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MLB
15 cc for attachments to reports.

Parsons-Jurden Corporation

A SUBSIDIARY OF THE RALPH M. PARSONS COMPANY

26 BROADWAY, NEW YORK, N. Y. 10004

June 25th 1971

Newmont Exploration Limited
300 - Park Avenue
New York, New York 10022

ATTENTION of Mr. Robert B. Fulton
Vice-President

SUBJECT Vekol Hills Copper Project
Papago Indian Reservation
Arizona
Parsons-Jurden Job 4832-01

Gentlemen:

Enclosed are 15 copies each of pages 2-15a and 2-19a which are to be inserted into the Parsons-Jurden Volume I Report dated March 15th 1971. These insertions will be to Section 2 - Geology and Ore Reserves.

This new material resulted from our analysis of additional Geological information which was not available at the time of our report. This additional information more clearly defines the ore body and does not materially change the total economic ore reserves presented by Newmont Exploration Limited.

If we can be of any further assistance we will be available at your convenience.

Yours very truly,

PARSONS-JURDEN CORPORATION

Fred J. Pisacane

Fred J. Pisacane

FJP:mec

Enclosures (As Above)

NEWMONT EXPLORATION LIMITED
OFFICE OF THE SECRETARY
300 PARK AVENUE
NEW YORK, N. Y. 10022

(212) PL 3-4800

July 12, 1971

Mr. Ben F. Dickerson, III
The Superior Oil Company
P. O. Box 12487
Tucson, Arizona 85711

Dear Ben:

On March 22nd last, we sent you three copies of Parsons-Jurden Report on the Vekol Hills property. On pages 2-15 and 2-19 in Section 2 (Geology and Ore Reserves) of Volume I of the Report are set forth calculations and an estimate of the various ore types on the 1400 Level horizon based on the bulk sampling results. We have felt that the reader of these pages might be left with the impression that these findings are representative of the percentages of sulphide-oxide copper ore within the entire deposit which, of course, is not the case. We have asked P.J. to consider this and to modify the statements by addenda if they can do so in good conscience. This has now been done and the enclosed sheets 2-15a and 2-19a carry the modifying language. Copies of P.J.'s covering letter are also enclosed. Will you please see that these are inserted in the copies of the Report which were sent to you.

Thanks very much.

Sincerely yours,

R B Fulton

R. B. Fulton

RBF/la

Filed in P-J reports
7-20-71

Copy No. 4

NEWMONT EXPLORATION LTD.

VOLUME I
REPORT

VEKOL HILLS PROJECT
ARIZONA

P-J File No. 4832-01
March 15, 1971

PARSONS-JURDEN CORPORATION
A WHOLLY OWNED SUBSIDIARY OF THE RALPH M. PARSONS COMPANY
ENGINEERS-CONSTRUCTORS
PARSONS
NEW YORK

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VOLUME II

CAPITAL AND OPERATING COST ESTIMATES

LETTER OF
TRANSMITTAL

Parsons-Jurden Corporation

A SUBSIDIARY OF THE RALPH M. PARSONS COMPANY

26 BROADWAY, NEW YORK, N. Y. 10004

March 15, 1971

Newmont Exploration Ltd.
300 Park Avenue
New York, New York 10022

ATTENTION of Mr. Robert B. Fulton
Vice President

SUBJECT Vekol Hills Copper Project
Papago Indian Reservation - Arizona
P-J Job No. 4832-01

Gentlemen:

We are pleased to transmit 15 copies of our report, in two volumes, which review and evaluate the feasibility reports and work developed by Newmont Exploration for a proposed mine and processing facility to be installed at the Vekol Hills site, located south of Casa Grande, Arizona. The report takes into consideration slight changes and additions to the original scope, all of which were agreed to by the Client at the start of the work.

Volume I is written commentary describing the actual mechanics of our review of this project. Notable variances in capital and operating cost estimates, basis used in calculating these costs, and differences in interpretation and conclusions between the Client and Parsons-Jurden, drawn from the available data, are pointed out. Our suggestions and alternates which Newmont may care to investigate in connection with this project are also included in this volume.

Volume II is devoted essentially to our capital and operating cost estimates for both the mine and processing facility and contains other supporting information used in completing our report.

Based on an agreed upon engineering and construction schedule, appropriate escalation and contingency factors have been applied so that the submitted total capital cost estimate for the project is sufficiently accurate to assure that the project will not overrun by more than 15 percent or underrun by more than 10 percent.

The work of this report was done under the direction of Mr. F. M. Stephens, Jr., Vice President and Technical Director. Major contributors to the report were Messrs. I. Hauser, Vice President and Chief Engineer, R. J. Brison, J. Engel, E. H. Gates, K. I. Mackenzie and E. A. Mills.

We shall be happy, at your convenience, to discuss and review any aspect of our study and cost estimate and look forward to assisting you in the subsequent completion of engineering and construction of this report.

Very truly yours,

PARSONS-JURDEN CORPORATION

Fred J Pisacane

Fred J. Pisacane

FJP:mh

SECTION 1

PARSONS

SUMMARY

This report covers Parsons-Jurden's review and evaluation of Newmont Exploration Ltd.'s feasibility reports and work for the Vekol Hills Project. In this review, Parsons-Jurden personnel have visited the project site, have discussed the project plans with the Newmont staff members and consultants who have been working on the project, and have carefully reviewed the reports and documents pertaining to the ore reserves, mining plans and proposed plant facilities.

GENERAL

The review of the ore reserves and geology has convinced Parsons-Jurden that the work done by Newmont Exploration Ltd. has been done competently and thoroughly using techniques and procedures which are acceptable in the industry. Parsons-Jurden agrees with Newmont calculations which indicate that the Vekol Hills orebody contains 109,083,000 tons of minable ore which can be delivered to the mill at an average grade of 0.543 percent of total copper.

The proposed mining plan as detailed in the Newmont mining report is considered by Parsons-Jurden's mining engineers to be completely feasible. The mining equipment proposed is adequate for the mining operation assuming that mining is carried out on a 15-shift per week basis. Parsons-Jurden believes that better utilization of mine equipment could be made by planning on a 20-shift per week operation during the first years of the project. This should permit a capital cost saving of over $2.3 million in mining equipment.

With the exception of some several items which have a minor effect on capital and operating costs, Parsons-Jurden believes that the proposed concentrator and ancillary surface facilities, as proposed by Newmont, are adequate to process the Vekol Hills ore and to obtain metallurgical results equivalent to those obtained in laboratory and pilot plant tests.

OPERATING COSTS

The review of Newmont's projected operating costs has indicated that the basic mining costs of 19 cents per ton of material moved are reasonable. The review also indicated that the projected mining cost per ton of ore delivered to the primary crusher will range from a high of 78 cents per ton in Year 3 to a low of 29 cents in Year 15, with the average cost being 46 cents per ton of ore for the life of the project. However, Parsons-Jurden believes this average cost should be increased to 52.5 cents per ton to include the replacement cost of light vehicles and the added cost of crushing and hauling waste in Years 6 through 16.

The review of the milling costs indicated that the figures of 93.34 cents per ton of ore used in the Newmont report may be somewhat low. An independent analysis by Parsons-Jurden develops a cost of $1.02 per ton of ore milled. This increase is primarily because of higher projected wear iron consumption and a higher base cost of grinding media.

Summation of these costs indicates that the average direct operating cost per ton of ore mined and milled will be $1.545. Based on the data available for Parsons-Jurden's analysis, it is believed that this operating cost is accurate within a range of plus or minus 10 percent.

CAPITAL COST

A review of the projected capital costs for the mine has shown no appreciable difference between the equipment and preproduction stripping costs outlined by Newmont and those which Parsons-Jurden would consider normal. It is suggested, however, that allowances for labor and materials escalation and a 10 percent contingency be added. This would increase the total capital cost requirement of $22,215,000, as estimated by Newmont, to $26,658,000, assuming a 15-shift per week mining operation. If further investigation proves a 20-shift per week operation to be feasible it is believed that the mine capital requirement could be reduced to below $24,500,000

Parsons-Jurden's capital cost estimate for the surface facilities is based on a slightly different criteria than that used in the Newmont reports. The major differences are as follows:

1. Plot Plan is the one originally proposed rather than the one used for Newmont's revised estimate.

2. The primary crusher has been increased in size from a 54-inch to a 60-inch unit.

3. A repair bay has been added to the fine crushing plant.

4. Reagent mixing facilities have been increased.

5. An additional water well has been added.

6. The molybdenum plant has been estimated on a more detailed basis.

7. Escalation has been based on a later completion date.

8. A contingency factor has been added.

Using these criteria Parsons-Jurden's estimate indicates the surface facilities will have a capital requirement of $42,735,000.

When adjustment is made for the contingency factor, which was not used in the Newmont estimate, the Parsons-Jurden estimate for the surface facilities is $9,628,000 higher than the Newmont estimate. Of this amount some $2 million is the result of increasing the primary crusher size and other scope changes; $2 million is an increase in craft labor benefits to realistically represent the current labor contracts; $2.5 million is caused by higher unit price figures for concrete and site excavation; and the balance of $3.5 million represents the increased escalation, field indirects and engineer costs associated with the above listed items.

The review has indicated that the estimated Vekol Hills Project capital costs should be:

Mine	$26,658,000
Surface Facility	42,735,000
TOTAL	$69,393,000

Parsons-Jurden believes that, based on the data available and estimating techniques used, this estimate has an accuracy of plus 15 and minus 10 percent.

SECTION 2

GEOLOGY AND ORE RESERVES

INTRODUCTION

The objectives of this portion of the report are to describe and assess Newmont's techniques and results in the delineation and evaluation of reserves at the Vekol Hills Copper Project.

The information pertinent to this review and assessment was gathered during Parsons-Jurden's field examination of the property, by discussions with Newmont's project personnel, and through review of project documents provided by Newmont.

Field examination of Newmont's exploration and evaluation work was performed during January 4 through 7, 1971 by Mr. K.I. Mackenzie, Parsons-Jurden Senior Geologist. During this period the underground exploration workings and sample plant facilities were examined and discussed with Mr. M.A. Enright, Newmont Exploration Ltd. Exploration Geologist, supervising the underground work. Techniques, records and work-sheets pertaining to the estimation of reserves were also examined and discussed with Messrs. H.J. Steele, Senior Geologist, and D.F. Hammer, Exploration Geologist for Newmont Exploration Ltd. on the project. Mr. J. Guthrie, Exploration Geologist for Newmont Exploration Ltd., described and demonstrated core logging procedures.

Parsons-Jurden's home office review and assessment of the acquired information was performed during January 7 to March 12, 1971 by Messrs. K.I. Mackenzie and B.E. Russell, Parsons-Jurden Senior Geologist. During this period of review and assessment further information and supporting documentation was obtained through discussions between Messrs. K.I. Mackenzie and R.B. Fulton, President, R. Denny, Senior Systems Engineer, and W.C. Hellyer, Senior Metallurgist for Newmont Exploration Ltd. Supporting documents supplied by Newmont Exploration Ltd. are listed as footnotes at the end of this Section.

GEOLOGY OF THE DEPOSIT

The geology of the Vekol Hills deposit is outlined here to facilitate the presentation of this review and assessment. The descriptions of rock types and mineralization are summarized from Newmont Exploration Ltd. reports by Messrs. V. Vellet(1) and W.C. Hellyer(2) and these descriptions are coordinated with information acquired during Parsons-Jurden's inspection and through discussions with Messrs. H.J. Steele, D.F. Hammer and M.A. Enright. Pertinent stratigraphic details are drawn from a thesis by Mr. D.F. Hammer[3], and geologic structures are described as depicted on Newmont Exploration Ltd.'s geologic plan[4] and sections.[5]

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 dykes, and a few small sills of Laramide (?) hornblende porphyry. A surficial deposit of Tertiary sediments blankets the topographic lows concealing much of the older rocks.

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

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, but reportedly is also very coarse and is sometimes a pebble conglomerate. The quartzite can contain appreciable quantities of potash feldspar and is sufficiently permeable to be an important aquifer. Several diabase sills occur in the Dripping Springs formation; this diabase is described below as a separate rock unit.

Pyrite and chalcopyrite are the main primary sulfides and occur as disseminated mineralization, as fracture-filling mineralization in a typical crackle breccia structure, and, in lesser amounts, within fracture filling quartz veinlets. 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 but is less abundant than the accompanying pyrite and chalcopyrite. Secondary 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 porosity and permeability of the unit.

The Precambrian Mescal limestone conformably overlies the Dripping Springs quartzite. The Mescal limestone is a light grey to reddish brown or white, finegrained 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 considerably more fractured than the other rock types and these fractures are commonly filled with calcite veinlets about 0.1 inch thick. Mineralization is largely confined to these very fine fractures, and calcite veinlets. Primary sulfides are mainly very fine-grained pyrite, chalcopyrite and minor pyrrhotite which are commonly accompanied by magnetite. Molybdenite-bearing quartz veinlets occur in the limestone as in the Dripping Springs quartzite. Moderately oxidized sections clearly exhibit cuprite and green copper staining.

Precambrian diabase sills 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 the fractures of the crackle breccia structure. However, quartz veinlets with these sulfides and molybdenite occur as in the Dripping Springs quartzite. Oxidized portions of the diabase are slightly limonite stained and do not clearly exhibit the contained copper minerals, aside from occasional green copper staining.

The Cambrian Bolsa quartzite overlies the Mescal limestone, perhaps with slight angular incomformity, 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 very 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 Laramide (?) 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 accessary amounts of mica and chlorite. Alteration of feldspar to montmorillonite, strong silicification and introduced calcite are common results of hydrothermal alteration. Moderate to strong agrillization, probably montmorillonitization, 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 crackle breccia 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.

Further information on the mineralogical composition of the important host rocks, from Progress Report No. 4(6), are given in the table below:

Mineralogical Analyses of Rock Types

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
% Sulphides	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 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.[7] The proportion of copper-bearing to barren ferrous oxides in the deposit is not known but is expected to be small.

TECHNIQUES USED TO DELINEATE AND EVALUATE RESERVES

Surface Exploration Drilling Program

Surface drilling was initiated in 1966 under the supervision of Mr. V. Vellet, Senior Geologist, with the assistance of Messrs. D.H. Osborne and R. Skiles, Exploration Geologists of Newmont Exploration Ltd. In February 1970 Messrs. Steele, Hammer, Enright and Guthrie joined the project replacing, Messrs. Vellet, Osborne and Skiles. Drilling within the current pit limits had been essentially completed at the time that project personnel changed. Current project geologists reviewed logs by previous personnel and were check-logged by Mr. Vellet on a few current holes so as to ensure a uniformity of geologic descriptions in the logs.

The surface drill hole sampling program consists 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. These holes were subsequently deepened by core drill holes, referred to as 'V' series holes. Since none of the current project personnel were present when the rotary drilling was performed, Mr. R.B. Fulton has supplied the following description of the rotary drill hole sampling practices and procedures:

> "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 5-1/2 to 4-3/4 inches in diameter. Large diameter bits were used to penetrate the alluvium section following which 5-1/4-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 are Nx size, reduced to Bx size where necessary. The logged core is split to obtain a 180 degree section of core for a sample. The core samples vary in length from about 2 to 15 feet depending on the continuity of estimated copper grade. Core and rotary samples were assayed at the San Manuel assay office for total copper and a few samples have been assayed for molybdenum, silver and gold. Oxide copper analyses are currently being performed on samples composited for bench elevation intervals. The assay methods employed by the San Manuel assay office are outlined in Volume II. The remaining 180 degree core section is stored in 5-foot water resistant core boxes. These boxes are stacked in core racks at Newmont Exploration Ltd.'s Casa Grande warehouse. Rotary sample cuttings, in cloth bags containing about 5 to 6 pounds of samples each, are also stored in Newmont Exploration Ltd.'s Casa Grande warehouse along with the pulps and rejects of the drill hole samples.

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

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

Underground Exploration Testing Program

The underground exploration consists of shaft sinking, drifting, raising and drilling. The sampling plant operation consisted of separately passing the muck from each round through a converted aggregate plant to obtain a sample fraction for assaying.

The objectives of the sinking, drifting and raising are to check the reliability of surface drill hole sampling data, drill hole spacing and the interpreted distribution of reserve types, grades, rock types and faults. The underground drilling program has the objectives of further extending these tests. Other objectives of the underground drilling, apparently completed, include: (1) a check for major faults on the northeast side of the pit; (2) a test of the competency of rock on the south-southwest side of the pit; and (3) a check for an interpreted, but not found, major fault on the west side of the pit. The complete results of these tests are not yet available.

Underground exploration work was initiated on March 10, 1970 with the sinking of a timbered, two-compartment (manway and skip) vertical shaft after completing 12 feet of cemented shaft collar. The location of the shaft at coordinates 200N, 200E was planned to provide a check on the reliability of drill hole data from an adjacent hole. The shaft is 7 by 12 feet in section and was completed at a 441-foot depth on May 12, 1970. The shaft station was partially cut during shaft sinking and was completed at 1400-foot elevation by mid-May 1970.

Crosscuts and drifts, nominally 5-1/2 by 8-1/2 feet in section, were completed on November 1, 1970 with 2809 feet of advance. Raising was attempted in October but was delayed to November 1970 due to production difficulties. Three vertical raises, timbered, untimbered and partially timbered, and each about 200 feet high, were completed by January 7, 1971. The dimensions of untimbered raise and timbered raise (measured inside the timber) are both 5 by 6 feet. Two additional raises are planned and will bring the total raise advance to about 950 feet. The headings were driven on or near section lines.

Raises designed to test surface drill holes are located at coordinates 600N - 200W, ON - 200E and ON - 200W. Two other raises for the same purpose are planned for 600N - 400E and 400N - 200E. This work was carried out using two-crew shifts per day with three men per crew. The crews consisted of two drillers and a mucker working alternately at two headings being driven simultaneously. Mucking was performed with an Eimco 911 LHD having a 1-cubic-yard bucket. The advance per round of shaft, drift and raise averaged about 6 feet (6.5 tons per foot estimated), 5 feet (5 tons per foot estimated) and 5 feet (4 tons per foot estimated), respectively. The muck passed through a grizzly with 9-inch spacing to a 50-cubic-foot capacity loading pocket.

Underground core drilling was started September 30, 1970 with 10,000 to 12,000 feet planned for completion by February 28, 1971. About 7,000 feet of drilling was completed by January 7, 1971. Less than 500 feet of this total is Nx size core and the remainder is Bx size. The underground drilling includes some horizontal sectional drilling and some inclined sectional fan drilling (as on Section OE in Crosscut 600N), but consists mainly of inclined up-holes drilled off section and at various azimuths.

Survey control of the underground openings was carried out by Harvey W. Smith, Engineer, who plumbed the shaft and established underground survey stations from which the advance per each round and the location for each core drill set-up were measured.

The sampling plant consists of a converted Cedar Rapids, Junior Commander, Portable Aggregate Plant, Model 442 with additions as shown on the flow diagram in Volume II. Muck from the storage bin at the shaft collar is screened to minus 2-inch, the oversize is choke-fed to a jaw crusher and rescreened to minus 2-inch. The minus 2-inch muck is further screened to 3/4-inch, and the plus 3/4-inch oversize is roll crushed to minus 3/4-inch and is returned to the screens.

The minus 3/4-inch undersize from the screens is sent to a drum sampler. The slot on the drum sampler is set to obtain a cut of 5 percent, 7-1/2 percent, and 10 percent fractions during shaft sinking, drifting and raising, respectively. The respective 95, 92-1/2 or 90 percent reject fractions contain an estimated 13,000 dry short tons of 0.65 to

0.70 percent sulfide copper (oxide copper deducted) and 1320 dry short tons of 1.13 percent oxide copper. There is no estimate of waste muck stockpile which is to be used to back-fill the shaft.

The 5, 7-1/2 or 10 percent sample fractions are fed to either or two 50-cubic-foot capacity storage bins. These bins feed muck to a roll crusher. The relatively dry, soft muck from the oxide zone is roll crushed to minus 3/16 inch (4 mesh), but the relatively wet, hard muck from the sulfide zone requires additional water for screening and is roll crushed to minus 3/8-inch.

The discharge from the roll crusher is sampled to obtain a 5 percent sample fraction using an automatic sampling arrangement similar to a pulp distributor. The 95 percent reject fractions from the sampler (about 1-ton of muck per round) are individually piled and marked near the plant. The 5 percent sample fraction is hand split in a Jones Splitter to a 2-pound sample. The reject from the Jones Splitter, about 150 to 200 pounds, is individually sealed in labled drums and stored at Newmont's Casa Grande warehouse.

The 2-pound sample is pulverized, mixed and split in a Jones Splitter to 4-1/2-pound samples. One 1/2-pound sample fraction is analyzed for total copper, oxide copper and occasionally molybdenum. The remaining 3-1/2-pound sample fractions are stored at Newmont's Casa Grande warehouse.

Wet muck from the sulfide zone (from below the water table) is reported to stick on the screens and it is general practice, therefore, to keep the screens clean by hosing them with water. The plant was not operational during the field examination; however, the introduced water is expected to result in some loss of fines due to spillage and leakage. The plant was cleaned between processing each round, but a few hundred pounds of material, not practical to remove, is reportedly left in the equipment.

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 up-dated 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 under the supervision of Mr. V. Vellet. 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 under the supervision of Mr. Steele to agree with the new plan.

On October 29, 1970 a computer check calculation, supervised by Mr. Denny, 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 procedure used to calculate the reserves in February 1970 are as follows:

1. Drill holes were plotted on northwest sections at a scale of 1 inch to 50 feet. Tro-Pari survey results were reportedly used to plot deflections of the holes.

2. Geological and assay data from the drill logs and data sheets were transferred to the plotted drill holes. This included data on rock types, structures (faults), type of mineralization (oxide or sulfide) and assay values (total copper; some molybdenum, silver and gold; and very few oxide copper).

3. Hole-to-hole correlations of rock types, structures, ore types, waste and alluvium were made. The criteria used for ore types are:[8]

   - Sulfide Ore - 0.30 percent or more total copper and 0.25 percent or more sulfide copper
   - Oxide Ore - 0.30 percent or more 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 are reportedly delineated as such based on visual estimates of the copper minerals noted in the drill logs. Spot checks of logs revealed that the presence of the various copper minerals are noted, but no estimate of percent content is shown.

4. Oxide copper assays, for samples composited at bench intervals, are currently being obtained and are posted on the sections as they become available.

5. Northeast sections were constructed as in Items 1. through 4. above, and differences in the hole to hole correlations between the northwest and northeast sections were resolved.

6. The correlated data, showing the boundaries of the reserve types, were transferred to bench plans at a scale of 1 inch to 200 feet.

7. Reserve measurements were made on the northwest set of sections. The interpreted boundaries of the reserve types were used as a basis for determining the quantity of waste rock and type of reserves on the 50-foot benches within a preliminary pit limit shown on the sections. However, the locations of the reserve type boundaries that were measured on the bench medians were not the same as the locations of the boundaries already interpreted in Item 3. above.

   Vertical cutoff lines, based on further geological interpretation and/or to facilitate measurement of reserve categories, were established at or near each boundary intercept of the bench median. The horizontal intervals between these vertical cutoff lines were taken as the measure of the quantity of each reserve type along the section of a given bench. The horizontal intervals of the reserve types between the cutoff lines were measured and their locations were transferred to the 1 inch to 200 feet bench plans. Each of these measurements was also tabulated by section number, bench elevation and reserve type.

8. Grades were obtained for each horizontal interval of the bench between the cutoff lines by first averaging, for each drill hole, a set of sample assays corresponding to the bench interval. These drill hole bench interval grades were next used to obtain the grade of each horizontal bench interval of a given reserve type, usually by calculating an area-weighted average of the bench interval grades between respective cutoff lines.

However, further geological interpretation was often used to select, project and weight the bench interval grades when determining the average grade of the reserve type between cutoff lines. Bench interval grades from the outside of the pertinent bench and cutoff lines were, therefore, averaged with bench interval grades for a reserve type between cufoff lines; some bench interval grades were consequently used more than once during reserve calculations.

9. The cutoff lines were used to establish polygon areas of waste and reserve types on each bench plan. The areas of the polygons were determined and the corresponding volume of the reserve type was calculated using the bench height, and was 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 composites of drill core.(9)

10. The tonnages of the reserve types for each bench level are tabulated and totaled on Newmont's reserve record sheets.

11. Grades, corresponding to the length, area volume and tonnage measurements described above, were calculated using length, area, volume, and tonnage weighted grades respectively.

The estimate of reserves in the sulfide category, resulting from the February 1970 calculation, is 89,710,000 dry short tons in place, at 0.560 percent total copper. This does not include dilution. A computer check of this estimate was completed on October 29, 1970. Input on the computer method consisted of the drill hole data, the February 1970 pit limits, and the geologically interpreted outlines of the reserve types for each bench. The October 29, 1970 computer-calculated reserves are 89,572,000 dry short tons in place, at 0.564 percent total copper without dilution.(10)

The manual reserve estimates, calculated during March 10 to May 23 and during November 1970, are reported in terms of undiluted tons and grade for each bench and in terms of diluted tons and grade for each year of production, respective to the proposal plan.(13)

ASSESSMENT OF TECHNIQUES USED TO DELINEATE AND EVALUATE RESERVES

Assessment of the Reliability of Drill-Sampling Data

The reliability of the surface drill hole data can be assessed by comparing the sample results from the drill holes with the sample results from the underground headings. This assessment is based on the assumption that the sampled headings are representative of a block of mineralized ground that is within 50 feet of the centerline of the headings, both vertically and laterally. This test block is essentially a 100-foot wide portion of the 1350 and 1400 benches that is centered on the section lines followed by the underground workings. Two reserve estimates were made for this block. One estimate is based on the sample data from surface drill holes and the other estimate is based on the sample data from the underground headings.

The drill hole indicated grade of the reserves in the tested block is determined from the grade of drill hole intercepts of the block. About one-third of the footage of these intercepts are from rotary holes and the remainder is from core holes. The grade of each drill intercept of the block is taken as the total copper grade value shown on the plans of the 1350 and 1400 bench drill hole intercepts.(14) The part of the test block that falls within the waste boundary as shown on the bench plans,(14) are accepted as waste in this Study. Each drill intercept grade is given a volume of influence extending within the test block to a vertical plane midway between holes, but within the ore outlines.

The average grade of the drill indicated reserves in the block is obtained by weighting the grade of each part of the block by the respective volumes of these parts. The drill indicated volume and grade for reserves within the block, calculated by the above method, is 21,242,350 cubic feet at 0.542 percent total copper. The reserves in the tested block, as well as all reserves on the 1350 and 1400 benches, are all classed as sulfide reserves.(15)

The sample results from the underground headings, as shown on the sample assay plan(16) of the headings on the 1400 level, are also used to obtain a volume and an average grade for the plus 0.30 percent total copper reserves of the tested block. The waste portions of the block are between 80 and 365 feet long and correspond well with the locations of waste sections as shown on the bench plans.(17) The grade for each part of the block is obtained by weighting each sample grade by the sample length. The average grade for the block is obtained by weighting each part of the block by the volume of the respective parts. The volume and grade of reserves above 0.30 percent total copper in the block as obtained using the above method on sample data from the underground heading, is 21,529,000 cubic feet at 0.563 percent total copper. These reserves in the tested block, as outlined by the sample results from underground headings, include significant qualities of non-sulfide reserves to be discussed below; however, the two reserve estimates of plus 0.30 percent total copper material, indicated by both sample data from the drill holes and sample data from underground heading, can be compared on a volume basis.

The volume of reserves estimated for the tested block using surface drill hole data is 1.33 percent low if the estimated reserve volume, obtained by using sample data from underground headings, is assumed to be truly representative of the same block. The reserve grade corresponding to the drill hole indicated reserve volume is also low by 3.73 percent if the reserve grade, indicated by sample data from the underground heading, is also assumed to be truly representative of the tested block.

Significant quantities of non-sulfide reserves have been outlined in the tested block using sample data from the underground headings. The drill indicated reserves for the tested block, and all reserves on the 1350 and 1400 benches are, however, classified as sulfide reserves on the bench plans(17) and in Pincock's tabulation of bench level reserves.(18) The quality of the drill indicated reserves in the tested block can be assessed if the sample data from the underground heading is assumed to be truly representative of this block. For the purpose of this assessment the reserves are further defined without changing basic criteria currently used to define the reserves.(18) The reserves in the test block are classified as follows:

1.a. Sulfide reserves are defined as containing over 0.30 percent total copper and 0.25 percent or more sulfide copper only when the oxide copper content does not exceed 10 percent of the total copper content.

1.b. Oxide-rich sulfide reserves are defined as sulfide reserves with an oxide copper content exceeding 10 percent of the total copper content.

PARSONS

Note: The following sentence should be added to the paragraph immediately following Item 4 on Page 2-16:

The bulk sample taken from the 1400 level is not representative as regards the sulfide-oxide ore ratios in this deposit. For explanation see statement in Conclusions on Page 2-19a.

See letter [illegible] of report

2. Oxide reserves are not defined further and consist of 0.30 percent or more oxide copper and less than 0.25 percent sulfide copper.

3. A marginal reserve category is added for material containing 0.30 to 0.55 percent total copper, but less than 0.30 percent oxide copper and less than 0.25 percent sulfide copper.

4. Waste rock is not defined further and consists of less than 0.30 percent total copper.

The parts of the block, defined by these criteria, were kept to a minimum volume of 500,000 cubic feet.

The quantity and grade of each of the reserve types within the test block, determined by the methods described above and using sample data from the underground heading, are as follows:

1.a. Sulfide reserves are 10,397,000 cubic feet at 0.584 percent total copper and .044 percent oxide copper.

1.b. Oxide-rich sulfide reserves are 6,657,000 cubic feet at 0.606 percent copper and 0.103 percent oxide copper.

2. Oxide reserves are 3,005,000 cubic feet at 0.482 percent total copper and 0.329 percent oxide copper.

3. Marginal reserves are 1,470,000 cubic feet at 0.387 percent total copper and 0.234 percent oxide copper.

4. The waste volume is 6,335,000 cubic feet.

Reserves of plus 0.30 percent total copper within the tested block are currently shown as sulfide reserves on Newmont's bench plans(19) and in Mr. Pincock's tabulation of reserves(20); however, the sample data from the underground headings indicate that on a volume basis these reserves actually consist of 48.3 percent sulfide reserves, 30.9 percent oxide-rich sulfide reserves, 14.0 percent oxide reserves, and 6.8 percent marginal reserves.

Evaluation of Rotary Drill Hole Sample Data

A number of core holes were drilled about 5 feet away from previously drilled rotary holes and provide an opportunity to check the reliability of rotary drill hole samples. The core holes were drilled in February

1970 and the drill hole sample assays for corresponding sample intervals from the core and rotary holes are reported on pages 15 through 17 of Newmont's Progress Report No. 2 of metallurgical testing.[21] Two core holes, DR-9A and DR-201A, are respectively adjacent to rotary holes R-9 and R-201. Together these drill holes provide sample data for nine corresponding pairs of core and rotary sampled intervals.

The nine corresponding intervals represent a check sampled drill hole distance totaling 386 feet and include 177 feet of diabase, 110 feet of shale, 51 feet of quartzite, and 48 feet of limestone. The nine rotary sampled intervals all have lower oxide copper assay values than the corresponding core sample intervals, and eight of the nine rotary sampled intervals have lower total copper assay values than the corresponding core sample intervals. The weighted averages of total copper assay values for core and rotary samples are 0.382 and 0.258 percent total copper, respectively, and the weighted averages of oxide copper assay values for core and rotary samples are 0.121 and 0.060 percent oxide copper, respectively. The total copper assay and the oxide copper assay for rotary samples are 67.5 percent low and 49.6 percent low, respectively, assuming that the core sample assay values represent 100 percent of the total and oxide copper in the mineralized ground drilled by the rotary holes.

Assessments of Techniques Used to Measure Reserves

The northwest-striking vertical section 200E was selected by Parsons-Jurden for a check of the methods of manual ore reserve calculations used by Newmont. This section passes close to the center of the proposed pit and the shaft. The main heading on the 1400 foot elevation are also located along the 200E section.

The records used by Parsons-Jurden in this check are a 1 inch to 50 foot scale blue-line print of section 200E and reserve tabulation sheets provided by Newmont. The information shown on the sections includes the location of drill holes with their indicated deflections, drill hole sample intervals with the assay values for total copper plus a few for oxide copper and molybdenum, the calculated bench interval average of total copper assay values for each drill hole, interpreted boundaries of rock types and reserve types dated February 1970, vertical cutoff lines, and a pit profile dated February 1970. The reserve tabulation sheets are dated February 1970 and list the measured and calculated quantities and grades of sulfide reserves, oxide reserves and waste and alluvium for each section and bench.

The procedures used by Parsons-Jurden to check the reserve measurements are essentially the same as used by Newmont. The outlines of reserve types established by Newmont on the basis of geological interpretation were accepted by Parsons-Jurden; however, the location of vertical cutoff lines established by Parsons-Jurden varied slightly from those established by Newmont. The check measurement of reserves made by Parsons-Jurden on section 200E are restricted to linear feet of each reserve type due to the method of reserve measurements. The Parsons-Jurden measurement of each reserve type and the corresponding grades are listed below with the measurement and grades shown on Newmont's data sheets for section 200E.

Linear Feet and Percent Total Copper

(measured in Section 200E)

	Newmont	P-J
Sulfide Reserves	16,724 ft. at 0.631%	16,608 ft. at 0.637%
Oxide Reserves	3,603 ft. at 0.621%	3,900 ft. at 0.600%
Total Reserves	20,327 ft. at 0.629%	20,508 ft. at 0.630%
Waste	12,194 ft.	11,900 ft.

The sum of all measurements by Parsons-Jurden is about 0.3 percent lower than the sum of all measurements obtained by Newmont and may be due to a slight shrinkage of the blue-line print.

CONCLUSIONS

The method and procedures used to extract muck from the underground heading at Vekol Hills is considered to provide material that is highly representative of the mineralization in the vicinity of the workings. The reduction of this material in the sample plant, to sample size proportions, was not observed; however, examination of the plant indicates that the automatic sampling devices proportionally reduce the full cross-sectional area of material flow in the sample plant. The automatic sampling devices plus the subsequent manual splits with the Jones Splitter are therefore expected to provide reliable sample cuts of the underground muck.

Some fines, however, are lost from the plant due to leakage and spillage when water is used to keep the screens clean. Since the fines carry a high proportion of the copper values in the muck, the loss of fines may result in samples that are slightly lower in percent copper than the material they represent. This condition is expected to occur mainly in relation to muck from the sulfide reserves since the screens generally required washing only when material from below the water table was processed.

The rotary sample values for total and oxide copper from two rotary holes are 67.5 percent and 49.6 percent, respectively, of the values obtained in the corresponding samples from two adjacent core holes. The size of the samples and the consistency with which the lower values are reported for the rotary samples indicates that the rotary samples are biased and consistently given lower copper grades than the material they represent. This bias appears to be stronger for the oxide copper grades than the total copper grades and is possibly due to a down-the-hole-loss of fines that contain a high proportion of oxide copper. The check sample data from the core holes is adequate to show the direction and approximate magnitude of the bias; however, sample data from the shaft and raises, when available, will probably provide more accurate estimate of the amount by which the rotary sample assay values are low.

The check on the reliability of surface drill hole data is based on the assumption that the sample resulting from the underground headings are truly representative of the quantities and grade of the material in the test block. On this basis the drill hole indicated volume of the test block is low by 1.33 percent of the true volume of reserves in the block, and the total copper grade is low by 3.73 percent of the true grade of the block. The footage sampled by rotary drilling in the test block is about one-third of the total footage in the block and the total footage drill sampled within the pit limits is estimated to consist of the same proportion of rotary and core footage.

The rather close agreement of the volume and grade values obtained by both methods indicates that the drill hole spacing and sampling methods probably provide reliable data for the estimation of reserves in terms of total copper. Further definition of the reserve in the test block, based on the results of underground sampling, indicate that 20.8 percent of the reserve in the test block, now considered to be sulfide reserves, contain 34 percent of its total copper in the form of oxide copper. The current drill hole data does not, therefore, adequately indicate the quality of the reserves. Oxide copper analyses of drill hole samples should, however, provide adequate data to reliably evaluate the quantity and quality of the reserves.

The tonnage factors presently being used are estimated values based on density measurements of samples composited from several rock and ore types. Further density measurements on specific rock types and ore types should, therefore, be made to substantiate the presently used values.

The quantity of copper-bearing ferrous oxides in the deposit is expected to be very small and perhaps is restricted largely to the non-sulfide reserves. However, some montmorillonite in the Santa Catalina formation has been determined to contain 1.74 percent copper. This copper may contribute significantly to the copper content of this formation since its montmorillonite content is about 10 or 20 percent.

Check measurements of the reserves performed by Parsons-Jurden on Section 200E confirms the reliability of the reserve measurement techniques used by Newmont. The resulting reserve estimate is, however, based on grade data and density data, both of which need further refinement and definition as indicated above. The reserves are adequately, perhaps slightly conservatively, estimated in terms of percent total copper; however, further definition of the quality of the reserves is necessary.

RECOMMENDATIONS

It is recommended that the drill hole sample data be further evaluated using the available sample data for the shaft and raises. Once the reliability of the drill hole data is established, the outlines of the reserve types can be refined on the 1 inch to 50 foot sections and the reserves revised accordingly, using the oxide copper assays currently being obtained. Oxide copper assay data for the reserves above the 1200-foot bench level should be at least 90 percent complete for the reliable estimate of the quality of the reserves.

Density measurements on bulk samples of the important host rocks are recommended to substantiate the tonnage factors presently being used, and further investigations can be made to check the amount of copper-bearing ferrous oxides and montmorillonite in the reserve tonnage.

Footnotes

(1) Megascopic Mineralogy of the Vekol Hills Orebody, Arizona, Newmont Exploration Limited, V.V. Vellet, September 23, 1966.

(2) Metallurgical Testing of Samples from Vekol Hills Deposit, Casa Grande, Arizona (Progress Reports Nos. 1 thru 4). File No. 240-01, W.C. Hellyer and S.W. Nabbs, Newmont Exploration Limited, Metallurgical Department, Danburg, Connecticut. Dated respectively[a]January 24,[b]March 18,[c]May 25 and[d]July 27, 1970.

(3) Geology and Ore Deposits of the Jackrabit Area, Pinal County, Arizona Graduate College, The University of Arizona, D.F. Hammer, dated 1961.

(4) Bedrock Geological Plan (showing topography and drill hole locations), Scale: 1 inch to 500 feet, not dated.

(5) Geological Sections (An incomplete set showing drill hold data, rock types and ore types on east-west and north-south section.) Scale: 1 inch to 100 feet (poor quality photo reduced reproductions of 1 inch to 50 foot scale sections), not dated.

(6) Addendum to Table IV, Mineralogical Analyses of Rock Types, Progress Report No. 4, under (2) above.

(7) Memorandum, Preliminary Studies of Non-Sulfide Copper Occurences at Vekol Hills, Arizona, Newmont Exploration Limited, J.W. Ahlrichs to W.C. Hellyer, dated August 24, 1970.

(8) Progress Report No. 3. See (2) above.

(9) Progress Report No. 1. See (2) above.

(10) Memo (results of the computer check of manually calculated reserves), Vekol Hills Project, Newmont Mining Corporation, J.R. Denny to W.K. Pincock, dated October 9, 1970.

(11) Feasibility Report, Mining Section, Vekol Hills Project, Newmont Mining Corporation, W.K. Pincock, P.E., dated December 13, 1970.

(12) See (11) above.

(13) a. Memo. See (10) above.
b. Memo.(results of the computer check of manually calculated reserves), Vekol Hills Project, Newmont Mining Corporation, J.R. Denny to W.K. Pincock, dated December 8, 1970.

(14) Bench Plans (computer plotter out-put plans for each bench elevation showing bench limits, drill hole locations, and drill hole grades). Scale: 1 inch to 200 feet, dated November 19, 1970.

(15) See (2) and (14) above.

(16) General Arrangement Plan, 1400 Level, Vekol Hills Project Newmont Exploration Ltd., (Geological, assay and ore reserve plan of the underground drifting and surface drill holls values at the 1400 bench). Scale: 1 inch to 50 feet, dated july, 1970.

(17) See (14) above.

(18) See (11) above.

(19) See (14) above.

(20) See (11) above.

(21) Progress Report No. 2. See (2) above.

SECTION 3

MINE PLAN

The "Feasibility Report, Mining Section, Vekol Hills Project," dated December 13, 1970, by Mr. W.K. Pincock describes a workable mine plan with safe pit slopes and a sequence of mining that provides the mill with a fairly constant copper value year by year. The maximum slope used in the alluvium cover is 40°. In the southwestern portion of the pit a maximum slope of 48° has been used and in the ends and northern area 50° has been used. The overall slopes vary from 35° 26' to 49° 38' according to rock types and pit configuration. These slopes compare favorably with those used by other mining operations in Arizona.

The mining plan has been laid out using 50-foot benches. It is detailed bench by bench and year by year for the first ten years and the final pit plan has been developed.

Because of the way the orebody is situated, the stripping and waste removal is heavy during the first few years of production and then rapidly decreases. There is no obvious way to smooth out the waste removal curve.

Proven equipment has been selected throughout and operating costs have been based on using 120-ton trucks. The 15-cubic yard electric shovels used in the estimate and the 12-1/4-inch drills have been used on similar operations in Arizona with good results.

Comments on the capital and operating cost estimates appear later in this report.

The following three items should be investigated further:

1. The advisability of putting the primary crusher in the pit during Year 5.

2. The possible savings in capital cost by starting the mine on a 20-shift week rather than on a 15-shift week as shown in the report.

3. The profitability of the first few production years - This can be done when the present sample analyses are completed. The reason for this is that the first few years mill feed may show a low copper recovery against a known high mining cost per ton of ore.

SECTION 4

PROCESS FACILITIES

The metallurgical basis for review of the process facilities has been taken from the results of the laboratory test work on a Master Composite Sample of Vekol Hills sulfide ore as reported in "Progress Report No. 3" by Newmont's Metallurgical Department at Danbury, Connecticut. Also utilized were reports by the Institute of Mineral Research, and by Messrs. Hellyer and Nabbs of Newmont on the pilot plant test work conducted at Houghton, Michigan.

The basis for review of the material handling and processing of the Vekol Hills ore in the proposed physical plant has been taken from the process flowsheet, plot plan, and layout drawings developed by Mr. D.M. Shaw from criteria outlined by Mr. D.J. Christie. The flowsheet has been arranged to indicate material flowrates and specific gravities, types and sizes of processing equipment, power, water and reagent requirements; the drawings show the proposed space and arrangement of equipment as specified in the process flowsheet. The review covers the compatibility of these facilities and the metallurgical requirements.

The following metallurgical balance was prepared from the flotation test results reported by the Danbury laboratory on the Master Composite Sample. It was used for the evaluation of the flowrates indicated on the Newmont flowsheet and as a starting point for the moly plant flowsheet and layout required for the estimate.

	Product Weight	Assays		Metal Content Tons/Day		Distribution	
	DST/D	% Cu	% Mo	Cu	Mo	% Cu	% Mo
Heads	20,000.0	0.543	0.014	108.60	2.80	100.0	100.0
Cu Concentrates	326.6	28.3	0.172	92.28	0.56	84.98	20.0
Mo Concentrates	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	-

*NOTE: For a conservative estimate of rougher concentrate loads to the cleaner circuits, the weight of this product was calculated on the expected rougher flotation performance at a pulp pH of 10.1. The test work has indicated that the optimum pH would be about 10.7, which would result in a higher grade and somewhat lower weight of product.

The above metallurgical balance and the flowsheet are based upon flotation results obtained on an ore sample having a low oxide copper content of about 0.05 percent. Poorer metallurgical results may be obtained by this flotation process if Vekol Hills ores containing larger amounts of copper oxides are treated. Laboratory and pilot plant tests have resulted in copper recoveries of only 52 percent on ores that contain 0.14 percent copper as oxide.

The review by plant areas of the process requirements for copper concentration, together with the physical facilities provided, follows.

COARSE CRUSHING - AREA 10

In order to accommodate a five-foot maximum feed, Newmont gave instructions to change the size of the 54-inch gyratory crusher shown on their flowsheet to 60 inches. The building and crane size were changed accordingly and the estimate reflects these changes.

The trucks hauling ore from the pit will dump directly into the feed of a 60-inch by 89-inch gyratory crusher driven by a 500 horsepower motor. The crusher has an open side setting of 6-1/2 inches and discharges into a 360-ton live surge pocket. Crushed product is then fed by a 60-inch by 20-foot apron feeder onto a 54-inch conveyor belt loading to the top of the coarse ore storage pile.

The trucks can dump into the primary crusher from two sides. An overhead crane has been supplied with both a 75-ton and 15-ton hoist to service the crushing equipment. A rock hook has been provided to help move material which may hang up in the feed to the crusher. A 10,000-cfm wet dust collector, air conditioning facilities for the control room, and a service elevator have been provided. An area for servicing mantles and concaves and a zinc melting furnace is included.

This crushing plant can make a 100 percent minus 9-inch product at an average feed rate of 2220 tons per hour. To accommodate the concentrator's requirement of 20,000 tons per day, the crusher will operate nine seven-hour shifts per week.

COARSE ORE STORAGE - AREA 20

Provision has been made for the storage of 150,000 tons of minus 9-inch primary crusher product. Of this total, 50,000 tons are live capacity. When necessary, the remainder can be recovered with the assistance of a dozer or front-end loader. This storage will allow the mine to operate when the concentrator is down for repairs and will allow the concentrator to run when ore is not delivered from the mine on weekends, or when production from the mine is curtailed for any reason.

The storage is in the form of a conical pile. The pile is approximately 280 feet in diameter at the base and 122 feet high and was calculated using a bulk density of 115 pounds per cubic foot.

A concrete tunnel has been provided under the pile to house six hydra-stroke feeders which discharge ore onto two 42-inch conveyors.

Each conveyor belt is protected by an electromagnet on a trolley, which facilitates removal of the tramp iron. In addition to the magnetic protection, a metal detector has been provided to detect nonmagnetic metal and material which has escaped the magnets.

Each belt is equipped with a belt scale to weigh the fine crusher feed. A 21,000-cfm wet-type dust collector has been provided.

Although the live storage provided calculates to be 50,000 tons based upon a draw-off angle of 60 degrees, experience has proved that variations in the ore effect the angle of draw-off and the actual live storage can be far less than the theoretical. When this occurs, cats or front-end loaders would have to be called upon to move the ore into position for draw-off. In order to avoid this, it would be preferable to install a pile with 100,000 tons of live storage. It would also be preferable to use pan feeders in lieu of hydra-stroke feeders provided. These modifications are suggested in Section 7 of this report.

FINE CRUSHING PLANT - AREA 30

The fine crushing plant is composed of two lines of secondary crushers and screens, followed by three lines of tertiary crushers and screens. The tertiary crushers are in closed circuit with their screens.

Each of the two conveyors bringing feed from the coarse ore storage feed a 7-foot standard cone crusher. The crushers discharge to 8 feet wide by 20 feet long vibrating screens with 1/2-inch square mesh openings. Screen undersize is the finished crushing plant product and conveyed to the fine ore bins. Screen oversize is conveyed to a surge bin ahead of the tertiary crushing circuit.

By means of three 48-inch by 60-inch vibrating feeders and three 36-inch belt conveyors, ore is fed to three 7-foot short-head cone crushers. Each tertiary crusher discharges to a 8-foot wide by

20-foot long vibrating screen with 1/2-inch square mesh openings. The tertiary screen oversize is the recirculated flow and joins the secondary circuit screen oversize which is conveyed to the surge bin ahead of the tertiary crushers. Screen undersize is the finished crushing plant product and joins the secondary cirucuit screen undersize which is conveyed to the fine ore bins.

A 31,800-cfm wet dust collector has been provided for the secondary crushing circuit and a 54,500-cfm wet dust collector has been provided for the tertiary crushing circuit. An air conditioned control room and a service elevator have also been provided.

To accommodate the 20,000-ton-per-day requirements of the concentrator, the fine crushing plant will operate 21 seven-hour shifts. The flowsheet just described is in accordance with that presented in Mr. D. Shaw's report. In estimating the structure, an additional 22-foot bay has been added to be used as a repair area. Additional inclusions are a tripper over the surge bin ahead of the tertiary crushers and raising the building roof in that area accordingly.

Although the flowsheet and design as described and estimated in Mr. Shaw's report are feasible, it would be preferable to rearrange the equipment so that the crane would have access to the screens. It would also be preferable to place 6-foot by 12-foot double-deck screens ahead of the secondary crushers and substituting 6-foot by 16-foot screens in lieu of the 8-foot by 20-foot screens used. The vibrating feeders at the surge bin could be eliminated and retractable belt feeders could be used instead, thereby reducing the amount of equipment needed. These modifications are suggested in Section 7 of this report.

FINE ORE STORAGE - AREA 40

Four fine ore bins having a total capacity of 7200 tons live load have been provided. The ore is distributed to the bins by a tripper conveyor and each bin is arranged to feed one of the four primary grinding circuits. A belt feeder conveyor and belt scale are provided to meter the ore from the bins to the grinding circuits.

The ore storage that has been provided amounts to 8.6 hours of operating time on a full bin and 5.6 hours at the end of the three-hour down-time period in the fine crushing plant. This amount of surge is considered adequate for normal operating conditions. In Section 7 of this report, however, a larger surge capacity has been suggested to provide for unforeseen contingencies, such as major problems in the crushing plant and unfavorable changes in angle of repose that may be caused by moisture content or changes in physical characteristics of the ore.

PRIMARY GRIND AND REGRIND - AREA 50

The primary grinding facilities comprise four identical grinding-classification circuits arranged in parallel. Each circuit contains one 16-1/2 by 24-foot ball mill and a cluster of five 20-inch hydrocyclone classifiers. In these circuits a single step reduction of the ore is made from the minus 1/2-inch crusher product to a particle size of about 65 percent minus 200 mesh which is required as feed to the rougher flotation circuits. The grinding energy consumption is taken at 11.2 kilowatt-hours per ton of ore.

On the basis of the above parameters, computations of mill size and power requirements check Newmont's figures as shown on the Computation Sheets in Volume II. It should be noted, however, that if the above energy requirements are correct, the ball mills and motor drives may not be large enough to produce the specified product size if slotted screens are used.

On the Newmont flowsheet, the circulating loads are indicated at 200 percent in the ball mill-classifier circuits. Parsons-Jurden believes this amount is too small. For this type of grinding operation where coarse, minus 1/2-inch feed is directly reduced in one grinding stage to a final 116 micron product, allowance should be made for a circulating load of at least 400 percent. Under comparable conditions, the Sierrita Concentrator is reported to be operating at 400 percent, and Silver Bell at between 250 and 400 percent circulating loads.

In Section 7 of this report a 400 percent recirculation is recommended, and the additional pumping and classifying capacity required is indicated. An additional suggestion is the installation of scoop feeders on the mills to reduce the pumping head to the hydrocyclones.

A particle size reduction of the copper rougher concentrate is required for liberation of the ore minerals from gangue before the upgrading of the concentrate is accomplished in the cleaner flotation sections. A 10-foot by 13-foot regrind mill with four 14-inch hydrocyclone classifiers has been provided for this purpose.

In the laboratory flotation test program this question of the ore quality was handled by segregating the rotary bench samples into sulfide or oxide categories on the basis of chemical analysis and further segregating the resulting sulfide samples into amenable and non-amenable materials on the basis of their response to flotation. As a result of this work a master composite sample was prepared representing all of the amenable sulfide ore. In essence, this master composite sample eliminated from the composite the sulfide ore above the 1550 level and the sulfide ore along the 10 west longitude line. If this procedure is followed in the actual mining of the deposit, its net effect will be to reduce the ore reserve by 6-1/2 million tons, or approximately 6 percent, which should ensure that the balance of the reserve would respond to treatment by flotation as predicted in the laboratory tests. This procedure would eliminate the need for considering selective mining, based on oxide content or flotation response in the upper portions of the pit.

The underground bulk sample taken from the 1400 level indicated an adverse oxide-sulfide ore ratio; however, this horizon has been accounted for in the master composite sample used for flotation testing and has not adversely affected the metallurgical results. Therefore, the ore from the 1400 level is not considered to be a potential problem in the overall flotation response of ore from this deposit.

Further evaluation of the oxide content of the "sulfide" ore in the upper sections of the pit - i.e., above the 1550 level, may permit the addition of part of the questionable 6-1/2 million tons of ore to the category of suitable mill feed.

See letters in front of report

The design data in the pilot plant report show that 4.53 kilowatt-hours per ton of solids are required to reduce the particle size of the concentrates from 55 percent to 90 percent minus 325 mesh. Calculations included in Volume II show that the regrind mill would supply a net energy input of 5.68 kilowatt-hours per ton to grind. This would indicate about 25 percent over-capacity based upon the amount of estimated rougher concentrate that will be produced.

The circulating load within the regrind circuit is indicated at 120 percent. Parsons-Jurden believes that the equipment should be designed to accommodate 150 percent circulating load to improve the grinding characteristics of the facility.

FLOTATION - AREA 60

The ground pulp from each of the grinding circuits is sampled and floated in four lines of rougher-scavenger flotation cells. Each line contains eleven 360-cubic foot cells and provides a nominal 13 minutes of retention time for the pulp at 30 percent solids. A separate head sample is taken by a sample cutter from the pulp stream to each of the four lines of cells. The tailings produced by the four lines are joined, sampled and then conducted to the tailing thickening facilities. All the concentrate produced by these cells is combined and serves as feed to the regrind circuit.

The reground product is cleaned in twelve 100-cubic-foot first cleaner flotation cells. The first cleaner concentrate product from these cells is then recleaned in three additional 100-cubic-foot cells. The recleaned concentrate produced from these last three cells, which contain both the copper and moly minerals, is then thickened to serve as feed to the moly recovery plant. The tailing from the recleaner cells is recycled to the first cleaner flotation circuit. The tailing produced by the first cleaners is scavenged in six 360-cubic-foot cells, to remove any residual floatable copper and moly minerals, and is then discarded to waste where it joins the rougher tailing stream flowing to the tailing thickening facilities.

The metallurgical balance covered earlier in this report was developed from the results of Newmont's laboratory work to estimate quantities of product to be treated in the moly plant process. On the basis of this balance some changes in the Newmont flowsheet involving quantities of solids and flowrates throughout the copper-moly cleaning circuits have been required. On the assumption that the floatable sulfide content of the Vekol Hills ore will not be substantially greater than that contained in the low-oxide wastes composite sample, it is believed unlikely that the quantity of rougher concentrate feed to the cleaning circuit will be greater than about 1608 tons of solids per day. This amounts to about 20 percent less material to treat than indicated in the Newmont flowsheet.

On the flowsheet in Volume II the flowrates have been re-balanced to reflect this difference and the number of cells have been reduced accordingly. The nominal flotation retention times for the facilities provided by Newmont in the cleaning section are as follows:

First Cleaner Flotation

12 - 100 C.F. Cells = 4.3 minutes @ 16% solids

Recleaner Flotation

3 - 100 C.F. Cells = 3.0 minutes @ 13% solids

Cleaner-Scavenger Flotation

6 - 360 C.F. Cells = 11.7 minutes @ 17% solids

NOTE: The percent solids shown is the net value in which the effect of recycling loads and launder water is reflected.

Computations and diagrams used for determining the revised materials balance are presented in Volume II.

COPPER CONCENTRATE THICKENING, FILTERING AND DRYING - AREA 70

The estimate of this facility will differ from that of Newmont's in that an additional thickener is provided to receive the combined copper and moly concentrate. This thickener partially dewaters the concentrate feed to the moly recovery plant. The copper-moly concentrates are sampled before entering the thickener to provide a concentrate sample for a metallurgical balance.

The thickener specified on the Newmont flowsheet for this material is 70 feet in diameter. With the material flowrates Parsons-Jurden is using, this thickener will provide about 11.7 square feet of settling area per ton of solids per day. It is believed that this area is adequate. The solids are thickened to an estimated 60 percent weight and the thickener overflow water is recovered in the mill water reservoir for reuse.

The tailing produced by the moly plant is the finished copper concentrate. The tailing stream is sampled to provide a final copper product sample (or moly plant tailing sample) and is then discharged into a second 70-foot diameter thickener identical to the copper-moly

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The tailing produced by the moly plant is the finished copper concentrate. The tailing stream is sampled to provide a final copper product sample (or moly plant tailing sample) and is then discharged into a second 70-foot diameter thickener identical to the copper-moly rate is then pumped to a 100,000-gallon steady-head tank for distribution to the concentrator circuits. Work-sheets in Volume II show the net process water balance for concentrator and the distribution of process water supplied by the steady-head tank for the copper circuits. From these balances the fresh water make-up for the process is estimated at 2227 gallons per minute for the copper concentrate and 95 gallons per minute for the moly plant, or a total requirement of 2322 gallons per minute. An allowance of about 5 percent additional water should be made for spillage, evaporation, and unaccounted for losses. The total process water requirement will then be approximately 2438 gallons per minute.

In the above estimate of water requirements, the water recovered from the tailing pond is estimated at 30 percent of the water contained in the pulp entering the pond. It is believed that this recovery, which is lower than that indicated on the Newmont flowsheet, will be more applicable , particularly during the development of the new pond. These differences are shown on the flowsheet in Volume II of this report. The tailing facility and mill water reclaim are diagrammed on the work-sheets in Volume II.

The underflows from the two tailing thickeners are discharged into an adjacent sump and are pumped through a 20-inch transite pipeline to the tailing empoundment area.

The tailing pond has two compartments which cover 400 acres of storage area. This area is sized on the basis of 20 acres per 1000 daily tons. Tailing distribution lines are placed along each of the two dams with outlet spigots and portable hydrocyclones for sand separation for dike building. Ten 10-inch hydrocyclones are provided for this service. It is believed that these cyclones may be undercapacity and the use of ten 15-inch cyclones is suggested in Section 7.

The dikes will be built initially to give an average pond depth of 30 feet and will provide enough storage capacity for 44 months of operation.

The estimate is based on dams with 1-1/2:1 slopes and a 25-foot wide top. The top of the dikes has been designed with a 0.25 percent slope so that the distribution pipes will be self draining.

A decant system is provided with two concrete decant towers in each compartment of the dam. Drainage from the towers is gathered into a sump at the southwest corner of the dam. The reclaimed water is pumped to the mill water reservoir.

FLOTATION REAGENTS - AREA 90

The reagents used in the copper concentrator are:

1.	Hydrated Lime	4.4 lbs/ton ore
2.	Aeropromotor No. 3302 (liquid used without dilution)	0.009 lbs/ton ore
3.	Aeropromotor No. 3501 (reactive, caustic white solid water soluble)	0.030 lbs/ton ore
4.	Dow Froth No. 250 (used without dilution)	0.025 lbs/ton ore
5.	Pine Oil (used without dilution)	0.025 lbs/ton ore
6.	Superfloc No. 55 (consumption: 5 p.p.m. solids in tailing)	0.010 lbs/ton ore

Provision for 400 tons of pebble lime has been indicated on the Newmont flowsheet. This amounts approximately to a 10-day supply at an estimated maximum rate of consumption indicated by the test work.

A screw feeder conveys the lime from storage to a 4-foot by 5-foot ball mill in closed circuit with two 8-inch hydrocyclone classifiers where hydration and dilution take place. The overflows from the hydrocyclones discharge a 10 percent lime slurry into a 15-foot 6-inch by 20-foot milk-of-lime storage tank equipped with an agitator. The storage tank has a six-hour capacity at the maximum anticipated rate of consumption.

The milk-of-lime is pumped through a loop in closed circuit with the storage tank. The loop is provided with automatic valves that deliver adjustable amounts of lime from the loop to points of addition in the primary grinding and flotation circuits.

The flowrates of solids, water for hydration and dilution, and milk-of-lime slurry are computed on work-sheets in Volume II.

The facilities indicated on the Newmont flowsheet for storing and mixing other reagents than lime appear to be incomplete. This facility has been expanded to accommodate the additional equipment and space believed to be necessary as shown on Drawing 4832-01-3 in Volume II. The use of Clarkson-type feeders on the Aeropromotor No. 3302 and the frothers has been indicated for estimating because of the very low feed rates.

Aeropromotor No. 3302 is an insoluble oily liquid added full-strength to the circuit. Only small amounts are used and a 30-day supply is provided by storage space for twelve 55-gallon drums. A small, portable transfer pump is provided to replenish a 100-gallon head-tank with one drum at a time as needed. This reagent would be metered to the four primary grinding circuits, with Clarkson-type feeders drawing their supply from the head-tank.

Aeropromotor No. 3501 is a reactive white solid that requires P.V.C.-lined equipment for handling and stainless steel agitators for mixing the solution. A thirty-day supply of this reagent requires a storage area to hold sixty 42-gallon drums having a net weight of 300 pounds each. Two combined mixing and storage tanks are provided to facilitate mixing in one tank while the mixed solution is stored in the other for use on demand. A small transfer pump delivers the solution to a 100-gallon head-tank fitted with a float and limit switch to actuate the pump. A safety overflow line is also installed for return of the solution to storage. The reagent is metered to addition points in the circuit by means of eight flowrator-type feeders.

Equal amounts of pine oil and Dowfroth No. 250 are used in the circuit. A thirty-day supply of these reagents requires space for a total of 66 55-gallon drums (33 drums each). On the assumption that these reagents can be pre-mixed before use, a 200-gallon mixing tank has been provided where one drum of each reagent can be mixed before transfer by pump to a 100-gallon head-tank. Clarkson-type feeders are provided to distribute the mixed frother from the head-tank to eight points in the rougher-scavenger flotation circuits.

A thirty-day supply of Superfloc will require a storage area for 20 42-gallon drums. Superfloc is slow to dissolve and is made up in 1 percent solution. Two 2400-gallon tanks, each having a capacity to hold a 24-hour supply of solution, are provided. While the Superfloc is being dissolved and mixed in one tank, the alternate tank will serve as storage and is arranged to supply the reagent head-tank by means of a small transfer pump. Two flowmeters will meter the Superfloc solution to the pulp entering the tailing thickeners, each at a rate of 0.82 gallons per minute.

SITE PREPARATION - AREA 100

Site preparation costs are based upon Mr. D. Shaw's Plant Layout Drawing 52-4-R-0. Included are plant roads, yards, culverts, fences, drainage, electrical distribution and yard lighting.

WATER SYSTEM - AREA 120

The water system estimated is essentially the same as described in Mr. D. Shaw's reports, except that an additional well was added to the two originally provided.

Fresh water will be obtained from an aquifer 5.75 miles east of the plant site. Three wells, each equipped with turbine pumps, will supply water through an 18-inch pipeline to a fresh water storage tank located on a hill above the plant. Power to the well pumps will be supplied by a 13.8-kilovolt line from the mill substation with 4.16-kilovolt stepdown at the well site.

Fresh water will be used for mill water make-up, pump seals, cooling water, dust control, potable water system and fire protection. Costs for potable water storage tank, potable water treatment, potable water yard piping, fire protection yard piping and hydrants are accounted for in Miscellaneous Piping - Area 130.

A 2-million-gallon mill water reservoir with pumping facilities and mill water steady head-tank are also provided.

MISCELLANEOUS PIPING - AREA 130

Miscellaneous piping includes potable water tank, potable water treatment equipment, potable water yard piping, plant air compressor, air yard piping, fire pump, fire yard piping, hydrants, diesel generator set, sewers, septic tank and drainage field.

ACCESS ROAD - AREA 140

The estimate for the plant access road includes installation of a new unpaved road connecting to the existing road. Drainage and culverts are provided.

SERVICE BUILDINGS - AREA 150

All buildings associated with the mill were estimated on a square foot basis and include interior finish, partitions, piping, plumbing, lighting, heating and air conditioning, where required.

Buildings included are:

- General Office
- Gate House
- Warehouse
- Change House
- Analytic and Metallurgical Lab
- Mill General Shops
- Flammable Storage
- Sample Preparation
- Core Storage
- Switchyard Area

POWER SUPPLY - AREA 170

Estimate included for incoming power supply and substation is based upon information furnished Newmont by The Papago Tribal Utility Authority.

MOLY PLANT - AREA 160

Introduction

Because the material submitted to Parsons-Jurden for review did not include design of a plant for separation of molybdenite from the copper-moly concentrate, preliminary design of this portion of the plant was undertaken by Parsons-Jurden to provide a basis for estimating its cost.

The design of the moly plant was based primarily on the Newmont Exploration Ltd.'s report entitled "Molybdenite Flotation Testing on Vekol Hills Concentrates from Pilot Plant Test," dated November 9, 1970. The flowsheet and the reagent combination are similar to those developed at the San Manuel plant.

The material balance is based in part on the projected metallurgical results indicated in the covering letter of the above Newmont report. Parsons-Jurden's review indicated that the overall recovery obtained in laboratory testing was about 45 percent, rather than the 60 percent used in this study. However, capital and operating costs will be essentially independent of molybdenum recovery and, therefore, this factor has no bearing on the preliminary design or the cost estimates in this report.

The moly plant design is based on a feed rate of 330 dry short tons per day, per the metallurgical balance at the beginning of this section, rather than 400 dry short tons per day indicated on Newmont Flowsheet 52-1-R-4.

As in the copper concentrator, no specific provision is made for overall plant expansion. However, the moly plant is designed for considerable flexibility in the flowsheet and for future addition of flotation cells or other equipment as may be required to accommodate changing technology, varying moly content and varying moly plant feed rates.

Flowsheet

The flowsheet, as shown in Drawing 4832-01-2, consists basically of a rougher flotation step in which the molybdenite is floated and the copper is depressed, followed by a series of six flotation cleaning steps. The molybdenum content is gradually increased from about 0.6 to 0.7 percent Mo in the moly plant feed to about 54 percent Mo (90 percent MoS_2) in the final moly concentrate.

The moly plant feed is pumped at 60 percent solids from the copper-moly concentrate thickener and conditioned in two 6-foot by 6-foot conditioning tanks, in series. Each tank is sized to provide an average retention time of 20 minutes.

The conditioned pulp is combined with recirculated middlings and pumped to the No. 1 Rougher Flotation Cells. Rougher concentrate is cleaned once, then reground in a 4-foot by 6-foot ball mill with a 40-horsepower motor in closed circuit with a 6-inch cyclone.

No. 1 Rougher Tails are further treated in the No. 2 Rougher Cells and the No. 1 Cleaner Tails are treated in the scavenger cells. Tails from the No. 2 Rougher Cells and the scavenger cells comprise the final moly plant tails and are pumped to the copper concentrate thickener. Froth from both of these flotation units is returned to the rougher feed.

The regrind cyclone overflow is subjected to five additional flotation cleaning steps (No. 2 Cleaner to No. 6 Cleaner). In each case, the tailing is returned to the feed of the preceding flotation step.

Nos. 4, 5 and 6 Cleaners are of the cell-to-cell type to permit froth transfer without pumps.

Pertinent design data on the various flotation steps are summarized below:

	No. Cells	Cell Vol. (Cu. Ft.)	% Solids	Flot. Time (Minutes)
No. 1 Rougher	4	40	30.0	6.1
No. 2 Rougher	4	40	27.8	6.7
No. 1 Cleaner	4	40	8.9	6.8
Scavenger	4	40	6.7	7.1
No. 2 Cleaner	4	22.5	6.2	5.3
No. 3 Cleaner	2	22.5	6.0	4.2
No. 4 Cleaner	2	12	5.0	5.6
No. 5 Cleaner	1	12	5.0	5.6
No. 6 Cleaner	1	12	5.0	7.0

The final moly concentrate is discharged to a 9-foot by 9-foot surge tank designed for a 24-hour concentrate storage. From here, it is pumped to a 4-foot diameter, 4-disc filter with an area of 88 square feet with a vacuum pump, filtrate receiver and air blower. The filter cake discharges at 18 percent moisture directly into a hollow-screw-type dryer, which utilizes an electrically-heated heat-transfer medium.

The dried moly product drops directly into a 55-gallon drum holding approximately 800 pounds 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.

Reagents

Reagent amounts and addition points are as listed in the Newmont report on molydbenite flotation.

It is anticipated that 50 percent hydrogen peroxide, sulfuric acid, 50 percent caustic soda and liquid chlorine will be received in tank truck shipments. The last two will be used in the preparation of sodium hypochlorite at the plant. All other reagents are expected to be received at the plant in bags or drums.

A minimum of 30-day storage for each reagent is provided at the moly plant. A flow diagram for the storage, handling and feeding of each reagent is given in the Reagent Flowsheet, Drawing 4832-01-3.

Sampling

Wet samplers are provided at three points:

1. The moly plant feed is sampled at the point of discharge of the copper-moly concentrate into No. 1 Conditioner.

2. The moly plant tailing (final copper concentrate) is sampled at the point of discharge of the combined No. 2 Rougher and Scavenger Tailing into the moly plant tailing sump.

3. The molybdenite concentrate is sampled at the point of discharge of No. 6 Cleaner Concentrate into the surge tank.

In addition, each moly concentrate drum may be sampled manually for lot-blending purposes.

Water System

Fresh water is used for pump gland seal water and for launder spray water on Cleaners Nos. 2 through 6, a total of about 100 gallons per minute.

In addition, about 180 gallons per minute of process water for use in the initial stages of the moly plant are provided from the copper thickener overflow. The excess copper thickener overflow water joins the tailing thickener underflow for disposal.

Instrumentation

A central control room is provided for operation of the moly plant. Adequate instrumentation is provided, including mass flow control of moly plant feed; automatic pH control at two points; automatic froth level control on all flotation units, except Nos. 4, 5 and 6 Cleaners; level alarms in all sumps, surge tanks, reagent storage tanks and head-tanks, as required; interlocks to prevent starting of horizontal pumps without gland seal water; and standard instrumentation for the regrind mill, the filter and dryer.

Plant Description

The general arrangement of the moly plant is shown in Drawing 4832-01-4.

As shown on the key plan, the moly plant is located in an area adjacent to the copper filter plant and near the two 70-foot concentrate filters.

The moly processing operations are contained in a building area 60 feet by 72 feet, serviced by a 5-ton crane. Approximately half of this area is allocated to flotation. On the flotation floor space is reserved for a future duplicate row of Cleaners Nos. 2 to 6 for use either as standby equipment, or for increasing the flotation time in these cleaners, or for handling larger than normal volumes of pulp in these cleaning stages. In addition, space is provided for a possible future row of 60-cubic-foot cells ahead of the existing row of roughers. These cells could be installed, if necessary, to increase flotation volume in each stage by changing the present roughers to No. 1 Cleaners, the present No. 1 Cleaners to No. 2 Cleaners, and so on.

The other half of this area is utilized for the regrind mill, a control room and electric room, product filter and dryer and product storage in drums.

The reagents are received, stored and prepared in a separate adjacent 60-foot by 37-foot "lean-to" area. A portion of this area consists of a covered unloading ramp. Reagent drums are stored in the covered area and reagent bags are stored in the enclosed area. Outside storage tanks are provided for hydrogen peroxide, caustic soda and sodium hypochlorite. Chlorine is used directly from the truck tank. All process pump stations in the moly plant will consist of one operating and one standby pump.

Fresh water for the moly plant is provided from the copper concentrator supply. Plant air and instrument air are also supplied from the copper concentrator. Normal heating and ventilation facilities are included. Sanitary facilities are provided.

Process computations are included in Volume II.

SECTION 5

PARSONS

CAPITAL COST ESTIMATES

MINE

I. General

This is a Preliminary Capital Cost Estimate for the preproduction work necessary to put the mine in readiness to start feeding ore to the mill at a rate of 7 million tons per year beginning on September 1, 1973.

II. Estimate Basis

The estimate is based on "Feasibility Report, Mining Section, Vekol Hills Project" by Mr. W.K. Pincock dated December 13, 1970. Mr. Pincock is a Registered Professional Engineer (Mining) in Arizona with offices in Tucson.

The scope of the Feasibility Report (hereinafter called the Report) was to:

1. Design an open pit to encompass the economic reserves utilizing all available data.

2. Establish the open pit mining reserves.

3. Make a long range mining plan to determine the amount of pre-production stripping and the operating waste-ore ratios required to keep a continuous flow of ore to the concentrator at a rate of 20,000 tons per day.

4. Determine the equipment requirements for the pit operation.

5. Make an estimate of the operating costs for mining at the rate specified.

Parsons-Jurden has checked the above items in detail and has the following comments on them:

1. The open pit design contains an adequate haul road, safe pit slopes and appears to extract the maximum amount of ore.

2. Mineable reserves of 109,083,000 tons averaging 0.543 total copper have been calculated by planimetering and checked by

computer. Additional analyses for oxide copper are now being made to confirm that all of this ore can be concentrated economically.

3. The long range mining plan, including 60 million tons of preproduction stripping, will provide the mill with a fairly constant copper value year by year.

4. The selection of equipment has been made from well known manufacturers and a reasonable availability factor has been applied in all cases.

III. Estimate Breakdown

The Pincock estimate is broken into the following areas:

- Mine Equipment
- Mine Shop
- Mine Electrical System
- Spare Parts
- Preproduction Stripping

The explosives storage was considered as part of the contingency. The warehouse, office and changeroom are included in the concentrator estimate.

The preproduction stripping is assumed to be done by the mining company with its personnel and equipment.

IV. Major Equipment

All mining equipment cost estimates were obtained in writing from vendors. These have been verified by the in-house records of Parsons-Jurden.

V. Spare Parts

The cost of spare parts is an allowance taking into consideration that the project is close to the warehousing facilities of equipment suppliers in Phoenix and that some suppliers will approve an arrangement of consignment parts.

VI. Labor Rates

Labor rates for the construction of the shop and pit electrical system are based on the schedule for construction workers shown under Labor Rates for the concentrator construction later in this Section.

Preproduction stripping labor is in accordance with the present mine operating labor contracts now in effect in Arizona and due to be re-negotiated this coming July. Escalation was not considered in the Pincock report but has been included in Parsons-Jurden's review.

VII. Indirect Costs

Indirect costs for construction are as shown under Indirect Costs for the concentrator later in this Section, and for preproduction stripping are as listed in the operating cost schedule.

VIII. Engineering Costs

Engineering costs for construction are as shown under Engineering Costs for the concentrator later in this Section, and for preproduction stripping are as listed in the operating cost schedule.

IX. Sales Tax

Arizona State Sales tax of 3 percent does not apply to the initial mining equipment purchased, but should be applied to the spare parts and supplies consumed during preproduction stripping.

X. Project Duration

For the purpose of this estimate, the preproduction stripping is scheduled for a duration of two years and will be completed at the time the concentrator is ready to start processing ore.

XI. Exclusions

The following items are not included in the Pincock estimate:

- Escalation
- Contingency
- Land acquisition or land easements

- Owner's expenses connected with this project
- Working capital, financing cost and interest during construction
- Premium time costs
- Federal, State and Local Taxes
- Local, State or Federal permits and fees
- Performance bond
- Legal expense
- Fee for engineering and construction.

For purposes of establishing cash flow needs Parsons-Jurden included escalation and contingency in the review and comparison of the capital cost estimates.

Equipment List - Mine

I. Major Equipment

Shovels

4	P&H 2100B - 15 Yd.

Drills

4	BE 60-R
1	Secondary Mobile Drill

Trucks

22	120-Ton Unit Rig or
18	150-Ton Wabco

Dozers, Graders, Loader

3	D9G - Two with rippers, One with Winch
3	824 Rubber-tired Dozers
3	Cat 16E Graders
1	Loader (15 Yd. LeTourneau)

Cranes

1	65-Ton
1	18-Ton

II. Auxiliary Equipment

1	Powder Truck
2	8,000 Gallon Water Trucks
1	Fork Lift - For Tire Handling
1	Lube Service Truck
1	Shovel Repair Truck
1	Supply Truck
1	Welding Truck
	Radio Equipment
12	Pickups - 1/2-Ton
4	Pickups - 3/4-Ton
1	Sedan (Manager)
1	Station Wagon (Superintendent)
2	Busses for transporting crew in mine area only
1	Tractor Lowboy
1	Compressor
1	Melroe Bobcat
6	Light Plants

Pincock Capital Cost Estimate - Mine

Mine Equipment including $41,083 Contingency	$12,181,600
Mine Shop	815,000
Mine Electrical System	368,400
Spare Parts, Allowance	750,000
Preproduction Stripping, 60,000,000 tons @ $0.135	8,100,000
	$22,215,000

CONCENTRATOR

I. General

This is a Preliminary Capital Cost Estimate for the construction of a copper concentrator capable of handling 20,000 tons of ore per day. The estimate includes all direct and indirect costs for equipment, materials, field labor, subcontracts, indirect field costs, engineering costs, sales tax, escalation and contingencies.

II. Estimate Basis

The estimate is based on Newmont's report and drawings prepared by Mr. D. Shaw dated May 1, 1970, and Revision 1 thereof dated November 1, 1970. The major differences between Mr. Shaw's latest revision and this Parsons-Jurden estimate are listed below.

1. Plot plan used is the one originally estimated in May 1, 1970 report.
2. Crushing plant size is increased to accommodate a 60-inch crusher.
3. Fine crushing building is increased to include a repair bay.
4. Reagent facilities for concentrator are increased.
5. An additional water well was added.
6. Molybdenum plant detail estimate is added.
7. Cost of primary power supply is included in the estimate details.
8. Power to the well pumps is brought in from the main substation.
9. The cost of site preparation is based on the assumption that site and structural excavation will consist of 10 percent rock, and that fill will be supplied to designated areas at the required time from the mine at no cost.
10. Construction schedule is assumed to start January 1, 1972.

III. Estimate Breakdown

The estimate is broken down into the following areas:

Area 10 - Coarse Ore Crushing
Area 20 - Intermediate Storage
Area 30 - Fine Crushing
Area 40 - F. O. Storage
Area 50 - Grinding
Area 60 - Flotation
Area 70 - Filtering
Area 80 - Thickening
Area 90 - Reagent and Lime
Area 100 - Site Preparation
Area 110 - Tailings Disposal
Area 120 - Water System
Area 130 - Misc. Piping
Area 140 - Access Road
Area 150 - Service Buildings
Area 160 - Molybdenum Plant
Area 170 - Power Supply

In addition, there are details for indirect costs, engineering, sales tax, escalation, insurance and contingency.

IV. Major Equipment

Most of the major process equipment costs are based on budget quotations from manufacturers. Where no quotations could be obtained, in-house data were used.

V. Other Materials

Other material costs, such as concrete, piping, steel, and electrical, were estimated, based on the information available, such as plot plans, preliminary general arrangement drawings, one-line diagrams, or are based on an experience factor. In-house data and quotations from local suppliers were used for pricing.

VI. Incoming Power Supply

The capital cost for the incoming power supply is based on a quotation received from Papago Tribal Utility Authority.

VII. Labor Rates

Direct labor costs are based on the base rates listed below. The regular work week considered in this estimate is five eight-hour days. The below listed labor rates are valid until mid-year 1972. Labor rates for boilermakers, electricians and painters were estimated since agreements expire during 1971.

	Hourly Wages		Daily
	Base Rate	Fringes	Subsistence
Boilermaker (estimated)	$8.20	$1.60	$12.00
Cement Mason	7.58	1.03	10.00
Carpenter	7.20	1.03	15.00
Electrician (estimated)	9.00	-	8.00
Laborer	5.20	0.75	10.00
Oper. Engineer	7.40	1.13	10.00
Millwright	7.50	1.03	15.00
Iron Worker	7.80	0.90	11.00
Painter (estimated)	6.80	0.30	8.00
Pipe Fitter	9.50	1.93	11.00
Teamster	6.06	0.77	10.00

No cost of premium time is included.

Labor fringes and daily subsistence are estimated under indirect costs item craft benefits.

VIII. Indirect Costs

These costs are broken down as follows:

- Field Supervision
- Field Travel Expenses
- Office Supplies
- Communications
- Temporary Facilities
- Utilities
- Watchmen
- Construction Equipment
- Gas, Oil, Repairs
- Construction Supplies
- Start-up (Mechanical run-in only)
- Labor Taxes
- Craft Benefits.

IX. Engineering Costs

This item represents all engineering costs connected with the project, such as management and administration; process and project engineering; construction support; design; drafting; accounting; estimating; scheduling; cost engineering; procurement; expediting; inspection; stenographic; clerical; and overhead and out-of-pocket expenses such as printing, reproduction, computer charges, communications and travel.

Also included are costs of outside consultants for soils and water investigation, as well as the cost of pre-design studies and estimates.

X. Sales Tax

Arizona Sales Tax of 3 percent was applied to all items purchased by engineer contractor which will not become part of the plant. All components of the plant will be purchased under Newmont's name and will be exempt from Arizona Sales Tax.

XI. Project Duration

For the purpose of this estimate it is assumed that engineering will start on April 1, 1971, with a construction schedule of 20 months, beginning January 1, 1972 and ending October 31, 1973.

XII. Escalation

Escalation has been applied for the duration of the project. Labor was escalated at 11 percent annually starting mid-1972, based on present-day labor agreements. Equipment, materials and engineering were escalated 6 percent per annum.

XIII. Contingency

A contingency of 10 percent has been applied to the project cost. Contingency is being defined as a specific provision for unforeseeable elements of cost within the defined project scope.

XIV. Estimate Accuracy

Due to the preliminary nature of this estimate, it is considered that the accuracy is such, that the final cost may be 10 percent under or 15 percent above this preliminary estimate.

XV. Exclusions

The following items were not included in the estimate:

- Land acquisition or land easements
- Owner's expenses connected with this project
- Raw materials, tools and supplies for initial operation (except ball charges)
- Working capital, financing cost and interest during construction
- Premium time costs
- Spare parts
- Federal, State and Local Taxes other than State Sales Tax
- Local, State or Federal permits and fees
- Performance bond and permits
- Legal expense
- Fees for engineering and construction.

CAPITAL COST SUMMARY

DIRECT COSTS BY AREA

CODE	ITEM	TOTAL	COARSE CR. 10	STORAGE 20	FINE CR. 30	F.O. STOR. 40	GRINDING 50	FLOT. 60	FILTER 70	THICKEN. 80
60	EXCAVATION	2 145 900	63 600	110 500	12 200	6 700	12 700	7300	9 000	205 500
61	CONCRETE	2 456 600	431 200	601 100	202 000	212 100	289 900	138 700	79 200	355 900
62	BUILDINGS	3 254 500	156 600	109 800	702 900	235 900	429 700	420 300	75 600	21 300
63	EQUIPMENT	9 041 100	950 200	390 500	1 586 600	224 400	3 259 400	616 900	377 700	449 400
64	PIPING	3 006 200	11 300	1 900	66 000	1 900	376 000	325 000	101 000	134 500
65	ELECTRICAL	3 333 100	137 100	12 400	261 100	32 200	502 700	287 900	78 500	58 600
66	PAINTING	146 000	4 700	3 500	22 500	9 800	15 000	15 300	3 000	6 100
67	INSTRUMENTS	347 100	7 200	2 600	37 300	9 800	188 700	4 800	11 600	24 000
	DIRECT COST TOTAL	23 732 800	1 756 900	1 232 300	2 890 600	732 800	5 074 100	1 816 200	735 600	1 255 300

CODE	ITEM	REAGENT 90	SITE PREP. 100	TAILINGS 110	WATER SYS. 120	MISC. PIPE 130	ACCESS RD. 140	BLDGS 150	MOLY PLT. 160	INC. PWR 170
60	EXCAVATION	1 300	897 300	582 900	117 500		115 300		4 100	
61	CONCRETE	22 800		26 900	41 100	8 300			47 400	
62	BUILDINGS	39 600			10 000			922 200	135 600	
63	EQUIPMENT	141 000		31 800	239 400	78 800		403 800	291 200	
64	PIPING	55 700		712 600	927 200	233 100			70 000	
65	ELECTRICAL	61 100	530 300	93 500	327 200	2 300			90 500	860 000
66	PAINTING	13 400		8 300	39 900				4 500	
67	INSTRUMENTS	5 600			9 500				46 000	
	DIRECT COST TOTAL	340 500	1 427 600	1 456 000	1 711 800	312 500	115 300	1 326 000	689 300	860 000

FIELD SUPERVISION	862 300
FIELD TRAVEL EXP.	131 700
OFFICE SUPPLIES	74 000
COMMUNICATIONS	44 500
TEMP. FACILITIES	434 100
UTILITIES	124 000
WATCHMEN	78 300
CONST. EQUIPMENT	1 300 000
GAS, OIL, REPAIRS	450 000
CONST. SUPPLIES	648 000
START UP	164 000
LABOR TAXES	1 385 300
CRAFT BENEFITS	2 311 800
INDIRECT COST TOTAL	8 008 000
ENGINEERING	3 900 000
SALES TAX	56 000
INSURANCE	180 000
ESCALATION	2 973 200
CONTINGENCY	3 885 000
TOTAL COST	42 735 000

PARSONS- JURDEN CORP.	
P-J JOB No. 4832	
PREPARED BY	J E 3-10-[illegible]
APPROVED BY	

DIRECT COST

LABOR-MATERIAL ANALYSIS

COST CODE AND ITEMS

AREA	AREA NAME	60 EXCAVATION MATERIAL	60 LABOR	60 TOTAL	61 CONCRETE M	61 L	61 T	62 BUILDINGS M	62 L	62 T	63 EQUIPMENT M	63 L	63 T
10	COARSE CRUSHING	500	63,100	63,600	186,700	244,500	431,200	109,600	42,000	151,600	873,000	77,200	950,200
20	STORAGE	1400	109,100	110,500	262,400	338,700	601,100	81,000	28,800	109,800	309,200	81,300	390,500
30	FINE CRUSHING	1100	11,100	12,200	71,700	130,300	202,000	506,500	196,400	702,900	1,389,500	197,100	1,586,600
40	FINE ORE STORAGE	400	6300	6700	74,600	137,500	212,100	152,300	83,600	235,900	176,900	47,500	224,400
50	GRINDING	1000	11,700	12,700	118,100	171,800	289,900	298,200	131,500	429,700	3,072,000	187,400	3,259,400
60	FLOTATION	1500	5800	7300	45,700	93,000	138,700	304,200	116,100	420,300	539,800	77,100	616,900
70	FILTER	1400	7600	9000	27,000	52,200	79,200	54,900	20,700	75,600	332,700	45,000	377,700
80	THICKENING	28,200	177,300	205,500	141,800	214,100	355,900	17,300	4,000	21,300	414,800	34,600	449,400
90	REAGENTS	300	1000	1300	7700	15,100	22,800	27,600	12,000	39,600	115,400	25,600	141,000
100	PLANT SITE PREP.	188,000	709,300	897,300									
110	TAILINGS DISPOSAL	12,400	570,500	582,900	12,500	14,400	26,900				28,700	3100	31,800
120	WATER SYSTEM	28,700	88,800	117,500	10,700	30,400	41,100	10,000		10,000	237,000	2400	239,400
130	MISC. PIPING				1700	6600	8300				73,000	5800	78,800
140	ACCESS ROAD	32,800	82,500	115,300									
150	SERVICE BUILDINGS							460,700	461,500	922,200	385,000	18,800	403,800
160	MOLY PLANT	300	3800	4100	19,000	28,400	47,400	77,200	58,400	135,600	260,200	31,000	291,200
170	INCOMING POWER												
	TOTAL DIRECT $	298,000	1,847,900	2,145,900	979,600	1,477,000	2,456,600	2,099,500	1,155,000	3,254,500	8,207,200	833,900	9,041,100
	% TOTAL DIRECT COST			9.0			10.4			13.7			38.1
	LABOR TO MTL. L/M		6.2			1.5			.55			.10	
	RATE PER HOUR		6.70			6.89			7.99			7.96	
	TOTAL MAN HOUR		276000			214,000			144000			105000	

AREA	AREA NAME	64 PIPING M	64 L	64 T	65 ELECTRICAL M	65 S/C	65 T	66 PAINTING M	66 L	66 T	67 INSTRUMENTS M	67 S/C	67 T
10	COARSE CRUSHING	5000	6300	11,300	78,200	58,900	137,100	1200	3500	4700	4800	2400	7200
20	STORAGE	1000	900	1900	4300	8100	12,400	900	2600	3500	1800	800	2600
30	FINE CRUSHING	40,000	26,000	66,000	148,400	112,700	261,100	5700	16,800	22,500	24,300	13,000	37,300
40	FINE ORE STORAGE	1000	900	1900	11,500	20,700	32,200	2500	7300	9800	6300	3500	9800
50	GRINDING	190,000	186,000	376,000	303,500	199,200	502,700	3800	11,200	15,000	121,700	67,000	188,700
60	FLOTATION	145,000	180,000	325,000	145,800	142,100	287,900	3900	11,400	15,300	3100	1700	4800
70	FILTER	45,000	56,000	101,000	35,800	42,700	78,500	800	2200	3000	7600	4000	11,600
80	THICKENING	60,000	74,500	134,500	33,100	25,500	58,600	1400	4700	6100	15,500	8500	24,000
90	REAGENTS	25,000	30,700	55,700	28,100	33,000	61,100	3400	10,000	13,400	3800	1800	5600
100	PLANT SITE PREP.				317,300	213,000	530,300						
110	TAILINGS DISPOSAL	457,200	255,400	712,600	56,100	37,400	93,500	1900	6400	8300			
120	WATER SYSTEM	497,000	430,200	927,200	153,000	174,200	327,200	10100	29800	39900	6500	3000	9500
130	MISC. PIPING	105,800	117,300	22,100	1000	1300	2300						
140	ACCESS ROAD												
150	SERVICE BUILDINGS												
160	MOLY PLANT	30,000	40,000	70,000	45,000	45,500	90,500	1200	3300	4500	.30,000	16,000	46,000
170	INCOMING POWER				860,000		860,000						
	TOTAL DIRECT $	1,602,000	1,404,200	3,006,200	2,221,100	1,114,300	3,335,400	36,800	109,200	146,000	225,400	121,700	347,100
	% TOTAL DIRECT COST			12.7			14.1			.6			1.4
	LABOR TO MTL. L/M		.88			.50			2.97			.54	
	RATE PER HOUR		9.31			13.00			6.90			13.00	
	TOTAL MAN HOUR		151000			87000			16000			9400	

PARSONS-JURDEN CORP.	
P-J JOB No. 4832	
PREP.	J.E 3-10-71
APP'D.	

MAJOR QUANTITIES

QUANTITIES BY AREA

ITEM	COARSE CR. 10	STORAGE 20	FINE CR. 30	F.O. STOR. 40	GRINDING 50	FLOT. 60	FILTER 70	THICKEN. 80	REAGENT 90	SITE PREP. 100	TAILINGS 110	WTR. SYS. 120	MISC. PIPE 130	ACCESS RD. 140	BLDGS. 150	MOLY PLANT 160	INC. PWR. 170	TOTAL
EXCAVATION - CY.	16,200	31,000	2,330	2,000	3,100	940	1,900	36,800	180	242,000	16,200	25,500	350	8,900		900		388,300
BACKFILL - CY	13,000	19,600	2,300	1,600	1,900	1,000	900	31,400	150	725,000	8,200	3,400	250	42,000		700		851,400
TAILINGS DAM - CY						-					1,630,000							1,630,000
CONCRETE - CY	3,200	4,510	1,424	1,492	2,763	900	573	1,690	162		300	185	34			390		17,623
REINFORCING - #	505,000	1,276,000	221,000	254,000	345,000	114,000	76,000	238,000	23,000		44,000	22,000	4,000			62,000		3,184,000
STRUCT. STEEL - #	175,000	-	710,000		836,000	820,000	114,000	49,400	66,000							220,000		2,990,400
CONVEYOR STEEL - #	132,000	210,000	430,000	23,000	60,000		45,000											900,000
BIN STEEL - #			265,500	640,000														905,000
HORSEPOWER CONN.	1,133	275	2,490	135	16,353	2,211	412	557	212			2,100	125			415		26,418

OPERATING COSTS

MINE

General

The mine operating costs in this section are those developed in the Pincock report. They are quite variable because waste removal starts at 28 million tons during the first production year and gradually reduces to 606,000 tons in Year 15. The average for the production period is 1.4 tons of waste to 1 ton of ore.

Provisions have been made to stockpile separately the 12.6 million tons of oxide copper rock which averages 0.53 percent total copper. No money has been put into the capital cost estimate for preparing the base of the oxide dump, nor has any income been forecasted for the recoverable copper.

The productivity of the equipment shown in the report has been attained by other similar operations and has been checked by in-house records of Parsons-Jurden and by manufacturers' performance curves.

It is noted that a 15-shift mining week is planned in the report. Because of the high capital cost per employee, it is suggested that a 20-shift mining week be investigated.

Provision has been made for the installation of a primary crusher in the pit on the 1250 bench during Year 5 and for conveying the ore and waste out of the pit to the coarse ore stockpile. Credit for this has been taken in the operating cost estimate but nothing has been done to offset this in the capital cost estimate.

The operating costs developed in the report are reasonable and comparable to others actually experienced by similar operations in Arizona. From in-house records Parsons-Jurden has been able to spot-check the Pincock operating cost estimates. These are as follows in cents per ton of material moved:

Item	Pincock	P-J Range
Drilling	3.50	2.4 - 3.9
Blasting	2.85	2.2 - 2.5
Loading	2.10	1.9 - 3.5
Hauling	6.50	9.5 - 11.3
Roads and Dumps	1.55	3.3 - 3.4
Pit Department	2.50	2.0 - 2.3

It should be noted that the Vekol Project has shorter hauls than the other two properties in the comparison which results in lower hauling and road costs. These are average costs and not year by year costs.

Parsons-Jurden has checked all of these calculations and has found them to be accurate with the exceptions noted. They range from $0.72 to $0.29 and average $0.46 per ton of ore delivered to the crusher on an annual basis.

The method of calculating the shovel loading cost in the report is unusual. Labor cost per hour is corrected by $8 \div 7$ and added to the other hourly costs. This total hourly cost is then multiplied by 6.5 to get the cost per shift.

Two water trucks are shown in the equipment list but they do not appear in the operating cost estimate.

Neither of the last two items will have any appreciable affect on the overall feasibility of the project.

In calculating the tonnage of material to be moved, a dilution factor of 5 percent at 0.2 percent copper has been used and appears to be satisfactory.

Exclusions

The Pincock operating cost estimate excludes:

- Royalties
- Insurance
- Depreciation and Resale Value
- Amortization
- Depletion
- Interest
- Taxes
- Duties
- Fees
- Licenses
- Overall Corporate Management
- General Management
- Escalation
- Contingency.

Manning Table

The following manning table pertains to the preproduction period and the first three years of production. It will be decreased after that to accommodate the reduction in waste to be moved.

Mine Operations

STAFF (outlined in report)	
Supervision (excluding mechanical foreman)	14
Engineering & Geology	9
Timekeepers & Clerks	3
Safety	3
Janitors - Change House	3
TOTAL	32
HOURLY PERSONNEL	
Drilling	17
Blasting	6
Loading	18
Haulage	
a) 17 Drivers x 3	51
b) Water Trucks 2	2
c) Dumpmen 3	3
Roads, Dumps, Cleanup	14
Labor Pool	6
TOTAL	117
TOTAL MINE OPERATIONS	149

Mine Maintenance

STAFF	
General Maintenance Foreman	1
Shovel & Drill Foreman	1
Welding Foreman	1
Shop Foreman	2
Clerk	1
TOTAL	6
HOURLY PERSONNEL	
Shovel & Drill Sections	10
Shop	22
Lubrication	7
Tires	7
Welding	14
Electrical	5
General & Utility	4
TOTAL	69
TOTAL MAINTENANCE OPERATION	75
TOTAL MINE & MAINTENANCE OPERATION	224

Labor rates are in accordance with the present mine operating labor contracts now in effect in Arizona and due to be renegotiated this coming July.

Miscellaneous Costs

Miscellaneous costs used in the Pincock report include the following and these compare favorably with Parsons-Jurden in-house costs.

Item	Cost (In Dollars)
12 1/4-inch drill bit	0.90 /foot
ANFO	0.0425/pound
Slurry	0.085 /pound
Fuel oil for explosives	.02 /pound
Fuse	.025 /foot
Boosters	1.00 each
Diesel fuel	.12 /gallon

Equipment Replacement

In the report it has been assumed that the shovels, drills and trucks will last the life of the mine. An allowance of $150,000 per year has been made for the replacement of light vehicles and dozers but is not included in the unit costs.

Operating Cost Summary

It should be noted that the preproduction unit costs are lower (13.5) because of less drilling and blasting due to the alluvium plus more favorable haul distances.

The Pincock production costs have been broken down year by year per ton of ore delivered to the crusher as follows:

Pincock Production Costs

Year	$/Ton Ore
1	.74
2	.74
3	.78
4	.60
5	.63
6	.56
7	.36
8	.31
9	.31
10	.32
11	.32
12	.32
13	.33
14	.33
15	.29
16	.30
Average	.46

CONCENTRATOR

The total direct operating cost for the complete Vekol Hills concentrator facility is estimated at $1.02 per ton of ore processed. Of this amount, the operating cost for the copper concentrator is $0.95, as compared with $0.883, and the cost for the moly recovery facility is $0.07, as compared with $0.05 on the Newmont estimate.

These costs do not include administrative costs, fringe benefits and other general expense items. These exclusions would cover supervision above the level of mill superintendent as well as labor, supplies, insurance and other expense items for the general office, warehouse, plant security, safety and other facilities and activities supporting the concentrator operation.

For the determination of operating and maintenance labor costs, the personnel required to man adequately the total concentrator have been listed together with the hourly base rates pertaining to the year 1970 as listed by Newmont in their operating cost estimate of March 27, 1970. These hourly rates were escalated by 10 percent per year for the estimated 1972 rates. Differentials of 12 and 16 cents per hour were used for the shift labor. An 8 percent increase was applied to the itemized labor cost to account for the cost of the replacement labor required to cover job vacancies resulting from vacations and sickness. The derivation of this increase is shown in the operating cost worksheets in Volume II.

Newmont's salary list, given in their estimate of March 27, 1970, was directly utilized in Parsons-Jurden's estimate without adjustment for the above 8 percent increase.

Operating costs for bowls, mantles, and mill liners have been derived from recent quotations, escalated at the rate of 4 percent per year. Wear rates for crusher parts are estimated from averages at similar operations. The estimated wear rates for the liners are based upon the kilowatt-hour energy input to the mills.

The largest single item in the operating cost estimate for the copper concentrator is the cost of grinding balls, principally in primary grinding.

P-J's estimated cost for supplies, materials and replacement are based upon an annual cost of 3 percent of the direct capital cost of the plant. For this purpose, the direct capital costs of the copper concentrator and moly plant have been assumed to be $21 million and $700,000 respectively.

The major direct operating cost items for the moly plant are labor and reagents. Of the total direct operating cost of $0.07 for the moly plant, each of these accounts for $0.03 per ton of ore milled.

Reagent and fuel oil costs for the copper concentrator have been taken from quotations by suppliers for material delivered to the Tucson area. Moly plant reagent costs have been based on the delivered costs at the San Manuel plant. In the case of chemicals delivered to San Manuel by rail tank car, the estimated price was increased by one cent per pound to allow for the higher cost of delivery by tank truck. All the above costs have been escalated at the rate of 4 percent per year through 1972.

The unit cost of electrical energy is calculated to be 8.71 mills per kilowatt-hour from the information supplied by Newmont on February 23, 1971. The work sheets for this calculation are included in Volume II. The electrical energy required per ton of ore milled is estimated at 18.68 kilowatt-hours for the copper concentrator and 0.34 kilowatt-hours for the moly plant.

P-J's operating cost estimates are significantly higher than those of Newmont for grinding media, mill liners and hourly labor. On the other hand, it is believed that Newmont's estimate may be high with regard to the overall cost of maintenance supplies, materials and replacements. The following is submitted on these four items.

1. It is estimated that the rate of consumption of grinding media is 0.2 pounds per kilowatt-hour of energy to grind; at 11.2 kilowatt-hours of energy required per ton of ore, the consumption rate would be 2.24 pounds of media per ton. This is 12 percent higher than the Newmont estimate. In addition, P-J has a quotation of 12.78 cents per pound (1970) on 2-1/2-inch "Moly-Copper" balls delivered in the vicinity of Tucson which is about 20 percent higher than the price used by Newmont. These result in about 35 percent higher cost for grinding media than shown on the Newmont estimate.

2. For the large primary mills, it has been assumed that a liner composition of chrome-moly will be used and it has been estimated that there will be a wear factor of 70 percent with no credit allowance for scrap. The cost of the liners on this basis is 32.6 cents per pound (Allis Chalmers quote), divided by 70 percent on 46.6 cents per pound, as compared with the cost of 35 cents used by Newmont.

3. In addition to the 8 percent increase in the hourly labor costs for vacation and sickness as explained earlier, P-J has added to operating labor shown in the Newmont estimates two helpers to cover

the concentrator load-out area and to help with reagent handling. Also added have been two men to help cover the tailing disposal area for a 3-shift per day, 7-day per week operation. To Newmont's maintenance labor estimate, an electrician to the evening and graveyard shifts and a carpenter and rubber repair man to the day shift have been added. These additions raise the total estimated labor costs from 9.6 cents to 11.3 cents per ton of ore milled.

4. It is believed that Newmont's estimate may be overly pessimistic regarding the cost of maintenance supplies, materials and replacements. In one example, an allowance of 5.5 cents per ton has been made for the cost of pump and cyclone maintenance materials which amounts to $35,692 per month (1972) in this area alone. This seems high.

For the overall concentrator, the sum of the maintenance material costs in the Newmont estimate amounts to about 15 cents as compared with P-J's estimated cost of about 9 cents per ton for these items.

The following two pages summarize the direct operating cost estimates for the copper and moly facilities. The computations for these estimates are shown in Volume II.

Copper Concentrator Direct Operating Cost Summary (1972)

(Parsons-Jurden's Estimate)

(In Dollars)

Item	Cost/DST Ore Milled	Cost/Mo.
Operating Labor (41 Men)	0.0518	31,064
Maintenance Labor (22 Men)	0.0308	18,502
Supervision (12 Men)	0.0230	13,817
Assaying (6 Men)	0.0069	4,162
Wear Iron:		
Crushers	0.0517	31,020
Ball Mill Liners	0.1163	69,780
Grinding Balls	0.3224	193,440
Flotation Reagents	0.0920	55,200
Maintenance Supplies, Materials and Replacements	0.0875	52,500
Electrical Energy:		
18.68 kW-Hr/DST at $0.00871/kW-Hr	0.1627	97,620
Fuel Oil	0.0048	2,856
	- 0.9499 -	
TOTAL	0.950	569,961

(Newmont's Estimate)

TOTAL	0.8834	530,057

Moly Plant Direct Operating Cost Summary (1972)

(Parsons-Jurden's Estimate)

(In Dollars)

Item	Per Month	Per DST Moly Plant Feed	Per DST Milled
Operating Labor (14 Men)	10,702	1.081	0.0178
Maintenance Labor (3 Men)	2,357	0.238	0.0039
Assaying (2 Men)	1,625	0.164	0.0027
Supervision @ 25% of Above	3,671	0.371	0.0061
TOTAL LABOR AND SUPERVISION	18,355	1.854	0.0305
Grinding Balls and Liners	831	0.084	0.0014
Flotation Reagents	18,263	1.845	0.0304
Maintenance Supplies	1,750	0.177	0.0029
Electrical Energy	1,793	0.181	0.0030
TOTAL	40,992	4.141	0.0682

(Newmont's Estimate)

TOTAL	30,000	-	0.0500

SECTION 7

SUGGESTED ALTERNATES

MINE

Capital Costs

The scope of the Pincock report was outlined earlier in this report. Parsons-Jurden has checked the Pincock report and found it accurate as to equipment selection, pricing and estimating. However, in order for Newmont to calculate cash needs and financial feasibility it has been necessary to review the concentrator estimate and to add escalation and contingency to the mine estimate. Some of the factors which could affect the total mine capital cost are discussed below.

The mine equipment cost estimate should be reliable if the equipment is ordered soon. It would be expected that some of the present budget-type quotations would be higher than competitive final bid prices.

By going to a 20-shift mining week (from 15), a decided saving can be made in equipment purchases. Assuming 50 weeks per year and 20 shifts per week, each shovel has a theoretical productivity of 15.5 million tons per year. Three shovels (rather than four) at 75 percent availability will therefore give the required 35 million tons per year for the first few years. One drill and four 150-ton trucks can also be saved by the longer work week. The following table shows the number of shovels needed for the proposed mining schedule:

Production Year	20 Shift Week	15 Shift Week
1	2+	4
2	2+	4
3	2+	4
4		2
5		2
6		2
7-15		1

As only one shovel and one drill will be needed for production after Year 6, it would seem prudent to start with the 20-shift week and reduce the initial number of shovels and drills. Then, as waste removal declines, the mine can go to a 15-shift week if it is so desired. The initial savings in capital cost would be $2,390,546 on the 20-shift week.

Parsons-Jurden estimates for the mine shop and mine electrical system are higher than those accompanying the Pincock report. Details are shown in Volume II. The mine shop went from $815,000 to $911,000 and the mine electrical from $368,402 to $432,700.

The operating costs have no contingency or escalation because it is assumed that the price of copper will fluctuate about the same as labor and materials. However, as the preproduction work is scheduled for a two-year period, and is based on operating costs, Parsons-Jurden has applied an escalation and contingency factor to this work. It is assumed that of the total projected cost, 50 percent is labor and 50 percent is materials and supplies. A 6 percent escalation has been applied to materials and supplies and 12 percent to labor, both yearly.

No escalation has been applied to the mine shop and electrical system as they will have to be completed before preproduction work starts.

The preproduction cost has also been increased to include the yearly equipment replacement cost and to reflect the 3 percent sales tax on materials and supplies.

In summary, the Pincock Basic Estimate is $22,215,000. To the Pincock Basic Estimate, Parsons-Jurden suggests adding the following:

Labor Escalation	$1,620,000
Materials & Supplies Escalation	648,000
Materials & Supplies Taxes	213,840
Equipment Replacement	700,000
Additional Cost-Mine Shop	96,000
Additional Cost-Mine Electrical	64,300
10% Contingency*	1,100,830
TOTAL	$4,442,970

* 10% contingency on preproduction stripping, mine shop and mine electrical basic estimates.

Operating Costs

The operating cost estimate should also be raised, as noted above, by adding the yearly equipment replacement cost and the 3 percent tax on materials and supplies.

Mr. Pincock suggests in his report that a primary crusher be installed on Bench 1250 during Year 5 and all material be crushed and conveyed out of the pit. He shows an operating cost saving of $3,270,000.

The estimated direct cost of the 60-inch primary crusher installation is $1,850,000. Add to this 1600 feet of inclined shaft at $140 per foot, and 3500 feet of conveyor at $250 per foot, and the total estimated capital cost is $2,899,000.

In the concentrator estimate, a cost of 3.2 cents per ton is developed for primary crushing. There are 35,804,000 tons of waste to be handled from Year 6 to the end of mining. At 3.2 cents per ton, this amounts to $1,146,000. In addition, the cost of moving the waste from the end of the inclined conveyor to the dump areas must be considered.

If some way can be devised to move the original primary to Bench 1250 without shutting down the mill for lack of ore, it is suggested this alternate be studied in detail.

In any case, as the estimate stands now, either $3,270,000 should be added to the operating costs for Years 6 through 16, or additional capital should be added in Year 5 plus crushing and handling costs of the waste for the later years.

Another scheme has been suggested whereby the crusher would be located in a shaft near the center of the orebody and periodically lowered as the pit deepens. Because of the variety and volume of materials to be handled in the early years of the mine, this was not considered feasible. Here again, if the crusher can be moved with no stoppage of the concentrator, this alternate should be studied in detail.

Mr. Pincock made an allowance of $150,000 per year for replacement of light vehicles and dozers. Parsons-Jurden has drawn up a schedule which results in an average expenditure of $350,000 per year for mine equipment replacement.

PROCESS FACILITIES

Capital Costs

Not included in the detailed capital cost estimate of the mill in this report are the following alternatives that Parsons-Jurden suggests should be evaluated more fully before the inception of final engineering and design. The plus or minus effect on the overall capital cost estimate presented is indicated by an order-of-magnitude value. Drawing 4832-01-1 in Volume II depicts the flowsheet recommended.

Alternative	Overall Effect on Capital Cost (in Dollars)	
Coarse Ore Crushing - Area 10		
. Substitute a hydraulic rock grapple for rock hook provided. Crusher down-time will be reduced.	+40,000	
. Reduce width of coarse ore conveyor to storage pile from 54" to 48" by utilizing 35° idlers.		-10,000
Intermediate Storage - Area 20		
. Increase live ore storage from 50,000 tons to 100,000 tons in order to eliminate possible use of surface equipment to move ore on pile into position for reclaiming.	+75,000	
. Substitute apron feeders for hydra-stroke feeders to reduce possible operating problems.	+45,000	
. Reduce width of two coarse ore conveyors to fine crushing plant from 42" to 36" by utilizihg 35° idlers.		-20,000

Alternative	Overall Effect on Capital Cost (in Dollars)

Fine Crushing - Area 30

- Revise fine crushing plant as follows:

  a) Provide two 6'x12' double-deck screens ahead of secondary crushers.

  b) Replace all 8'x20' screens with 6'x16' screens.

  c) Replace vibrating feeders and belt conveyors feeding tertiary crushers with belt feeders.

  d) Arrange equipment so that 6'x16' screens could be serviced by crane.

The above revisions would eliminate the troublesome 8'x20' screens and would reduce crushing plant operating time from 21 hours to 18 hours per day.	No change

Fine Ore Storage - Area 40

• Increase live ore storage from 7200 tons to 12,000 tons in order to have more time available for maintenance in fine crushing plant.	+50,000

Grinding - Area 50

• Revise pumps and cyclone requirements to accommodate a 400% circulating load in the primary ball mill circuit.	+40,000
• Include scoop feeders on ball mills in order to reduce cyclone pumping head. The mill motors would then be located at discharge end of mill. Mill spacing would have to be increased and grinding area would have to be lengthened by two bays.	+185,000

Alternative	Overall Effect on Capital Cost (in Dollars)	
• Reduce size of regrind mill from 10' dia. x 13' long to 9' dia. x 13' long; and reduce motor from 600 HP to 500 HP		-30,000
Flotation - Area 60		
• Reduce first cleaner cells from 12 to 10 and reduce cleaner-scavenger cells from 6 to 5.		-15,000
Tailing Disposal - Area 110		
• In order to accommodate flow, use 10 - 15" hydrocyclones in place of 10 - 10" hydrocyclones for tailings dam construction.	+ 8,000	
Flotation Reagents - Area 90		
• Provide two milk-of-lime storage tanks with agitators, each having 24-hour storage capacity in place of one having six hour storage.	+70,000	
Conc. Thickening, Filtering, Drying & Moly Plant - Areas 70 and 160		
• Relocate these areas so that filter and moly plants are part of the concentrator and thickeners, dryer and load-out are nearby. Piping, electrical and building costs should be reduced. With this arrangement the copper and moly reagent facilities can be combined.		-50,000
TOTALS	+513,000	-125,000
NET DIFFERENCE	+388,000	

Operating Costs

The above suggested alternatives will not appreciably affect the operating costs as indicated in Section 6.