfide copper in doubt. Flotation tests were conducted in order to get a better picture of the actual percentage of sulfides in these upper ore areas.

The ore outlines, from which the following tabulation (Table 5-1) of reserves was made, were established in accordance with sound and accepted geological and engineering practices. The procedures used are famil-iar and these outlines have been re-examined several times. There is no doubt that this is a realistic estimate of reserves. Although there may be variations experienced geographically from the outlines shown, it is expected that the overall tonnage and grade of reserves will be recovered by mining.

VEKOL RESERVES - REVISED PIT LIMITS

Tonnages (in thousands)

Before Dilution

Level	Ore Tons	Ore Grade	Oxide Tons	Oxide Grade	Alluvium Tons	Waste Tons	Total Tons
1900 & up						1,109	1,109
1850						1,504	1,504
1800					8,965	3,740	12,705
1750					23,157	6,030	29,187
1700			559	.56	15,756	12,550	28,865
1650			2,645	.54	9,540	15,157	27,342
1600			6,409	.58	4,548	15,120	26,077
1550	1,892	.546	5,295	.52	1,611	15,770	24,568
1500	6,598	.530	1,943	.49		14,177	22,718
1450	8,187	.520				12,686	20,873
1400	8,710	.511				10,262	18,972
1350	8,635	.532				8,488	17,123
1300	8,393	.569				7,014	15,407
1250	8,327	.543				5,328	13,655
1200	7,380	.586				3,986	11,366
1150	7,230	.551				2,773	10,003
1100	6,803	.579				1,891	8,694
1050	6,302	.642				1,250	7,552
1000	5,867	.614				560	6,427
950	4,640	.581				510	5,150
900	3,667	.599				379	4,046
850	2,940	.488				267	3,207
800	2,150	.630				250	2,400
750	1,225	.617				384	1,609
700	722	.447				38	760
Total	99,668	.559	16,851	.54	63,577	141,223	321,319
Dilution	4,983	.20				-4,983	
Total	104,651	.543	16,851	.54	63,577	136,240	321,319

%Mo = 0.014 after dilution. Waste-Ore Ration 2.07 to 1

Table 5-1

The oxide tonnage listed in the tabulation includes some oxides in limestone which have not been delineated completely. Provision has been made to segregate the nonmillable material including oxides and mixed ore on a dumping area. Blast hole assays and flotation tests will be the primary control in the mining operation and will determine the final disposition of all material.

C. Dilution

The ore outlines are based on composite bench assays and as such include some vertical dilution in order to make the orebody conform to a bench type configuration. The grades shown include this type of dilution.

Another type of dilution is related to the shovel operation and is proportionate to the amount or length of ore-waste contacts. This total interface length was measured on all levels. Allowance was made for a strip of material equivalent to a 10-foot width to be taken with ore as mining dilution along this ore-waste contact zone. The tonnage resulting from this study was approximately 4.5 percent of the total ore. As a result, a dilution factor of 5 percent was used.

Since this diluting material comes from the contact zone and the ore cut-off is 0.30 percent Cu., it is assumed that it will also have some copper values. A figure of 0.20 percent Cu., has been assigned to this diluting material.

5.5 Mining Plan

The primary purposes of a mining plan are to determine the extent of pre-production tonnage that will have to be moved to allow mining to start and the amount of waste that will have to be mined concurrently with the ore to allow continuing ore production at the rate specified. This latter value will be referred to as the operating waste-ore ratio.

The plan will indicate at what period in time this operating waste-ore ratio can be reduced. It will also point up any difficulties that might be encountered with respect to maintaining continued access to all areas as well as other problems that could not be foretold.

A preliminary plan was developed along the following lines:

1. A milling rate of 20,000 tons per day, 355 operating days per year, 7,100,000 tons per year.

2. Selecting the ore tonnage to insure a maximum grade in the early years.

3. Blocking out the required ore tonnage and taking the necessary amount of waste to make this tonnage available.

4. Maintaining the level development in an orderly pattern allowing for maximum flexibility.

5. Making sure that room for access ramps is available at all times to all operating levels.

6. Tying in the final or permanent ramp system as soon as practicable.

This plan was only carried through ten years of mining because at this point there is very little waste remaining and the remaining ore grade could be maintained at a uniform level with no difficulty.

After studying the results of the preliminary plan the annual tonnages for the final or operating mining plan were set up and are shown in the following summary (Table 5-2).

5.6 Equipment Selection

The basis for the selection of the major pieces of equipment (shovels, drills, trucks) is outlined below:

1. Shovels

The daily tonnage to be moved during the pre-production and the first three years of operation amounts to approximately 100,000 tons per day on the basis of a seven-day week and 20 operating shifts per week.

The 1900 P&H shovel with an 11 yard dipper has the capability of producing at the rate of 12,000 tons per shift. A schedule of nine shovel shifts per day would require the minimum number of trucks and fits the total tonnage requirements.

MINING PLAN - Vekol Hills

(tonnage in thousands)

Period	Ore (+0.30% Cu.) tons	Grade	Alluvium tons	Waste tons	Total tons	Daily tons	Shovel Shifts per day	Shovel Shifts per wk	Operating days/wk
Pre-Prod.			29,172	32,828	62,000	99.1	9	55	7
Year 1	7,100	.529	7,220	19,180	33,500	99.1	9	55	7
Year 2	7,100	.517	12,029	14,371	33,500	99.1	9	55	7
Year 3	7,100	.514	10,232	16,168	33,500	99.1	9	55	7
Year 4	7,100	.551	3,739	13,161	24,000	71.0	6	40	7
Year 5	7,100	.560	1,027	15,873	24,000	71.0	6	40	7
Year 6	7,100	.527	158	9,742	17,000	65.4	6	28	5+
Year 7	7,100	.547		8,400	15,500	60.0	5	25	5
Year 8	7,100	.561		5,900	13,000	50.0	4	21	5
Year 9	7,100	.535		4,700	11,800	45.4	4	19	5
Year 10	7,100	.593		3,400	10,500	40.4	3	17	5+
Year 11	7,100	.541		2,800	9,900	38.1	3	16	5+
Year 12	7,100	.541		2,400	9,500	36.5	3	16	5+
Year 13	7,100	.541		2,200	9,300	35.8	3	15	5
Year 14	7,100	.541		1,768	8,868	35.3	3	15	5
Year 15	5,251	.541		200	5,451	29.6	3	13	5
Totals	104,651	.543	63,577	153,091	321,319				

Note: When the work schedule drops to a 5-day week, an ore shovel would always be scheduled for at least Saturday or Sunday. The crew would be held to a 5-day schedule with the exception of Years 6, 10, 11, 12, which would be handled on overtime.

Table 5 - 2

The mining plan also dictates that following the pre-production period operations will be on 9 to 10 benches annually. In this case the geography of the operation also has a bearing on the number of units required for good operations. This fact together with the tonnages to be moved dictated the selection of the 1900 P&H with 11 and 12 yard dippers--the 12 yard dipper to be used in the alluvial areas and the 11 yard used in rock.

Shovel utilization should normally be held in the range of 67 percent to a maximum of 75 percent. The mining plan at Vekol sets a maximum of 75 percent for the initial five years of operation (two years pre-production and three years of actual mining). Since the location of the operation is close to both Tucson and Phoenix as opposed to being in a remote area, parts and services should be readily available. This maximum rate will only be required for five years, after which six shovel shifts per day will be sufficient.

Total shovel requirements:

> Four (4) P&H 1900 AL, two with 12 yard dippers and two with 11 yard dippers.

II. Drills

Samples of the various rock types in the Vekol pit were sent to both the Hughes and Security Bit manufacturers for microbit testing. They determined the penetration rates that could be expected in the various rock types with the 60-R drill.

The geological group developed the relative percentages of the various rock types encountered by years as outlined by the mining plan. With this information the average penetration rate was calculated. The period requiring the greatest drill capability is during the first three years of operation as shown in the following tabulation:

Year	Pen. Rate Feet/Hour	Alluvial Tons	Rock Tons	Total Tons
Pre-Prod.	40.2	29,172,000	32,828,000	62,000,000
1	45.7	7,220,000	26,280,000	33,500,000
2	36.9	12,029,000	21,471,000	33,500,000
3	38.3	10,232,000	23,268,000	33,500,000
4	35.7	3,739,000	20,261,000	24,000,000
5	34.1	1,027,000	22,973,000	24,000,000

Basic data used in calculation:

1.	Hole size	12¼ inch
2.	Pattern	27' x 30' x 62'
3.	Yards per hole	1,500 yards
4.	Tons per hole	3,240 tons
5.	Tons per foot drilled	52.25 tons

Drill footage and tonnage per shift shown below:

Year	Penetration Rate	Feet Drilled per Shift	Tons per Drill Shift
Pre-Prod.	40.2	261	13,600
1	45.7	297	15,500
2	36.9	240	12,500
3	38.3	249	13,000
4	35.7	232	12,100
5	34.1	221	11,550

Drill Requirements

Year	Annual Tons Rock	Tons per Day	Tons Broken per Drill Shift	Drill Shifts per Day	Drills Require
1	26,280,000	77,800	15,500	6	3
2	21,471,000	63,400	12,500	6	3
3	23,268,000	68,800	13,000	6	3
4	20,261,000	60,000	12,100	5	3
5	22,973,000	68,000	12,100	6	3

It is anticipated that the actual drill performance in the field will exceed the rates indicated by the lab tests. However, additional drilling will probably be required in some of the alluvial areas. This particular situation is difficult to quantify but in any event the three 60-R drills will be required and will be adequate.

III. Trucks

The geographic location of the Vekol operation places it in an area where temperatures will be relatively high. As a result, these calculations have been based on the 120-ton Unit Rig truck which has a more favorable loading ratio with the 30:00 x 51 tire. This truck will handle a 120 ton load up to 550 foot lift without exceeding approved limits. This means that it could operate at rated capacity from the 1250 level with the full rated load of 120 tons. This point will not be exceeded until the 6th year of operation. It also will handle a 100 ton load for a lift of 850 feet which in the case of a waste haul would be down to the 1050 level.

A detailed haulage study was made for the pre-production period, Year 3, Year 7, Year 10, and Year 15. In this study the actual haul profiles were measured from each bench and cycle times were calculated. From this data both the unit cost and the number of operating units were calculated.

The maximum truck requirements are needed through the third year of operation while operating at 9 shovel shifts per day. This came to twenty 120 ton units.

During pre-production this same number of units will be required in order to handle the long haul to the tailing area to construct the initial tailing dams.

Following the third year of operation six shovel shifts per day will be required and the twenty trucks will be adequate since it will only be necessary to cover two operating shovels. Later, as haul distance continues to increase three shovel shifts per day will suffice and truck requirements are still adequate with allowance being made for lower utilization rates.

WABCO has a 150 ton truck available which was put in service in the Tucson area early in 1971. This unit has improved electric capabilities but the tire situation as related to Vekol is still in doubt at this time. The units presently in service have not been loaded to the 150 ton capacity and hence we have no reliable data on tire costs. There are also some reservations with respect to the engine life and cost of maintenance as opposed to the one used in the 120 ton unit.

The WABCO 120 ton unit is too heavy to handle the 120-ton payload in a hot area such as that at Vekol.

The complete list of mine equipment for purposes of this study and the capital cost is summarized in Section 8. Final selection of equipment will be by competitive bid between equipment proven capable of fulfilling operating and maintenance requirements.

SECTION 6

[illegible] — MILL AND OPTIONS

[illegible]

[illegible] several evaluations have been [illegible] since the initiation of this project. [illegible] plan in this [illegible] based on a conventional [illegible] operations, with only one or two changes in plant [illegible] evaluations. [illegible] estimates [illegible] to the [illegible] possible in design [illegible] the estimates, and [illegible] with refinements in the mining plan and better [illegible] data.

A [illegible] exception to this was the estimate of [illegible] which was to evaluate the use of a primary crusher in the [illegible] after the first [illegible] year of production, [illegible] being [illegible] removed from the pit by conveyor from this point on. [illegible] it was found that, considering expected completion of [illegible] of a [illegible] installation, this approach would [illegible] economical. It is expected [illegible] that, as the [illegible] approaches the fourth year of operation, this concept will [illegible].

This evaluation has been based on a [illegible] design. This concept [illegible] maximum advantage [illegible] and heavy [illegible]. All of the engineers who have reviewed this proposed plant in detail are convinced that [illegible] and believe that the [illegible] design concept during detailed engineering will be [illegible].

[illegible] FACILITIES

Facilities for the project include primary crusher, [illegible] secondary and tertiary crushing [illegible] flotation plant, [illegible], warehouse, laboratory, [illegible] disposal system, water supply and [illegible].

[illegible] ARRANGEMENT

The processing facilities, [illegible] approximately [illegible]

SECTION 6

PROJECT DESCRIPTION - MILL AND SERVICES

6.1 General

As mentioned in Section 1, several evaluations have been made since the initiation of this project. Evaluations previous to this one were based on a conventional mill and related facilities, with only slight or no changes in plant concept between evaluations. Hence estimates subsequent to the original were primarily to refine and update the estimates, and coincided with refinements in the mining plan and better metallurgical data.

A noteworthy exception to this was the estimate of November 1970 which was to evaluate the use of a primary crusher in the mine pit after the fourth year of production. Ore would have been removed from the pit by conveyor from this point on. However, it was found that, considering expected escalation of costs of a crusher-conveyor installation, this approach could not be proven economical. It is expected, though, that, as the mine approaches the fourth year of operation, this concept will be re-evaluated.

This evaluation has been based on a somewhat new concept in concentrator design. This concept takes maximum advantage of the climate and physical conditions at Vekol and, as described later, minimizes the use of buildings, overhead cranes and heavy supporting structures. Processing units are large in size, minimum in number. All of the engineers who have studied this proposed plant in detail are convinced that this is a practical realistic approach and believe that any alterations in design concept during detailed engineering will be inconsequential.

6.2 Plant Facilities

Facilities for the project include primary crusher, coarse ore conveying and storage, secondary and tertiary crushing plant, concentration plant, offices, warehouse, laboratory, maintenance shops, tailings disposal system, water supply and electrical supply.

6.3 Plant Arrangement

The ore processing facilities, plus office and general shops, will be located approximately 1500' east of the pit exit.

The main office, warehouse, laboratory, machine shop and pit truck repair facilities are grouped together in a "T" shaped building conveniently located to the open pit and process facilities. See Exhibits, drawing 069-2.

6.4 Yard and Non-Process Facilities

The construction site will be prepared by removing surface gravel, etc., down to solid ground. Rock is outcropping only in a few areas; consequently equipment and building foundations will be on spread footings and pads where necessary. The yard area will be graded, drained and surfaced with selected pit waste gravel. The main plant access road will be an improved dirt road leading from Highway 93. Double lane access roads will lead to all plant facilities and single lane along pipelines for maintenance purposes. In general, the structures will be on soil foundations with concrete floor and steel superstructures, with exterior metal siding. Certain areas will be air-conditioned and some will be provided with sprinkler systems for fire protection. The facilities will be serviced by: a) sewers and a septic system; b) yard water system, including potable water and fire loop; c) fuel oil tanks for process requirements; d) an electrical service substation and power distribution system.

6.5 Primary Crushing, Conveying and Coarse Ore Storage-Dwg 069-4

Ore will be delivered from pit to the primary crushing plant in 120-ton rear dump trucks. Arrangements have been made to dump into the 54" gyratory crusher from two sides. The ore, reduced to minus 9 inches by the 54" gyratory crusher, discharges into a surge pocket under the crusher. An 8-foot wide belt draws the ore from the surge bin and deposits it on a 54" stacker-conveyor, which transports the ore to a 30,000-ton live coarse ore stockpile. The primary crusher structure will be uncovered and crane service will be provided with a mobile unit.

6.6 Secondary and Tertiary Crushing and Fine Ore Storage-Dwg 069-5

The decision to use single stage grinding dictated a closed circuit crushing plant. The plant will consist of two 7-foot standard Symons secondary crushers, and two 7-foot shorthead crushers. Each secondary crusher will be fed by a separate feeder and conveyor system from the coarse ore stockpile. 48" Hydrostroke feeders draw ore from the stockpile and deposit it onto two 42" conveyors, which feed the standard crushers. A 48" conveyor collects the discharge from the two standard crushers and two 7' shorthead tertiary crushers, and delivers it to a screening tower. At the screening tower five 6' x 16' screens remove the minus ½" material, which is conveyed by a 36" conveyor and tripper to the fine ore storage. The plus ½" material from the screens is collected and conveyed by a 36" conveyor to a surge bin over the tertiary crushers. Variable speed belt feeders draw the ore from the surge bin and feed the two tertiary crushers.

The layout of the plant is such that, if necessary, an additional tertiary crusher or screen could be added at minimum cost and with practically no loss of production. The plant is also designed in such a manner that it is possible to operate at a rate in excess of 50 percent of plant capacity when a tertiary or secondary crusher is down for repair or one of the screens is out of service.

As a matter of economy, overhead cranes have been eliminated from the crushing and screening plants and only the necessary siding and roofing required to protect the equipment has been used. Servicing of the equipment will be by a mobile crane which will have ready access to all major equipment items.

The high operating availability of the crushing plant made possible the need for only a minimum of fine ore storage. The enclosed 9,000-ton live fine ore storage will be essentially rectangular in shape with sides made from compacted fill. The area will be covered by roof trusses supported on concrete slabs in the compacted fill. The tripper-conveyor feeding the fine ore storage will be enclosed and syspended from the roof trusses.

6.7 Grinding and Flotation Dwg 069-6 & 069-7

The grinding plant is unique in that all mills will be located in the open. The overhead service crane has been eliminated and repairs, when necessary, will be accomplished by means of a mobile crane. A simple prefabricated structure, equipped with a 5-ton suspended crane, covers the flotation area.

The grinding operation will be single stage and is performed in three parallel 18' diameter x 25'3" long ball mills, each mill driven by a 5,000 h.p. low speed synchronous motor through an air clutch. Each ball mill is in closed circuit, with a next of four 24" diameter cyclones. The cyclone underflow, making up the circulating load, is returned to the feed end of the ball mill. The cyclone overflow, essentially all minus 65 mesh and 65% minus 200 mesh, is the rougher flotation feed.

The overflow from the primary cyclone will be delivered to the rougher flotation section by an inverted syphon to avoid pumping this product. A rougher scavenger flotation section will maintain the 3-line processing concept, allowing a single mill and rougher line to be shut down without affecting the other circuits. Design of the rougher scavenger flotation circuit is based on the use of 365 cu. feet flotation cells. Each rougher scavenger line will consist of eleven 365 cu. feet cells.

A single concentrate regrind line will be provided and concentrate from all three rougher and scavenger lines will be combined and sent to a single sump located at the discharge end of the 10' x 13' regrind ball mill. A pump on this sump will deliver material to a nest of classifying cyclones. The cyclone underflow is the feed to the regrind mill. Cyclone overflow is the feed to cleaner flotation, which consists of three 365-cu. foot flotation machines. The regrind mill has been sized to produce a finished product of 95% minus 325 mesh for cleaning.

The cleaner and recleaner circuits are conventional copper flotation circuits except that at Vekol Hills the cleaner tails will be scavenged in six 365-foot cleaner scavenger flotation cells. The cleaner scavenger concentrate is returned to the head of the cleaner circuit and the cleaner scavenger tails joins the rougher tails and flows by gravity to the tailings thickeners. The cleaner concentrate is recleaned in three 100 cu. foot recleaner cells. The recleaner concentrate is the final copper-moly concentrate and the recleaner tails are recirculated to the head of the cleaner circuit.

Reagents required for the flotation circuits are lime for pH adjustment and Aeropromotors No. 3302 and 3501 for copper collection. The indicated frothers are Dow Froth No. 250 and pine oil. The reagent preparation and storage facilities permit batch preparation daily for 24 hours of plant operation.

6.8 Copper Concentrate Thickening, Filtering and Drying

The recleaner copper-moly concentrate is the feed to the molybdenum plant. The tailing from the molybdenum plant is the final copper concentrate. Dewatering of this material is done in a conventional 70' concentrate thickener, an 8'10" diameter by 8 disc filter, and a 70" diameter 40' long rotary dryer. The dryer discharge will be transported by conveyor to an enclosed concentrate building. At present it is contemplated that the concentrate will leave the property by truck.

The trucks will be loaded by a front-end loader and a truck weigh scale has been provided.

6.9 Tailings Disposal

The total concentrator tailing pulp at 30% solids is sampled ahead of the tailings thickeners. It is then dewatered in two 350' tailings thickeners to 50% solids before it is pumped to the tailings pond. The overflow from the two tailings thickeners is combined and pumped to a two million gallon mill water reservoir.

The underflow from the tailings thickeners, at 50% solids, is pumped through a 20" transite pipeline to the tailings impoundment area. The tailings pond has two compartments, each covering an area of 200 acres. Tailings distribution lines are placed along each of the two dams with outlets, spigots, and portable cyclones provided for sand separation and dike building.

The starter dams will be built essentially from selected overburden from the open pit. The starter dams will be built to an average height of 30' and will provide enough storage capacity for 44 months of operation. Presently, the decant system is provided with two concrete decant towers in each compartment of the dam. Drainage from the towers is gathered into a sump in the southwest corner of the dam. The reclaimed water is pumped to the mill water reservoir. Pumps mounted on barges are being considered as an alternative to the decant towers.

6.10 Instrumentation

Basic process instrumentation has been included to permit substantially automatic process control and to alarm when control cannot be held within preset limits. Additionally, a complement of weigh scales, samplers and meters will be provided to measure production quantities and grades, and power requirements for accounting purposes. In general, instruments will be electronic type.

6.11 Molybdenum Plant

The moly flowsheet consists basically of a rougher flotation step in which the molydenite will be floated and the copper will be depressed. The floated molybdenite rougher concentrate will be cleaned in a series of five flotation cleaning steps. The molybdenum content gradually increases from about 0.6 to 0.7

percent Mo in the moly plant feed to about 54% Mo (90% MoS_2) in the final molybdenum concentrate. The rougher concentrate, after the first cleaning, is reground in a 4' x 6' ball mill in closed circuit with a cyclone. The final moly concentrate is discharged to a 9' x 9' surge tank, designed for 24 hours of concentrate storage. From here, it is pumped to a 4' diameter filter. The filter cake discharges at approximately 18% moisture directly into a hollow-screw type dryer.

The dried moly product will drop directly into a 55-gallon drum holding approximately 800 lbs. of product. When filled to the required weight, the drum is moved to the storage area on a steel roller-type conveyor. It is anticipated that under normal conditions the filter and dryer will be operated only one or two shifts per day.

Automatic sampling has been provided to sample the molybdenum plant feed, concentrates and tails. In addition, each moly concentrate drum may be sampled manually for lot blending processes. A central control room will be provided for the operation of the moly plant in which adequate instrumentation has been installed.

6.12 Water Supply

A Water Lease was granted to Newmont Exploration Limited by the Papago Tribe of Arizona on May 5, 1971, and has subsequently been assigned to Vekol Copper Mining. This gives Vekol the right to explore for, develop, extract, and use water from certain prescribed tribal lands in Santa Rose, Wash., located several miles northeast of the Mining Leases. The Water Lease stipulates that 1) the right to explore for and develop water wells terminates on January 14, 1972, and 2) adequate protection shall be given to the residents of the Indian Village of Kohakt from loss of their water supply resulting from lessees withdrawals. The term of the lease is twenty-five years and is renewable subject to renegotiations. The lease stipulates that lessee shall conserve water used in its construction and operations by all practical means including the installation of thickeners and dams for reclamation of water from mill tailings, and shall keep records of all water consumed and furnish a report of water used, annually, to the tribe.

The lease also provides that the lessee will be granted a right-of-way to construct, use, and maintain a pipeline between the well sites and Lessee's Mining Leases. An annual rental of $100, for each well site included under the lease, is payable to the tribe.

Three wells have been established, subject to the lease, which are described as follows:

1) No. 32-1
Location: S¼ Sec. 32, T 9S R4E
Pumping rate: 3,000 gpm at a depth of 260'
Yield: 30 gpm/ft. of drawdown of water level.

2) No. 5-2
Location: NW¼ Sec. 5, T10S, R4E
Pumping rate: 3,000 gpm at a depth of 320'
Yield: 21+ gpm/ft. of drawdown of water level. Subsequently down rated to 2,000 gpm plus.

3) No. 5-3
Location: ½ mile south of well 5-2
Pumping rate: 3,000 gpm at depth of 291'
Yield: 26 gpm/ft. of drawdown of water level.

On a sustained basis, it is estimated the three wells will provide in excess of 5,000 gallons per minute which is adequate for the estimated demand. The three wells are located approximately 5½ miles from the proposed plant site.

Analysis of the water from each of the three wells have shown that the water is of excellent quality, suitable for both potable and operational usage.

The three wells will each be equipped with a turbine pump which will supply water to a collecting tank. From this tank water will be pumped by horizontal pumps through an 18" pipeline to a fresh water storage tank located on a hill above the plant. The bulk of the fresh water used for process will be added to the 2 million gallon mill water reservoir. Water from this reservoir is pumped to a steady head tank which supplies all the mill process water.

Fresh water will be used for pump seals, cooling water and dust control systems and fire protection. Adequate capacity has been allowed in the fresh water storage tank for these requirements.

6.13 Power Supply

Several sources for the supply of power to the Vekol Copper Mining Company plant complex have been investigated. After a preliminary survey of the situation, it was obvious that both the Lakeshore and Vekol operations would benefit economically if the power for both properties was supplied from the same source. Therefore, Vekol worked with Hecla Mining Company (operator of the Lakeshore property) for the supply of this power.

Trico Electric Company, a cooperative utility, had at that time a franchise to operate on the Papago Reservation and to supply the small amounts of power required by little villages and pumping stations.

Trico was approached initially with regard to supplying the power for Lakeshore and Vekol. It was determined, however, that their's would be a costly source of power as they were unable to generate any part of the power demand necessary for Vekol and Lakeshore and would therefore have to purchase these amounts from another source, probably Arizona Public Service Company. Their sole function would be the wholesale purchase and distribution for resale.

At this stage, a very preliminary check was made on the cost and availability of fuel for a common power generating plant for the two properties. This was dropped because gas was not available and the economics of operating a thermal plant in this location using oil were prohibitive.

In the meantime, the Papago Tribe formed a utility (Papago Tribal Utility Authority) to supply and distribute electrical power to users on the Reservation. PTUA then negotiated a wholesale power contract with Arizona Public Service for the amount of power necessary to supply the two mining properties.

For clarification, the following are the basic and important points with respect to the supply of this power:

1. The PTUA have negotiated a wholesale power contract with APS and will obtain this power from a tap on a 230 KV line running roughly east and west across the Reservation and some twelve miles north of the Lakeshore property.

2. Because of probable jurisdictional problems APS might have if it operated on the Reservation, APS will build a tap off Reservation, on the 230 KV line from which point the PTUA must take the power for delivery to the Reservation's users.

3. The capital cost of the APS facilities which include a tap and short section of line, will be included in their rate structure and charged to PTUA.

4. PTUA will in turn contract separately with Vekol and Lakeshore for the resale of the power that the PTUA has purchased from APS. The power rates in these contracts will be the same as those charged PTUA by APS; however, PTUA will receive directly from both Lakeshore and Vekol an annual charge to cover its administration expenses. This annual charge may be reduced in later years from diversity credits resulting from the difference between the demand charge paid by PTUA to APS and that paid to PTUA separately by Vekol and Lakeshore. PTUA will be metered on a single meter registering the combined demands from Lakeshore and Vekol, whereas the demands of Vekol and Lakeshore will be metered separately.

5. Based on the amount of power PTUA has contracted for with APS, Vekol's power cost is estimated to be between 20 and 27 mills per kilowatt hour.

6. It would not be prudent in the present situation for Vekol to execute this power contract with PTUA. The minimum power charges and cancellation clause would affect the economics drastically. Vekol should wait until such time as a decision is made to proceed with the Vekol sulfide ore operation.

7. The estimate of Vekol's share of the capital cost for the delivery of power to the Vekol site is outlined in the following table. Although the method to determine the prorating of cost to Vekol and Lakeshore for the common line facilities has not been settled yet, it will probably be based on the amount of power used by each.

6.14 Estimated Cost of Power Delivery System to Vekol Plant Site

		Total Cost	Vekol Share	Lakeshore Share
230 KV Line from APS Tap to Lakeshore	11.25 miles	552,000	276,000	276,000
230 KV Line from Lakeshore to Vekol	9.4 miles	455,000	455,000	-
Vekol Substation		707,000	707,000	-
Lakeshore Substation		1,362,000	-	1,362,000
	Total	$3,076,000	$1,438,125	$1,638,000

1) The division of cost Lakeshore/Vekol on common line from APS tap to Lakeshore is based on the same power requirements for both plants.

MANPOWER REQUIREMENTS

7.1 [illegible]

Manpower requirements during construction and pre-production mining are expected to be drawn principally from the Tucson and Phoenix areas. The consensus of opinion from several mining and construction people knowledgeable in the area is that there should be no major difficulties in attracting the required people. It is expected that some training of mining personnel will be required, but that an experienced cadre will be available. Some manpower will be available from the Papago Tribe.

Hourly construction and pre-production mining manpower will average about 140 men on plant and related facilities and 100 on pre-production mining. Construction manpower will be employees of the contractor and various subcontractors while pre-production mining personnel will be [illegible] Copper Mining Company "permanent" employees.

7.2 Manpower for Production

The expected staff and hourly employees during the early production years are tabulated below. Requirements in the mine will reduce during the fourth year of production, again at the seventh through tenth years, where the manpower should stabilize for the remaining five years.

[illegible] Mine - Personnel Schedule

Department	Staff	No.	Annual [illegible]	[illegible]
Management:	General Manager	1	[illegible]	[illegible]
	[illegible] Secretary	1	[illegible]	[illegible]
		3	[illegible]	[illegible]
Safety:	Safety Engineer	1	[illegible]	[illegible]
	First Aid Men	4	[illegible]	[illegible]
	Security Guard	4	[illegible]	[illegible]
		9	[illegible]	[illegible]
Personnel:	Personnel [illegible]	1	[illegible]	[illegible]
	[illegible]	1	[illegible]	[illegible]
		[illegible]	[illegible]	

[illegible]

[illegible]

MANPOWER REQUIREMENTS

7.1 Pre-Production

Manpower requirements during construction and pre-production mining are expected to be drawn principally from the Tucson and Phoenix areas. The concensus of opinion from several mining and construction people knowledgeable in this area is that there should be no major difficulties in attracting adequate people. It is expected that some training of mining personnel will be required, but that an experienced nucleus will be available. Some manpower will be available from the Papago Tribe.

Hourly construction and pre-production mining manpower will average about 160 men on plant and related facilities and 194 on pre-production mining. Construction manpower will be employees of the contractor and various subcontractors while pre-production mining personnel will be Vekol Copper Mining Company "permanent" employees.

7.2 Manpower for Production

The expected staff and hourly employees, during the early production years are tabulated below. Requirements in the mine will reduce during the fourth year of production, again in the seventh through tenth years, where the manpower should stabilize for the remaining five years.

Vekol Hills - Personnel Schedule

			Annual Rates$(000)	
Department	Staff	No.	Preprod(1)	Prod(2)
Management:	General Manager	1	44.9	46.6
	G. M. Secretary	1	10.1	10.5
		2	55.0	57.1
Safety:	Safety Engineer	1	22.5	23.3
	First Aid Man	4	53.9	56.0
	Security Guard	4	40.4	42.0
		9	116.8	121.3
Personnel:	Personnel Supervisor	1	20.2	21.0
	Clerk/Rec.	1	10.1	10.5
		2	30.3	31.5

(1) Escalated approximately 1½ years @ 8% per annum (x 1.123) +- mid point of Pre-production

(2) Escalated 2 years @ 8% per annum (x 1.166)

Department	Staff	No.	Annual Rate$ (0 Preprod(1)	 Prod
Mine:	Mine Superintendent	1	27.5	28.5
	General Pit Foreman	1	21.3	22.1
	Drill & Blast Foreman	1	21.3	22.1
	Asst. D & B Foreman	1	18.3	19.0
	Shift Foreman	4	69.2	72.0
	Asst. Shift Foreman	4	59.6	62.0
	Drill Foreman	4	66.0	68.4
	Road & Dump Foreman	1	17.3	18.0
	Prim. Powder Foreman	1	14.9	15.5
	Gen. Mech. Foreman	1	21.3	22.1
	Field Rep. Foreman	1	19.9	20.6
	Clerks & Timekeeper	4	54.0	56.0
		24	410.6	426.3
Engineering & Geology:	Chief Engineer	1	25.8	26.8
	Planning Engineer	1	22.5	23.3
	Ore Control Engineer	1	20.2	21.0
	Junior Engineer	1	16.8	17.5
	Surveyors	3	42.0	43.8
	Draftsman	1	14.0	14.6
	Geologist	1	20.2	21.0
		9	161.5	168.0
Concentrator:	Conc. Superintendent	1		27.0
	Chief-Metallurgist	1		22.0
	Chief Chemist	1		20.0
	Chemist	1		18.0
	X-ray operator	2		22.0
	General Mill Foreman	1		19.0
	Shift Foreman, Mill	4		68.0
	Shift Foreman, Moly.	4		68.0
	Clerk	1		11.0
		16		275.0
Comptroller:	Chief Accountant	1	22.4	23.3
	Accountant	1	18.0	18.6
	Payroll Clerk	1	13.5	14.0
	Clerk	1	11.2	11.7
	Chief Warehouseman	1	19.1	19.8
	Asst. Warehouseman	3	43.8	45.5
	Purchasing Agent	1	19.1	19.8
		9	147.1	152.7

Department	Staff	No.	Annual Rate$(00 Preprod[1]	Prod[2]
Maintenance:	Maintenance Supt.	1	29.2	30.3
	Repair Shop Foreman	1	20.2	21.0
	Maint. Shift Foreman	4	80.8	84.0
	Mech. Shop Foreman	1	19.1	19.8
	Conc. Repair Foreman	1	19.1	19.8
	Chief Electrician	1	19.1	19.8
	Instrument Engineer	1	20.2	21.0
	Maint. Planning Eng.	1	19.1	19.8
	Service Foreman	1	19.1	19.8
	Clerk	2	22.4	23.3
		14		278.6
Total Staff		85		1510.5

Department		No.		
Mine:	Rotary Drill Operator	8	115,000	120,60
	Rotary Drill Helper	8	93,200	97,70
	Shovel Operator	12	189,000	198,00
	Shovel Oiler	12	149,900	157,10
	F. E. Loader Operator	1	14,400	15,10
	Rubber T. Dozer Oper.	8	110,900	116,20
	Tractor Operator	5	69,300	72,60
	Truck Driver H.D.	60	806,400	844,80
	Grader Operator	4	55,400	58,10
	Water Truck Operator	4	51,700	54,10
	Secondary Drill Oper.	2	25,000	26,20
	Lead Blaster	2	23,800	30,10
	Blaster Helper	4	46,600	48,80
	Laborer	6	64,900	68,00
	Janitor	4	43,300	45,30
		140	1,863,800	1,952,70
Mill:	Primary Crusher Oper.	2		27,45
	Primary Crusher Helper	2		24,96
	Crusher & Mill Control Operator	4		54,91
	Crushing & Grinding Helper	4		49,92
	Flotation Helper	4		49,92
	Conc. & Reagent Oper.	4		54,91
	Relief Operator	2		27,45
	Tails Disposal Oper.	4		54,91
	Laborer	4		48,25
	Moly Plt. Operator	4		54,91
	Moly Plt. Helper	4		49,92
		38		497,53

Department	Staff	No.	Annual Rate$ (00 Preprod[1]	 Prod(
Maintenance:	Mechanic	30	402,600	421,700
	Welder	13	192,500	201,600
	Machinist	6	88,800	93,000
	Electrician	12	166,800	174,800
	Carpenter	1	14,800	15,500
	Painter	2	29,600	31,000
	Tireman	7	103,600	108,600
	Lubeman	7	87,500	91,600
	Oiler	1	11,700	12,200
	Laborer (Fool)	8	86,500	90,600
		87		1,240,600
Engineering:	Surveyor Helper	4	50,000	52,360
	Total Hourly	269		$3,743,19(

Total Payroll = 354 (At start of production) = $5,253,696
Y

Note: Hourly rates are based on 1976 copper industry/steelworker agreements in the Tucson area.

7.3 Housing

It is expected that permanent employees will reside in the Casa Grande area. Casa Grande is a rapidly growing area, with recent impetus given by the opening of the Lakeshore Mine and the Skyline Mobile Home Manufacturing Company. A survey of the housing situation indicates that, at present, there are very few houses available. There are a number of developers active in the area, providing single family units, apartments and mobile home sites. There is ample building space for development.

It is believed that the opening of Vekol would create a shortage of housing. However, there is an active and expanding housing construction industry which should be able to meet demand. Temporary housing and commuting to the Tucson and Phoenix areas will probably be required in the initial phase, but should not be a major problem in operating Vekol.

SECTION 8

COST ESTIMATES

8.1 Capital Cost Estimate

The estimate covers [illegible] construction and pre-production operations. The construction and [illegible] is planned to start in [illegible] 1, 1977 and July 1, 1977 respectively and be completed by July 1, 1978. Engineering and equipment procurement would be initiated in 1976. The estimated initial capital costs are as follows:

	$000
Pre-production Stripping	[illegible]
Mine Equipment (includes mine site)	[illegible]
Mine Shop	[illegible]
Site Drainage	[illegible]
Total Mine	[illegible]
Plant Construction (inc. coal plant)	[illegible]
Owners' Cost	[illegible]
Total	[illegible]
Contingency 10%	[illegible]
Land Rental and Other Payments	[illegible]
Advanced Royalty	[illegible]
[illegible]	[illegible]
Working Capital	[illegible]
Total Initial Capital	[illegible]

Engineering, escalation and contingencies are included.

8.2 Pre-Production Stripping

These costs cover moving [illegible] million tons of waste in a twenty-four month period. Costs include operating and maintenance labor and supplies, mine supervision and an allowance for [illegible]. The average cost per ton of [illegible] amounted to $[illegible]. [illegible] included in [illegible] Capital Cost.

8.3 [illegible]

The mine equipment is required for the pre-production stripping and will [illegible]

SECTION 8

COST ESTIMATES

8.1 Capital Cost Estimate

The estimate covers cost during construction and pre-production stripping. The construction and stripping is assumed to start in January 1, 1977 and July 1, 1976 respectively and be completed by July 1, 1978. Engineering and equipment procurement would be initiated in 1976. The estimated initial capital costs are as follows:

	$000
Pre-production stripping	15,395
Mine Equipment (includes mine elec.)	19,934
Mine Shop	870
Area drainage	30
Total Mine	36,229
Plant Construction (inc. moly plant)	44,393
Owners' Cost	5,921
Total	86,543
Contingency 10%	8,600
Land Rental and other Payments	549
Advanced Royalty	550
Operating Inventory	1,500
Working Capital	8,266
Total Initial Capital	106,008

Engineering, escalation and contingencies are included.

8.2 Pre-Production Stripping

These costs cover moving 62 million tons of waste over a twenty-four month period. Costs include operating and maintenance labor and supplies, with supervision and an allowance for drilling and blasting a portion of the alluvium. The average cost per ton of waste amounts to $0.2483. Owners' administration cost and payroll fringes are included under "Owners' Cost".

8.3 Mine Equipment

The mine equipment is required for the pre-production stripping, and will have to be procurred prior to July 1976. The estimated total cost of the equipment $19,933,610 based on current quotations with allowances for tax and erection, and escalation to July 1, 1976. The summary of these costs follows:

	Total Cost
Shovels	
4 P&H 1900 AL	4,906,000
2 with 11-yard dipper	
2 with 12-yard dipper	
Drills	
3 BE 60-R	1,995,000
1 Secondary Mobil drill	150,000
Trucks	
20 Unit Rig 120-ton	9,243,600
Dozers, Grader, Loader	
3 Cat D9G, two with rippers one with winch	673,750
3 824 Rubber tired dozers	425,700
2 Cat 16E Graders	284,240
1 Loader 15 yd. Dart or LeTourneau 12-yard bucket on Loader	390,300
Cranes	
1 75T	234,700
1 18T Hydraulic	86,700
Pit Electrical System	
4 1500 KVA portable subs complete with all switch gear; 20,000 ft. of cable for shovels and drills including couplers; pit rim circuit with laterals into pit area.	714,000
Auxiliary Equipment	
1 Powder Truck	44,000
2 8000 gal. water trucks	125,500
1 Fork lift for tire handling	77,500
1 Lube service truck	40,500
1 Shovel repair truck	19,500
1 Welding truck	8,900
1 Fork lift (warehouse)	14,850
1 Supply truck	7,900
2 Sedans (manager & pit Superintendent)	10,000
12 One-half ton pick ups	51,800

	Total Cost
Auxiliary Equipment (cont'd)	
4 Three-quarter ton pick ups	18,200
2 Busses (transporting men to mine area)	27,770
1 Tractor Lowboy	50,000
1 Melroe Bobcat	26,350
6 Light Plants	54,000
1 Compressor	13,000
Radio Equipment	105,000
Mine Equipment Requirements which cannot be detailed at this time.	102,000
Total	19,900,760
Sales Tax	32,850
	19,933,610

8.4 Processing Plant and Ancillaries (see Section 6)

The estimate includes cost of engineering, procurement, construction and technical services required to design and construct the plant facilities. It is based on a project schedule of 24 months from start of engineering in July 1976 to on-stream production by July 1978.

The estimate is made on the basis that construction will be by a general contractor. Principal subcontracts, steel erection, prefabricated buildings and electrical work are estimated on the basis of all inclusive quotations, thus have not been included in calculating indirect costs. The contractor's fee is included under "Owners' Cost".

Costs have been estimated at levels believed realistic for the third quarter of 1975. Forward escalation has been built into all costs making them valid for a start of construction date of January 1, 1977 and to the start of production in July 1, 1978. Escalation rates applied for this period are 10 percent per annum for labor and 8 percent per annum for materials and equipment.

See Table 8-1 for a summary of costs and Table 8-2 for the construction schedule.

8.5 Owner's Costs

This item covers administration, personnel and payroll fringe costs during the construction period other than those personnel costs which are included in the pre-production stripping estimate. Cost of owners' representatives at the construction site and the contractor's fee are included. There is an allowance of $1,595,000 for capital spares and basic warehouse inventory.

PROCESSING PLANT AND ANCILLARIES COST SUMMARY

	$000
Yard and Facilities	
Access Road	254
Site Preparation	1,561
Service Buildings	1,463
Water System	2,111
Miscellaneous Piping	348
Incoming Power	1,438
Crushing Facilities	
Coarse Crushing	2,200
Storage	888
Fine Crushing	3,699
Storage	539
Concentrator	14,235
Molybdenum Plant	1,176
Tailings Disposal	1,331
Sub-Total	31,243
Engineering	3,232
Indirect Cost	7,385
Sales Tax and Insurance	235
*Escalation 1/1/77 - 7/1/78	2,298
Total without contingency	44,393

*All other items escalated to start of construction - 1/1/77.

Table 8 - 1

Estimated Owner's Costs

Account	Dollars 1976-78	
1. Site Costs	---	Covered separately in total capital cost statement
(a) Royalty		
2. Temporary Offices	30,000	
3. Construction Supervision		
(a) Owner's representative at design office	45,000	
(b) Owner's representative at site	70,000	
(c) Accountant at design office	45,000	
(d) Travel & Accommodation	85,000	
4. Contractor's fee	1,325,000	Approximately 3% of Plant Cost
5. Capital Spares & Inventory		
(a) Concentrator	710,000	
(b) Mine	885,000	
6. Vehicles		
(a) Station Wagon (1)	5,000	
(b) Pickups (2)	9,000	
7. Overhead Personnel Preproduction	962,000	
8. Payroll fringes	1,750,000	
Total Estimated Owner's Cost	5,921,000	

8.6 Contingency

A general contingency of 10% has been applied to the total cost of the mine and mill. Contingency allowances have not been included in the individual cost items.

8.7 Land Rental and Advanced Royalty

Under terms of the new lease, $485,000 becomes payable to the Papago Indians at the time of governmental approval, and $614,250 during the remaining preproduction period of which $550,000 represents advance royalties which will be deducted from production royalties and the remainder non returnable land rental. An additional $323,280 of advanced royalties paid under terms of the expired leases will also be credited against production royalties.

Preproduction Owners Personnel Costs
(Exclusive of Stripping)

		Annual Cost $(000)	2 Yr. Cost
Overhead Personnel Preproduction			
Management:	1 Gen. Mgr.	44.9	
	1 Secretary	10.1	
		55.0	110.0
Safety:	4 Security	40.4	80.8
Personnel:	1 Supervisor	20.2	
	1 Clerk	10.1	
		30.3	60.6
Concentrator:	Conc. Supt.	27.0	54.0
	Gen. Mill Foreman	19 (1 yr.)	19.0
	Chief Met.	22.0	22.0
Comptroller:	Chief Acct.	22.4	
	Accountant	18.0	
	Payroll Clerk	13.5	
	Clerk	11.2	
	Chief Warehouseman	19.1	
	Asst.	43.8	
	Purchasing Agent	19.1	
		147.1	294.2
Maint.:	Maint. Supt.	29.2	
	Repair Shop Foreman	20.2	
	Maint. Shift Foreman	80.8	
	Maint. Planning Eng.	19.1	
	1 Clerk	11.4	
		160.7	321.4
Total Overhead Personnel-Preproduction 2 yrs.			$ 962.0

#8 Payroll Fringes

	2 Yr. Total Payroll		
Mines Hrly	3,727,600		
Mines Staff	1,144,200		
Overhead	962,000		
Total	5,833,800	x 30% =	$1,750,000

Table 8-2

8.8 Operating Inventory

This item covers operating supplies needed for day to day operations. These are not included in the warehouse inventory of spare parts.

8.9 Working Capital

The working capital required to cover the cash costs from start-up of operations to sale of concentrates has been estimated at five months operating costs.

SECTION 9

OPERATING COSTS, ROYALTY AND TREATMENT CHARGES

9.1 Summary

Average operating, freight, smelting and refining costs per pound of payable copper.

	¢ per pound payable Cu
Concentrate:	
Freight	1.462
Smelting	8.113
Refining	6.000
Total treatment charges	15.575
Royalty (av. value at 80¢ Cu.)	6.4
Mining	9.81
Milling	18.65
Overheads	8.04
Total operating costs	36.50
Total Costs	58.475¢

Based on 9.0 pounds payable copper per ton of ore. (Average grade of ore 0.543%, mill recovery 87.6% and a smelter deduction of 30 pounds copper per ton of concentrate assaying 28% copper.)

9.2 Operating Cost Estimate

The operating cost estimate includes all operating, maintenance and administrative costs for mining, concentrating and preparing concentrates for shipment.

Operating costs listed herein are segregated into three accounts: (1) Mining, (2) Milling and (3) Overhead. A description of the accounts follows:

Mining (See Table 9.1)

The mining account includes direct operating and maintenance labor and supplies, drilling, blasting, loading and hauling of ore to the primary crusher and delivery of waste and overburden to waste or stockpile areas. Supplies and cost of pit sampling and assaying are included.

Summary of Yearly Mining Costs
(cents per ton of material)

Year	Drilling	Blasting	Loading	Haulage	Road and Dump	Pit Dept.	Total
Pre-Prod.	3.21	2.68	4.20	9.81	2.47	2.46	24.83
1	4.15	3.56	4.40	11.01	1.87	2.36	27.35
2	3.93	3.11	4.40	11.75	1.87	2.36	27.42
3	4.08	3.28	4.40	12.50	1.87	2.36	28.49
4	4.99	3.92	4.40	13.34	1.94	3.29	31.88
5	5.62	4.27	4.40	14.23	1.94	3.29	33.75
6	5.91	4.43	4.40	15.12	2.73	4.64	37.23
7	5.87	4.48	4.40	16.06	2.99	4.12	37.92
8	5.99	4.61	4.40	17.57	2.22	4.92	39.71
9	6.05	4.69	4.40	19.04	2.45	5.42	42.05
10	6.13	4.80	4.40	20.54	2.76	5.31	43.94
11	6.17	4.85	4.40	22.01	2.32	5.63	45.38
12	6.20	4.91	4.40	23.46	2.42	5.87	47.26
13	6.23	4.92	4.40	24.92	2.47	6.00	48.94
14	6.24	4.93	4.40	26.33	2.59	6.34	50.83
15	6.40	5.15	4.40	27.92	4.22	10.31	58.40

Table 9 - 1

Milling (See Table 9.2)

The concentrating account includes direct operating and maintenance labor and supplies for ore crushing, concentration, and concentrate dewatering and storage. Tailings dam operating personnel are included, as are chemical and assay laboratory labor and supplies.

Overhead (See Table 9.3)

The account includes all general services, direct supervision and labor costs not specifically chargeable to mining or concentrating and includes direct administration and general office costs, fringe benefits and assessments on the entire payroll, insurance, property taxes and miscellaneous other expenses.

Basis for Operating Costs

Labor rates for hourly paid workers are based on the 1976-77 rates for workers in the Arizona copper industry escalated at 10% per annum to the preproduction mid point and to the start of production. Salaried personnel rates are based on the median figures in a July 1975 survey of 25 mining companies escalated forward at a rate of 8% per annum to the start of production. The overhead account contains allowances for medical and unemployment insurance, vacation, holidays, pensions and workman's compensation based on experience at the Magma Copper Company.

Electrical energy is to be supplied by the Papago Tribal Utility Authority at an estimated cost of 27.0 mills per kwh.

Delivered costs of consumable materials used in the milling cost estimate are as follows:

Steel	$ Cost per pound of material
Crusher liners	0.49
Ball mill liners	0.50
Regrind mill liners	0.50
2" balls	0.20
1" balls	0.25
Reagents	
Aero Promotor 3501	0.50
Dowfroth 250	0.35
Pine Oil	0.26
Line	0.015

MILLING COST ESTIMATE

Based on a 20,000 tpd Mill

	Cost per ton Milled ($)
Operating labor	0.0553
Maintenance labor	0.0387
Supervision	0.0246
Total labor	0.1186
Crusher liners	0.1000
Ball mill liners	0.1250
Grinding balls	0.4250
Flotation reagents	0.1000
Maintenance supplies and spares	0.1808
Fuel oil	0.0150
Electric power	0.5490
	1.6134
Molybdenum plant cost estimate	0.0650
Total mill operating cost	1.6784

Table 9 - 2

VEKOL HILLS - OVERHEAD COSTS

Mining @ 33,500,000 tpy & 20,000 tpd Mill !?
($000)

	Labor	Supplies	Other	Total
Management[1]	57.1	7.5		64.6
Safety[2]	42.0	7.5		49.5
Personnel	31.5	7.5		39.0
Engineering[3]	-			
Comptroller	152.7	30		182.7
Maintenance	278.6	15		293.6
			-In 1975 Dollars-	
Major Medical			148	148
Legal & Professional fees			52	52
Tel. & Tel.			43	43
Administration Building			7	7
Miscellaneous dues & contributions			35	35
New York Office			21	21
Insurance general			205	205
Vacation pay @ 3.5%	<Incl Above in Labor>		-	-
Pensions			525	525
Holiday pay @ 1.5%	<Incl Above in Labor>		-	-
State Unemployment Ins.			63	63
Workmen Compensation Ins.			192	192
Group Life Insurance			198	198
Taxes, F.I.C.A.			283	283
Taxes, Property			1,539	1,539 !?
Taxes, School			310	310
Miscellaneous			400	400
				4,632.4

$\frac{4,632,400}{7,100,000}$ = 65.2 Say 65¢ per ton ore in 1975 Dollars
" 75¢ " " " if escalated to 1978.

1. See Section 7.2 for tabulation of expected payroll.
2. Includes 4 security guards only. Safety Engineer and First Aid men are carried in mining costs in this estimate.
3. Carried in mining costs.

Table 9 - 3

C O P Y

VEKOL ROYALTY SCHEDULE

NET SMELTER RETURN (#/TON OF ORE)	PERCENTAGE ROYALTY OF NET SMELTER RETURNS.
BELOW 4.00	5%
4.01 TO 4.25	6%
4.26 TO 4.50	7%
4.51 TO 4.75	8%
4.76 TO 5.00	9%
5.01 TO 7.00	10% ← AT #.80 COPPER
7.01 TO 11.00	12%

AT CURRENT COPPER PRICES THE ROYALTY RATE IS PROBABLY BETWEEN 7 AND 8%.

Net Smelter Return ($/ton of ore)	Percentage Royalty of Net Smelter Returns
Below 4.00	5%
4.01 to 4.25	6%
4.26 to 4.50	7%
4.51 to 4.75	8%
4.76 to 5.00	9%
5.01 to 7.00	10%
7.01 to 11.00	12%

ESTIMATED OPERATING COST
BY YEAR

Milling at 7,100,000 tons per year rate

	Mining		Milling	Overhead	Contingency	Total
	(¢/ton material)	($/ton ore)	($/ton ore)	($/ton ore)	($/ton ore)	($/ton ore)
Pre-production	24.83	-	-	-	-	-
Mill Year 1	27.35	1.29	1.68	.75	0.20	3.92
2	27.42	1.29	1.68	.75	0.10	3.82
3	28.49	1.34	1.68	.75	-	3.77
4	31.88	1.08	1.68	.75	-	3.51
5	33.75	1.14	1.68	.75	-	3.57
6	37.23	.89	1.68	.75	-	3.32
7	37.92	.83	1.68	.75	-	3.26
8	39.71	.73	1.68	.70	-	3.11
9	42.05	.70	1.68	.70	-	3.08
10	43.94	.65	1.68	.70	-	3.03
11	45.38	.63	1.68	.70	-	3.01
12	47.26	.63	1.68	.70	-	3.01
13	48.94	.64	1.68	.70	-	3.02
14	50.83	.63	1.68	.70	-	3.01
15	58.40	.61	1.68	.70	-	2.99

Table 9 - 4

SECTION 10

ECONOMIC EVALUATION

Tables 10-1 through 10-5 show the Analysis of Project Economic, Summary Operating Statement, Operations Tax Statement and a Cash Flow Summary by Year with a copper price of 80¢ per pound. A sensitivity analysis of operating costs, capital costs and copper price is also provided.

10.1 Exploration and Property Acquisition Costs

Expenditures to date (Dec. 1975) equal $4,274,010. Of this amount, $323,280 is to be credited against production royalties.

10.2 Capital Requirements

The total capital requirement is estimated at $106,008,000, including pre-production stripping, contingencies and working capital (see Section 8). Prior exploration and property acquisition costs totaling $4,274,010 are excluded.

10.3 Financial Outcome

The result was calculated at 80¢ copper price, assuming an all equity investment. Summarized results follow:

	Copper Price	
	80¢/lb.	$1.15/lb.
Capital Requirements (000)	$ 106,008	$ 106,008
Net Cash Flow after Taxes from the 15 years of operations (000)	94,399	263,293
Payback	8.47 years	4.54 years
True (discounted) rate of return	7.01%	16.33%

10.4 Increase of Copper Price above 80¢ Per Pound

A 1¢ increase in copper price over the life of the mine increases the total net cash flow after taxes by approximately $4.8 million.

10.5 Tax Treatment of Pre-Production Development Costs

The evaluations were made as if Vekol Hills were an independent corporation, opting to write-off for tax purposes the pre-production development costs over the life of the orebody.

10.6 Taxes

Local Mine Taxes. Arizona severance tax amounting to 2% of net smelter returns is included. This tax is deductible from taxable income for Federal and State income taxes.

Federal and State Income Taxes. Current tax rates of 48% of taxable income for Federal tax, and 8% of taxable income for Arizona state tax are used.

Federal Preference Tax. Preference tax at 10% of the excess of percentage depletion over cost depletion, less the Federal income tax liability, is included. The cost basis is $4,274,010 consisting of property acquisition and exploration costs.

V E K O L

80 CENT COPPER PRE-MINE 62 MILLION TONS

ECONOMIC EVALUATION DECEMBER 31, 1975

OPEN PIT OPERATION

SUMMARY OPERATING STATEMENT

PLANT CHARACTERISTICS			
CONCENTRATOR SIZE 20,000 TONS PER DAY		104,651,000	
CONCENTRATOR FEED COPPER CONTENT IN PRCT		0.543	
GRADE OF PLANT CONCENTRATE COPPER CONTENT		28.00	
METAL DISTRIBUTION			
STON OF COPPER IN THE CONCENTRATE FEED		567,866	
RECOVERED STON OF COPPER 87.62 RECOVERY		497,566	
STON OF CONCENTRATE SHIPPED 28.00 GRADE		1,777,020	
CONC. RECEIVED AT SMELTER 0.50 FREIGHT LOSS		1,768,136	
SALABLE COPPER NET 30.0 DEDUCTION PER STON		468,560	
GROSS VALUE OF SALABLE PRODUCTS			
GROSS VALUE OF SALABLE COPPER AT $ 0.8000			$749,696,00
VALUE OF MOLY 0.014 GRADE X 0.6000	$ 2.60	$ 45,483,030	
VALUE OF SILVER 2.180 GRADE X 0.9500	$ 4.10	$ 15,013,419	
CHARGES BEYOND CONCENTRATING PLANT			
FREIGHT $ 7.20 PER WET STON 7.0 MOISTURE		$ 13,690,164	
SMELTER CHARGES $ 43.00 PER DRY STON TREATED		$ 76,029,848	
REFINING AND MARKETING PER STON CONC. $ 31.80		$ 56,226,723	
ROYALTY AT 10.00 PERCENT NET SMELTER VALUE		$ 66,424,573	
RECEIPTS AFTER CHARGES $ 5.71 PER TON OF ORE			$597,821,14
OPERATING COSTS			
MINING COSTS $ 0.8766 ORE $ 0.0000 WASTE		$ 91,740,110	
METALLURGICAL COSTS $ 1.68 PER TON TREATED		$175,813,680	
OVERHEADS-INDIRECTS $ 0.72 PER TON OF ORE		$ 74,888,700	
TOTAL MINE, MILL COSTS $ 3.27 PER TON ORE			$342,442,49
OPERATING PROFIT $ 2.44 PER TON OF ORE			$255,378,65
LOCAL ARIZONA MINE TAXES AT 2.00 PERCENT		$ 13,284,915	
NET OPERATING PROFIT AFTER LOCAL MINE TAXES			$242,093,73

V E K O L

80 CENT COPPER [illegible] 62 MIL [illegible] TONS

ECONOMIC EVALUATION DECEMBER 31, 1975

OPEN PIT OPERATION

OPERATIONS TAX STATEMENT 15 OPERATING YEARS

*** ***** *** **** ***** ***** ** *********** ******

NET OPERATING INCOME BEFORE TAXES AND INTEREST	242,093,736
DEPRECIATION	
EQUIPMENT DEPRECIATED OVER 10 YEARS OR MINE LIFE IF LESS	88,497,905
AMORTIZATION OF MINE DEVELOPMENT, BUILDING DEPRECIATION	16,494,000
FEDERAL DEPLETION	
DEPLETION AT 50. PERCENT OF NET AFTER STATE TAX	29,499,318
PRIMARY PRODUCT COPPER AT 15. PERCENT OF NET SMELTER	25,838,542
PRODUCT NO. 2 MOLY AT 15. PERCENT OF NET SMELTER	2,193,791
PRODUCT NO. 3 SILVER AT 15. PERCENT OF NET SMELTER	721,974
FEDERAL INCOME TAXES	
TAXABLE INCOME FOR FEDERAL TAX CALCULATION	73,892,173
FEDERAL LOSS CARRY-FORWARDS USED	
FEDERAL INCOME TAXES AT A 48.00 PERCENT RATE	35,468,242
EXPLORATION, PROPERTY ACQUISITION RECAPTURE	
PREFERENCE TAX AT 10.00 PERCENT OF DEPLETION LESS INCOME TAX	2,278,538
STATE INCOME TAXES IN ARIZONA	
DEPLETABLE INCOME FOR THE STATE OF ARIZONA	94,399,011
DEPLETION ALLOWABLE AT THE RATE OF 50.00 PERCENT	47,199,508
TAXABLE INCOME FOR THE STATE OF ARIZONA	47,199,503
STATE INCOME TAX FOR ARIZONA AT THE RATE OF 10.50 PERCT	4,955,950
STATE LOSS CARRY-FORWARD USED	
NET AFTER DEPRECIATION AND INTEREST BUT BEFORE TAXES	137,101,741
LESS FEDERAL AND STATE INCOME TAXES	42,702,730
NET INCOME AFTER FEDERAL AND STATE TAXES	94,399,011
ADD BACK DEPRECIATION AND BOOK DEVELOPMENT	104,991,905
LESS CAPITAL EXPENDITURES DURING THE PLANT LIFE	8,750,000
ADD BACK WORKING CAPITAL THE LAST YEAR	8,266,000
CASH FLOW AVAILABLE FOR CORPORATE PURPOSES	198,907,006

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80 CENT COPPER   PRE-MINE 62 MILLION TONS
ECONOMIC EVALUATION   DECEMBER 31, 1975
OPEN PIT OPERATION

ANALYSIS OF PROJECT ECONOMICS
******** ** ******* *********

CAPITAL FUNDS REQUIRED DURING PRE-PRODUCTION
EQUITY $106,008,000   LOANS $106,008,000

RATIOS OF OPERATION PROFIT AND AFTER TAX CASH FLOW
****** ** ********* ****** *** ***** **** **** ****

OPERATIONS PROFIT BEFORE TAXES, DEPRECIATION	$242,093,736
RATIO OF BEFORE TAX PROFIT TO TOTAL CAPITAL	2.28
CASH FLOW AFTER TAXES, LOAN REPAYMENTS	$198,907,006
RATIO OF AFTER TAX CASH FLOW TO TOTAL CAPITAL	1.87

PROJECT CASH FLOW ANALYZED AGAINST TOTAL CAPITAL
******* **** ****   ***** *******

CASH AVAILABLE FOR LOANS AND EQUITY	$198,907,006	
RATIO OF TOTAL CASH AVAILABLE TO CAPITAL	1.87	
PAYOUT PERIOD ON TOTAL PROJECT CAPITAL	8.47	YEARS
TRUE RATE OF RETURN ON TOTAL PROJECT CAPITAL	7.01	PERCEN
AVERAGE YEARLY TOTAL CASH AVAILABLE AS A PERCENT OF TOTAL PRE-PRODUCTION CAPITAL	12.50	PERCEN

PROJECT CASH FLOW ANALYZED AGAINST EQUITY CAPITAL
******* **** ****   ****** *******

TOTAL CASH AVAILABLE FOR CORPORATE PURPOSES	$198,907,006	
RATIO OF CASH AVAILABLE TO EQUITY CAPITAL	1.87	
PAYOUT PERIOD ON EQUITY CAPITAL	8.47	YEARS
TRUE RATE OF RETURN OF CASH FLOW ON EQUITY.	7.01	PERCEN
AVERAGE YEARLY CASH AVAILABLE AS A PERCENT OF EQUITY CAPITAL SUPPLIED	12.50	PERCEN

V E K O L

80 CENT COPPER PRE-MINE 62 MILLION TONS

ECONOMIC EVALUATION DECEMBER 31, 1975

OPEN PIT OPERATION

ANALYSIS OF PROJECT ECONOMICS
******** ** ******* **********

CAPITAL FUNDS REQUIRED DURING PRE-PRODUCTION $106,008,000
EQUITY $106,008,000 LOANS

PROJECT CASH FLOW ANALYZED AGAINST TOTAL CAPITAL
******* **** **** ***** *******

CASH AVAILABLE FOR LOANS AND EQUITY	$198,907,006	
PAYOUT PERIOD ON TOTAL PROJECT CAPITAL	8.47	YEARS
TRUE RATE OF RETURN ON TOTAL PROJECT CAPITAL	7.01	PERCEN

PROJECT CASH FLOW ANALYZED AGAINST EQUITY CAPITAL
******* **** **** ****** *******

TOTAL CASH AVAILABLE FOR CORPORATE PURPOSES	$198,907,006	
PAYOUT PERIOD ON EQUITY CAPITAL	8.47	YEARS
TRUE RATE OF RETURN OF CASH FLOW ON EQUITY.	7.01	PERCEN

V E K O L

80 CENT COPPER PRE-MINE 62 MILLION TONS

ECONOMIC EVALUATION DECEMBER 31, 1975

OPEN PIT OPERATION

EVALUATION CRITERIA
*********** ********

ORE RESERVES AND PLANT CHARACTERISTICS
*** ******** *** ***** ***************

TOTAL ORE RESERVES	104,651,000	GRADE OF COPPER	0.543
CONCENTRATOR SIZE	20,000	CONCENTRATOR RECOVERY	87.62
YEARS OF OPERATION	15	PRICE OF COPPER	$ 0.80
CONCENTRATE GRADE	28.00	CONCENTRATE MOISTURE	7.0
BYPRODUCT MOLY		MARKET PRICE	$ 2.60
GRADE MOLY	0.014	PERCENT PAID FOR	60.00
BYPRODUCT SILVER		MARKET PRICE	$ .4.10
GRADE SILVER	2.180	PERCENT PAID FOR	95.00
ORE MINING COSTS	$ 0.8766	WASTE MINING COSTS	$ 0.0000
CONCENTRATOR COSTS	$ 1.6800	OVERHEADS + INDIRECTS	$ 0.7156

CHARGES BEYOND MILLING
******* ****** *******

SMELTER CHARGES	$ 43.00	REFINING + MARKETING	$ 31.80
CONCENTRATE FREIGHT	$ 7.20	SMELTER DEDUCTION UNITS	30.0
CONC. TRANSIT LOSS	0.5	ROYALTY PERCENTAGE	10.00

CAPITAL REQUIREMENTS
******* ************

TOTAL CAPITAL COST	$106,008,000	EQUITY CAPITAL	$106,008,000
WORKING CAPTIAL	$ 8,266,000		

TAX SPECIFICATIONS

FEDERAL TAX RATE	48.00	DEPLETION ON COPPER	15.00
ORIGINAL EQUIPMENT	$ 79,748,000	ORIGINAL BUILDINGS	$ 16,494,000
CAPITAL ADDITIONS	$ 8,750,000	SMELTER OR HOUSING	
STATE TAX RATE	10.50	STATE SEVERANCE TAX	2.00

V E K O L

80 CENT COPPER PRE-MINE 62 MILLION TONS

ECONOMIC EVALUATION DECEMBER 31, 1975

OPEN PIT OPERATION

COSTS PER UNIT OF PRIMARY PRODUCT COPPER
****** **** ***** *** ******** ******* *******

80.000 CENTS COPPER ANALYSIS IN CENTS

OPERATING COSTS
********* *****

MINING	$ 91,740,110	9.790		
METALLURGICAL	$175,813,680	18.761		
OVERHEADS	$ 74,883,700	7.991		
OPERATIONS			36.542	

CHARGES EX PLANT
******* ** *****

SMELTING	$ 76,029,848	8.113		
REFINING	$ 56,226,723	6.000		
FREIGHT	$ 13,690,164	1.461		
ROYALTY	$ 66,424,573	7.088		
EX-PLANT			22.662	

TAXES
*****

DEPRECIATION	$104,991,995	11.204		
FEDERAL TAX	$ 37,746,780	4.028		
STATE TAX	$ 4,955,950	.529		
TAXES			15.761	

TOTAL BEFORE CREDITS 74.965
***** ****** *******

BYPRODUCT CREDITS
********* *******

BYPRODUCT MOLY	$ 45,483,030	4.853		
BYPRODUCT SILVER	$ 15,013,419	1.602		
BYPRODUCTS			6.455	

NET COST OF COPPER 68.510
*** **** ** ******

PROFIT PER UNIT 11.490
****** *** ****

VEKOL

80 CENT COPPER PRE-MINE 62 MILLION TONS

ECONOMIC EVALUATION DECEMBER 31, 1975

OPEN PIT OPERATION

	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	BALANCE	TOTAL
PLANT CHARACTERISTICS												
TONS OF ORE MILLED (000 OMITTED)	7,100	7,100	7,100	7,100	7,100	7,100	7,100	7,100	7,100	7,100	33,651	104,651
CONCENTRATOR FEED COPPER CONTENT IN PRCT	0.529	0.517	0.514	0.551	0.560	0.527	0.547	0.561	0.535	0.593	0.541	0.542
METAL DISTRIBUTION IN STON OF COPPER												
STON OF COPPER IN THE CONCENTRATOR FEED	37,559	36,707	36,494	39,121	39,760	37,417	38,837	39,831	37,985	42,103	182,052	567,866
RECOVERED STON OF COPPER 87.62 RECOVERY	32,909	32,163	31,976	34,278	34,838	32,785	34,029	34,900	33,282	36,891	159,515	497,566
STON CONCENTRATE SHIPPED 28.00 GRADE	117,532	114,868	114,200	122,421	124,421	117,089	121,532	124,643	118,864	131,754	569,696	1,777,020
CONC. RECEIVED AT SMELTER 0.50 FREIGHT LOS	116,944	114,294	113,629	121,809	123,799	116,504	120,924	124,020	118,270	131,095	566,848	1,768,136
SALABLE COPPER 30.0 DEDUCTIONS PER STON	30,990	30,288	30,112	32,280	32,807	30,873	32,045	32,866	31,342	34,741	150,216	468,560
GROSS VALUE OF SALABLE PRODUCTS (000 OMITTED)												
SALABLE COPPER GROSS VALUE AT $ 0.8000	49,584	48,461	48,179	51,648	52,491	49,397	51,272	52,586	50,147	55,586	240,344	749,695
VALUE MOLY 0.014 GRADE X 0.6000 $ 2.60	3,086	3,086	3,086	3,086	3,086	3,086	3,086	3,086	3,086	3,086	14,626	45,486
VALUE SILVER 2.180 GRADE X 0.9500 $ 4.10	993	970	965	1,034	1,051	989	1,027	1,053	1,004	1,113	4,815	15,014
CHARGES BEYOND CONCENTRATOR (000 OMITTED)												
SMELTER CHARGES $ 43.00 PER STON OF CONC.	5,029	4,915	4,886	5,238	5,323	5,010	5,200	5,333	5,086	5,637	24,375	76,032
CONCENTRATE FREIGHT CHARGES $ 7.20 PER STON	905	825	820	943	959	902	936	960	916	1,015	4,389	13,690
REFINING,MARKETING $ 31.80 PER STON CONC.	3,719	3,635	3,613	3,874	3,937	3,705	3,845	3,944	3,761	4,169	18,025	56,227
ROYALTIES 10.00 PERCENT OF NET SMELTER	4,401	4,308	4,285	4,571	4,641	4,386	4,540	4,649	4,447	4,896	21,300	66,424
RECEIPTS AFTER CHARGES $ 5.71 PER TON ORE	39,609	38,774	38,566	41,142	41,768	39,469	40,864	41,839	40,027	44,068	191,696	597,822
OPERATING COSTS												
MINING COSTS AVERAGING $ 0.87 PER TON ORE	9,159	9,159	9,514	7,668	8,094	6,319	5,893	5,183	4,970	4,615	21,166	91,740
METALLURGICAL COSTS $ 1.68 PER TON TREATED	11,928	11,928	11,928	11,928	11,928	11,928	11,928	11,928	11,928	11,928	56,534	175,814
OVERHEADS-INDIRECTS $ 0.72 PER TON OF ORE	4,473	5,325	5,325	5,325	5,325	5,325	5,325	4,970	4,970	4,970	23,556	74,859
TOTAL OPERATING COSTS $ 3.27 PER TON OF ORE	25,560	26,412	26,767	24,921	25,347	23,572	23,146	22,081	21,868	21,513	101,256	342,443
OPERATING PROFITS $ 2.44 PER TON OF ORE	14,049	12,362	11,799	16,221	16,421	15,897	17,718	19,758	18,159	22,555	90,440	255,379
LOCAL ARIZONA MINE TAXES 2.00 PERCENT	880	862	857	914	928	877	908	930	889	979	4,261	13,285
OPERATING PROFITS NET LOCAL MINE TAXES	13,169	11,500	10,942	15,307	15,493	15,020	16,810	18,828	17,270	21,576	86,179	242,094

V E K O L

80 CENT COPPER PRE-MINE 62 MILLION TONS

ECONOMIC EVALUATION DECEMBER 31, 1975

OPEN PIT OPERATION

	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	BALANCE	TOTAL
NET BEFORE TAXES AND INTEREST	13,169	11,500	10,942	15,307	15,493	15,020	16,810	18,828	17,270	21,576	86,179	242,094
DEPRECIATION												
EQUIPMENT 10 YEARS OR MINE LIFE IF LESS	8,010	8,080	8,150	8,220	8,290	8,362	8,440	8,528	8,628	8,745	5,047	88,500
MINE DEVELOPMENT, BUILDING DEPRECIATION	1,119	1,119	1,119	1,119	1,119	1,119	1,119	1,119	1,119	1,119	5,304	16,494
FEDERAL DEPLETION	9,129											
NET AFTER STATE TAX AT 50.00 PERCENT	1,945	1,108	805	2,873	2,929	2,667	3,491	4,420	3,622	5,638		29,498
PRODUCT COPPER AT 15. PERCENT NET SMELTER											25,840	25,840
PRODUCT MOLY AT 15. PERCENT NET SMELTER											2,194	2,194
PRODUCT SILVER AT 15. PERCENT NET SMELTER											721	721
FEDERAL INCOME TAXES												
TAXABLE INCOME FOR FEDERAL TAX CALCULATION	1,945	1,108	805	2,873	2,929	2,667	3,491	4,420	3,622	5,638	44,393	73,891
FEDERAL INCOME TAXES AT 48.00 PERCENT	934	532	387	1,379	1,406	1,280	1,675	2,122	1,738	2,706	21,308	35,467
EXPLORATION, PROPERTY COST RECAPTURE												
PREFERENCE 10.00 DEPLETION LESS INCOME	101	58	42	149	152	139	182	230	188	293	744	2,278
STATE INCOME TAXES IN ARIZONA												
DEPLETABLE INCOME FOR ARIZONA	2,855	1,627	1,182	4,218	4,300	3,915	5,124	6,489	5,317	8,277	51,094	94,398
DEPLETION ALLOWABLE AT 50.00 PERCENT	1,428	814	591	2,109	2,150	1,958	2,562	3,245	2,658	4,138	25,547	47,200
STATE LOSS CARRY FORWARDS USED •												
TAXABLE INCOME FOR ARIZONA	1,428	814	591	2,109	2,150	1,958	2,562	3,245	2,658	4,138	25,547	47,200
TAX FOR ARIZONA 10.50 PERCENT	150	85	62	221	226	206	269	341	279	435	2,682	4,956
NET AFTER DEPRECIATION, INTEREST-BEFORE TAXES	4,040	2,301	1,673	5,968	6,084	5,539	7,251	9,181	7,523	11,712	75,828	137,100
LESS FEDERAL AND STATE INCOME TAXES	1,185	675	491	1,749	1,784	1,625	2,126	2,693	2,205	3,434	24,734	42,701
NET INCOME AFTER FEDERAL AND STATE TAXES	2,855	1,626	1,182	4,219	4,300	3,914	5,125	6,488	5,318	8,278	51,094	94,39[illegible]
ADD BACK DEPRECIATION, BOOK DEVELOPMENT	9,129	9,199	9,269	9,339	9,409	9,481	9,559	9,647	9,747	9,865	10,351	104,9[illegible]5
LESS CAPITAL EXPENDITURES	700	700	700	700	700	700	700	700	700	700	1,750	8,750
ADD BACK WORKING CAPITAL THE LAST YEAR											8,266	8,266
CASH FLOW AVAILABLE FOR CORPORATE PURPOSES	11,284	10,125	9,751	12,858	13,009	12,695	13,984	15,435	14,365	17,443	67,961	198,910
CUMULATIVE CASH FLOW AVAILABLE	11,284	21,409	31,160	44,018	57,027	69,722	83,706	99,141	113,506	130,949	198,910	198,910

VEKOL
ONE DOLLAR COPPER PRE-MINE 62 MILLION TONS
ECONOMIC EVALUATION DECEMBER 31, 1975
OPEN PIT OPERATION

ANALYSIS OF PROJECT ECONOMICS
******** ** ******* *********

CAPITAL FUNDS REQUIRED DURING PRE-PRODUCTION EQUITY $106,008,000 LOANS	$106,008,000

RATIOS OF OPERATION PROFIT AND AFTER TAX CASH FLOW
****** ** ********* ****** *** ***** *** **** ****

OPERATIONS PROFIT BEFORE TAXES, DEPRECIATION	$389,993,464
RATIO OF BEFORE TAX PROFIT TO TOTAL CAPITAL	3.67
CASH FLOW AFTER TAXES, LOAN REPAYMENTS	$296,189,180
RATIO OF AFTER TAX CASH FLOW TO TOTAL CAPITAL	2.79

PROJECT CASH FLOW ANALYZED AGAINST TOTAL CAPITAL
******* **** **** ***** *******

CASH AVAILABLE FOR LOANS AND EQUITY	$296,189,180	
RATIO OF TOTAL CASH AVAILABLE TO CAPITAL	2.79	
PAYOUT PERIOD ON TOTAL PROJECT CAPITAL	5.72	YEARS
TRUE RATE OF RETURN ON TOTAL PROJECT CAPITAL	12.71	PERCEN
AVERAGE YEARLY TOTAL CASH AVAILABLE AS A PERCENT OF TOTAL PRE-PRODUCTION CAPITAL	18.62	PERCEN

PROJECT CASH FLOW ANALYZED AGAINST EQUITY CAPITAL
******* **** **** ****** *******

TOTAL CASH AVAILABLE FOR CORPORATE PURPOSES	$296,189,180	
RATIO OF CASH AVAILABLE TO EQUITY CAPITAL	2.79	
PAYOUT PERIOD ON EQUITY CAPITAL	5.72	YEARS
TRUE RATE OF RETURN OF CASH FLOW ON EQUITY.	12.71	PERCE
AVERAGE YEARLY CASH AVAILABLE AS A PERCENT OF EQUITY CAPITAL SUPPLIED	18.62	PERCE

V E K O L

ONE DOLLAR COPPER PRE-MINE 62 MILLION TONS

ECONOMIC EVALUATION

DECEMBER 31, 1975

OPEN PIT OPERATION

	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	BALANCE	TOTAL
PLANT CHARACTERISTICS												
TONS OF ORE MILLED (000 OMITTED)	7,100	7,100	7,100	7,100	7,100	7,100	7,100	7,100	7,100	7,100	33,651	104,651
CONCENTRATOR FEED COPPER CONTENT IN PRCT	0.529	0.517	0.514	0.551	0.560	0.527	0.547	0.561	0.535	0.593	0.541	0.542
METAL DISTRIBUTION IN STON OF COPPER												
STON OF COPPER IN THE CONCENTRATOR FEED	37,559	36,707	36,494	39,121	39,760	37,417	38,837	39,831	37,985	42,103	182,052	567,866
RECOVERED STON OF COPPER 87.62 RECOVERY	32,909	32,163	31,976	34,278	34,838	32,785	34,029	34,900	33,282	36,891	159,515	497,565
STON CONCENTRATE SHIPPED 28.00 GRADE	117,532	114,868	114,200	122,421	124,421	117,089	121,532	124,643	118,864	131,754	569,696	1,77[illegible],020
CONC. RECEIVED AT SMELTER 0.50 FREIGHT LOSS	116,944	114,294	113,629	121,809	123,799	116,504	120,924	124,020	118,270	131,095	566,848	1,768,136
SALABLE COPPER 30.00 DEDUCTIONS PER STON	30,990	30,288	30,112	32,280	32,807	30,873	32,045	32,866	31,342	34,741	150,216	468,560
GROSS VALUE OF SALABLE PRODUCTS (000 OMITTED)												
SALABLE COPPER GROSS VALUE AT $ 1.0000	61,980	60,576	60,224	64,560	65,614	61,746	64,090	65,732	62,684	69,482	300,432	937,120
VALUE MOLY 0.014 GRADE X 0.6000 $ 2.60	3,086	3,086	3,086	3,086	3,086	3,086	3,086	3,086	3,086	3,086	14,626	45,486
VALUE SILVER 2.180 GRADE X 0.9500 $ 4.10	993	970	965	1,034	1,051	989	1,027	1,053	1,004	1,113	4,815	15,014
CHARGES BEYOND CONCENTRATOR (000 OMITTED)												
SMELTER CHARGES $ 43.00 PER STON OF CONC.	5,029	4,915	4,886	5,238	5,323	5,010	5,200	5,333	5,086	5,637	24,375	76,032
CONCENTRATE FREIGHT CHARGES $ 7.20 PER STON	905	8[illegible]5	8[illegible]0	943	959	902	936	960	916	1,015	4,389	13,690
REFINING,MARKETING $ 31.80 PER STON CONC.	3,719	3,635	3,613	3,874	3,937	3,705	3,845	3,944	3,761	4,169	18,025	56,2[illegible]7
ROYALTIES 12.00 PERCENT OF NET SMELTER	6,769	6,624	6,588	7,035	7,144	6,744	6,987	7,156	6,841	7,543	32,769	102,200
RECEIPTS AFTER CHARGES $ 7.16 PER TON ORE	49,637	48,573	48,308	51,590	52,388	49,460	51,235	52,478	50,170	55,317	240,315	749,471
OPERATING COSTS												
MINING COSTS AVERAGING $ 0.87 PER TON ORE	9,159	9,159	9,514	7,668	8,094	6,319	5,893	5,183	4,970	4,615	21,166	91,74[illegible]
METALLURGICAL COSTS $ 1.68 PER TON TREATED	11,928	11,928	11,928	11,928	11,928	11,928	11,928	11,928	11,928	11,928	56,534	175,81[illegible]
OVERHEADS-INDIRECTS $ 0.72 PER TON OF ORE	4,473	5,325	5,325	5,325	5,325	5,325	5,325	4,970	4,970	4,970	23,5[illegible]6	74,8[illegible]
TOTAL OPERATING COSTS $ 3.27 PER TON OF ORE	25,560	26,412	26,767	24,921	25,347	23,572	23,146	22,081	21,868	21,513	101,256	342,4[illegible]
OPERATING PROFITS $ 3.89 PER TON OF ORE	24,077	22,161	21,541	26,669	27,041	25,888	28,089	30,397	28,302	33,804	139,059	407,02[illegible]
LOCAL ARIZONA MINE TAXES 2.00 PERCENT	1,128	1,104	1,098	1,173	1,191	1,124	1,164	1,193	1,140	1,257	5,460	17,03[illegible]
OPERATING PROFITS NET LOCAL MINE TAXES	22,949	21,057	20,443	25,496	25,850	24,764	26,925	29,204	27,162	32,547	133,599	389,9[illegible]

V E K O L

ONE DOLLAR COPPER PRE-MINE 62 MILLION TONS

ECONOMIC EVALUATION DECEMBER 31, 1975

OPEN PIT OPERATION

	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	BALANCE	TOTAL
NET BEFORE TAXES AND INTEREST	22,949	21,057	20,443	25,496	25,850	24,764	26,925	29,204	27,162	32,547	133,599	389,
DEPRECIATION												
EQUIPMENT 10 YEARS OR MINE LIFE IF LESS	8,010	8,080	8,150	8,220	8,290	8,362	8,440	8,528	8,628	8,745	5,047	83,
MINE DEVELOPMENT, BUILDING DEPRECIATION	1,119	1,119	1,119	1,119	1,119	1,119	1,119	1,119	1,119	1,119	5,304	16,
FEDERAL DEPLETION												25,
NET AFTER STATE TAX AT 50.00 PERCENT	6,654	5,710	5,380	7,121	7,238	7,358	7,068	7,251	6,912	7,668	33,130	76,
PRODUCT COPPER AT 15. PERCENT NET SMELTER				463	463		463	463	463	463	2,194	4,
PRODUCT MOLY AT 15. PERCENT NET SMELTER				155	158		154	158	151	167	721	1,
PRODUCT SILVER AT 15. PERCENT NET SMELTER												
FEDERAL INCOME TAXES												
TAXABLE INCOME FOR FEDERAL TAX CALCULATION	6,654	5,710	5,380	7,821	7,975	7,358	9,047	10,986	9,258	13,587	83,037	16
FEDERAL INCOME TAXES AT 48.00 PERCENT	3,194	2,741	2,582	3,754	3,828	3,532	4,343	5,273	4,444	6,522	39,857	80
EXPLORATION, PROPERTY COST RECAPTURE	346	297	280	398	403	383	334	260	308	178		3
PREFERENCE 10.00 DEPLETION LESS INCOME												
STATE INCOME TAXES IN ARIZONA												
DEPLETABLE INCOME FOR ARIZONA	9,768	8,382	7,898	11,407	11,602	10,802	12,056	13,325	12,031	15,185	79,227	191
DEPLETION ALLOWABLE AT 50.00 PERCENT	4,884	4,191	3,949	5,703	5,801	5,401	6,028	6,662	6,016	7,593	39,613	95
STATE LOSS CARRY FORWARDS USED												
TAXABLE INCOME FOR ARIZONA	4,884	4,191	3,949	5,703	5,801	5,401	6,028	6,662	6,016	7,593	39,613	9
TAX FOR ARIZONA 10.50 PERCENT	513	440	415	599	609	567	633	700	632	797	4,159	1
NET AFTER DEPRECIATION, INTEREST-BEFORE TAXES	13,820	11,858	11,174	16,157	16,441	15,283	17,366	19,557	17,415	22,683	123,248	28
LESS FEDERAL AND STATE INCOME TAXES	4,053	3,478	3,277	4,751	4,840	4,482	5,310	6,233	5,384	7,497	44,016	9
NET INCOME AFTER FEDERAL AND STATE TAXES	9,767	8,380	7,897	11,406	11,601	10,801	12,056	13,324	12,031	15,186	79,232	19
ADD BACK DEPRECIATION, BOOK DEVELOPMENT	9,129	9,199	9,269	9,339	9,409	9,481	9,559	9,647	9,747	9,865	10,351	10
LESS: CAPITAL EXPENDITURES	700	700	700	700	700	700	700	700	700	700	1,750	
ADD BACK WORKING CAPITAL THE LAST YEAR											8,266	
CASH FLOW AVAILABLE FOR CORPORATE PURPOSES	18,196	16,879	16,466	20,045	20,310	19,582	20,915	22,271	21,078	24,351	96,099	29
CUMULATIVE CASH FLOW AVAILABLE	18,196	35,075	51,541	71,586	91,896	111,478	132,393	154,664	175,742	200,093	296,192	29

V E K O L

1.15 COPPER [illegible] MILLION TONS

ECONOMIC EVALUATION DECEMBER 31, 1975

OPEN PIT OPERATION

ANALYSIS OF PROJECT ECONOMICS
******** ** ******* *********

CAPITAL FUNDS REQUIRED DURING PRE-PRODUCTION $106,008,000
EQUITY $106,008,000 LOANS

RATIOS OF OPERATION PROFIT AND AFTER TAX CASH FLOW
****** ** ********* ****** *** ***** *** **** ****

OPERATIONS PROFIT BEFORE TAXES, DEPRECIATION	$510,881,944
RATIO OF BEFORE TAX PROFIT TO TOTAL CAPITAL	4.81
CASH FLOW AFTER TAXES, LOAN REPAYMENTS	$369,301,014
RATIO OF AFTER TAX CASH FLOW TO TOTAL CAPITAL	3.48

PROJECT CASH FLOW ANALYZED AGAINST TOTAL CAPITAL
******* **** **** ***** *******

CASH AVAILABLE FOR LOANS AND EQUITY	$369,301,014	
RATIO OF TOTAL CASH AVAILABLE TO CAPITAL	3.48	
PAYOUT PERIOD ON TOTAL PROJECT CAPITAL	4.54	YEARS
TRUE RATE OF RETURN ON TOTAL PROJECT CAPITAL	16.33	PERCENT
AVERAGE YEARLY TOTAL CASH AVAILABLE AS A PERCENT OF TOTAL PRE-PRODUCTION CAPITAL	23.22	PERCENT

PROJECT CASH FLOW ANALYZED AGAINST EQUITY CAPITAL
******* **** **** ****** *******

TOTAL CASH AVAILABLE FOR CORPORATE PURPOSES	$369,301,014	
RATIO OF CASH AVAILABLE TO EQUITY CAPITAL	3.48	
PAYOUT PERIOD ON EQUITY CAPITAL	4.54	YEARS
TRUE RATE OF RETURN OF CASH FLOW ON EQUITY.	16.33	PERCENT
AVERAGE YEARLY CASH AVAILABLE AS A PERCENT OF EQUITY CAPITAL SUPPLIED	23.22	PERCENT

V E K O L

$1.15 COPPER PRE-MINE 62 MILLION TONS DECEMBER 31, 1975

ECONOMIC EVALUATION

OPEN PIT OPERATION

	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	BALANCE	TOTAL
PLANT CHARACTERISTICS												
TONS OF ORE MILLED (000 OMITTED)	7,100	7,100	7,100	7,100	7,100	7,100	7,100	7,100	7,100	7,100	33,651	104,6
CONCENTRATOR FEED COPPER CONTENT IN PRCT	0.529	0.517	0.514	0.551	0.560	0.527	0.547	0.561	0.535	0.593	0.541	0.5
METAL DISTRIBUTION IN STON OF COPPER												
STON OF COPPER IN THE CONCENTRATOR FEED	37,559	36,707	36,494	39,121	39,760	37,417	38,837	39,831	37,985	42,103	182,052	567,
RECOVERED STON OF COPPER 87.62 RECOVERY	32,909	32,163	31,976	34,278	34,838	32,785	34,029	34,900	33,282	36,891	159,515	497,
STON CONCENTRATE SHIPPED 28.00 GRADE	117,532	114,868	114,200	122,421	124,421	117,089	121,532	124,643	118,864	131,754	569,696	1,77[illegible]
CONC. RECEIVED AT SMELTER 0.50 FREIGHT LOSS	116,944	114,294	113,629	121,809	123,799	116,504	120,924	124,020	118,270	131,095	566,848	1,768,
SALABLE COPPER 30.000 DEDUCTIONS PER STON	30,990	30,288	30,112	32,280	32,807	30,873	32,045	32,866	31,342	34,741	150,216	468,
GROSS VALUE OF SALABLE PRODUCTS (000 OMITTED)												
SALABLE COPPER GROSS VALUE AT $ 1.1500	71,277	69,642	69,258	74,244	75,456	71,008	73,704	75,592	72,087	79,904	345,496	1,077,
VALUE MOLY 0.014 GRADE X 0.6000 $ 2.60	3,086	3,086	3,086	3,086	3,086	3,086	3,086	3,086	3,086	3,086	14,626	45,
VALUE SILVER 2.180 GRADE X 0.9500 $ 4.10	993	970	965	1,034	1,051	989	1,027	1,053	1,004	1,113	4,815	15,
CHARGES BEYOND CONCENTRATOR (000 OMITTED)												
SMELTER CHARGES $ 43.00 PER STON OF CONC.	5,029	4,915	4,886	5,238	5,323	5,010	5,200	5,333	5,086	5,637	24,375	76,
CONCENTRATE FREIGHT CHARGES $ 7.20 PER STON	905	885	880	943	959	902	936	960	916	1,015	4,389	13,
REFINING,MARKETING $ 31.80 PER STON CONC.	3,719	3,635	3,613	3,874	3,937	3,705	3,845	3,944	3,761	4,169	13,025	56,
ROYALTIES 12.00 PERCENT OF NET SMELTER	7,804	7,714	7,672	8,197	8,325	7,856	8,140	8,339	7,970	8,794	38,177	119,
RECEIPTS AFTER CHARGES $ 8.34 PER TON ORE	57,819	56,569	56,258	60,112	61,049	57,610	59,696	61,155	58,444	64,488	279,971	873
OPERATING COSTS												
MINING COSTS AVERAGING $ 0.87 PER TON ORE	9,159	9,159	9,514	7,648	8,094	6,319	5,893	5,183	4,970	4,615	21,166	91
METALLURGICAL COSTS $ 1.68 PER TON TREATED	11,928	11,928	11,928	11,928	11,928	11,928	11,928	11,928	11,928	11,928	56,534	175
OVERHEADS-INDIRECTS $ 0.72 PER TON OF ORE	4,473	5,325	5,325	5,325	5,325	5,325	5,325	4,970	4,970	4,970	23,556	74
TOTAL OPERATING COSTS $ 3.27 PER TON OF ORE	25,560	26,412	26,767	24,921	25,347	23,572	23,146	22,081	21,868	21,513	101,256	342
OPERATING PROFITS $ 5.07 PER TON OF ORE	32,259	30,157	29,491	35,191	35,702	34,038	36,550	39,074	36,576	42,975	178,715	530
LOCAL ARIZONA MINE TAXES 2.00 PERCENT	1,314	1,286	1,279	1,366	1,387	1,309	1,357	1,390	1,328	1,466	6,365	19
OPERATING PROFITS NET LOCAL MINE TAXES	30,945	28,871	28,212	33,825	34,315	32,729	35,193	37,684	35,248	41,509	172,350	51

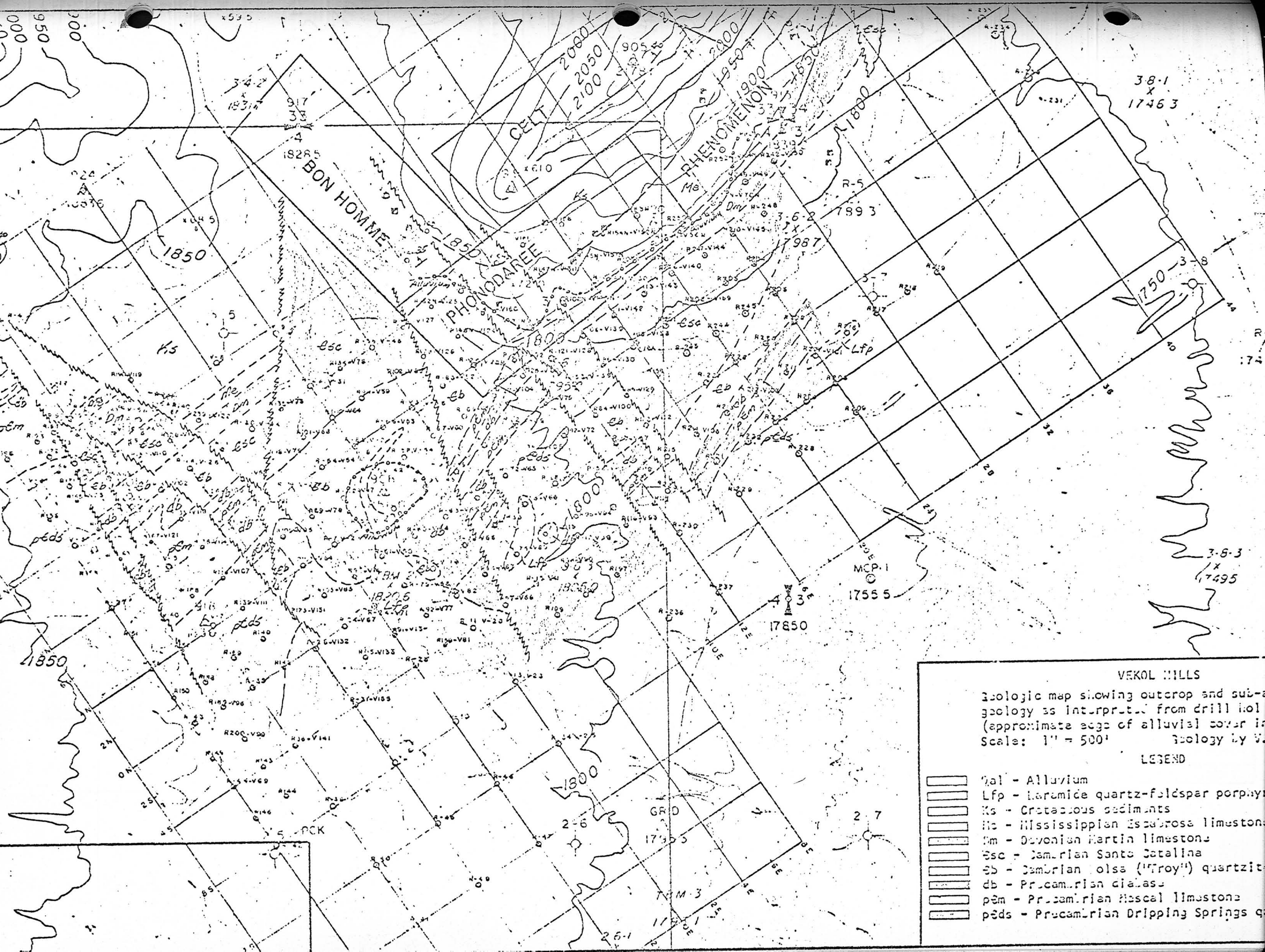
$1.15 COPPER PRE-MINE 62 MILLION TONS

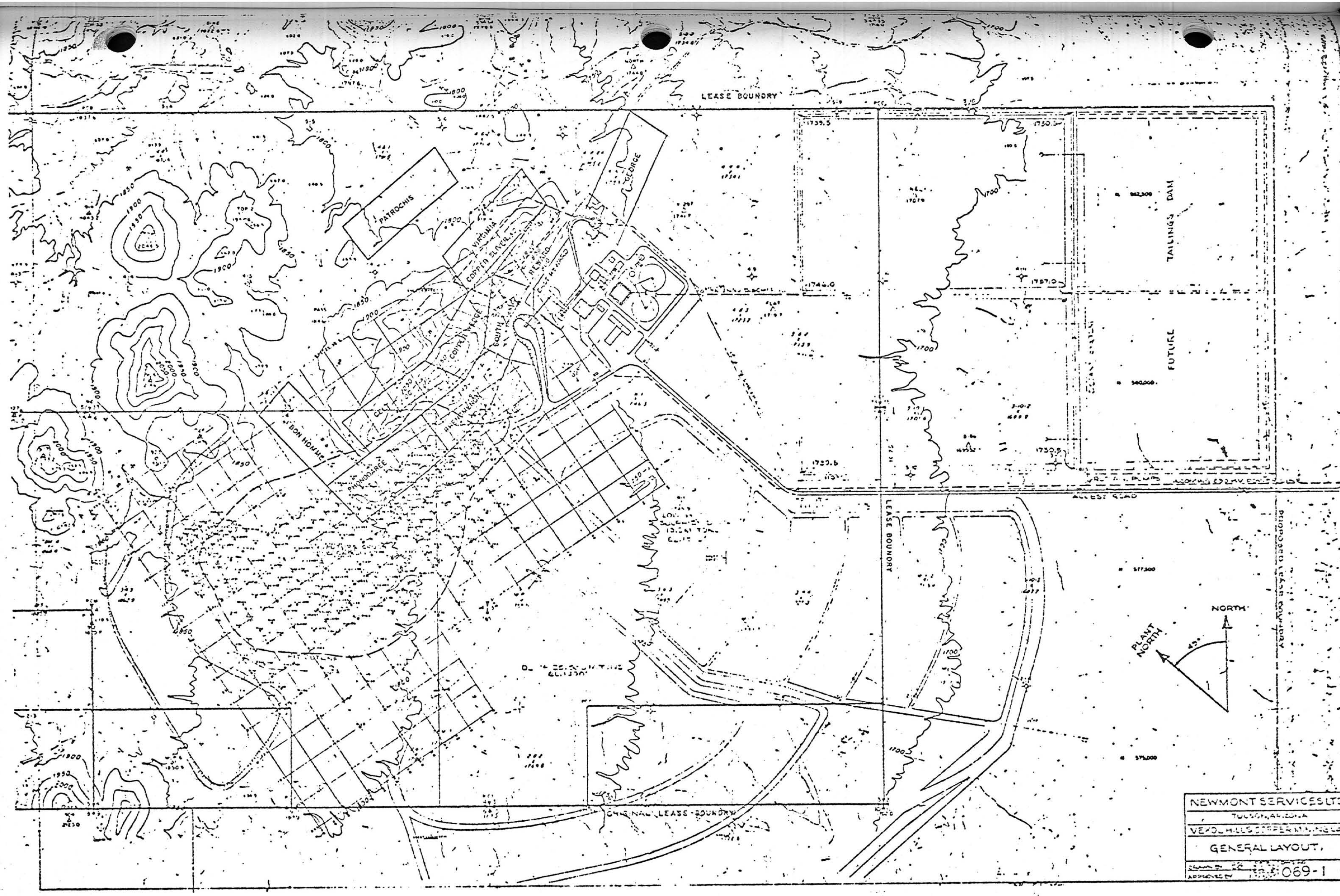
ECONOMIC EVALUATION DECEMBER 31, 1975

OPEN PIT OPERATION

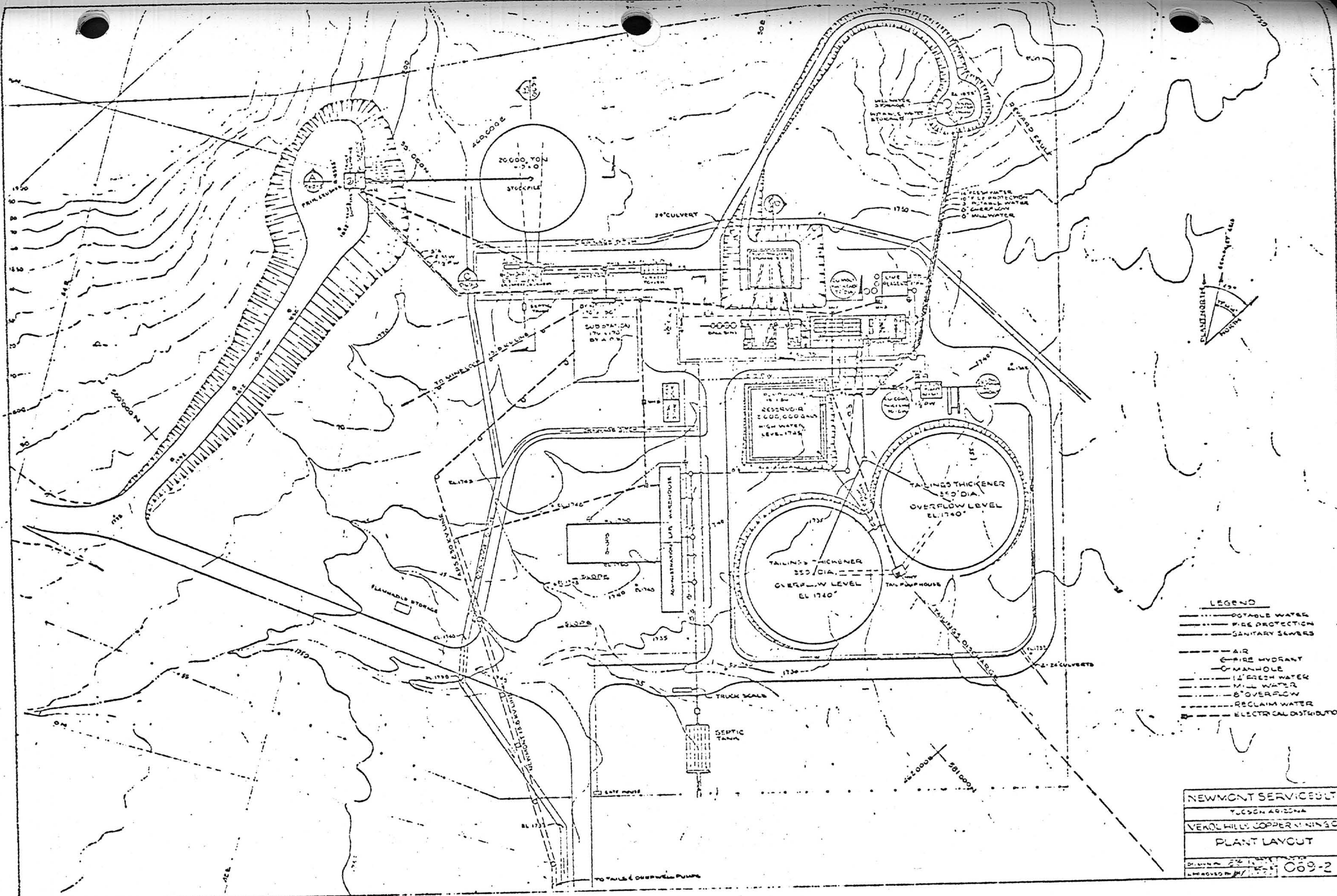
	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	BALANCE	TOTAL
NET BEFORE TAXES AND INTEREST	30,945	28,871	28,212	33,825	34,315	32,729	35,193	37,684	35,248	41,509	172,350	510,8[illegible]1
DEPRECIATION												
EQUIPMENT 10 YEARS OR MINE LIFE IF LESS	8,010	8,080	8,150	8,220	8,290	8,362	8,440	8,528	8,628	8,745	5,047	8[illegible],500
MINE DEVELOPMENT, BUILDING DEPRECIATION	1,119	1,119	1,119	1,119	1,119	1,119	1,119	1,119	1,119	1,119	5,304	16,494
FEDERAL DEPLETION												
PRODUCT COPPER AT 15. PERCENT NET SMELTER	8,061	7,877	7,831	8,399	8,537	8,030	8,337	8,552	8,153	9,043	39,078	121,898
PRODUCT MOLY. AT 15. PERCENT NET SMELTER	463	463	463	463	463	463	463	463	463	463	2,194	6,824
PRODUCT SILVER AT 15. PERCENT NET SMELTER	149	146	145	155	158	148	154	158	151	167	721	2,252
FEDERAL INCOME TAXES												
TAXABLE INCOME FOR FEDERAL TAX CALCULATION	12,364	10,475	9,813	14,608	14,873	13,787	15,785	17,897	15,848	20,894	114,669	261,013
FEDERAL INCOME TAXES AT 48.00 PERCENT	5,935	5,028	4,710	7,012	7,139	6,618	7,577	8,590	7,607	10,029	55,042	125,287
EXPLORATION, PROPERTY COST RECAPTURE												
PREFERENCE 10.00 DEPLETION LESS INCOME	274	346	373	201	202	202	138	58	116			1,910
STATE INCOME TAXES IN ARIZONA												
DEPLETABLE INCOME FOR ARIZONA	14,828	13,587	13,169	16,412	16,689	15,609	17,025	18,421	16,892	20,538	101,624	264,79[illegible]
DEPLETION ALLOWABLE AT 50.00 PERCENT	7,414	6,793	6,584	8,206	8,345	7,804	8,512	9,211	8,446	10,269	50,812	132,396
STATE LOSS CARRY FORWARDS USED												
TAXABLE INCOME FOR ARIZONA	7,414	6,793	6,584	8,206	8,345	7,804	8,512	9,211	8,446	10,269	50,812	132,396
TAX FOR ARIZONA 10.50 PERCENT	778	713	691	862	876	819	894	967	887	1,078	5,376	13,90[illegible]
NET AFTER DEPRECIATION, INTEREST-BEFORE TAXES	21,816	19,672	18,943	24,486	24,906	23,248	25,634	28,037	25,501	31,645	161,999	405,8[illegible]
LESS FEDERAL AND STATE INCOME TAXES	6,987	6,087	5,774	8,075	8,217	7,639	8,609	9,615	8,610	11,107	60,378	141,09[illegible]
NET INCOME AFTER FEDERAL AND STATE TAXES	14,829	13,585	13,169	16,411	16,689	15,609	17,025	18,422	16,891	20,538	101,621	264,78[illegible]
ADD BACK DEPRECIATION, BOOK DEVELOPMENT	9,129	9,199	9,269	9,339	9,409	9,481	9,559	9,647	9,747	9,865	10,351	104,9[illegible]
LESS CAPITAL EXPENDITURES	700	700	700	700	700	700	700	700	700	700	1,750	8,75[illegible]
ADD BACK WORKING CAPITAL THE LAST YEAR											8,26[illegible]	8,26[illegible]
CASH FLOW AVAILABLE FOR CORPORATE PURPOSES	23,258	22,084	21,738	25,050	25,398	24,390	25,884	27,369	25,938	29,703	118,488	369,30[illegible]
CUMULATIVE CASH FLOW AVAILABLE	23,258	45,342	67,080	92,130	117,528	141,918	167,802	195,171	221,109	250,812	369,300	369,30[illegible]

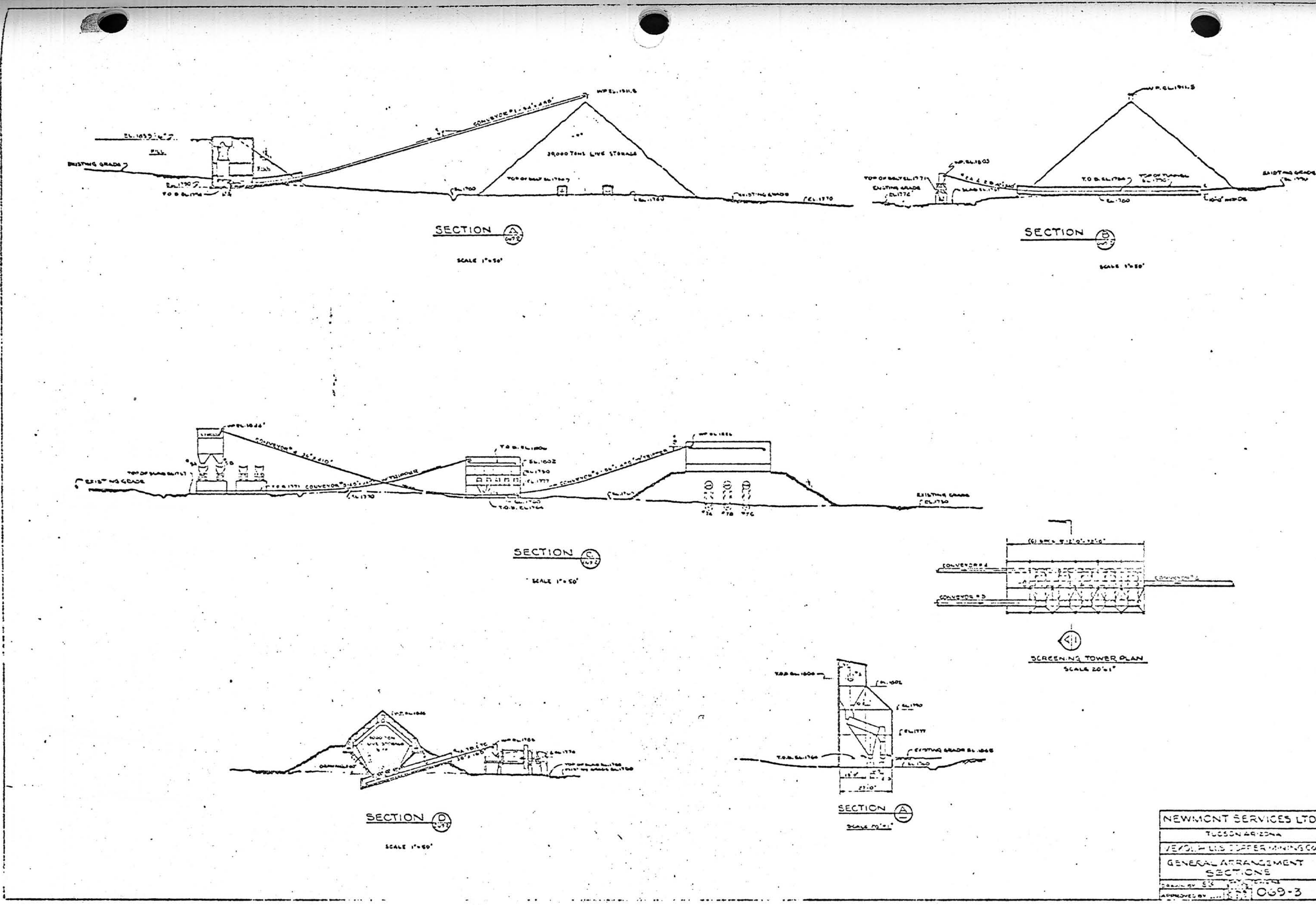
APPENDIX


VEKOL HILLS
Geologic map showing outcrop and sub-a
geology as interpreted from drill hol
(approximate edge of alluvial cover i
Scale: 1" = 500'
Geology by V.
LEGEND
Qal - Alluvium
Lfp - Laramide quartz-feldspar porphy
Ks - Cretaceous sediments
Me - Mississippian Escabrosa limestone
Dm - Devonian Martin limestone
Esc - Cambrian Santa Catalina
Eb - Cambrian Bolsa ("Troy") quartzit
db - Precambrian diabase
pEm - Precambrian Mescal limestone
pEds - Precambrian Dripping Springs q
BON HOMME
PHONODAREE
PHENOMENON
CELT
MCP-1
17555
17850
GRID
1850
1800
1750
2000
2050
2100
1950
1900

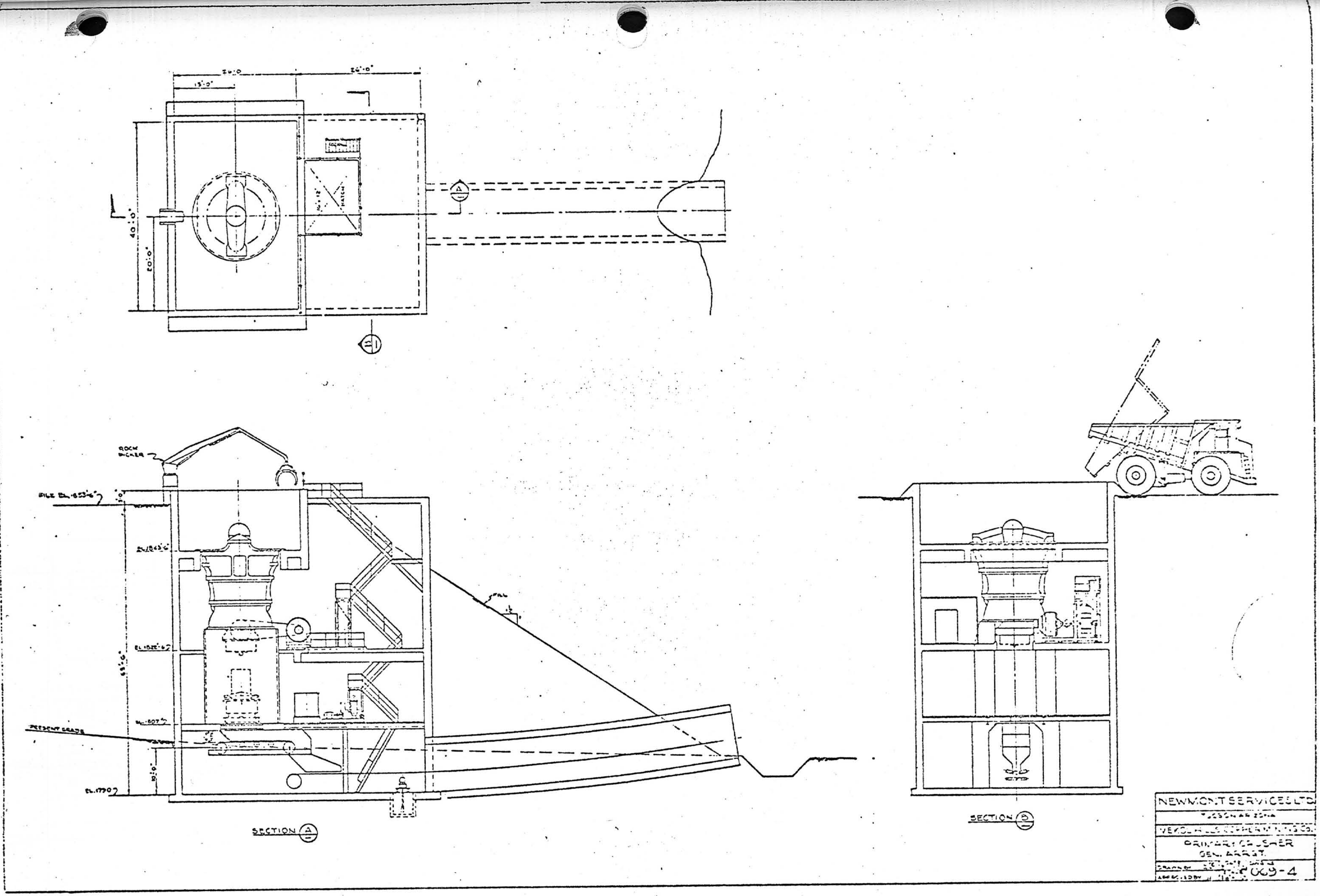


LEASE BOUNDRY
PATROCHIS
GEORGE
VIRGINIA
COPPER SILVER
COPPER VERGE
SOUTH
BON HOMME
PHONODAREE
TAILINGS DAM
FUTURE
ACCESS ROAD
LEASE BOUNDRY
PROPOSED LEASE BOUNDRY
NORTH
PLANT NORTH
45°
ORIGINAL LEASE BOUNDRY
NEWMONT SERVICES LTD
TUCSON, ARIZONA
GENERAL LAYOUT,
069-1

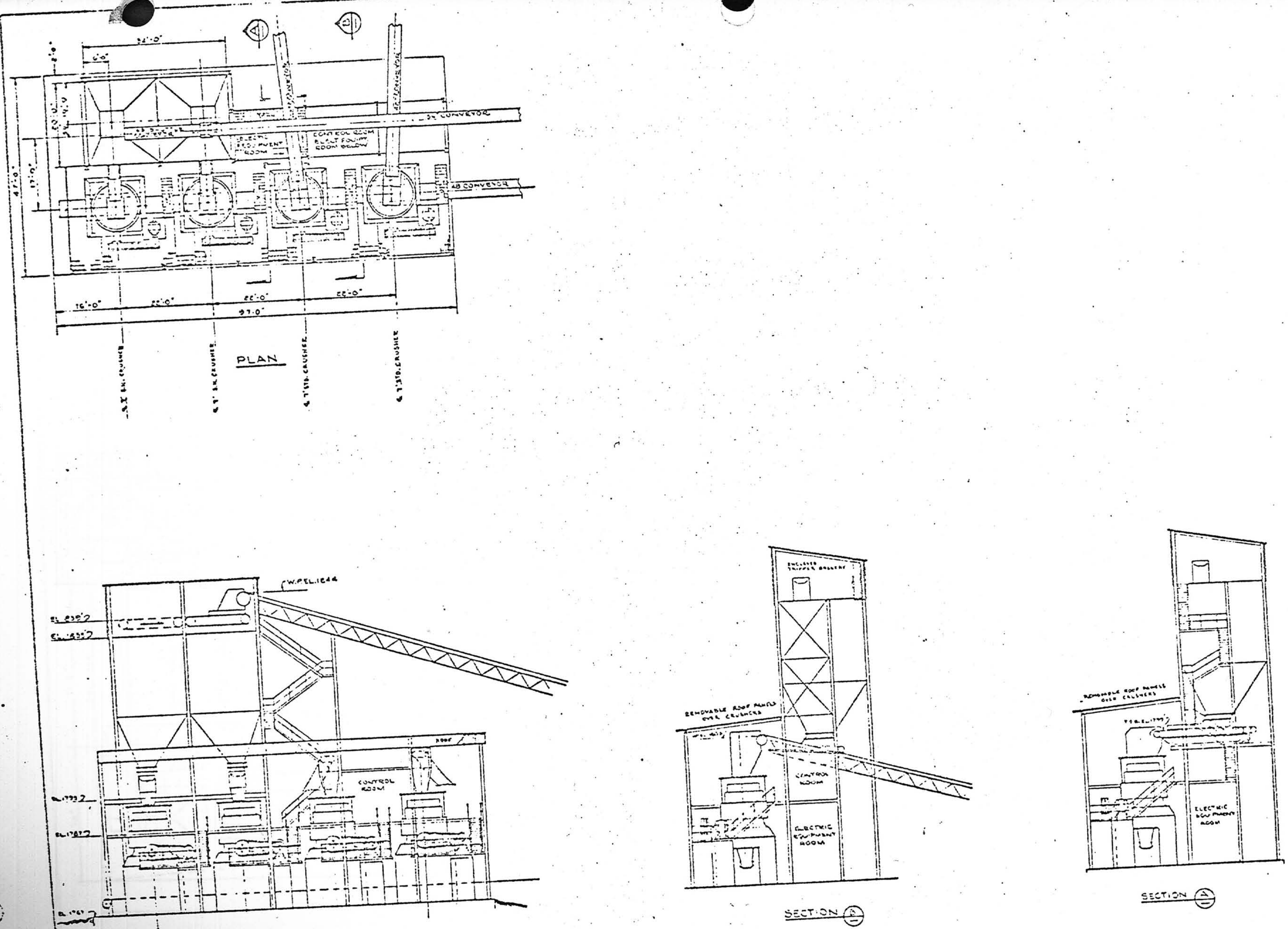

NEWMONT SERVICES LTD
TUCSON ARIZONA
VEKOL HILLS COPPER MINING CO.
PLANT LAYOUT
C09-2
LEGEND
POTABLE WATER
FIRE PROTECTION
SANITARY SEWERS
AIR
FIRE HYDRANT
MANHOLE
12" FRESH WATER
MILL WATER
8" OVERFLOW
RECLAIM WATER
ELECTRICAL DISTRIBUTION
PLANT NORTH
NORTH
TAILINGS THICKENER
350' DIA.
OVERFLOW LEVEL
EL.1740'
TAILINGS THICKENER
350' DIA.
OVERFLOW LEVEL
EL 1740'
TAIL PUMP HOUSE
TAILINGS DISCHARGE
RESERVOIR
2,000,000 GALS
HIGH WATER
LEVEL 1745'
ADMINISTRATION LAB. WAREHOUSE
SUBSTATION
20,000 TON
STOCKPILE
24" CULVERT
2-24" CULVERTS
FLAMMABLE STORAGE
TRUCK SCALE
SEPTIC TANK
GATE HOUSE
TO TAILS & OVERFLOW PUMPS
MILLWATER STORAGE
POTABLE WATER STORAGE
REWARD FAULT
12" FRESH WATER
10" F.P. PROTECTION
3" POTABLE WATER
8" OVERFLOW
8" MILLWATER
PRIMARY CRUSHER
LIME PLANT
SLOPE
581,000 N
580,000 N



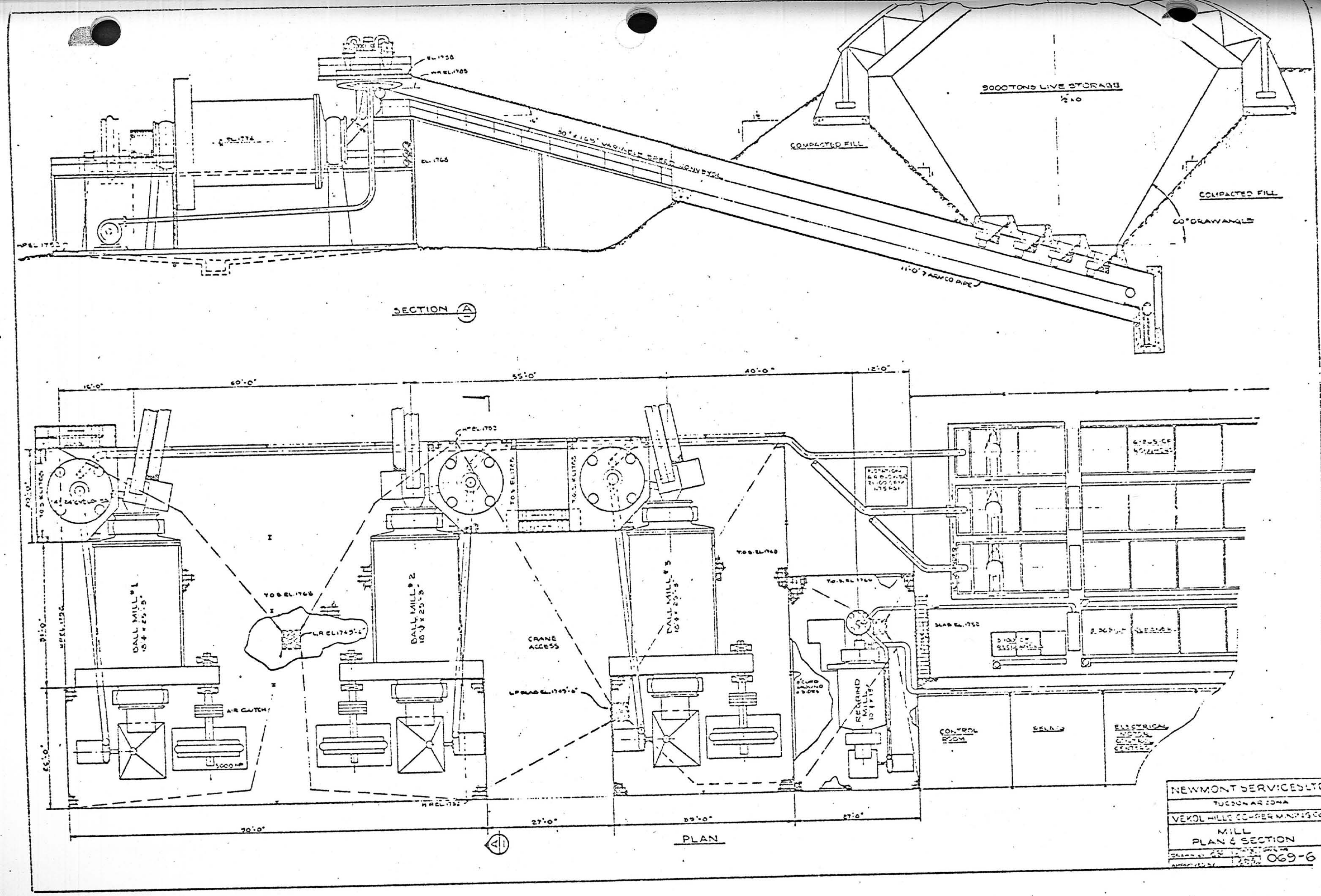
30,000 TONS LIVE STORAGE
EXISTING GRADE
SECTION A
SCALE 1"=50'
SECTION B
SCALE 1"=50'
SECTION C
SCALE 1"=50'
SCREENING TOWER PLAN
SCALE 20'=1"
SECTION D
SCALE 1"=50'
SECTION E
NEWMONT SERVICES LTD.
TUCSON ARIZONA
GENERAL ARRANGEMENT
SECTIONS
069-3



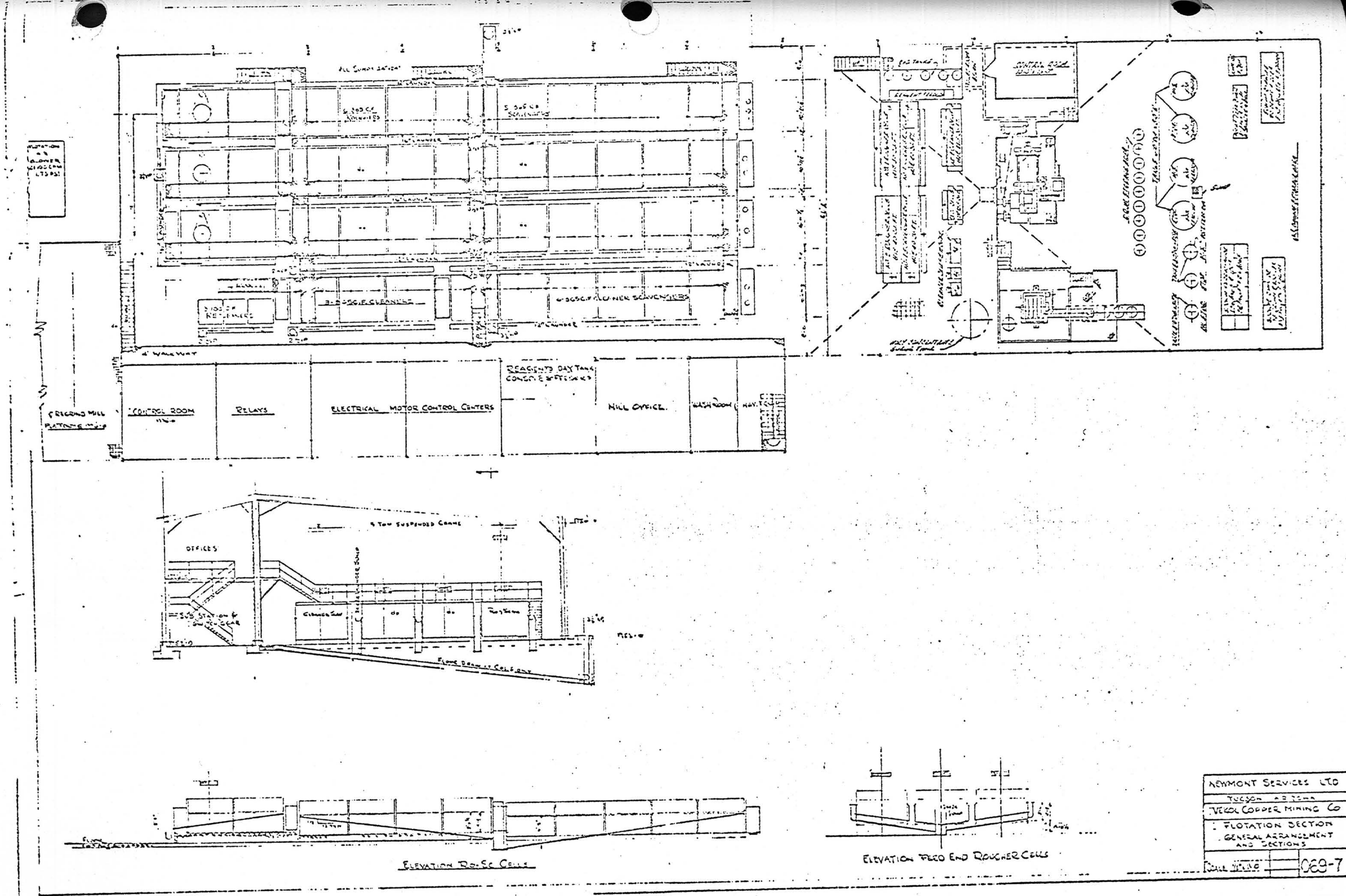
24'-0"
24'-0"
15'-0"
40'-0"
20'-0"
HATCH
ROCK PICKER
PILE EL.1853'-6"
EL.1807'-6"
65'-6"
10'-0"
EL.1790'
PRESENT GRADE
SECTION A
SECTION B
NEWMONT SERVICES LTD.
TUCSON ARIZONA
PRIMARY CRUSHER
GEN. ARRGT.
069-4



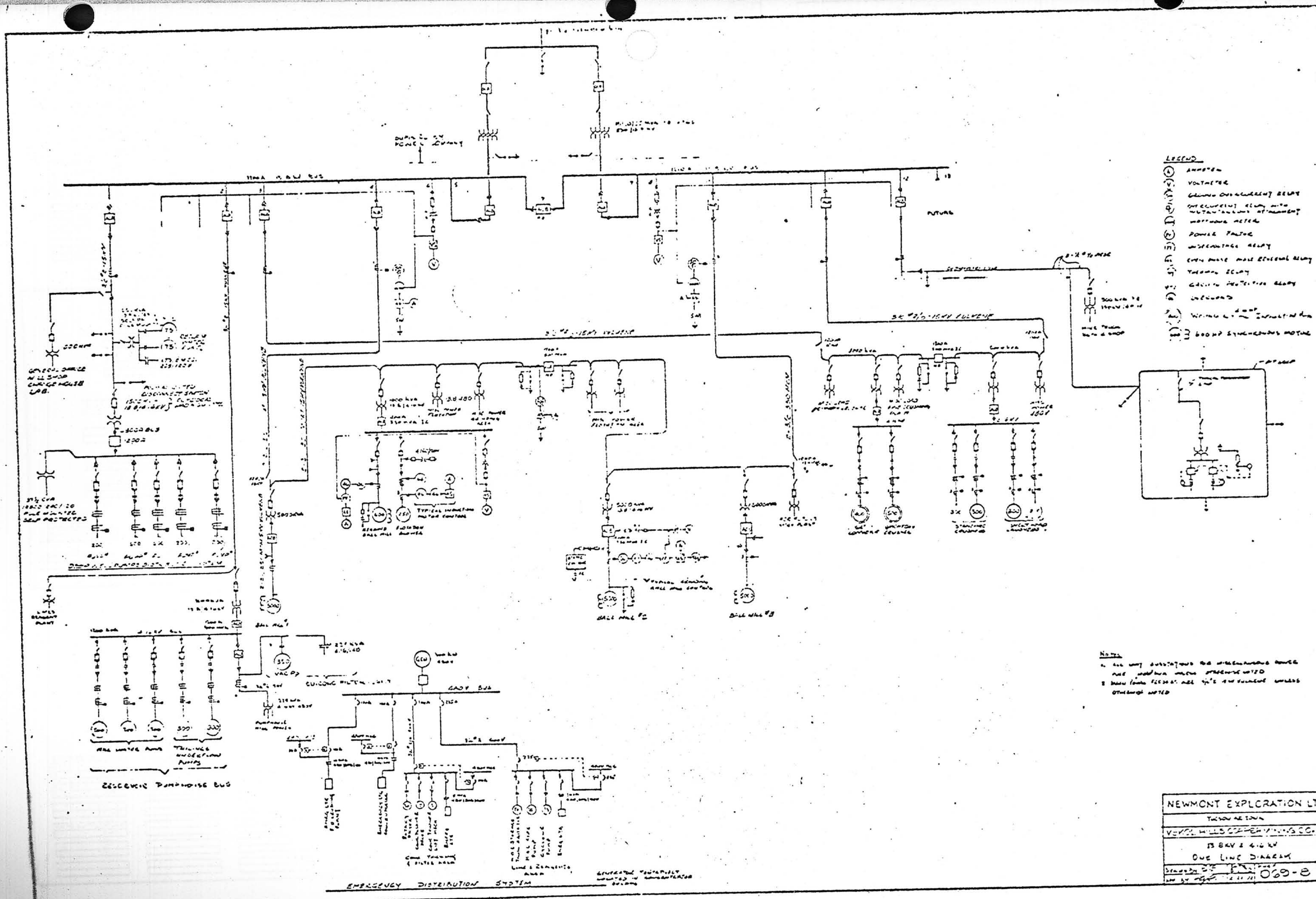
NEWMONT SERVICES LTD.
TUCSON ARIZONA
NEWCOL HILLS COPPER MINING CO.
FINE CRUSHING PLANT
PLAN & SECTIONS
069-5



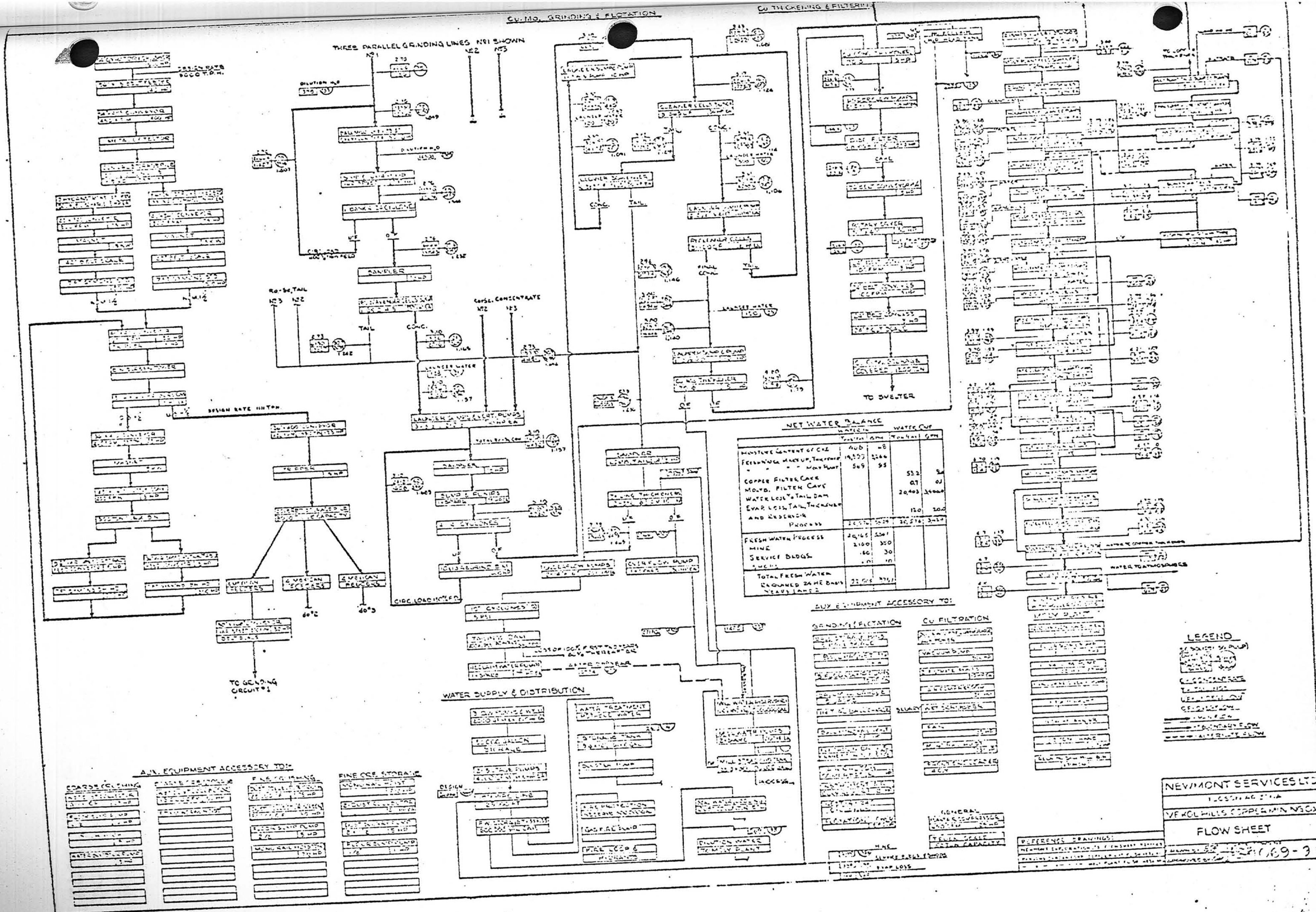
3000 TONS LIVE STORAGE
COMPACTED FILL
COMPACTED FILL
60° DRAWANGLE
11'-0" ARMCO PIPE
SECTION A
55'-0"
40'-0"
12'-0"
15'-0"
60'-0"
BALL MILL #1
BALL MILL #2
BALL MILL #3
CRANE ACCESS
AIR CLUTCH
REGRIND MILL
CONTROL ROOM
RELAY
ELECTRICAL
70'-0"
27'-0"
27'-0"
PLAN
NEWMONT SERVICES LTD
TUCSON ARIZONA
VEKOL HILLS COPPER MINING CO.
MILL
PLAN & SECTION
069-6



CONTROL ROOM
RELAYS
ELECTRICAL MOTOR CONTROL CENTERS
REAGENTS DAY TANK CONCENTRATE FEEDERS
MILL OFFICE
WASH ROOM
4' WALKWAY
REGRIND MILL PLATFORM
ALL SUMPS
5-300 CF SCAVENGERS
3-300 CF CLEANERS
4-300 CF CLEANER SCAVENGERS
OFFICES
SUB STATION & SWITCHGEAR
5 TON SUSPENDED CRANE
CLEANER SCAV
ROUGHER
FLOOR DRAIN TO CELLS ONLY
ELEVATION RO-SC CELLS
ELEVATION FEED END ROUGHER CELLS
NEWMONT SERVICES LTD
TUCSON ARIZONA
VEKOL COPPER MINING CO
FLOTATION SECTION
GENERAL ARRANGEMENT AND SECTIONS
069-7



LEGEND
NOTES
EMERGENCY DISTRIBUTION SYSTEM
NEWMONT EXPLORATION LTD
13.8 KV & 4.16 KV
ONE LINE DIAGRAM
069-B



Cu.Mo. GRINDING & FLOTATION
Cu THICKENING & FILTERING
THREE PARALLEL GRINDING LINES Nº1 SHOWN
Nº1
Nº2
Nº3
DILUTION H2O
SAMPLER
TAIL
CONC.
Ro-Sc TAIL
CONSL. CONCENTRATE
DESIGN RATE IN TPH.
TO GRINDING CIRCUIT
CIRC. LOAD
TO SMELTER
FINAL CONC.
NET WATER BALANCE
WATER IN
WATER OUT
MOISTURE CONTENT OF ORE
COPPER FILTER CAKE
MOLYB. FILTER CAKE
WATER LOSS TO TAIL DAM
AND RESERVOIR
PROCESS
FRESH WATER PROCESS
MINE
SERVICE BLDGS.
TOTAL FRESH WATER
REQUIRED 24 HR BASIS
AUX EQUIPMENT ACCESSORY TO:
GRINDING & FLOTATION
Cu FILTRATION
WATER SUPPLY & DISTRIBUTION
AUX. EQUIPMENT ACCESSORY TO:
COARSE ORE STORAGE
FINE ORE STORAGE
DESIGN
ACCESS
LEGEND
ALTERNATE FLOW
REFERENCE DRAWINGS:
NEWMONT SERVICES LTD
FLOW SHEET

REPORTS ISSUED ON VEKOL HILLS PROJECT

Metallurgical; from Danbury Metallurgical Laboratory:

Progress Report No. 1	January 24, 1970
Progress Report No. 2	March 18, 1970
Progress Report No. 3	May 25, 1970
Progress Report No. 4	July 27, 1970
Pilot Plant Investigations on a composite sample from the Vekol Hills deposit, Casa Grande, Arizona	September 14, 1970
Molybdenite Flotation Testing on Vekol Hills concentrate from pilot plant test	November 9, 1970
Metallurgical Investigations on ores from the Northeast Extension of the Vekol Hills Main Pit	February 7, 1972
Summary of Results of Metallurgical Testing of ores from the Vekol Hills Deposit, Casa Grande, Arizona	April, 1972

Concentrating Facilities:

N.E.L. Job 52, Vekol Hills, 20,000 tpd concentrator, D. M. Shaw	April 1, 1970
1st Revision	November 1, 1970
2nd Revision	March, 1971
3rd Revision	April, 1971
4th Revision	October 1, 1971
5th Revision	February, 1972
"Notes on Estimates", D. M. Shaw	February 8, 1972
"Capital Cost Estimate, 20,000 tpd Copper Concentrator and Associated Facilities", D. M. Shaw	March 1, 1972
"Vekol Hills Plant Design Criteria", D. J. Christie	December , 1969
Revised	May 20, 1971 and January 18, 1972

Mining:

"Report Vekol Hills Project", W. K. Pincock	February 24, 1969
"Interim Report, Vekol Hills Project", W. K. Pincock	March 10, 1970
"Feasibility Report, Mining Section, Vekol Hills Project", W. K. Pincock	December 13, 1970
"Combined Stripping Operations, Scrapers Plus Shovels and Trucks", W. K. Pincock	September 15, 1971
"Feasibility Report, Mining Section, Vekol Hills Project", W. K. Pincock	February 16, 1972

Geology:

Lease Boundary Plan (Showing Topography and Drill Hole Locations), N.E.L.	
Drill Hole Logs, N. E. L.	
Bedrock Geological Plan, N.E.L.	
Vekol Hills Geologic Plan and Sections, N.E.L.	
General Arrangement Plan, 1400 Level, Vekol Hills Project, N.E.L.	July, 1970
Vekol Hills Project, Bench Plans Showing Ore Outlines	June 8, 1971
Vekol Hills, Average Molybdenum Content, N.E.L.	March 21, 1972