CONTACT INFORMATION
Mining Records Curator
Arizona Geological Survey
1520 West Adams St.
Phoenix, AZ 85007
602-771-1601
http://www.azgs.az.gov
inquiries@azgs.az.gov

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PRINTED: 05/31/2002

ARIZONA DEPARTMENT OF MINES AND MINERAL RESOURCES AZMILS DATA

PRIMARY NAME: ORACLE RIDGE

ALTERNATE NAMES:
- CONTROL
- CONTROL COPPER PROPERTY
- MARBLE PEAK
- GEESMAN
- COPPER PRINCESS
- HARTMAN - HOMESTAKE
- LEATHERWOOD
- DAILY
- SOUTHERN COPPER COMPANY
- SOUTH ATLANTIC VENTURES

PIMA COUNTY MILS NUMBER: 881

LOCATION: TOWNSHIP 11 S RANGE 16 E SECTION 16 QUARTER W2
LATITUDE: N 32DEG 28MIN 34SEC LONGITUDE: W 110DEG 43MIN 39SEC
TOPO MAP NAME: BELLOTA RANCH - 15 MIN

CURRENT STATUS: PRODUCER

COMMODITY:
- COPPER SULFIDE
- SILVER
- GOLD
- IRON MAGNETITE

BIBLIOGRAPHY:

- ADMMR ORACLE RIDGE FILE
- USBM RI 5650 P. 113-4, 116, & 130
- BRAUN, ERIC, 1969, UA MS THESIS
- PRODUCTION POSSIBLITIES OF MARGINAL COPPER DEPOSITS IN ARIZONA, P. 63 ADMMR
- USBM MIN. PROP. FILE 21.227, DMEA-2314
- USBM MIN. PROP. FILE 21.64, PROJ. NO. 1418
- LEHNER, R.E. 1973 ADMMR FIELD REPORT
- E&MJ 9/72 P. 272, 6/74 P. 212, 4/75 P. 170 6/78 P. 36-40
- PAY DIRT MAGAZINE 3/26/75, 5/78 P 24, 8/23/76 P. 23, 1/24/77 P. 10, 8/77, 4/26/76.
- SKILLINGS MINING REVIEW 8/14/76, 8/13/77 P. 6 1/14/78, P. 7,14.
- MINING ENGINEERING 6/75 P. 15
- WORLD MINING 10/77, P. 171

CONTINUED ON NEXT PAGE

CONTINUATION OF ORAC. E RIDGE

MINING CONGRESS JOURNAL 9/77 P. 6
CONTINENTAL MATERIALS CORP 1975 ANNUAL REPORT
CONTINENTAL MATERIALS CORP 1976 ANNUAL REPORT
ROSCOE, JOHN G. 1971, STATEMENT BEFORE THE
IMPACT SURVEY TEAM OF THE CATALINA MULTIPLE
USE PLAN BY CONTINENTAL COPPER INC.
ADMMR "U" FILE CU 58
AGS 1995 SPRING FIELD TRIP GUIDEBOOK

Feasibility Study
ORACLE RIDGE PROJECT

Prepared for
Continental Materials Corporation

Volume 1
Technical Report

Feb/1975

Copy No. 15

Feasibility Study

ORACLE RIDGE PROJECT

Prepared for

CONTINENTAL MATERIALS CORPORATION

VOLUME I

TECHNICAL REPORT

Job No. 5382-1

February 12, 1975

The Ralph M. Parsons Company

Parsons-Jurden Division

The Ralph M. Parsons Company
ENGINEERS • CONSTRUCTORS / PASADENA, CALIFORNIA 91124
PARSONS-JURDEN DIVISION

February 14, 1975

PLEASE REPLY TO
655 N. ALVERNON WAY
TUCSON, ARIZONA 85711

Continental Materials Corporation
Post Office Box 662
Oracle, Arizona 85623

ATTENTION of Mr. C. H. Reynolds
Vice President

SUBJECT Oracle Ridge Project
Feasibility Study
P-J Job Number 5382-1

Gentlemen:

Pursuant to our letter proposal for Engineering Services dated August 6, 1974, and authorized August 12, 1974, we are pleased to submit our Feasibility Study on the Oracle Ridge Project. Ten (10) hard-bound copies and forty (40) soft-bound copies are being supplied.

This completes the work in accordance with the terms of reference set forth in the original proposal letter and discussed in subsequent conferences held with the Continental Materials Corporation staff.

The major portion of our work was accomplished by James P. Pollock, Chief Geologist and Project Manager, assisted by Ellis Gates, Process Engineer, L. S. Hill, Economist, D. E. Miall, Mining Engineer, and Norman Parker, Project Engineer.

It was a pleasure to work with the Continental Materials Corporation staff on this interesting project. When you have completed your review of this report, our staff will be pleased to meet with you to discuss it.

Very truly yours,

THE RALPH M. PARSONS COMPANY
PARSONS-JURDEN DIVISION

W. T. Pettijohn
Project Sponsor

WTP:vn
Enclosures

CONTENTS

CONTENTS (Contd)

CONTENTS (Contd)

TABLES

SECTION 1

INTRODUCTION AND CONCLUSIONS

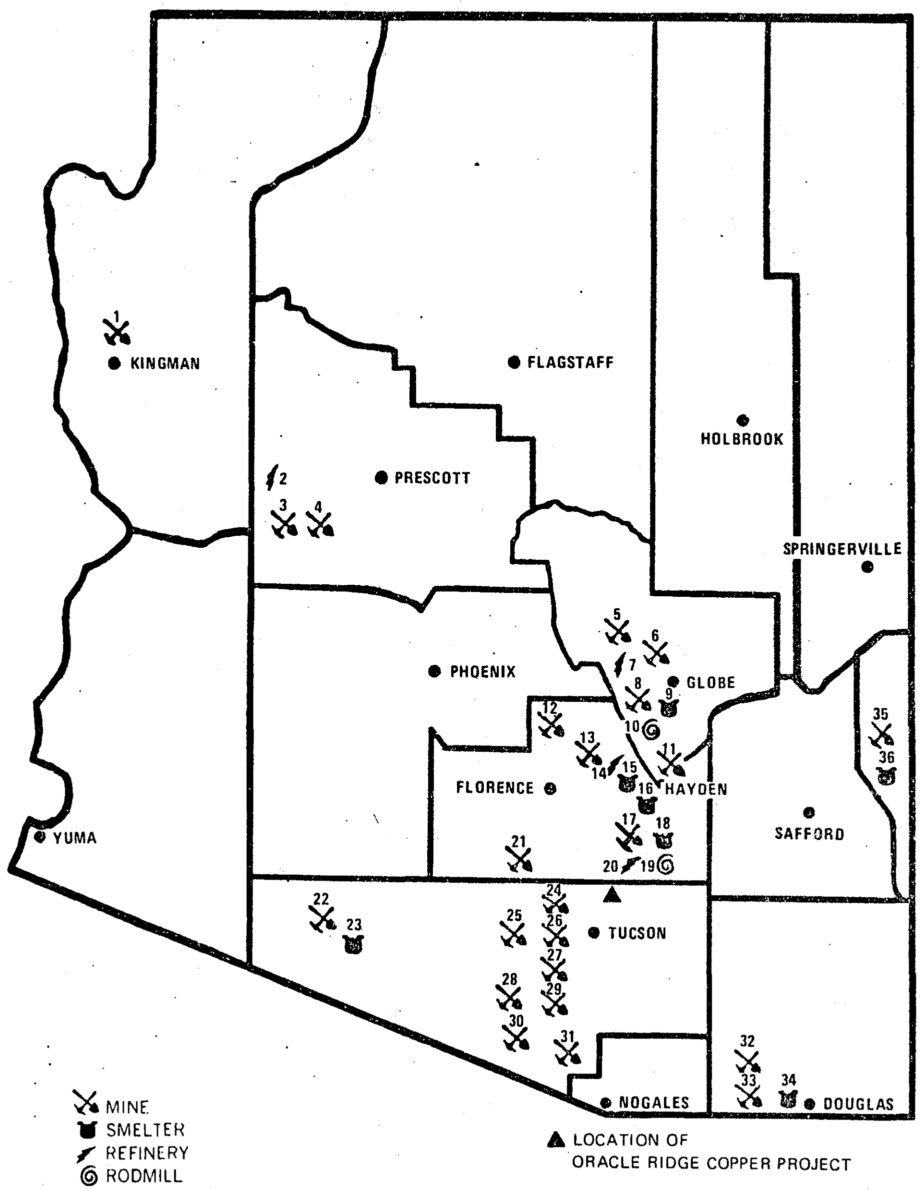
Continental Materials Corporation's Oracle Ridge Project lies 16 miles north-northeast of Tucson on the north flank of the Catalina Range. The property that will be the source of the ore is called the control mine. To the north-east, 53 miles by road, is the Hayden custom smelter operated by the American Smelting and Refining Company, which is a logical market for the planned copper concentrate production; see Figure 1-1. Just to the north of Oracle, the nearest town, is the San Manuel mine and smelter complex, a major copper producer; see Figure 1-2. Its recently negotiated labor contract has been used as a basis for Parsons estimate of the cost of labor in this study.

The principal source of data for Parsons study was an in-house feasibility report by the staff of Continental Materials Corporation and its consultants. See Volume II for a listing of the documents obtained from Continental. The report was supplemented by site visits by Parsons engineers, on-site study of the tailing pond, pipeline, and mill site, and consultation with Continental's staff. Metallurgical test data conducted under the direction of Mr. G. W. Bossard, consultant to Continental, presented in Volume II, were used by Parsons in metallurgical design and calculation of net smelter value. These tests were made on composites prepared by Continental from the assay rejects of diamond-drill core splits from seven areas and one master composite of all seven areas. Additional metallurgical tests during Parsons study to aid in metallurgical design were performed under the supervision of Mr. Bossard.

Parsons has included the results of the metallurgical test work mentioned above and much other data critical to the design of the plant and mine, Volume II. Based upon this data, Parsons has investigated the following:

- Ore reserves as calculated by Continental's staff.
- Mining plans at the rate proposed by Continental.
- Milling plans, based on metallurgical test work supplied by Continental, including tons and grade of concentrate production.
- Capital cost estimate for mine equipment and development and and milling plant.
- Average annual operating cost estimate for mining, milling, and general administration.
- Economic evaluation on the basis of discounted cash flow rate of return at various head grades of ore sent to the mill and various prices of copper.

Arizona Copper Producers



1 Duval Corporation's Mineral Park Mine
2 Bagdad Copper Corporation's Electrowinning Plant
3 Bagdad Copper Corporation's Mine
4 Cyprus Mines Corporation's Bruce Mine
5 Cities Service Company's Pinto Valley Mine
6 Cities Service Company's Miami Copper Operations Mine
7 Inspiration Consolidated Copper Company's Electrolytic Refinery
8 Inspiration Consolidated Copper Company's Mines
9 Inspiration Consolidated Copper Company's Smelter
10 Inspiration Consolidated Copper Company's Rod Mill
11 Inspiration Consolidated Copper Company's Christmas Mine
12 Magma Copper Company's Superior Mine
13 Kennecott Copper Corporation's Ray Mine
14 Kennecott Copper Corporation's Electrowinning Plant
15 Kennecott Copper Corporation's Hayden Smelter
16 American Smelting and Refining Company's Hayden Smelter
17 Magma Copper Company's San Manuel Mine
18 Magma Copper Company's San Manuel Smelter
19 Magma Copper Company's Rod Mill
20 Magma Copper Company's Electrolytic Refinery
21 Hecla Mining Company's Lakeshore Mine
22 Phelps Dodge Corporation's New Cornelia Mine
23 Phelps Dodge Corporation's Ajo Smelter
24 American Smelting and Refining Company's Silver Bell Mine
25 American Smelting and Refining Company's San Xavier Mine
26 American Smelting and Refining Company's Mission Mine
27 Pima Mining Company's Pima Mine
28 Banner Mining Company's Properties
29 The Anaconda Company's Twin Buttes Mine
30 Duval Corporation's Sierrita Mine
31 Duval Corporation's Esperanza Mine
32 Phelps Dodge Corporation's Lavender Pit Mine
33 Phelps Dodge Corporation's Copper Queen Mine
34 Phelps Dodge Corporation's Douglas Smelter
35 Phelps Dodge Corporation's Morenci Mine
36 Phelps Dodge Corporation's Morenci Smelter

ARIZONA MINING ASSOCIATION MAP

Figure 1-1 - Oracle Ridge Project Location

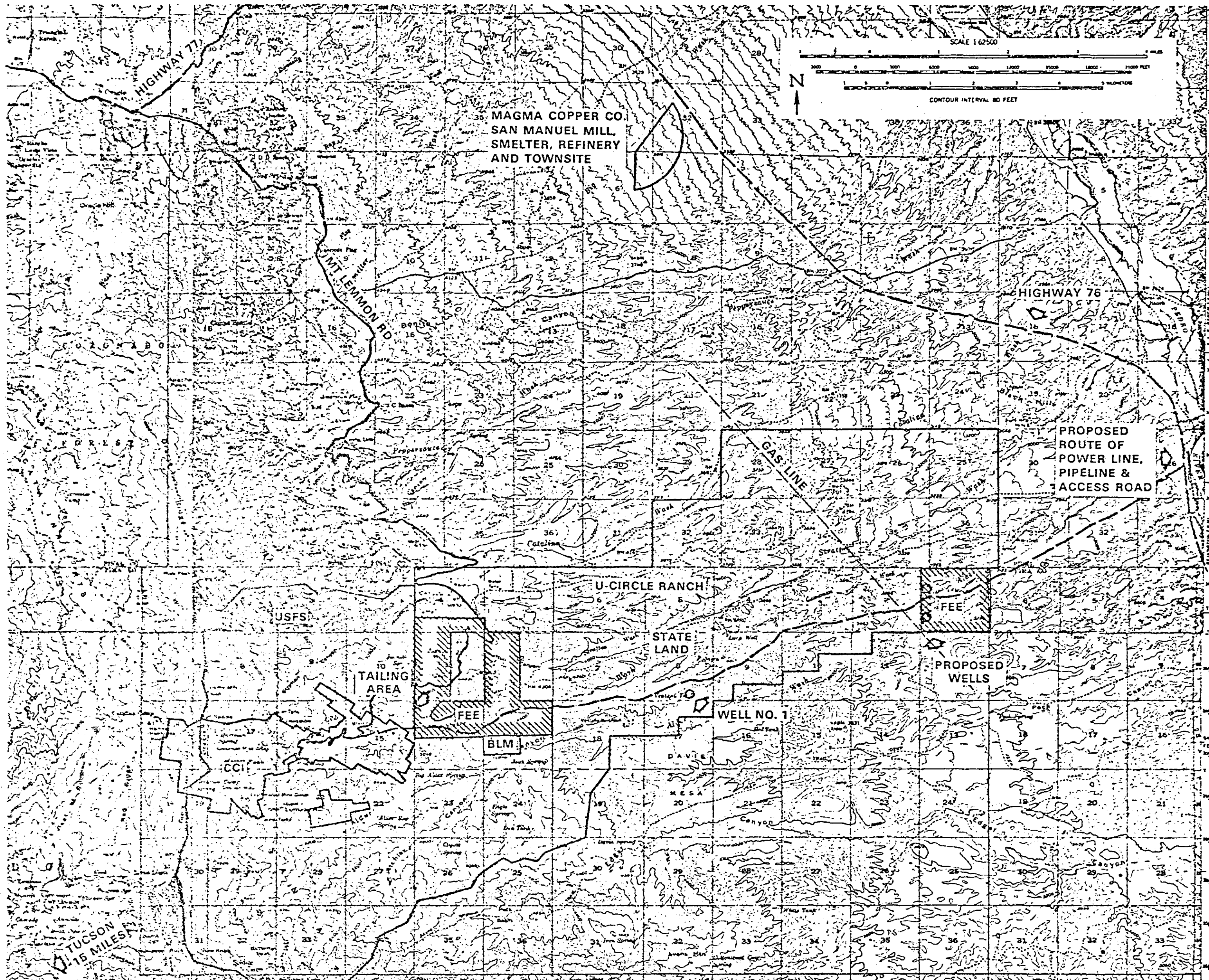


Figure 1-2 - Oracle Ridge Area Map

Based upon the above study, Parsons believes that the ore reserve and geological works has been done competently and agrees with the Continental Materials Corporation's calculations that indicate that the control mine ore body contains 11,270,800 tons of in-place reserves with a grade of 2.28% Cu and 0.64 oz Ag undiluted.

Continental's proposed mining plan appears to be feasible, but will require tight cost control and work scheduling.

The concentrator designed by Parsons has included sufficient grinding capacity to attain a finer primary grind than was used in the metallurgical test work to help assure the probability that the copper recovery indicated in the metallurgical balance can be attained in practice with a minimum generation of slimes.

Environmental considerations have not been examined in depth, but the legal opinion obtained by Continental Materials Corporation at Parsons request indicates the probability of serious problems is slight. The standards for emission of effluent into dry creek beds are as yet ill-defined. See Volume II.

CONCLUSIONS

CAPITAL COST

Parsons estimates the total capital cost as follows:

Concentrator and ancillary facilities		$14,942,800
Major mine equipment		2,336,740
Preproduction preparation		3,207,160
	Total	$20,486,700

Because of the considerable distance of the concentrator from the present sources of available power and location of the freshwater supply, the capital cost for providing these utilities is 20% of the total capital cost of the concentrator and ancillary facilities.

OPERATING COST

Parsons estimates the total operating cost, not including corporate burden charges, as follows:

	Annual	Per DST Mined
Mining cost	$3,565,725	$5.100
Milling cost	1,909,090	2.727
Indirect general staff, misc	361,350	0.516
	$5,836,165	$8.343

Productivity of direct underground labor is estimated at 41.2 tons per manshift. Productivity of all mine manpower is estimated at 27.7 tons per manshift.

FINANCIAL OUTCOME

A computer analysis of the cash flow rate of return on investment is presented in Section 8 for various prices of copper and for three postulated copper grades of mill feed. All cases are calculated at 86% recovery and 32 Cu grade in the concentrate.

SECTION 2

GEOLOGY AND ORE RESERVES

The objectives of this portion of the report are to describe and assess the techniques used by the geological staff of Continental Materials Corporation and the results obtained in the delineation and evaluation of the ore reserves at the Oracle Ridge Project.

The information utilized herein was gathered during Parsons initial field examination of the deposit and during subsequent site visits by Messrs. J. P. Pollock (the Parsons geologist assigned to this project), and V. P. Bluege, and N. H. Parker of the Parsons-Jurden staff to discuss various aspects of the operations and the ore reserve. The C. J. Orback estimate of November-December 1974 has actually been in progress during the entire Parsons study. The purpose of this final estimate by Orback is to replace the original provisional estimate contained in Continental Materials Corporation's Feasibility Report of July 15, 1974 by a detailed study making total use of the diamond drilling, long-hole steel drilling, and other data collected in the exploration phase. As the studies of the various ore blocks became available, Parsons geologist checked the work. At Parsons request, during the study Mr. J. Fritts of Continental's geological staff also independently checked a number of the ore blocks estimated by Orback to verify tonnage and grade by the standard polygonal method, as described below.

Parsons geologist also discussed the diamond drilling, the 6,400-foot level adit, cross-cut sample results, and the long-hole steel drilling from the adit with Messrs. J. G. Roscoe, Manager; G. C. Heikes, Consulting Geologist; C. J. Orback, Consulting Geologist; and J. J. Fritts, Resident Geologist. A number of diamond drill holes plotted on random sections by Mr. Heikes were checked with the cross sections and horizontal projections of various ore blocks and lenses prepared by Mr. Orback. Drill logs were checked with the plotted ore runs to determine thickness, grade, and elevation of tops and bottoms of ore runs in a number of holes. The results of this checking shows the ore reserve calculation has been conducted with meticulous care and accuracy. As a background to a discussion of the ore reserves, the following description of the local geology of the Oracle Ridge deposit has been prepared by the geological staff of the Continental Materials Corporation.

GEOLOGY OF THE ORACLE RIDGE DEPOSIT

REGIONAL SETTING

The deposit is located some 20 miles north-northeast of Tucson in the northern part of the Catalina Mountain range. The Catalina Mountains consist primarily of a Tertiary quartz monzonite batholith and associated injection gneiss. The older rock was apparently pre-Cambrian quartz monzonite.

Along the northeast flank of the Catalina Mountains there is an elongate stock called the Leatherwood "diorite." Recent mapping by the United States Geological survey shows that the quartz monzonite batholith is younger than the Leatherwood stock. The mapping also shows hydrothermal alteration associated with the Leatherwood intrusive, but apparently not with the quartz monzonite batholith.

The Control Mine ore bodies occur in a roof pendant of Paleozoic sedimentary rocks at the north end of the Leatherwood stock. The roof pendant measures approximately 1 mile north-south by 1-1/2 miles east-west. Surface mapping and diamond drilling show the roof pendant to have an irregular, bowl-shaped bottom in contact with the stock. Although called the Leatherwood "diorite," the intrusive immediately under the roof pendant is primarily a porphyritic quartz monzonite, with attendant sill-dike complexes of granodiorite to granite composition.

STRATIGRAPHY

The roof pendant is a remnant of the local sequence of Paleozoic sedimentary rocks, with the Cambrian Bolsa Quartzite at the base and less than half of the Pennsylvanian Horquilla Limestone at the top of the section. Major disconformities are present in the section, but there are no angular unconformities.

At the base of the section, along the western margin of the roof pendant, most of the 300 feet of Cambrian Bolsa Quartzite is exposed. The Bolsa is a relatively pure, medium- to coarse-grained, light gray, massive to thick-bedded orthoquartzite, with local cross-bedding.

The Cambrian Abrigo Formation, overlying the Bolsa Quartzite, is locally divided into three members aggregating approximately 400 feet. The lower member, about 300 feet thick, consists mainly of carbonaceous, calcareous, and silty shale, with several thin limestone and sandstone beds.

A medium-grained sandstone, about 40 feet thick, with abundant carbonate cement, comprises the middle member.

At the top of the formation are 65 feet of limestone and magnesian limestone characterized by undulatory laminations or thin beds.

The Devonian Martin Formation consists of about 250 feet of carbonaceous limestone, sandy carbonate, and an orthoquartzite 35 feet thick near the top of the formation. In the lower half, much of the carbonate is dolomite or magnesian limestone, alternating with limestone.

Overlying the Martin Formation is the Escabrosa Limestone, approximately 580 feet thick. It is a rather massive, coarse-grained, relatively pure limestone with only local bedding.

Less than 700 feet of Pennsylvanian Horquilla Limestone is present at Marble Peak, the highest elevation on the property. This is less than half of the total formation thickness in the area. Carbonaceous, argillaceous, silty and sandy limestone with abundant chert and good bedding characterize the formation. A calcareous shale about 30 feet thick is present some 500 feet above the base. The base of the formation is marked by a thin argillaceous sandstone.

STRUCTURE

At the north end of the property, the Geesaman fault trends west-northwest, dips steeply to the south, and has a throw of more than 3,000 feet down on the south. This fault antedates the Leatherwood stock, and the pre-Cambrian rocks on the north side of the fault formed a buttress against which the Leatherwood was intruded.

Most structure within the roof pendant constitutes adjustment of the sedimentary rocks to the intruding magma of the Leatherwood stock. Part of the adjustment was by flexure. In general, the sedimentary units describe an arc, with a northeast trend at the north, and a southeast trend at the south. The western half was eastward dips of 45 to 50 degrees, but toward the east the rock dips only 5 to 10 degrees east. This reflects the generally concave, or bowl-shaped, upper surface of the stock in this vicinity.

The intruding magma produced compressive stress acting in a north-south direction. There was probably also a nearly contemporaneous east-west compressive stress. The result was a complex of premineralization faults and joints, with attendant intense shattering. In general, the faulting consists of thrusts, two systems of high-angle northeast and northwest strike-slip faults and two systems of high-angle north-south and east-west tension faults. Displacement is normally less than 30 feet.

Two types of presulfide felsic dikes occupy the position of a number of these faults, with the thrust and tension faults predominating. In addition, a post-sulfide diorite magma was injected mainly along northwest strike-slip faults.

It is evident that the region was stressed subsequent to mineralization. The principal effect was of minor recurrent movement on older structures. Probably the source of the stress was the intrusion of a post-Leatherwood batholith, southwest of the Leatherwood stock.

ALTERATION AND MINERALIZATION

Alteration of the carbonates is variable, but quite widespread. Over about 40% of the 1-1/2-square-mile area, more than 100 feet of carbonate rock has been completely silicated. Centrally within this area, 200 to 300 feet of carbonate rock has been so altered. The silicate alteration is not normally concordant with stock surface, and is not related to the stock contact except in a broad, general way.

These effects mean that a large volume of hydrothermal fluid passed through the crystallized shell of the stock through a number of loci and into the overlying carbonates.

Early magnesium silicate alteration produced diopside-quartz rock in three sandy limestone or dolomite beds in the Martin Formation, and to a lesser extent in a bedding plane zone in the center of the upper Abrigo limestone. These essentially bedding concordant zones thus became unavailable as replaceable hosts for the later mineralizing hydrothermal fluid.

The silicate alteration minerals include two main groups: The higher-temperature suite of dense minerals, and the lower-temperature suite of light, hydrated minerals. The higher-temperature group includes epidote, garnet, quartz, and diopside. The lower-temperature group includes tremolite, talc, serpentine, and chlorite.

Copper ore minerals include chalcopyrite, bornite, and minor covellite as primary sulfides, and chalcocite as a locally important secondary sulphide. Magnetite commonly accompanies copper mineralization, and some ore sections contain more than 50% magnetite, by weight. Replacement textures of chalcopyrite and bornite after tremolite and chlorite have been noted in many drill core samples of ore. Approximately half of the total ore mineralization occurs as fillings in random fractures, in joint sets, and in minor faults. The rest of the ore mineralization occurs as disseminations and replacements, primarily controlled by bedding. Where the chemical reactivity (replaceability) of the rock has been diminished by prior alteration to diopside-quartz, sparse sulfide mineralization occurs only as thin fracture fillings, and the rock does not achieve ore grade.

There is thus strong bedding control, and lateral continuity dependent on fluid volume and composition, on both ore and waste zones. Ore zones occur in the upper Abrigo limestone, in the Martin formation at four stratigraphic locations and in the Escabrosa limestone. (See stratigraphic section in Volume II.)

ORE CONTROLS

Ore controls other than bedding include fractures, joint sets, faults, and the "damming" or "ponding" effect of such impermeable barriers to fluid flow as the Martin quartzite and some of the sill-like bodies of igneous rock. The "roots" of the sill-like units comprise portions of the stock surface from which greater amounts of the hydrothermal fluid flowed. Such areas are indicated by increased abundance of vein quartz and vein sulfides in the stock surface. "Blossoming" of alteration and mineralization zones in the carbonate beds occurs up-dip from such areas of more abundant hydrothermal fluid flow.

Because the Escabrosa limestone is nearly massive, bedding control of alteration and mineralization is much less pronounced in this formation than in the Martin and upper Abrigo ore zones. The Escabrosa ore bodies are therefore much more irregular in overall shape than any of the ore bodies in the Martin formation and the upper Abrigo limestone.

[End of Quote]

EXPLORATION METHODS AND PROCEDURES

INITIAL SURFACE EXPLORATION

The property was initially examined by C. H. Reynolds and C. J. Orback in 1968. The presence of old workings with an estimated past production of 115,000 tons of 3.5% Cu ore in the lower Escabrosa limestone and the existence of an extensive surface area of contact metasomatic-type alteration in the upper limestones, which could represent commercial grade mineralization in more receptive host rocks in the Abrigo, Martin, and lower Escabrosa limestones at depth, attracted their interest. Three drill holes had reportedly been put down by an earlier owner, one of which (in Hartman Gulch) had encountered sulfide mineralization. This was later confirmed by Continental holes 1 and 3.

Original mapping was on a base of aerial photographs which were transferred to a scale of 1 inch to 100 feet for use as a drill base map. The volume and type of alteration on the periphery of the roof pendant block was mapped; this map has been updated as required. A ground magnetometer survey of the hole 1 ore body area showed an anomaly; in it there is close association of ore with magnetite.

The entire area has been photographed from the air in black and white and in color. Color enlargements have been used for field mapping, and the black-and-white photos, after establishment of a triangulation network, were used as the basis of a 1-inch to 500-foot topographic map. The topographic triangulation net also forms a control and base for the underground maps, as well as the various surface maps.

DIAMOND DRILLING

The first hole drilled was guided by a magnetic anomaly and interesting alteration. As the results were encouraging, the program was continued. To date, 138 diamond drill holes totaling 103,656 feet have been drilled on this deposit. Because of the rugged topography and the cost of building access roads, some drill sites have multiple inclined holes, and a total of 24 inclined holes were drilled. The balance of the holes (114) were drilled vertically. In a number of instances, substantial deflections were experienced and in-hole camera surveying with an Eastman single-shot camera and magnetic determination of azimuth was found necessary to determine the true position of the drill hole. Roughly 75% of the holes have been surveyed, and in plotting the ore reserve, this deflection data have been used.

Silicate alteration, especially garnet and quartz, visible at the surface in the higher horizons guided the exploration drilling to test additional favorable deeper horizons. The drill core was NQ size, with reduction to BQ size if casing was found necessary. Core recovery in both ore and waste was essentially 100%, except in strongly fractured zones. These favorable drilling conditions are also reflected in the fact that in the majority of holes the occasional cavernous condition of the limestone caused loss of circulation, but the drillers successfully continued the hole using fresh water drilling without water return.

All core has been retained and stored. Ore sections were split on a wedge-type core splitter, generally in runs from 4 to 10 feet long. The split core was sent to Southwestern Assayers and Chemists in Tucson for determination of copper and silver. At the assayer, the core was crushed to 3/8 inch and a split made for analysis. The balance of the minus 3/8-inch crushed core was returned to the Oracle office for preparation of the master metallurgical sample, while the split for analysis was treated in the customary fashion.

LOGGING

The core was logged visually and, in many instances, has been relogged several times, as increasing knowledge has been gained concerning the stratigraphy and the association of various types of alteration with specific horizons. Based on this meticulous work, the stratigraphic section has been broken down into relatively thin units, which materially assist in recognition of mineralized horizons and interpretation of structure. The items and categories recorded and presented on a graphic log are: elevation of observation in multiples of two feet, length of run noted, core recovery, rock type, mineralization with magnetite and metallic sulfides in histogram form (see Volume II, Geologic Techniques and Results at the Control Mine Property, Oracle, Arizona, by John J. Fritts) assay for copper and silver, alteration, and gangue minerals (histograms of concentration of specific minerals) carbonates and silicates, oxidation, structure (fractures or fault) graphic illustration of structure, intensity, and comments. The system used results in a large amount of information being presented in a small space and readily assimilable form.

ADIT LEVEL 6400

An adit has been driven into the F-1 ore zone at the 6,400-foot elevation. It is 2,653 feet long and, at draft mark 2,427 feet, a 132 foot cross-cut has been driven with another crosscut at draft mark 2,567 feet, which is 159 feet in length. From this base, 47 long steel drill holes, which total more than 6,000 feet, have been drilled for the purposes of checking the geology projected from diamond drilling and surface geology and check sampling. See Volume II for a description of long-steel sample handling and geologic mapping of the adit and Summary of Long Hole Drilling for analyses.

The long-steel drill holes have been plotted on projections of the ore reserves and principally tested blocks 1, 2, and 4. As they were not surveyed, some deflection of these holes may have occurred. The holes were directed as closely as possible at a direction normal to the dip so as to give information about the thickness of grade of ore. Because of the possible deflection of the drill hole and changes of dip, the ore runs encountered in the drill holes tabulated can only be used in a general way to confirm the reserves. It is worth noting that in each drill run some waste could be included at the start and finish of the 5-foot sludge sample series and considerable dilution is possible. The arithmetical averages of the long-steel hole runs in ore segregated by geological horizon are as follows:

No. Runs	Arithmetic av	Horizon
18	2.06% Cu	Cambrian Upper Abrigo (Cau)
6	1.97	Devonian Martin L2 Bed
6	2.17	Devonian Martin L3 & L4 beds
11	2.09[1]	Miss. Escabrosa (Me)

While these averages have only general significance, nevertheless the above diluted grades confirm the undiluted in-place grade of block 1 (2.05% Cu).

An attempt was made to correlate the grades determined by a long-hole drill, carbulk samples, and drill cuttings collected from the rounds drilled to advance the crosscut, as reported in Continental's Feasibility Study.

[1]One run of 30 feet of 6.78% Cu omitted from calculation.

In crosscut 1, the following information was gained; the tests are stratigraphically comparable:

98' of long hole 1	1.14% Cu
99' of bulk samples	0.97% Cu
99' of drill cuttings	0.83% Cu

In crosscut 2, better correlation was attained in these three stratigraphically comparable tests:

90' of long hole 2	1.74% Cu
91' of bulk samples	1.76% Cu
91' of drill cuttings	1.71% Cu

The data are too limited to be significant in either case, but it seems possible that crosscut 1 was on the fringes of the ore, which accounts for the erratic results, and that the longhole, directed at a plus angle, was more in the center.

TECHNIQUES USED TO DELINEATE AND EVALUATE RESERVES

SURFACE DIAMOND DRILLING

The 138 drill holes mentioned above supplied essentially all the data from which the ore reserve has been calculated, and the accuracy of the ore reserve depends largely on the continuity of ore between these diamond drill holes and the adequacy of the number of drill holes to test this contact metasomatic deposit; a type of deposit in which there can be considerable lateral variation in grade.

In considering the number of tons of ore that rest on one drill hole, it should be remembered that a number of the holes penetrated as many as five separate stacked lenses of ore. Thus, the number of drill penetrations of ore lenses substantially exceeds the number of drill holes. The difficulty in presenting the drill-hole-penetration data was caused by rugged terrain, deep holes that deviated from original angles, and the lack of ability to drill out consistent significant sections, which prevented construction of series of regular cross sections. As described below, this problem was overcome by projecting the drill information for each lens onto a horizontal projection, and then projecting all lenses in any one ore body to a master horizontal projection that also showed strategic cross sections. See Figures 2-1 through 2-5, the projection of all ore lenses in the F1 ore body and the individual data on the ore in the Cambian upper Abrigo limestone. Volume and grade were calculated as described in the following subsections.

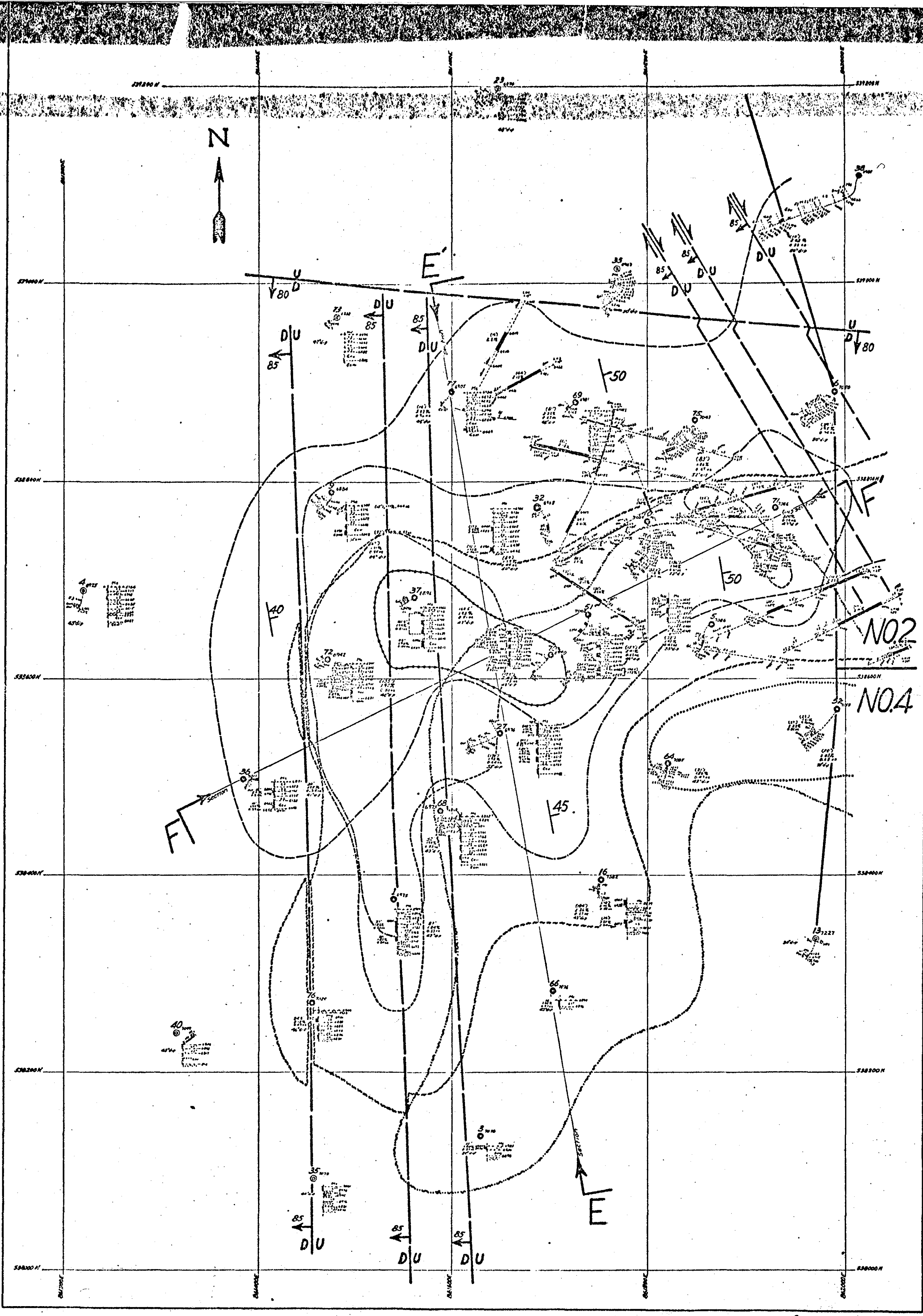


Figure 2-1 - Plan Showing Ore Body Zone

PLATE F1

ZONES COMPRISING NO. 1 ORE BODY

by C. J. Orback, August, 1974

EXPLANATION

ORE-ZONE PERIMETERS

············	Me
—·—·—·—	Dml_1-Me
— — — —	Dml_2
—··—··—	Dml_3
- - - - - -	Dml_4
————	Ꞓau
•—•—•—•	Central "core" zone (part of L2, L3 and L4)

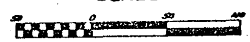
TONNAGE ESTIMATE

21,000	± 30%	Me
374,000	± 15%	Dml_1-Me
510,000	± 10%	Dml_2
300,000	± 10%	Dml_3
100,000	± 15%	Dml_4
900,000	± 10%	Ꞓau
150,000	± 5%	Central "core" zone (part of L2, L3, and L4)
2,355,000 T	± 11%	TOTAL

GRADE ESTIMATE

1.9 %	0.5 oz.	Me
1.9 %	0.5 oz.	Dml_1-Me
2.45 %	0.75 oz.	Dml_2
2.0 %	0.6 oz.	Dml_3
1.8 %	0.5 oz.	Dml_4
1.9 %	0.5 oz.	Ꞓau
2.2 %	1.0 oz.	Central "core" zone (part of L2, L3, and L4)
2.05 % Cu	0.6 oz. Ag	± 10% WT. AVGE.

SCALE



FEET

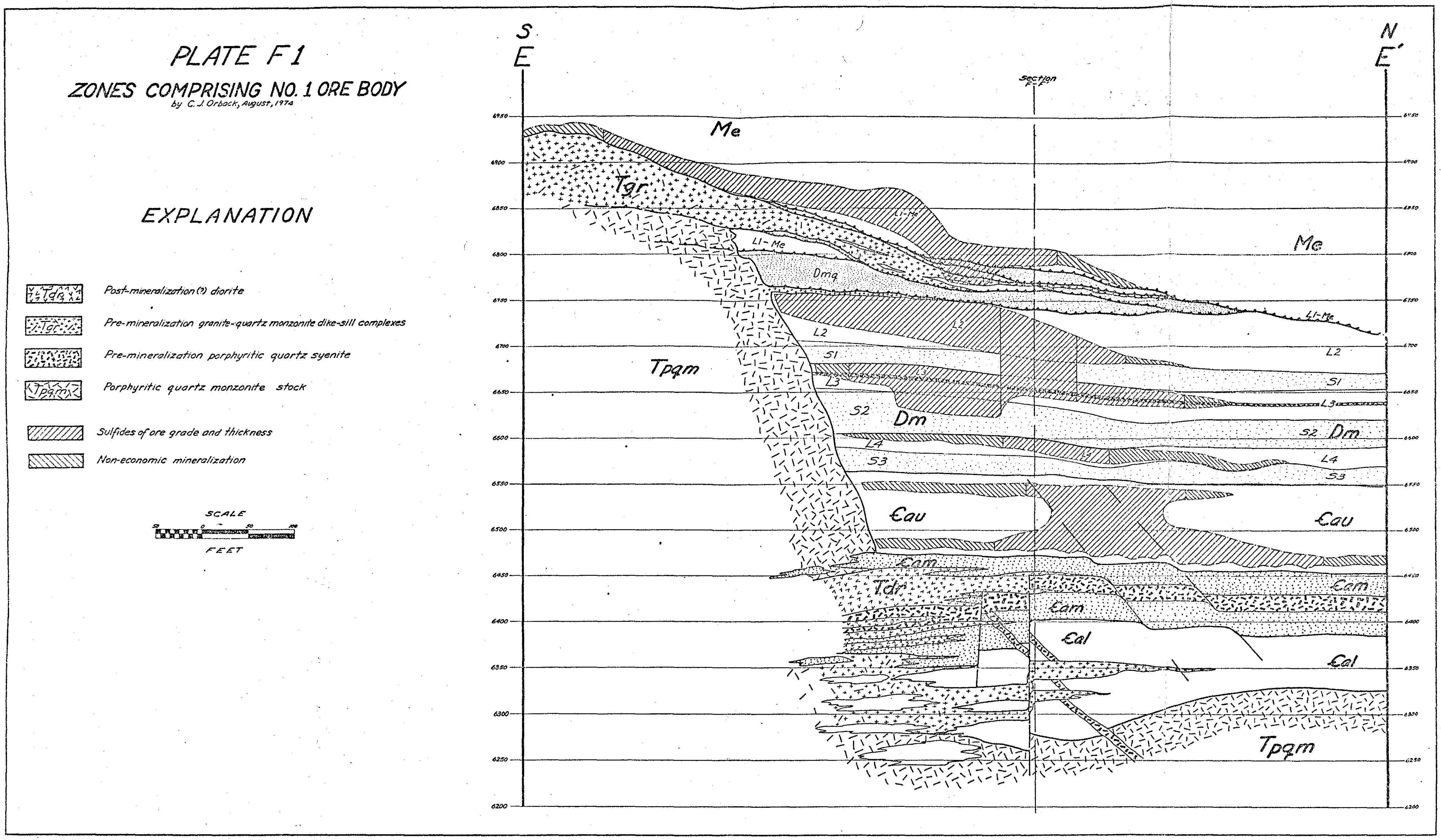


Figure 2-2 - Cross Section E-E'

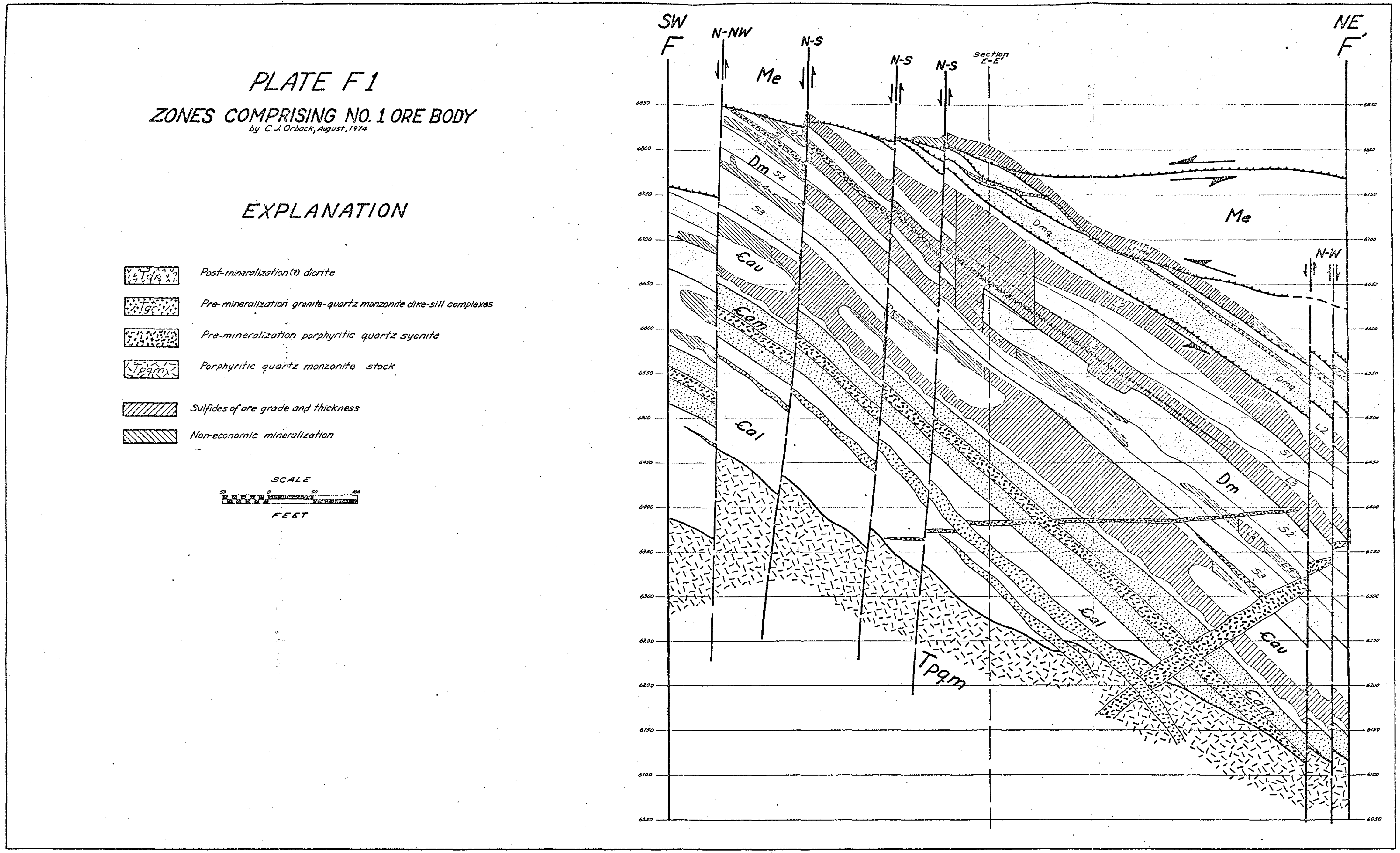


Figure 2-3 - Cross Section F-F'

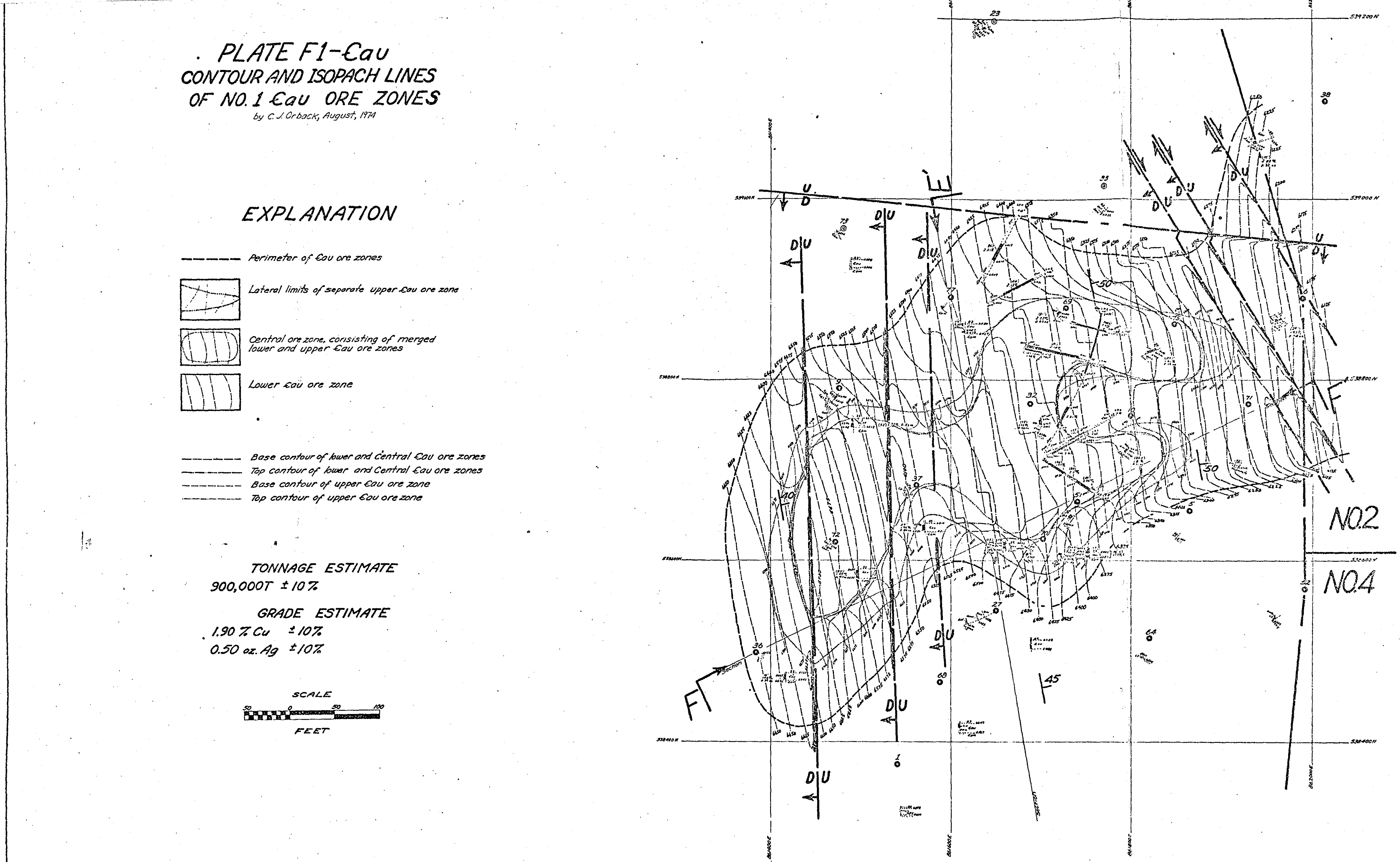


Figure 2-4 - Structural Contour Map

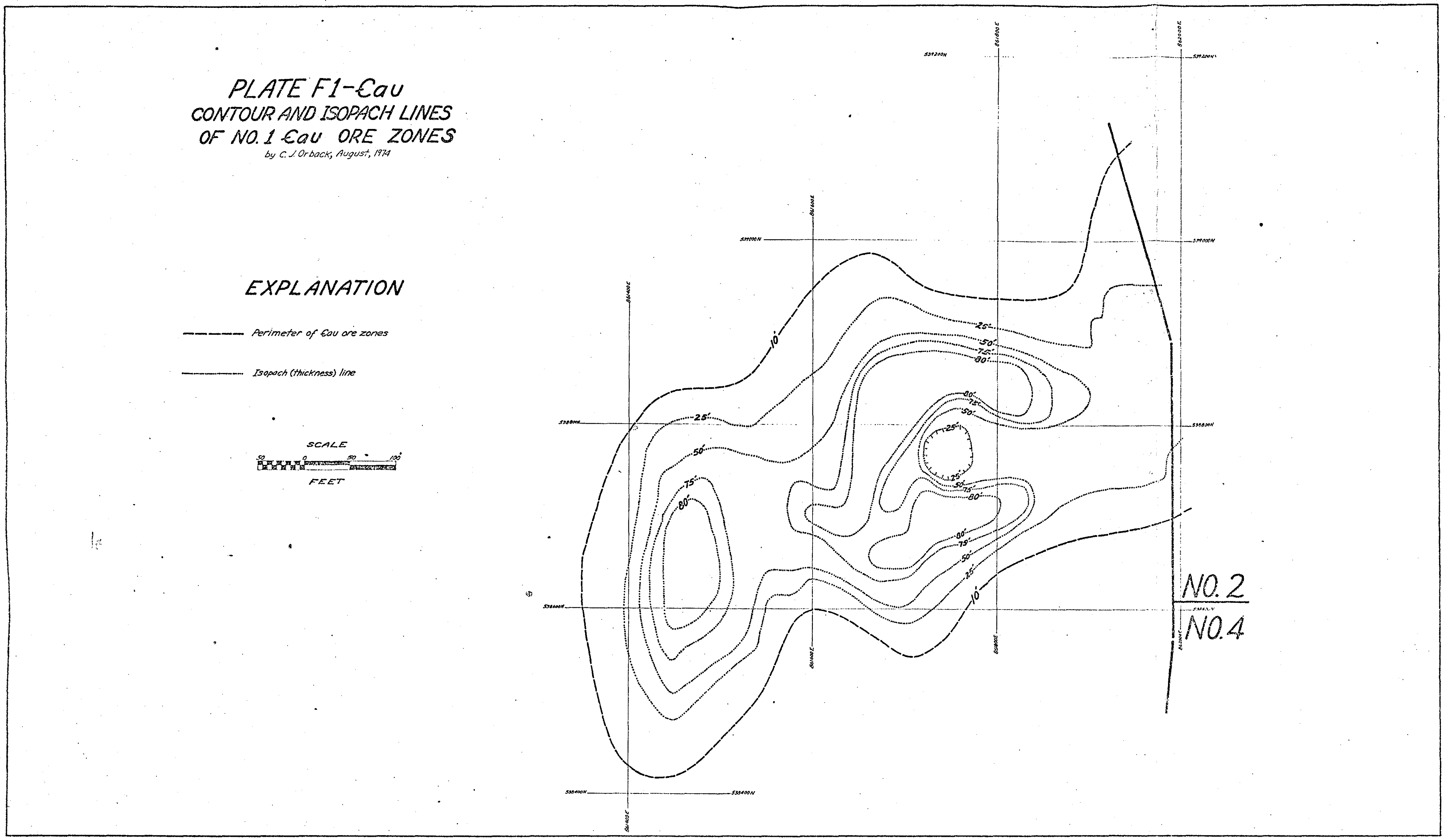


Figure 2-5 - Isopach Map

CALCULATION OF VOLUME AND GRADE OF ORE

The method used by C. J. Orback to calculate the volume of ore appears to be satisfactory where there is an adequate number of drill holes with spacing between on the order of 150 to 200 feet, and when this information is combined with the detailed stratigraphic breakdown and the zoning picture of the silicate minerals. This amount of information is exhibited for example in ore body F1 (see Figures 2-1 to 2-3) on an original scale of 1 inch to 50 feet. Parsons has classified this type of ore as drill-indicated.

Where only a few scattered holes penetrate an ore block such as F10 (where there are only three drill penetrations of the zone), then the quality of the data is insufficient to accurately define the perimeter and average thickness of the zone. This ore, particularly in blocks F10 and F11, can only be classified as drill-inferred.

This problem has been recognized in Continental Materials Corporation's Feasibility Study of July 15, 1974 (page 46), wherein the total ore reserves of 10,660,000 tons have been broken down into 7 million tons of drill-indicated ore with a plus or minus "certainty factor" of from 10% to 25%, and 3,660,000 tons of drill-inferred ore with a plus or minus "certainty factor" ranging from 40% to 60%.

In addition to the drill holes, the plotting of the perimeter of the ore zones is guided in a broad way by the outline of favorable silicate alteration zones in upper horizons, or found on the surface as described below. To thoroughly study the silicate and other significant mineralization, features intersected in the drill hole have been plotted on a series of maps at a scale of 1 inch to 50 feet. This permits more detailed study of silicate zoning associated with metallic-sulfide mineralization and more controlled plotting of the perimeter of the ore (see Figures 2-1 and 2-2).

The sequence of steps in the preparation of the ore reserve are as follows:

(1) A thickness of 10 feet of 1.7% copper is established as the cutoff grade x thickness, and this factor is used to delimit the ore zone.

(2) From the 200-scale exploration map, the general shape and size of the ore zone and contiguous ore zones influencing the study are noted.

(3) The floor or basement of the roof pendant is studied for structure by examining the structure contours on the surface of the stock (Plates C-1 and C-2 prepared by the Continental's geological staff; but not included in this report). Correlation is also made with surface structure, if possible. Stratigraphy is correlated from hole to hole utilizing the stratigraphic subdivisions in the Martin and Abrigo to assist in structural interpretation.

(4) On a horizontal projection of the ore zone, with a scale of 1 inch to 50 feet, all drill intersections, the pertinent structure, a graphic log with pertinent assay data for ore runs, and the projection of

the entire drill hole, if inclined or found to be deflecting, are plotted. If there is more than one ore lens penetrated in the drilling, all lenses are projected to this horizontal section. See Figure 2-1 for example.

(5) The perimeter of each ore lens is then sketched using as guides the following:

- The width of the ore zone is weighted with respect to the amount of sulfides in the drill holes and, to some extent, the amount and nature of silicate alteration.

- Minor fractures that controlled mineralization are considered, also proximity to a major fault, both of which lead to downgrading of the continuity and extent of ore. It is likely that the ore is spatially closely associated with the structure and is not pervasive replacement of the altered sedimentary horizon.

- Drilling and mapping experience, which has shown that there is a fairly smooth transition from the thicker center of the ore lens toward the perimeter, with gradually diminishing sulfides and quantity of silicate alteration. The ratio of dense-to-lighter silicates or wet-to-dry silicates is felt to represent the quantity and quality of hydrothermal alteration that has introduced the ore-bearing minerals, and this is also considered.

(6) One or more cross sections projecting the diamond drill information to these sections are constructed on this sheet; these sections aid in the sketching of the perimeter.

(7) Sections are also influenced by such structure as the top of the stock, major normal or thrust faults, and sills such as the marker sill in the 1-3 horizon of the Martin Limestone, used as stratigraphic horizons; ore intercepts are correlated joining the same stratigraphic units into continuous zones.

(8) This procedure results in one or more perimeter zones stacked vertically; these perimeters are then plotted on a scale of 1 inch to 50 feet on separate sheets according to stratigraphic zone. See Figures 2-1 and 2-2, which show the perimeter of the Cambrian upper Abrigo formation ore in the F1 ore body.

(9) Ore intercepts in drill holes are correlated on this separate sheet for each lens and a contour map is made of the top and bottom of the ore lens, using information such as elevations in diamond drill holes and strikes and dips determined in holes that did not intersect ore.

(10) On the same sheet overlying the structure contours an isopach map that gives vertical thickness of ore within the established perimeter is prepared. The area with a given average thickness can be

planimetered or, in the case of an ore zone with relatively constant thickness, an average may be estimated.

(11) The volume of ore is determined by the product of area of horizontal projection of ore zone multiplied by the vertical thickness.

(12) To obtain tons in place, the volume is multiplied by a factor of cubic feet per ton, which is determined by specific gravity. The highest specific gravity used gives 9 cubic feet per ton, which corresponds to a normal silicate content with 6% magnetite content and 3% voids. This is a specific gravity of 3.59; the specific gravity factor most generally used is 10 cubic feet per ton.

(13) To express the degree of confidence a plus-or-minus limit of error is assigned subjectively. Confidence depends upon the degree of uniformity of geologic structure, and uninterrupted continuity of strata without faults and folds. In addition to the confidence-level approach, a blanket reduction in the calculated tonnage in place reserve may be made where geological evidence indicates a lesser quantity of favorable alteration.

(14) Although based on the original drill hole grades, the determination of grade is also subjective, it is believed necessary to correct for the fact that drilling was directed to cut ore and not to test the fringe areas of ore zones in a representative fashion. Therefore, the lower grade in fringe areas is not adequately represented in the diamond drill sample. To correct this bias, grades may be assigned by inspection based on the general principle that the grade tends to diminish in a regular manner from the thicker center of the lens towards the fringes. This interpolation may also be modified by structural considerations. Grades are reduced because of this correction for bias and never increased.

The C. J. Orback ore reserve calculation completed in December 1974 is shown in in Table 2-1.

RECALCULATION OF THE ORE RESERVES

Upon studying the ore reserves calculation prepared by Mr. C. J. Orback, Parsons accepted the method of determination of the volume of ore establishing a perimeter to a cutoff of 10 feet of 1.7% Cu, contouring the tops and bottoms of the respective lenses, and based on these drawing isopachs, as described above. The geologic interpretations based upon detailed stratigraphy appeared sound. It was believed necessary to check the average grade of the blocks by constructing polygons around the diamond drill intersections and weighting the diamond drill grade by the tonnage determined for that polygon in a geometric average of the tonnage. At Parsons request, this work was performed by J. H. Fritts and C. H. Reynolds on randomly selected blocks. The following is a detailed description of the technique used (essentially, the standard polygonal method):

(1) Perpendicular bisectors were constructed on the lines joining pairs of drill holes, and these were prolonged to their intersection,

Table 2-1 - Orback Ore Reserve Estimate

Class[a]	Ore Block	Low Tonnage	Median Tons	High Tons	Cu Median (%)	Ag Median (oz)
Ind	1	2,000,000	2,355,000	2,600,000	2.05	0.6
Ind	2	290,000	360,000	430,000	2.00	0.5
Ind	3	140,000	175,000	210,000	2.20	0.7
Ind	4	440,000	550,000	660,000	2.20	0.6
Inf	5	510,000	735,000	950,000	2.10	0.55
Ind	6	700,000	935,000	1,170,000	2.93	0.75
Ind	7	170,000	200,000	230,000	2.50	0.8
Ind	8	1,510,000	1,925,000	2,340,000	2.35	0.8
Inf	9	1,330,000	1,905,000	2,480,000	2.20	0.6
Inf	10	390,000	644,800	900,000	2.60	0.6
Inf	11	650,000	1,186,000	1,720,000	2.20	0.6
Inf	12	190,000	300,000	400,000	2.25	0.7
Total Tonnage		8,320,000	11,270,800	14,090,000	2.28	0.64
Total Ind			6,500,000		2.29	0.68
Total Inf			4,770,800		2.24	0.60

[a]Ind = Drill-Indicated; Inf = Drill-Inferred.

NOTES: Parsons classifies those blocks with uncertainty factors above ±30% average as drill-inferred.

The estimates of high and low tonnage are calculated by application of Orback uncertainty factors to median tonnage.

either with the ore perimeter or with other perpendicular bisectors, forming a polygon with the drill hole at the center.

(2) Areas surrounding each drill hole were assigned a grade from the drill hole. Where anomalously high grades were encountered, they were reduced by the method used by C. J. Orback to remove drill bias. Grades were never increased, and they were so reduced approximately 10% of the cases.

(3) Compensating polar planimeter determinations of the area of the polygons were made. The thicknesses were accepted from C. J. Orback volume plots.

(4) Density factors were determined independently and varied from 9 to 11 cubic feet per ton, according to the mineralogical character of the rock. The quantity of magnetite was the principal variable.

(5) Tonnage and grade for each block in one stratigraphic unit were entered in a tabulation and a geometrical average summation made.

(6) All stratigraphic totals were added together geometrically for one given ore zone. Silver assays were not considered.

MINEABLE RESERVES

In Continental Materials Corporation's in-house feasibility study, the diluted grade of ore fed to the mill was 1.7% Cu which "...is considered an average diluted grade for the expected production." As the original ore reserve estimate had an in-place grade of 2.2% Cu, this was a dilution factor of 29%.

Parsons used the grade of 1.7% Cu feed to the mill in calculating the metallurgical balance and sizing the mill equipment, since the final ore reserve was not available until December 1974 (late in this study).

The final in-place ore reserve estimate has a median tonnage of 11,270,800 at an average grade of 2.28% Cu and 0.64 oz/ton Ag. This estimate is subject to an uncertainty of ±10%.

Parsons has adopted a dilution factor of 25% with zero grade material. Mineable reserves are therefore 14,088,500 tons of 1.82% Cu and 0.51 oz Ag.

If we apply the 10% uncertainty factor to this median grade, we obtain:

		Cu%	Ag oz
Case A	Low (-10%)	1.64	0.46
Case B	Median	1.82	0.51
Case C	High (+10%)	2.00	0.56

At the request of Continental Materials Corporation, Parsons has used the above grades as varying feed grades to the mill in the financial analysis - discounted cash flow computer runs to determine the effect on the outcome at various prices of copper. For the purposes of this report, it can be assumed that the mill equipment is adequately sized to handle the increased grade at a constant feed tonnage, but prior to final design additional metallurgical study will be necessary to confirm this. In Volume II, metallurgical balances have been calculated for the above Cases A to C, and the net smelter value of the resulting concentrate.

At the proposed mining rate of 700,000 DST/yr, the life of the diluted reserves will be 20.13 years. For the financial analysis, a life of 20 years has been used.

The results of this recalculation work are shown in Table 2-2. It is apparent from the table that the original Orback technique is conservative both as to tonnage and grade.

Table 2-2 - Fritts Ore Reserve Estimate

Block	Horizon	Short tons		Grade of Copper %	
		Fritts Check	Orback Original	Fritts Check	Orback Original
1	(all)	2,451,000	2,355,000	2.06	2.05
5	(all)	813,000	735,000	2.30	2.10
6	(all)	1,001,000	935,000	2.87	2.93
8	(all)	2,338,000	1,925,000	2.54	2.35
9	(all)	2,200,000	1,905,000	2.28	2.20
10	(Cau)	653,100	578,100	2.51	2.70
10	Dm14	15,800	14,700	1.90	1.90
10	Dm13	56,500	52,000	2.10	2.10

GENERAL COMMENTS

Because of the tendency to erratic sulfide mineralization on the fringes of the ore lenses, Parsons believes that a major effort will be required in the area of grade control to maintain the desired mill feed. One aspect of this will be adequate staffing of the geological department and provision for rapid analysis of samples. Another aspect will be the probable necessity of a continuing underground diamond drill program to guide development headings.

With the above provisos, Parsons believes the grade and tonnage production targets set by Continental Materials Corporation are realistic.

It should also be noted that additional ore exploration targets exist south of drill hole 121, and undoubtedly in other areas, which will be disclosed by the mining process.

SECTION 3

MINING PROGRAM

Present available information indicates that the series of lenses that constitute the Control Property mineralized zones lend themselves favorably to an overhand cut-and-fill mining system. It is anticipated that this system will require modification to a room-and-pillar configuration where lens width demands. Pillar recovery does not appear to present a serious problem as waste material from development headings could be used as structural fill.

From available data, it is apparent that a closely controlled, systematic and continuing probe drilling program will be a prerequisite of a successful mining operation. Rapid analysis of cuttings and detailed logging of probe holes must be available for grade control as well as updating the interpretation of geological occurrence, forecasting of and preparation for changing ground conditions, and mine planning for efficient use of personnel, equipment, and facilities.

Concentrator design demands mine production of 700,000 DST/yr. This production will be achieved on a two-shift per day, 5-day per week, 250-day per year schedule, requiring a 2,800-TPD performance from the mining operation.

The following subsections describe the various operations and equipment involved in the mining operation. Assumptions made where data are either unavailable or incomplete are also defined.

SUMMARY AND CONCLUSIONS

The mining portion of this study has considered the significant logistics of mining at a rate of 700,000 DST/yr of ore. The number, shape, size, and attitude of the various ore zones in the Oracle Ridge property owned by Continental Materials Corporation clearly defines the requirements for aggressive and imaginative supervision.

Grade control in this system of complex ore bodies will be readily accomplished with the detailed sampling and analytical and interpretive techniques presently being employed by the technical staff.

The mining of these ore zones by a hanging wall and footwall ramping system allows large, highly mobile mining equipment to be used. With tight controls on maintenance, repair, and servicing, this equipment will be highly productive and efficient.

A 2-1/2-year preproduction preparation period is anticipated. This amount of time will be required partly because of equipment delivery time and partly because of mill construction scheduling. In the event that equipment delivery is improved and the mill requires feed earlier than scheduled, mine development planning has enough flexibility to accommodate a 6-month early start. Total manpower for the mine will be 101 at full production.

Total capital cost for the mine is estimated at $5,543,900, consisting of $2,336,740 for equipment purchase and $3,207,160 for preproduction mine preparation.

The annual direct operating cost is estimated at $3,565,725, which will result in a unit cost of $5.10/ton. The cost does not include equipment replacement expenditure which, over the long term, is estimated to average out at $0.42/ton of ore.

In considering the above costs, the following conclusions have been reached: Significant areas for seeking economics in capital costs are preproduction development and underground mobile equipment. As a part of the final design and during the development of schedules for the mining methods, it is recommended that a concerted effort would be made to research ways to minimize both these items.

Significant areas for reducing operating costs are personnel, mobile equipment, and explosives, which account for 82% of the operating cost. As operating experience is gained, it will be necessary for supervision to be persistent in adjusting the method of operations to suit conditions encountered so that the most efficient use is made of manpower and equipment.

MINING METHOD

Drill hole data have been sufficient for Continental's staff engineers to develop a general mining method for the control property. In the case of Zone 1, the quantity of drill hole data and the penetration of this zone by the 6,400 adit, which resulted in the driving of a footwall drift and two crosscuts in ore, have provided valuable correlative data for geologic interpretation and mine planning. This information has also provided operating personnel with direct and specific observations of ground conditions, and has reasonably defined the magnitude of grade control requirements. Probe drilling for ore delineation will be required on a regular ongoing basis and will constitute a priority activity for continuing definition and refinement of the mine plan.

Primary access to the ore zones will consist of levels driven from adits located at the 5,900- and 6,400-foot elevations. Except for ore transport, both levels will be essentially operated as independent production facilities because of the physical orientation of the various ore zones. The 6,400-foot level will be a trackless operation feeding ore through a system of ore passes to the 5,900-foot level for track haulage to the mill. A major advantage of having independent production facilities is that this provides flexibility and adaptability when differing conditions are encountered.

Preliminary to stoping, ramps will be driven in the hanging wall and footwall surrounding the various lens structures. The ramps will be driven from the main levels and provide not only production access, but also ventilation and escape routes as well. During ore block development, ore passes and ventilation raises will be reamed or driven.

The mining method will conform with the following preliminary design criteria:

(1) Ramps, drifts, and crosscuts will be nominally 11 feet high by 15 feet wide.

(2) Ramp grades will not exceed 15%, and will be flatter when stope lengths permit. Ramps will be laid out in a zigzag pattern, with switchbacks at continuing ends 30 feet wide by 50 feet long at 0 grade. Sublevels will be established at not less than 50-foot nor more than 100-foot vertical intervals.

(3) Raises about 50 feet high will be driven conventionally at predesignated attitudes. Raises longer than 50 feet between levels will be drilled; they will be designed to discharge vertically or into the side of drawpoint drifts. Raise personnel will be carried on this roster as longhole drillers.

(4) Longholing from lower levels will precede the advance of development headings.

(5) Stope drifts will be driven along the footwall from the footwall ramps, with appropriate crosscuts and longholes to the hanging wall to establish stope limits.

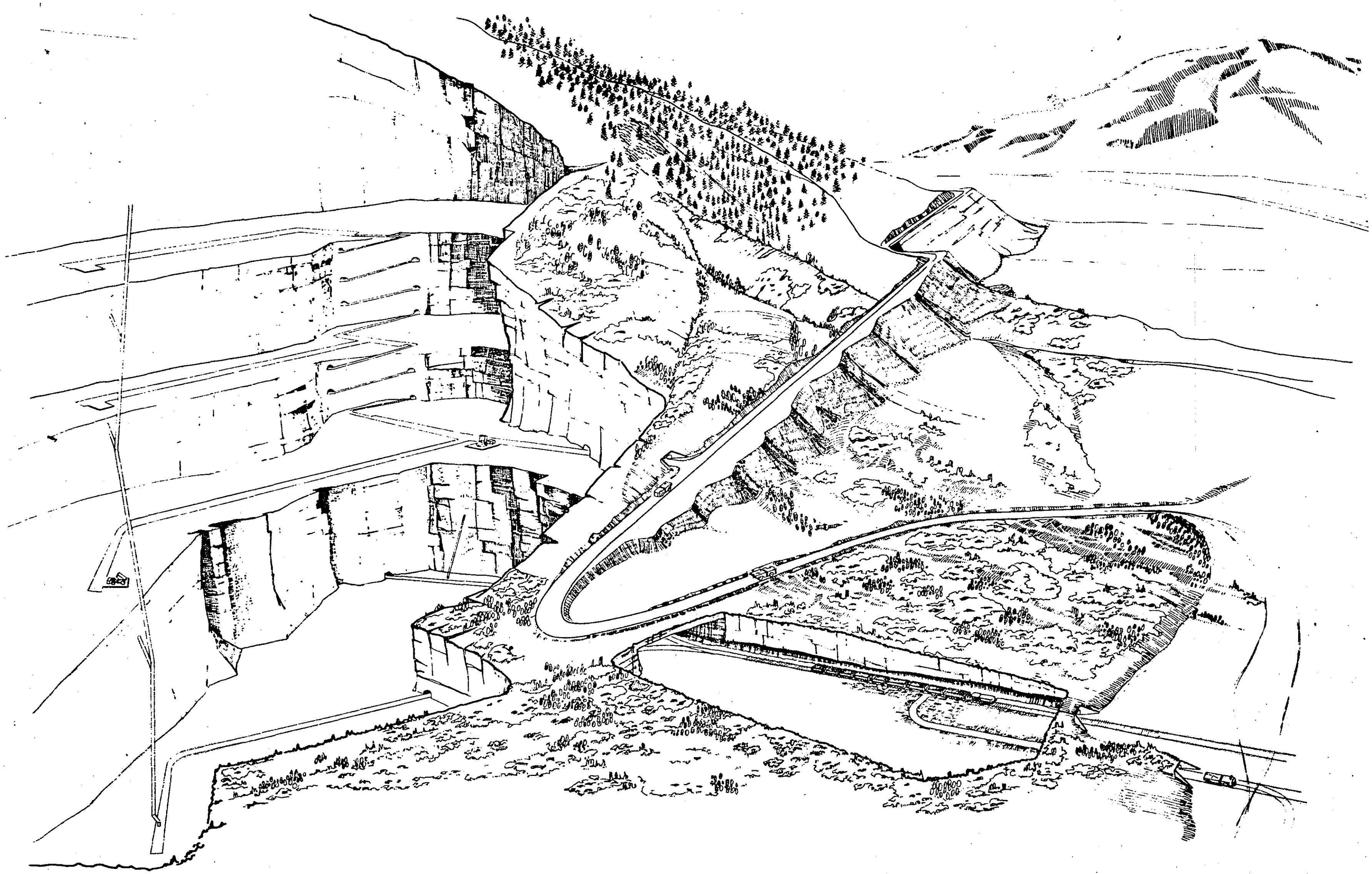
(6) Trackless stoping and stope development using cut-and-fill or room-and-pillar techniques will be used where width of ore allows. Selective mining for grade control will require provision of jackleg drills and slushers while the bulk of production will be accomplished with large drill jumbos and load-haul-dump (LHD) equipment. Backfill material will be waste development rock and/or sized mill tailings.

MINING PLAN: REVIEW AND COMMENTS

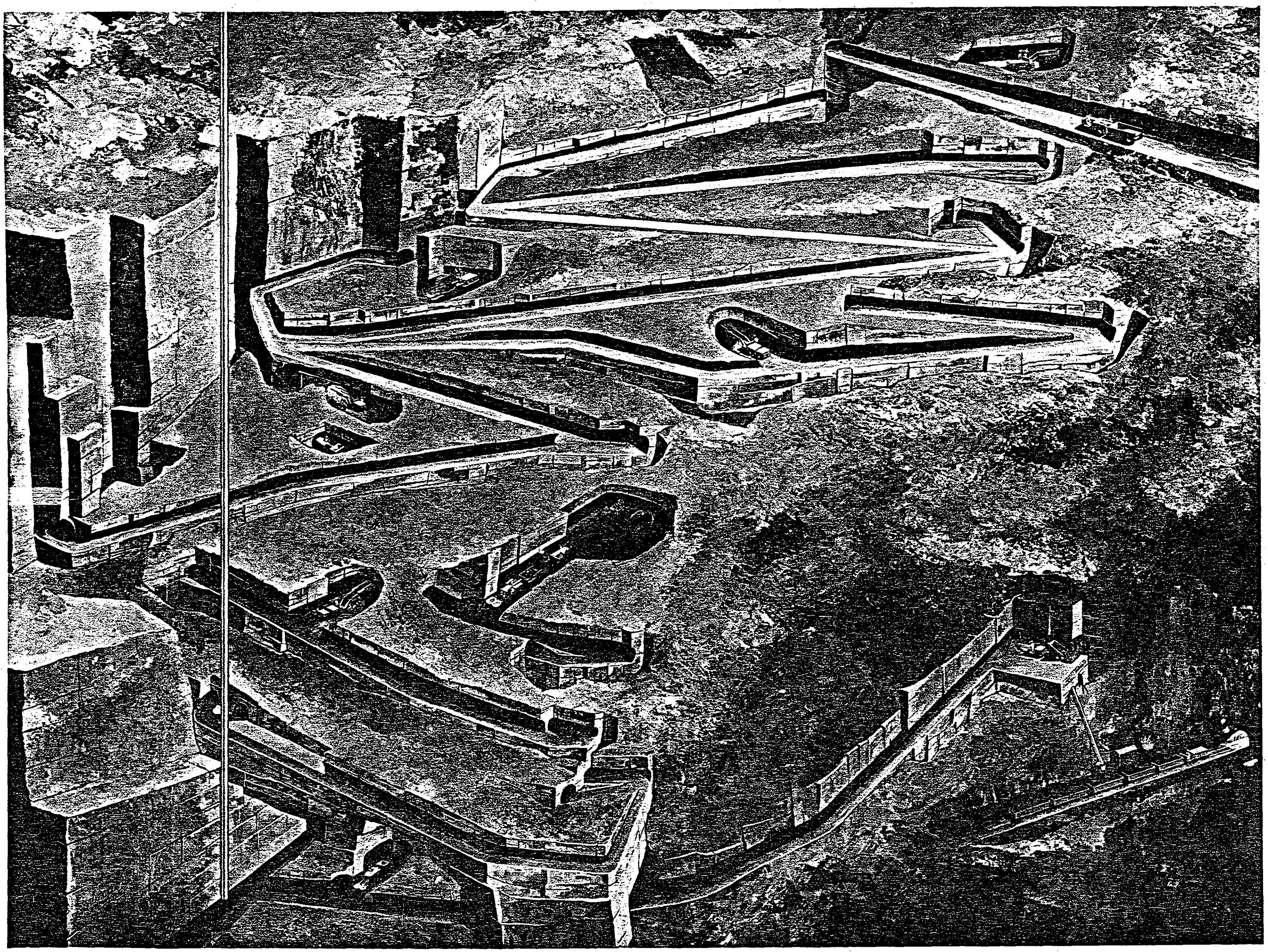
Mine planning drawings engineered by Continental correlate closely with geological data. The mining design has resulted from extensive and serious consideration of many factors and shows a thorough knowledge of the various ore zones.

Since engineering drawings of this type of complex mining system involve much detail and may tend to confuse a reader who is not well aquainted with the Oracle Ridge project, the following two sketches are included in the report for clarity.

The first sketch is an artist's overview of the property showing the general layout of the mining operation; the second (in color) details a ramp system similar to the one that will be used to mine the Number 1 ore zone.



Artist's Overview of Oracle Ridge Copper Mine



Mine Ramp System

PREPRODUCTION MINE PREPARATION SCHEDULE

The following listing is a schedule that shows the sequence of PREPRODUCTION MINE PREPARATION by quarter years. This is followed by tables that summarize tonnages of ore and waste excavated in the various zones.

1st Quarter - 5,900 Level

Zone 7 - prepare portal access road

Zone 6 - no advance

6,400 Level

Zone 1 - 1 heading
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #1) = 7,175 ton

Zone 8 - excavate shops and sumps
1 heading
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #2) = 7,175 ton

Zone 9 - no advance

2nd Quarter - 5,900 Level

Zone 7 - prepare portal site

Zone 6 - no advance

6,400 Level

Zone 1 - 2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #1) = 14,350 ton

Zone 8 - shops and sumps
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #2) = 7,175 ton

Zone 9 - no advance

3rd Quarter - <u>5,900 Level</u>

Zone 7 - construct portal

Zone 6 - no advance

<u>6,400 Level</u>

Zone 1 - 2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #1) = 14,350 ton

Zone 8 - shops and sumps
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #2) = 7,175 ton

Zone 9 - no advance

4th Quarter - <u>5,900 Level</u>

Zone 7 - 1 heading (5,900 adit)
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #3) = 7,175 ton

Zone 6 - no advance

<u>6,400 Level</u>

Zone 1 - 2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #1) = 14,350 ton

Zone 8 - shops and sumps
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #2) = 7,175 ton

Zone 9 - no advance

5th Quarter - <u>5,900 Level</u>

Zone 7 - single heading (5,900 south #7 access)
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #3) = 7,175 ton

<u>5,900 Level</u> (contd)

Zone 6 - 1 heading (5,900 west #6 access)
2 rounds/shift/advance
= 62.5 x 14 = 875'
(Jumbo #5) = 14,350 ton

<u>6,400 Level</u>

Zone 1 - 1 heading (6,550 crosscut south)
1 round/2 shifts/advance
= 62.5 x 7 = 437.5'
(Jumbo #1) = 7,175 ton

2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #1) = 14,350 ton

Zone 8 - 1 heading (6,400 south)
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #2) = 7,175 ton

Zone 9 - no advance

6th Quarter - <u>5,900 Level</u>

Zone 7 - 1 heading (5,900 south #7 access)
1 round/shift/advance
= 62.5 x 14 = 875'
(Jumbo #3) = 14,350 ton

Zone 6 - 1 heading (5,900 west #6 access)
2 rounds/shift/advance
= 62.5 x 14 = 875'
(Jumbo #3) = 14,350 ton

<u>6,400 Level</u>

Zone 1 - 1 heading (6,550 crosscut south)
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #1) = 7,175 ton

2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #1) = 14,350 ton

6th Quarter - <u>6,400 Level</u> (contd)

Zone 8 - 1 heading (6,400 south)
1 round/2-shift day/advance
= 62.5 x 14 = 875'
(Jumbo #2) = 14,350 ton

Zone 9 - no advance

7th Quarter - <u>5,900 Level</u>

Zone 7 - 2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #3) = 14,350 ton

Zone 6 - 1 heading (west to #2)
1 round/2-shift day/advance
= 62.5 x 14 = 875'
(Jumbo #3) = 14,350 ton

<u>6,400 Level</u>

Zone 1 - 1 heading (6,550 crosscut south)
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #1) = 7,175 ton

2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #1) = 14,350 ton

Zone 8 - 2 headings (HW & FW ramps)
1 round/day/advance
62.5 x 14 = 875'
(Jumbo #2 = 14,350 ton

Zone 9 - 1 heading (6,400 south)
1 round/2-shift day/advance
= 62.5 x 14 = 875'
(Jumbo #2) = 14,350 ton

8th Quarter - <u>5,900 Level</u>

Zone 7 - 2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #3 = 14,350 ton

8th Quarter - <u>5,900 Level</u> (contd)

Zone 6 - 1 heading (west to #2)
1 round/2-shift day/advance
= 62.5 x 14 = 875'
(Jumbo #3) = 14,350 ton

<u>6,400 Level</u>

Zone 1 - 1 heading (6,550 crosscut south)
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #1) = 7,175 ton

2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #1) = 14,350 ton

Zone 8 - 2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #2) = 14,350 ton

Zone 9 - 2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #2) = 14,350 ton

9th Quarter - <u>5,900 Level</u>

Zone 7 - 2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #3) = 14,350 ton

Zone 6 - 2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #3) = 14,350 ton

<u>6,400 Level</u>

Zone 1 - 1 heading (6,950 access)
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #1) = 7,175 ton

2 headings (stope development)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #1) Ore = 14,350 ton

9th Quarter - 6,400 Level (contd)

Zone 1 - 2 headings (stope development)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #4) Ore = 14,350 ton

Zone 8 - 2 headings (stope development)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #2) Ore = 14,350 ton

Zone 9 - 2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #2) = 14,350 ton

10th Quarter - 5,900 Level

Zone 7 - 2 headings (1 stope development)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #3) Ore = 14,350 ton

Zone 6 - 2 headings (HW & FW ramps)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #3) = 14,350 ton

6,400 Level

Zone 1 - 1 heading (6,950 access)
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #1) = 7,175 ton

2 headings (1 stope development)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #1) Ore = 14,350 ton

2 headings (1 stope development)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #4) Ore = 14,350 ton

Zone 8 - 2 headings (stope development)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #2) Ore = 14,350 ton

10th Quarter - <u>6,400 Level</u> (contd)

Zone 9 - 1 heading (6,550 access)
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #2) = 7,175 ton

11th Quarter - <u>5,900 Level</u>
(full production)

Zone 7 - 1 stope (production)
2 rounds/2-shift day/advance
= 62.5 x 28 = 1,750'
(Jumbo #3) ore = 28,700 ton

Zone 6 - 2 headings (1 stope development)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #3) ore = 14,350 ton

<u>6,400 Level</u>

Zone 1 - 2 headings (1 stope development)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #1) ore = 14,350 ton

2 stopes (production)
2 stopes x 2 headings/2-shift day/advance
= 62.5 x 8 x 7 = 3,500'
(Jumbo #5) ore = 57,400 ton
(Jumbo #1)

Zone 8 - 2 headings (stope development)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #2) ore = 14,350 ton

1 stope (production)
2 rounds/shift/2-shift day/advance
= 62.5 x 28 = 1,750'
(Jumbo #2) ore = 28,700 ton

Zone 9 - 1 heading (6,550 access)
1 round/2-shift day/advance
= 62.5 x 7 = 437.5'
(Jumbo #6) = 7,175 ton

2 headings (1 stope development)
1 round/day/advance
= 62.5 x 14 = 875'
(Jumbo #6) ore = 14,350 ton

Table 3-1 - Preproduction Mine Development Excavation by Quarters (Tonnage)

Quarter	Zone 1 Waste	Zone 8 Waste	Zone 9 Waste	Zone 7 Waste	Zone 6 Waste	Total Waste	Zone 1 Ore	Zone 8 Ore	Zone 9 Ore	Zone 7 Ore	Zone 6 Ore	Total Ore
1	7,175	7,175	-	-	-	14,350	-	-	-	-	-	
2	14,350	7,175	-	-	-	21,525	-	-	-	-	-	
3	14,350	7,175	-	-	-	21,525	-	-	-	-	-	
4	14,350	7,175	-	7,175	-	28,700	-	-	-	-	-	
5	21,525	7,175	-	7,175	7,175	43,050	-	-	-	-	-	
6	21,525	14,350	-	14,350	14,350	64,575	-	-	-	-	-	
7	21,525	14,350	14,350	14,350	14,350	78,925	-	-	-	-	-	
8	21,525	14,350	14,350	14,350	14,350	78,925	-	-	-	-	-	
9	7,175	-	14,350	14,350	14,350	50,225	28,700	14,350	-	-	-	43,050
10	7,175	-	7,175	-	-	14,350	28,700	28,700	-	14,350	14,350	86,100
Subtotal						416,150						129,150
11 (full production)	-	-	7,175	-	-	7,175	71,750	43,050	14,350	28,700	14,350	172,200
Total						423,325						301,350

Table 3-2 - Preproduction Development Schedule

Quarter	Excavation Ore	Excavation Waste	% Total	Cum %
1		14,350	2.7	2.7
2		21,525	3.9	6.6
3		21,525	3.9	10.5
4		28,700	5.3	15.8
5		43,050	7.9	23.7
6		64,575	11.8	35.5
7		78,925	14.5	50.0
8		78,925	14.5	64.5
9	43,050	50,225	17.1	81.6
10	86,100	14,350	18.4	100.0

Notes: Total Waste Excavated in 10 Quarters = 416,150 tons
Total Ore Excavated in 10 Quarters = 129,150 tons

Total = 545,300 tons

MINE PRODUCTIVITY

MINE PRODUCTIVITY

CRITERIA FOR "STANDARD ROUND"

The criteria used in this report are as follows:

Dimensions: 11 ft high x 15 ft wide x 7 ft deep (drilled)
11 ft high x 15 ft wide x 6.8 ft deep (broken)

It has been the experience of Continental Material Corporation's operating staff that the 11- by 15-foot drift size is the most practical when the size of equipment, space requirements for services, and road maintenance are considered. Maximum equipment and schedule control can be obtained by achieving a cyclical rate of advance each shift. Continental's on-site experimentation and test drilling operations have recently indicated that the 6.8-foot (7-foot drilled) round can be consistently cycled in 11- by 15-foot headings.

The desired advance rate for depth drilled and fragmentation required for efficient equipment utilization was stated to result from drilling a 32-hole round, which includes a 3.5-inch burn hole. The standard bit size is 1-7/8 inch diameter with Grade G tungsten carbide inserts. All holes except lifters are drilled with a slight upward inclination to facilitate cutting removal and to provide a dry hole for blown ammonium nitrate/fuel oil mixture explosives. Lifters are drilled as flat as possible, and blasting grade is carried 1 foot below mucking grade. All holes in the round are drilled parallel to the long axis of the drift.

Because of various factors, it is probable that actual rounds drilled in the different development and production areas will differ from the standard round. For the purposes of this study, however, all rounds are assumed to be the same as far as drilling and blasting are concerned.

Continental Material Corporation's project staff has indicated that grade control and heading development will be determined from drill cuttings by an x-ray fluorescence method. The drillers will be expected to gather samples, and analytical counts will be run by supervisors who, in turn, will determine and mark up the dimension and direction of the mining advance in the various headings. This method of operation appears to eliminate much of the expense and time consumed where the conventional fire and wet assay determinations are employed. The mine manning schedule reflects the use of the x-ray fluorescence method in that no allowance has been made for extensive sampling, sample preparation, data plotting, interpretation, and correlation.

Probe drilling will be accomplished by two long-hole drills. The evaluation of cuttings obtained will determine stope limits.

The production requirements and work schedule will be as follows:

- Production Requirements

Ore		
	Annual	700,000 DST
	Daily	2,800 DST

- Work Schedule

Mining	250 days/yr 5 days/wk 2 shifts/day
Haulage	250 days/yr 5 days/wk 2 shifts/day
Equipment Maintenance	250 days/yr 5 days/wk 2 shifts/day

DAILY MINE PERFORMANCE TO MEET CONCENTRATOR DEMAND

With due consideration to logistic, service, maintenance and supply functions encountered in a producing mine, it is prudent to limit the performance criteria of the various types of headings to those described in the following paragraphs:

- Ramp Development - This operation includes level development, to be scheduled as required by mine supervision. All ramp and level development is expected to be in waste. It is expected that a three-boom jumbo will advance two separate headings at a rate of one round per shift.

- Stope Development - All stope development is expected to be in ore. In order to provide an adequate number of working faces, concurrent development of one stope will be required for each producing stope. This development will advance at a rate of two rounds per jumbo per shift.

- Stope Production - Apparent configuration of the ore zones realistically provides the working space necessary for multiple face stoping activities. By using highly mobile drilling and mucking equipment, and because travel distances will not be lengthy between working faces in individual stopes, stope performance is rated at three standard rounds per shift.

The standard rounds required per day are:

	Ramp Development (Waste)	Stope Development (Ore)	Stope Production (Ore)
5,900 Level	1	3	4
6,400 Level	2	6	12

Other criteria are:

- Tonnage Factors

  Ore in Place:

  10 cu ft/ton
  200 lb/cu ft

  Broken Ore:

  14 cu ft/ton
  143 lb/cu ft

  Waste (granite and limestone) in Place:

  12 cu ft/ton
  167 lb/cu ft

  Broken Waste:

  18 cu ft/ton
  111 lb/cu ft

- Broken Round

  Ore:

  $$\frac{11 \text{ ft} \times 15 \text{ ft} \times 6.8 \text{ ft}}{10 \text{ cu ft/ton}} = 112 \text{ tons}$$

  112 tons x 14 cu ft/ton = 1,568 cu ft (58 cu yd)

  Waste:

  $$\frac{11 \text{ ft} \times 15 \text{ ft} \times 6.8 \text{ ft}}{12 \text{ cu ft/ton}} = 94 \text{ tons}$$

  94 tons x 18 cu ft/ton = 1,692 cu ft (63 cu yd)

EQUIPMENT PERFORMANCE

In a standard round, the equipment selected is expected to perform as follows:

Scooptram Wagner ST-5A

Operation	Sec
At the Muckpile	
Approach	10
Lower	5
Crowd	5
Buck (8 times @ 2.5 sec)	20
Level	4
Lift	6
Reverse	10
	60 (1 min)
At Dump Point	
Turn and position	15
Dump	30
Return and advance	15
	60 (1 min)
Travel	
1,000 ft max, one-way down loaded/up empty:	
Down loaded, av 6 mph 1,000 ft/8.8 ft/sec	114
Up empty, av 4 mph 1,000 ft/5.9 ft/sec	169
	283 sec (4.7 min)
Total	6.7 min

- Capacity

Bucket:

4.5 cu yd struck; max tramming wt = 7.5 tons

Broken ore:

$$14 \text{ cu ft/ton} = \frac{4.5 \text{ cu ft/load} \times \dfrac{27 \text{ cu ft}}{\text{cu yd}}}{14 \text{ cu ft/ton}} = 8.7 \text{ tons/load}$$

(use 7.5 tons/load)

115 tons/round/7.5 tons/load = 16 loads/round

Waste @ 18 cu ft/ton = 6.75 tons/load

96 tons/6.75 = 15 loads/round

Mucking Schedule for a Standard Round

Assume muckpile is washed down and heading is scaled and haul distance is 1,000 feet at maximum grade of 15%, using one Wagner ST-5A Scooptram:

Operation	Min
Travel to face	5.0
Clean road and dress muckpile	10.0
Load and dump 16 loads x 6.7	107.2
Clean up road	5.0
Travel to next assignment	5.0
	132.2 (2.2 hr, use 2.5 hr)

Mucking Schedule for Standard Round of Ramp Development (Truck Haulage)

Assuming the muckpile is washed down and heading is scaled, that the main level haul distance is 3,000 feet one way, and that grade is ± 1/2%, for maximum use, scoop will load two trucks. Turnouts will be at maximum 500-foot centers (average distance, 250 feet). If the ramp haul average distance is 1,000 feet at ± 15%, a truck bypass will be established 500 feet up the 15% ramp from the truck turnaround closest to face, where the empty truck will wait until one that is full has passed on its way down to dump. Three struck buckets of the Scooptram will load the truck (Wagner MT-F25-35-Articulated Rear Dump Truck).

Using the ST-5A Scooptram, the mucking schedule is as follows:

Operation	Sec
• From Loading Point	
Turn and advance	15.0
Up 15% empty (avg. 4 mph) 250 ft @ 5.9 ft/sec	42.4
• At the Muckpile	
Approach	10.0
Lower	5.0
Crowd	5.0
Buck (8 times @ 2.5 sec)	20.0
Level	4.0
Elevate	6.0
Reverse	10.0
Down 15% loaded (av 6 mph) 250 ft @ 8.8 ft/sec	28.4
• At Loading Point	
Turn and position	15.0
Dump into truck	30.0
	190.8 (3.2 min)

Wagner MT-F25-35 Articulated Rear Dump Truck

- Capacity

Dump Bed: 12.8-cu-yd struck; max rated payload = 25 tons

Broken ore: 14 cu ft/ton = 12.8 x 27/14 = 24.7 tons/load
115 tons/24.7 = 5 loads/round

Waste: 18 cu ft/ton = 19.2 tons/load
96 tons/19.2 = 5 loads/round

Operation	Sec
• Travel	
Down 15% loaded (av 6 mph), 1,000 ft @ 8.8 ft/sec	113.6
Down 1/2% loaded (av 15 mph), 3,000 ft @ 22 ft/sec	136.4
Turn at dump	20.0
Back to dump	10.0
Raise dump	30.0
Lower dump	20.0
Return to main haulage	10.0
Up 1/2% empty (av 15 mph), 3,000 ft @ 22 ft/sec	136.4
Up 15% empty to bypass (av 4 mph), 500 ft @ 5.9 ft/sec	84.7
Delay for bypass	65.7*
Delay start	10.0
Up 15% to loadout point (av 4 mph), 500 ft @ 5.9 ft/sec	84.8
Position for loadout	15.0
	736.6 (12.3 min)

Muck and Haul Schedule for Standard Round (Truck Haulage)

The following schedule is obviously the ideal case. Because mucking and haulage cycles encounter so many varieties of delay situations in a producing operation, a real but still optimistic schedule time of 2 hours per round will be used in this study.

*When first truck arrives at bypass, second truck still has 8.9 seconds in loadout cycle and 56.8 seconds to clear bypass, resulting in 65.7 seconds dead time for first in bypass.

Operation	Min
Travel to face	5.0
Clean road and dress muckpile	10.0
Load first truck	9.5
Load second truck	9.5
Fill bucket and return to loadout point	2.7
Wait on first truck	0.1
Load first truck	6.9
Fill bucket and return to loadout	2.7
Wait on second truck	0.2
Load second truck	6.9
Fill bucket and return to loadout	2.7
Wait on first truck	0.3
Load first truck	6.9
Scale ribs and face, clean bottom	10.0
Second truck dumps and all equipment Travel to new assignment	17.0
Total	90.4 (use 2 hr)

Gardner Denver Universal III Drill Jumbo (three Drills)

The average drilling cycle for a 32-hole round (averaging 11 holes per machine) is as follows:

- Ramp Development and Stope Development Round

	Min
Tram to face, 1,000 ft @ 2.5 mph	5
Position Jumbo and set stabilizers	10
Connect services	10
Position boom, 11 times x 1 min	11
Drill equivalent 1-7/8-in.-dia hole, 83 ft x 1 min/ft*	83
Blow and wash hole, retract steel, 11 x 1 min	11
Disconnect services and rig for travel	10
Total	140 (2.3 hr, use 2.5 hr)

Cut hole, @ 3.5-in. dia = $7\text{ ft} \times \frac{3.5^2}{1.875^2}$	25 ft
32 holes, @ 1-7/8-in. = 32 ft x 7 ft	224 ft
Total	249 ft

The footage drilled that is equivalent to a 1-7/8-in.-dia drill hole will be 83 linear feet for each of the three machines on the Jumbo.

*Equivalent footage drilled in a standard round

- Two-Face Drill-out for Stope Production Shift

	Min
Tram to face	5
Position Jumbo and set stabilizers	5
Position boom, 11 x 1 min	11
Drill 83 ft equivalent 1-7/8-in.-dia hole @ 1 ft/min	83
Blow and wash hole, retract steel, 11 x 1 min	11
Position jumbo and set stabilizers	5
Position boom 11 x 1 min	11
Drill, 83 ft equivalent @ 1 ft/min	83
Blow and wash hole, retract steel, 11 x 1 min	11
	225 (3.75 hr)

With the 33-hole round, a three-boom jumbo is capable of drilling out three stope faces per shift.

Rail Haulage Cycle

Ten 10-ton-capacity cars will haul ore on the 5,900-foot level to the mill. This train will haul from both the western and southern portions of the mine. With an average one-way distance of 6,000 feet and an overall average travel speed of 10 miles per hours, the following will constitute a cycle for one ore train trip:

Operation	Sec
Travel from Dump to loading chute 6,000 ft @ 14.7 ft/sec	408
Move position under chute 10 times x 15 sec	150
Load car from chute 10 times x 90 sec	900
Travel loaded to dump 6,000 ft @ 14.7 ft/sec	408
Dump cars 10 times x 30 sec	300
	2,166 (36 min)

Since the 5,900-foot level will provide access to stoping areas as well as trackage for the ore train, adequate bypass openings and a signaling system will be required for an efficient operation. Multiple use of the level will require consideration for delays in the ore-haulage cycle.

It is therefore assumed that the train will be able to make one trip in 45 minutes and that the available operating time is 7 hours per shift. Train haulage would then account for about 1,900 tons of ore per day to the coarse ore storage at the mill. The remaining 900 tons required to achieve full production would be delivered by truck, or by an additional car on the ore train.

COMPRESSED AIR REQUIREMENTS

Compressed air requirements will be met by two Atlas Copco ER-9 reciprocating compressors (3,494-CFM standard rating @ 125 psi) and the rotary screw portable (1,200 CFM rating) presently at the mine site. The criteria are as follows (operating factor for 15 drills = 63%):

Ramp development, Jumbo 1 x 1,150 CFM x 0.63	=	725
Stope development, Jumbo 2 x 1,150 CFM x 0.63	=	1,450
Stope production, Jumbo 3 x 1,150 CFM x 0.63	=	2,175
Long-hole machines, 2 x 385 CFM x 0.63	=	485
15-hp air slusher, 4 x 100 CFM	=	400
Misc chutes, tuggers, and shop equipment	=	200
Total		5,435 CFM
Add 15% for line loss		815
Add 3%/1,000 ft el x		6,250
7,000 ft = compressor efficiency		1,313
		7,563 CFM

WATER REQUIREMENTS

Each drill requires 7.3 U.S. gallons per minute when operating, each Jumbo requires 21.9 U.S. gallons per minute.

Each compressor requires 2,100 gallons per hour cooling water; if recycled using a cooling tower arrangement, makeup should not exceed 10%.

- <u>Drill Water</u>

Ramp Development:

3 rounds/day x drill, 83 min	=	249 min
3 rounds/day x blow and wash, 11 min	=	33 min

Stope Development:

9 rounds/day x drill, 83 min	=	747 min
9 rounds/day x blow and wash, 11 min	=	99 min

Stope Production:

16 rounds/day x drill, 83 min	=	1,328 min
16 rounds/day x blow and wash, 11 min	=	176 min
		2,632 min

Daily drill water demand by Jumbos:

21.9 gal/min/Jumbo x 2,632 Jumbo min = 57,641 gal

Daily long-hole drill water demand:

2 machines x 2 shifts x 6 hr x 50 min x 7.3 gpm = 8,760 gal

- Compressor Makeup Water

2 compressors x 2 shifts x 6 hr x 85% demand x 2,100 gal/hr x 10% makeup = 4,284 gal/day

Total 70,685 gal (use 75,000 gal)

Because water resources are limited in mine area, it will be necessary to excavate a system of drainage, decantation, and clear water sumps to gather water from drilling and backfilling to be reused in the mining operation.

VENTILATION REQUIREMENTS

Arizona Mining Code 4:29 states that diesel equipment requires 150 CFM/hp. The following equipment will be used in the mining operation:

Equipment	Hp
Locomotive, 120 hp x 1 @ 50% use	60
ST-5A, 180 hp x 6	1,080
MT-F35, 25 @ 270 hp x 2	540
Universal III drill Jumbo, 66 hp x 6 @ 10% use	40
Utility truck, 52 hp x 2	104
Powder truck and cap wagon, 52 hp x 4 @ 10% use	21
Personnel carrier, 52 hp x 10 @ 20% use	104
Grader, 125 hp x 1	125
Personnel truck, 52 hp x 1 @ 50% use	26
Shop tools and welders	150
	2,250

Therefore, the estimated ventilation required is 337, 500 CFM; use 350,000 CFM to allow for efficiency factors and friction losses.

ELECTRICAL POWER REQUIREMENTS

Use	Rating (hp)
Compressors, 2 x 700 hp	1,400
Vent fans, 2 x 100 hp	200
5 x 20 hp	100
8 x 40 hp	320
Pumps, 4 x 100 hp	400
Underground shop equipment	250
Dry, office, and warehouse	100
Lighting, signals, and communications	100
Misc	100
	2,970

Peak Demand	Rating
2,970 hp x 0.746	2,216 kW
Estimated Averate Demand, 2,216 kW x 85% x 2 shifts/day x 6 hr/shift	22,603 kW-hr/day
Annual Average Usage, 22,603 kW-hr/day x 250 days	5,650,750 kW-hr/yr
Monthly usage (av)	470,896 kW-hr/mo

MINE SERVICES

COMMUNICATIONS

Two independent communication systems will be installed between the surface and undergound as it is anticipated that these will be required by federal mine safety regulations in the future.

COMPRESSED AIR

The maximum volume of compressed air would be demanded if six three-boom Jumbos and two long-hole machines were in operation during one shift. However, with this many machines at various faces, only a certain number of the total will be in operation at any one time. The compressor plant has been estimated on the basis that the actual demand for compressed air reflects an operating factor of 63% of the ideal requirements.

In order to assure delivery of the actual demand volume, a 15% line loss is considered reasonable; a 3%-per-1,000-foot increase in elevation from sea level is also considered acceptable to make up for loss in compressor efficiency.

The compressed air requirements of the mine in full production, with due consideration to the above parameters, will be about 8,000 CFM at the intake of the compressors.

The pressure requirements of the drills are of similar significance to compressor and multiple demand efficiencies. Most Jumbo-mounted drills are designed to operate most efficiently around 85 psi at the drill. In this mine, with 8,000 CFM required, 3,000 feet of 8-inch-diameter main air line, 500 feet of 6-inch-diamter feeder line to individual faces, and 100 feet of bull hose, the resultant pressure losses of the delivery system demand that compressed air leave the compressor at 125 psi.

The requirements of the system will be met by two reciprocating compressors of the Atlas Copco ER-9 type, with a standard rating of 3,494 CFM at 125 psi, and the portable rotary screw compressor at the mine site. This machine is rated at 1,200 CFM and has an operating range up to 125 psi.

EXPLOSIVES STORAGE

An approved explosives magazine will be provided by a supplier; a cap magazine will also be provided. The explosives magazine will be located in accordance with local, state, and federal regulations. A suitable excavation into the hillside at the dead end of an access road is expected to suit the purpose. This excavation will be surrounded by a 7-foot cyclone fence, tapped with three strands of barbed wire, which will be adequately grounded to earth.

The cap magazine will be located according to regulations and will be similarly fenced.

Appropriate warnings will be displayed on access roads to both magazines.

HOISTING

Emergency escape is the only function that will require a hoist. A portable fuel-driven hoist will be mounted over a ventilation raise serving the production area central to the largest number of personnel.

So as to take maximum advantage of having a portable hoisting unit on the property, the design of surface ventilation facilities dictates readily demountable fans.

PUMPING SYSTEM

To date, there is no indication that large amounts of water will be encountered with the advance of the mining operation. Effluent water will result mainly from drilling and backfill percolation. Reuse of mine water for drilling will be required because of the scarcity of water resources in the immediate area. Therefore, a series of ditches, drilled drain holes, gathering sumps, decant sumps, and clear water sumps with pumping facilities will be provided. Makeup cooling water for compressors will be minor compared with water required for drilling.

SAFETY AND TRAINING

The safety, ventilation, and training Engineer on the underground staff will have space available in the changehouse to conduct classes in these functions. This space will also be available for use by federal and state inspectors for the conducting of the various courses they offer that are applicable to mining. The whole mining operation should benefit greatly when maximum participation in these programs is encouraged.

SANITATION FACILITIES

Eight portable underground toilets will be provided. Three will be located on the 5,900-foot level and five on the 6,400-foot and above levels, as required. These units will be skid-mounted and will employ an enzyme action for disposal of waste material.

SHOP AND FUEL SUPPLY

The underground shop facility to be located on the 6,400-foot level will require the excavation of about 10,700 cubic yards of rock (about 28,750 tons). A shop facility will be located in the mine building at the 5,900-foot level portal. These shops will be outfitted with welding machines, small tools, benches, compressed air, and water for maintenance and servicing. Major overhaul and rebuilds will be done by outside contractors.

Fuel will be supplied in the mine by tank truck from a storage tank located either on the 6,400- or 5,900-foot level dump area. A lube truck will travel to the equipment work site to supply lube, grease, and hydraulic service.

MINE VENTILATION

Life support and equipment exhaust will require ventilation on the order of 500,000 CFM. Fresh air will be forced into the portals and boosted where needed. To keep the velocity within an acceptable limit, a system of ventilation raises and drift breakouts will be provided. In stoping areas, both intake and exhaust raises will be drilled to supplement the intake air from the portals.

HYDRAULIC BACKFILL

Arrangements will be made in the mill to supply the mine with hydraulic backfill on reasonable notice. Mill tailings would be run through a cyclone so that all slimes and a majority of fines can be removed, providing the mine with material of consistent particle size to ensure adequate percolation after placement.

Backfill will be piped to a yet to be determined high point in the mine where the backfill plant will be located.

Cement could be added to the top 1 foot of each filled lift to provide a hard surface for equipment travel and to reduce dilution of the ore.

It is reasonable to expect that a slurry containing 60% solids with a specific gravity of 1.68 could be deposited at a rate of 167 gallons per minute on a 24-hour-per-day basis. This would result in the placement of solids at a rate of 1,014 DST/day. The bulk density of the material after percolation will be about 140 pounds per cubic foot. Further data can be obtained from the flowsheet and material balance presented at the end of Section 4.

MINE PLANT
Equipment
Manning

MINE PLANT AND SURFACE BUILDINGS

The principal portion of the mining plant proper will be located on the 5,900-foot level dump and will consist of:

Mine office, changehouse, and mine shop

Compressor plant

Mine substation

Assay and environmental laboratory

Mine water plant

Facilities located on the 6,400-foot level dump will consist of:

Shop and warehouse

Gravel plant

In order to produce at a rate of 2,800 DST/day, the mining facility will require the major equipment listed on the following pages.

MAJOR EQUIPMENT LIST

Equipment	Qty
Wagner ST-5A Scooptram	
Development	2
Production	4
Spare	1
Wagner MT-F25-35 Articulated Rear Dump Truck	
Development	2
Three-Boom Drill Jumbo	
Development	3
Production	3
Spare	1
Motor Grader	1
Personnel Truck (MF-30 tractor w/homemade trailer)	1
Personnel Carrier (MF-30 tractor w/homemade trailer)	10
Utility Vehicle (MF-30 tractor w/homemade trailer)	2
Powder Truck (modified - used 4-wheel drive, 3/4-ton truck)	4
Air Compressor (ER-9 @ 3494 CFM)	2
(Joy Airscrew @ 1200 CFM) on site	1
Diesel Locomotive, 10-ton	1
Ore Cars, 10-ton	10
Spare	2
Mine Water Pump, 100-hp	4
Spare	2

MAJOR EQUIPMENT LIST (Contd)

Equipment	Qty
Portable Pump, 10-hp	8
Vent Fan, 100-hp	2
Spare	1
Vent Fan, 20-hp	5
Spare	1
Booster Fan, 40-hp	8
Spare	2
Longhole Drill	2
Jackleg Drill	8
Stoper Drill	8
Air Slusher	6
Bit Grinder	2
Welding Machine	3
Overhead Crane, 5-ton	2
Escape Hoist and Facility	1
Communications	Lot
Electrical and Signals	Lot
Backfill Plant	Lot
Chutes, chain gates, castings, air cylinders, wear plates, shop equipment, and small tools	As Required

MINE MANNING REQUIREMENTS SUMMARY

Staff	
Underground Production	4
Technical Services	11
Maintenance, Supply and Servicing	6
Labor	
Underground Production	68
Maintenance, Supply and Servicing	12
Total	101

MINE MANNING REQUIREMENTS

To produce 2,800 TPD will require four production stopes, four development stopes, and two development headings operating two shifts per day, and the personnel listed below.

Personnel	Men/Shift	Shifts/Day	Total
Underground Production Supervisory Staff			
Mine Superintendent	1	1	1
Shift Foreman	1	2	2
Mine Clerk	1	1	1
Subtotal			4
Technical Services Staff			
Chief Engineer	1	1	1
Mine Planning Engineer	1	1	1
Training, Safety, and Vent Engineering	1	1	1
Draftsman	1	1	1
Surveyor	2	1	2
Surveyor Helper	2	1	2

Personnel	Men/Shift	Shifts/Day	Total
Technical Services Staff (Contd)			
Chief Geologist	1	1	1
Geologist	1	1	1
Sampler	1	1	1
Subtotal			11
Underground Maintenance, Supply, and Servicing Staff			
Mechanical/Electrical Superintendent	1	1	1
Mechanical Foreman	1	2	2
Mine Warehouseman	1	1	1
Warehouse Clerk (day)	1	1	1
Warehouse Clerk (evening)	1	1	1
Subtotal			6
Underground Production Labor			
Tram Operator	1	2	2
Brakeman (chute puller)	1	2	2
Truck Operator	2	2	4
Scoop Operator	6	2	12
Miner (dayshift)	6	1	6
Miner (evening)	6	1	6
Longhole Driller	2	2	4
Helper	2	2	4
Trackman	1	1	1
Powderman	3	2	6
Helper	3	2	6

Personnel	Men/Shift	Shifts/Day	Total
Underground Production Labor (Contd)			
Road Maintenance	1	1	1
Backfill	1	2	2
Laborer	6	2	12
Subtotal			68
Maintenance, Supply, and Servicing Labor			
Lampman/Janitor	1	1	1
Drill Doctor	1	1	1
Mechanic (day)	2	1	2
Assistants (day)	2	1	2
Mechanic (evening)	2	1	2
Assistants (evening)	2	1	2
Electrician	1	1	1
Warehouse Driver	1	1	1
Subtotal			12

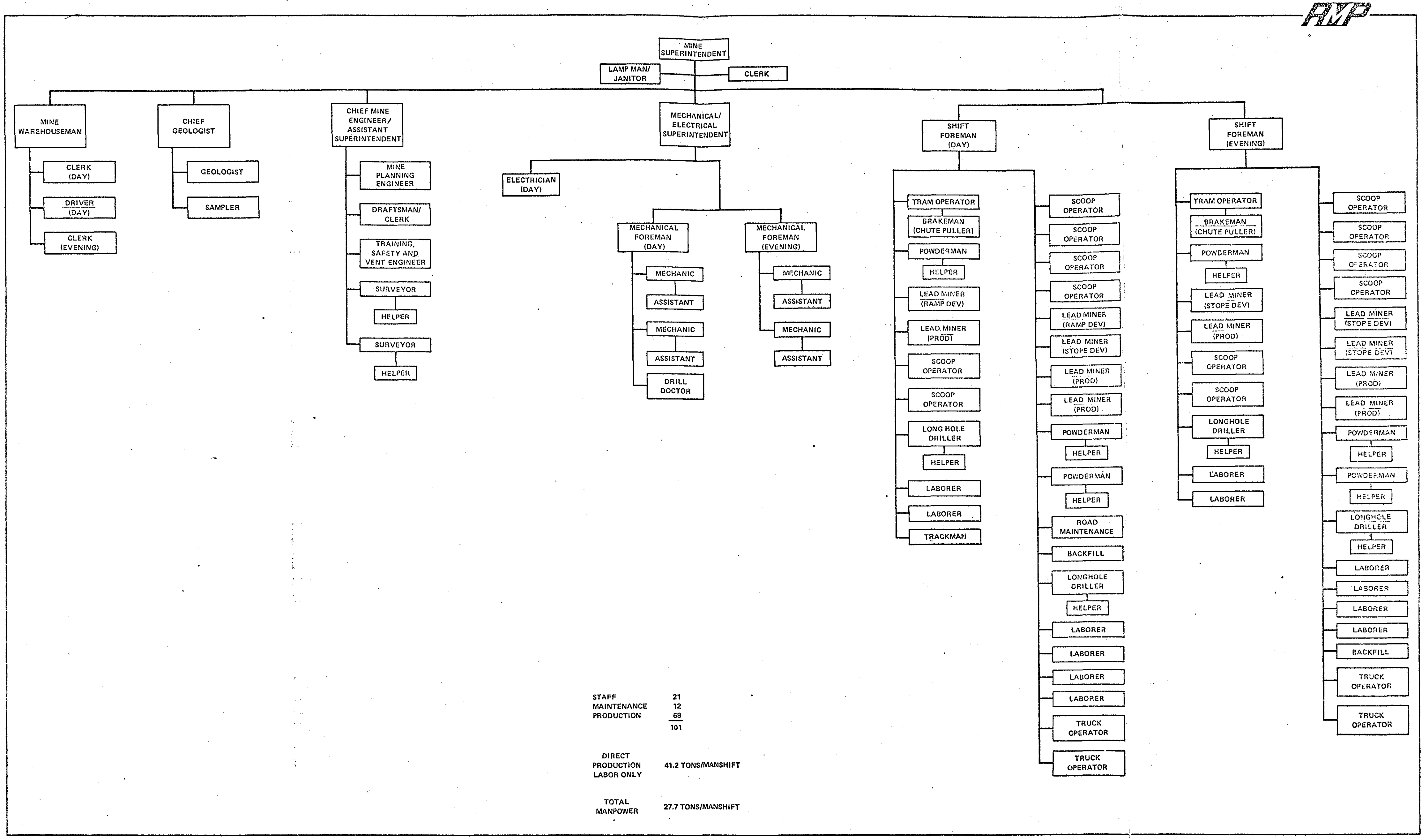


Figure 3-1 - Mine Manning Schedule for 2,800 TPD

CAPITAL COST

CAPITAL COST ESTIMATE SUMMARY - MINE

	Year 1	Year 2	Year 3
Major Mine Equipment*	$ 917,569	$1,049,223	$ 369,948
Preproduction Preparation**	657,456	1,561,705	987,999
Subtotal	$1,575,025	$2,610,928	$1,357,947
		Total	$5,543,900

*See Tables 3-3 and 3-4.

NOTE: The items detailed in Table 3-3 are not included in the capital cost estimate.

**The details for this item follow the mine equipment capital cost data.

Table 3-3 - Schedule of Expenditures by Quarter - Initial Purchase

Major Mine Equipment	Qtr 1	Qtr 2	Qtr 3	Qtr 4	Qtr 5	Qtr 6	Qtr 7	Qtr 8	Qtr 9	Qtr 10
Jumbo #1 (on site)	no cost									
Jumbo #2 (used)	45,000									
Jumbo #3 (used)			50,000							
Jumbo #4 (used)				53,000						
Jumbo #5 (used)				53,000						
Jumbo #6						98,000				
Jumbo #7 (spare)								98,000		
Air Compressor-1200 CFM (on site)	no cost									
Air Compressor-1200 CFM (rental)	7,641	7,641	7,641	7,641	7,641					
Air Compressor-1200 CFM (rental)			7,641	7,641	7,641					
Air Compressor (ER-9)				133,000						
Air Compressor (ER-9)				133,000						
Scooptram #1 (on site)	no cost									
Scooptram #2			81,000							
Scooptram #3					81,000					
Scooptram #4					81,000					
Scooptram #5										81,000
Scooptram #6										81,000
Scooptram #7 (spare)								81,000		
Custom Dump Truck (on site)	no cost									
Wagner Dump Truck #1					105,000					
Wagner Dump Truck #2					105,000					
Powder Truck #1 (used)		8,000								
Powder Truck #2 (used)				8,000						
Powder Truck #3 (used)								8,000		
Powder Truck #4 (used)										8,000
Powder Truck #5 (spare - used)										8,000
Personnel Truck	no cost									
Personnel Carrier #1		8,250								
Personnel Carrier #2				8,250						
Personnel Carrier #3				8,250						
Personnel Carrier #4					8,250					
Personnel Carrier #5						8,250				
Personnel Carrier #6							8,250			
Personnel Carrier #7							8,250			
Personnel Carrier #8								8,250		
Personnel Carrier #9									8,250	
Personnel Carrier #10									8,250	
Utility/Maintenance Vehicle #1		8,250								
Utility/Maintenance Vehicle #2					8,250					
Shop Built Trailer @ 700 ea		1,400		1,400	1,400	700	1,400	700	1,400	

Table 3-3 (Contd)

Major Mine Equipment	Qtr 1	Qtr 2	Qtr 3	Qtr 4	Qtr 5	Qtr 6	Qtr 7	Qtr 8	Qtr 9	Qtr 10
Motor Grader (used)		20,000								
10-Ton Locomotive						20,000				
10-Ton Granby Ore Cars (12 used)						24,000				
Mine Water Pump #1		10,000								
Mine Water Pump #2		10,000								
Mine Water Pump #3						10,000				
Mine Water Pump #4						10,000				
Mine Water Pump #5 (spare)										10,000
Mine Water Pump #6 (spare)										10,000
Portable Pumps @ $1,000 ea		2,000		2,000		2,000		2,000		
100-Hp Vent Fans (used @ $2,500)		5,000								
100-Hp Vent Fans (new @ $18,825)						37,650				
Spare										18,825
20-Hp Vent Fans (new @ $4,183)				8,366				8,366		4,183
Spare										4,183
40-Hp Booster Fans (new @ $3,355)		6,710		6,710		6,710		6,710		
Spare										6,710
Jackleg Drill (new @ $2,500)	5,000	5,000		5,000		5,000				
Stoper Drill (new @ $2,000)	4,000	4,000		4,000		4,000				
Longhole Drill (new @ $5,000)				5,000				5,000		
Air Slusher (new @ $4,000)	4,000	4,000				8,000				8,000
Escape Hoist (temporary)						5,000				
Bit Grinder (new @ $3,000)	3,000					3,000				
Welding Machine (new @ $2,500)	5,000									2,500
5-Ton Crane (new @ $5,000)	5,000									
Communications and Electricals	5,000					5,000				
Chutes and Furnishings				20,000			10,000	10,000	10,000	
Backfill Plant										50,000
Quarterly Total	83,641	100,251	146,282	464,258	405,182	247,310	27,900	228,026	27,900	292,401
Semi-Annual Total		183,892		610,540		652,492		255,926		320,301
Add 5% for Spare Parts, etc.		9,195		30,527		32,625		12,796		16,015
Semi-Annual Total		193,087		641,067		685,117		268,722		336,316
Annual Subtotal				834,154				953,839		336,316
Contingency @ 10%				83,415				95,384		33,632
Annual Total				917,569				1,049,223		369,948
Grand Total										2,336,740

Table 3-4 - Schedule of Additional Annual Investment with No Escalation

Equipment	Life Rating (Operating Hr)	Annual Operating Hr	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15
Jumbo #1 (Used)	7,500	3,000			98,000 (New)					98,000					98,000		
Jumbo #2 (Used)	7,500	3,000				98,000 (New)					98,000					98,000	
Jumbo #3 (Used)	7,500	3,000				98,000 (New)					98,000					98,000	
Jumbo #4 (Used)	7,500	3,000					98,000 (New)					98,000					98,000
Jumbo #5 (Used)	7,500	3,000					98,000 (New)					98,000					98,000
Jumbo #6	15,000	3,000					98,000					98,000					98,000
Jumbo #7	15,000	3,000							98,000					98,000			
Air Compressors	Life of Mine																
Scooptram #1 (Used)	7,500	3,000			81,000 (New)					81,000					81,000		
Scooptram #2	15,000	3,000					81,000					81,000					81,000
Scooptram #3	15,000	3,000						81,000					81,000				
Scooptram #4	15,000	3,000						81,000					81,000				
Scooptram #5	15,000	3,000							81,000					81,000			
Scooptram #6	15,000	3,000							81,000					81,000			
Custom Dump Truck On Site - No Information																	
Wagner Dump #1	15,000	3,000							105,000					105,000			
Wagner Dump #2	15,000	3,000							105,000					105,000			
Personnel Truck (Used)	7,500	3,000			8,250 (New)					8,250					8,250		
Personnel Carrier #1	15,000	3,000					8,250					8,250					8,250
Personnel Carrier #2	15,000	3,000						8,250					8,250				
Personnel Carrier #3	15,000	3,000						8,250					8,250				
Personnel Carrier #4	15,000	3,000							8,250					8,250			
Personnel Carrier #5	15,000	3,000							8,250					8,250			
Personnel Carrier #6	15,000	3,000							8,250					8,250			
Personnel Carrier #7	15,000	3,000							8,250					8,250			
Personnel Carrier #8	15,000	3,000							8,250					8,250			
Personnel Carrier #9	15,000	3,000								8,250					8,250		
Personnel Carrier #10	15,000	3,000								8,250					8,250		

FMP

Table 3-4 (Contd)

Equipment	Life Rating (Operating Hr)	Annual Operating Hr	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15
Powder Truck #1 (Used)	7,500	3,000			8,000 (Used)			8,000 (Used)			8,000 (Used)			8,000 (Used)			8,000 (Used)
Powder Truck #2 (Used)	7,500	3,000			8,000 (Used)			8,000 (Used)			8,000 (Used)			8,000 (Used)			8,000 (Used)
Powder Truck #3 (Used)	7,500	3,000				8,000 (Used)			8,000 (Used)			8,000 (Used)			8,000 (Used)		
Powder Truck #4 (Used)	7,500	3,000				8,000 (Used)			8,000 (Used)			8,000 (Used)			8,000 (Used)		
Shop Built Trailers	15,000	3,000						1,400	4,200	1,400			1,400	4,200	1,400		
Utility Vehicle #1	15,000	3,000						8,250					8,250				
Utility Vehicle #2	15,000	3,000							8,250					8,250			
Motor Grader (Used)	7,500	1,500					50,000 (New)										50,000 (New)
Locomotive	30,000	3,500								20,000							
Ore Cars (Used)	15,000	3,500						24,000 (Used)				24,000 (Used)				24,000 (Used)	
Mine Water Pump #1	30,000	3,000											10,000				
Mine Water Pump #2	30,000	3,000											10,000				
Mine Water Pump #3	30,000	3,000												10,000			
Mine Water Pump #4	30,000	3,000												10,000			
Vent Fan (Used)	15,000	3,000					18,825 (New)										18,825
Vent Fan (New)	30,000	3,000										18,825					
Vent Fan (New)	30,000	3,000										18,825					
Annual Subtotal					203,250	212,000	452,075	228,150	539,700	225,150	212,000	460,900	208,150	559,700	221,150	220,000	468,075
Spare Parts @ 5%					10,163	10,600	22,604	11,408	26,985	11,258	10,600	23,045	10,408	27,985	11,058	11,000	23,404
Annual Total					213,413	222,600	474,679	239,558	566,685	236,408	222,600	483,945	218,558	587,685	232,208	231,000	491,479

Grand Total 4,420,818

Total Ore Produced @ 700,000 TPY for 15 years = 10,500,000 tons

Equipment Replacement Cost = $0.42/ton

RMP

CAPITAL COST ESTIMATE - MAJOR MINE EQUIPMENT - DETAILS

Equipment	Units Reqd	Unit Price ($)	Budget Estimate ($)
ST-5 Scooptram (used on site)	1	no cost	no cost
ST-5A Scooptram	6	81,000	486,000
Three-Boom Jumbo (used on site)	1	no cost	no cost
Three-Boom Jumbo (used Hecla)	1	45,000	45,000
Three-Boom Jumbo (used Hecla)	1	50,000	50,000
Three-Boom Jumbo (used Hecla)	2	53,000	106,000
Three-Boom Jumbo	2	98,000	196,000
Longhole Drill	2	5,000	10,000
Custom Dump Truck (used on site)	1	no cost	no cost
MT-F25-35 Truck	2	105,000	210,000
Personnel Truck (used on site)	1	no cost	no cost
Powder Truck (used modified)	5	8,000	40,000
Personnel Carrier (MF-30 tractor)	10	8,250	82,500
Utility/Maintenance Vehicle (MF-30 tractor)	2	8,250	16,500
Trailer (shop built)	12	700	8,400
Air Compressor (1200 CFM on site)	1	no cost	no cost
Air Compressor (ER-9)	2	133,000	266,000
Motor Grader (used)	1	20,000	20,000
10-Ton Locomotive	1	20,000	20,000
10-Ton Ore Cars (used)	12	2,000	24,000
Mine Water Pump (100 hp)	4	100/hp	40,000
Spare	2		20,000
Portable Pump (10 hp)	8	100/hp	8,000

CAPITAL COST ESTIMATE - MAJOR MINE EQUIPMENT - DETAILS (Contd)

Equipment	Units Reqd	Unit Price ($)	Budget Estimate ($)
Vent Fan (100 hp used)	2	2,500	5,000
Vent Fan (100 hp)	2	18,825	37,650
Spare	1	18,825	18,825
Vent Fan (20 hp)	5	4,183	20,915
Spare	1	4,183	4,183
Booster Fan (40 hp)	8	3,355	26,840
Spare	2	3,355	6,710
Jackleg Drill	8	2,500	20,000
Stoper Drill	8	2,000	16,000
Air Slusher	6	4,000	24,000
Escape Hoist and Facility	1	5,000	5,000
Bit Grinder	2	3,000	6,000
Welding Machine	3	2,500	7,500
5-Ton Crane (overhead)	1	5,000	5,000
Communications and Electricals	Lot	5,000	10,000
Backfill Plant	Lot	50,000	50,000
Chutes and Furnishings	Lot	50,000	50,000
Subtotal			1,962,023
Small tools and spare parts @ 5%			98,101
Total			2,060,124

NOTE: This capital cost estimate is expended per Table 3-3.

CAPITAL COST ESTIMATE SUMMARY - PREPRODUCTION MINE PREPARATION

		Year 1	Year 2	Year 3
(1)	Raise Drilling	$ 82,500	$ 192,500	-
(2)	Equipment Rental	45,846	15,282	-
(3)	Expendable Materials	105,042	323,880	$ 236,345
(4)	Equipment Operation	93,849	289,368	211,161
(5)	Power	9,471	29,202	21,310
(6)	Surface Buildings	90,000	-	-
(7)	Payroll Supervision	65,436	201,761	147,231
	Labor	165,312	509,712	371,952
	Subtotal	$657,456	$1,561,705	$ 987,999
	Total			$3,207,160

CAPITAL COST ESTIMATE - PREPRODUCTION MINE PREPARATION - DETAILS

(1) Raise Drilling by Contractor

The contractor estimate is $150/linear ft for 60-inch-diameter upreamed raise, including pilot.

- Reamed Raises
  - Ore pass below 6,400 level dump: 500 linear ft
  - Vent raise below 6,400 level pump: 500 linear ft
  - Vent raise #1 zone @ 6,400 level to surface: 500 linear ft
- Drill-Blast Raises
  - Block 1 2 stubs @ 150 linear ft
  - Block 6 1 stub @ 50 linear ft
  - Block 8 1 stub @ 50 linear ft
  - Block 9 1 stub @ 100 linear ft

Total raise reaming required in the preproduction phase consists of 1,500 ft at a rate of $150/ft. This rate was approximated by a raise drilling contractor knowledgeable of the area. Total drill-blast raise would consist of 1,000 linear ft at an estimated cost of $50.00/linear ft.

Resultant capital cost would be $275,000. Year 1 would require 30% expenditure while 70% would be expended during Year 2 of the project.

(2) Equipment Rental

For the purposes of this report, the only rental equipment anticipated during the preproduction schedule is portable air compressors, which would be required only until the stationary compressors are in operation. Rental rates were obtained from the local (Pasadena, Ca) Joy Equipment dealer: the capital cost would be $45,846 expended in Year 1 and $15,282 expended in Year 2.

(3) Expendable Materials and Supplies

Capital cost for this item during the preproduction period is estimated on the basis of the per-ton cost derived for the production estimate and applied to the tons of development rock excavated.

Year 1:	86,100 tons x $1.22/ton	$105.042
Year 2:	265,475 tons x $1.22/ton	323,880
Year 3: (partial)	129,150 tons (ore) x $1.22/ton	157,563
	64,575 tons (waste) x $1.22/ton	78,782
	Total	$665,267

(4) Equipment Operation

This expense is prorated as in item (3) above.

Year 1:	86,100 tons x $1.09/ton	$ 93,849
Year 2:	265,475 tons x $1.09/ton	289,368
Year 3: (partial)	129,150 tons (ore) x $1.09/ton	140,774
	64,575 tons (waste) x $1.09/ton	70,387
	Total	$594,378

(5) Power

This expense is prorated as in item (3) above.

Year 1:	86,100 tons x $0.11/ton	$ 9,471
Year 2:	265,475 tons x $0.11/ton	29,202
Year 3: (partial)	129,150 tons (ore) x $0.11/ton	14,207
	64,575 tons (waste) x $0.11/ton	7,103
	Total	$59,983

(6) Surface Buildings

The building presently located in the 5,900-foot level dump at the mine parking lot has been designated as the changehouse. This structure will be enlarged to accommodate the full complement of personnel, and to serve as dry and changehouse; lamproom and mine offices; and training, safety, and first aid lecture room. In addition, it is recommended that space be considered for a small warehousing area and small shop. It is estimated this expansion would cost in the range of $40,000 and would be done during Year 1.

A 25- by 40-foot steel building located at the 6,400-foot elevation will serve as shop for routine maintenance and light repairs, bit and steel service and warehouse for mine supplies.

An additional expense under this item would be concrete foundations for compressors, pumps, etc; an allowance of $50,000 to be expended in Year 1 is estimated. The total estimated cost is $90,000.

(7) Payroll Cost

This expenditure is prorated as in item (3) above.

- Supervision

Year 1:	86,100 tons x $0.76/ton	$ 65,436
Year 2:	265,475 tons x $0.76/ton	201,761
Year 3: (partial)	129,150 tons (ore) x $0.76/ton	98,154
	64,575 tons (waste) x $0.76/ton	49,077
	Subtotal	$414,428

- Labor

Year 1:	86,100 tons x \$1.92/ton	\$ 165,312
Year 2:	265,475 tons x \$1.92/ton	509,712
Year 3: (partial)	129,150 tons (ore) x \$1.92/ton	247,968
	64,575 tons (waste) x \$1.92/ton	123,984
	Subtotal	\$1,046,976
	Total	\$1,461,404

EQUIPMENT OPERATION COST DETAILS

- Wagner Scooptram (ST-5A)

Development:

3 waste rounds x 2 hr x $9.82/hr	$ 58.92	
3 ore rounds x 2 hr x $9.82/hr	58.92	
5 ore rounds x 2.5 hr x $9.82/hr	122.75	

NOTE: With truck haulage.

Production:

16 ore rounds x 2.5 hr x $9.82/hr	392.80	
Total Daily Cost	$ 633.39	
Annual Cost = $633.39 x 250 days =		$158,347.50

- Wagner Rear Dump Truck (MT-F25-35)

Development:

3 waste rounds x 2 trucks x 2 hr x $12.27/hr	$ 147.24	
3 ore rounds x 2 trucks x 2 hr x $12.27/hr	147.24	
Total Daily Cost	$ 294.48	
Annual Cost = $294.48 x 250 days =		$ 73,620.00

- Personnel Truck (MF-30 Tractor)

1 unit x 6 hr x $2.07/hr =	$ 12.42	
Annual Cost = $12.42 x 250 days =		$ 3,105.00

- Personnel Carrier (MF-30 Tractor)

10 units x 6 hr x $2.07 =	$ 124.20	
Annual Cost = $124.20 x 250 days =		$ 31,050.00

- Utility/Maintenance Vehicle (MF-30 Tractor)

2 units x 6 hr x $2.07 =	$ 24.84	
Annual Cost = $24.84 x 250 days =		$ 6,210.00

- Road Grader

1 unit x 6 hr x $5.97	$ 35.82	
Annual Cost = $35.82 x 250 days =		$ 8,955.00

- 3-Boom Jumbo

Development:

2 units x 12 hr x $24.77/hr =	$ 594.48	

Production:

4 units x 12 hr x $24.77/hr =	$1,118.96	
Total Daily Cost	1,783.44	
Annual Cost = $1,783.44 x 250 days =		$445,860.00

- Powder Truck

4 units x 6 hr x $2.07 =	$ 49.68	
Annual Cost = $49.68 x 250 days =		$ 12,420.00

- Locomotive

1 unit x 14 hr x $2.09	$ 29.26	
Annual Cost = $29.26 x 250 days =		$ 7,315.00

- Major Electric Motor Rewind (@ $30/rated hp and 20,000 hr)

Compressors:

2 units @ 700 hp = 1400 hp

Ventilation Fans:

2 @ 100 hp = 200 hp
8 @ 40 hp = 320 hp
5 @ 20 hp = 100 hp

Mine Pumps:

4 @ 100 hp = 400 hp

Welding Machines and Overhead Crane

@ 150 hp = 150

Total hp for rewind = 2,570

Annual Cost

$2{,}570 \text{ hp} \times \$30 \times \frac{4000 \text{ hr/yr}}{20{,}000 \text{ hr}} =$	$ 15,420
Grand Total	$762,302.50
Unit Cost	$1,09/ton

OPERATING COST

OPERATING COST ESTIMATE SUMMARY - ANNUAL MINE OPERATING COST

Function	Total Cost	Unit Cost/Ton
Underground Payroll Supervision	$ 523,600	$0.76
Underground Labor	1,347,650	1.92
Equipment Operation	762,302	1.09
Expendable Materials and Supplies	853,062	1.22
Power	79,111	0.11
Total	$3,565,725	$5.10

MINE OPERATING COST ESTIMATE SUMMARY - PAYROLL

	Annual	Unit Cost/Ton
Underground Supervision		
Production Staff	$ 108,800	$0.16
Technical Staff	263,840	0.38
Services Staff	150,960	0.22
Subtotal	$ 523,600	$0.76
Underground Labor		
Production	$1,134,907	$1.62
Service	212,743	0.30
Subtotal	$1,347,650	$1.92
Total	$1,871,250	$2.68

Table 3-5 presents details of this summary.

Table 3-5 - Payroll Operating Cost Details

Salaried Personnel	Pay Level	Base Annual Pay	Classification cost/yr
Underground Production Staff			
Mine Superintendent		32,000	
Shift Foreman		20,000	
Shift Foreman		20,000	
Clerk		8,000	
Subtotal		80,000	
Plus fringes		28,800	
Subtotal		108,800	$0.16/ton
Technical Services			
Chief Mining Engineer		29,000	
Mine Planning Engineer		25,000	
Training, Safety, and Ventilation Engineer		20,000	
Draftsman		15,000	
Surveyor		15,000	
Surveyor		15,000	
Surveyor's Assistant		8,000	
Surveyor's Assistant		8,000	
Chief Geologist		27,000	
Geologist		20,000	
Sampler		12,000	
Subtotal		194,000	$0.38/ton
Plus fringes		69,840	
Subtotal		263,840	
Underground Maintenance, Supply and Servicing			
Mechanical/Electrical Superintendent		30,000	
Mechanical Foreman		20,000	
Mechanical Foreman		20,000	
Mine Warehouseman		25,000	
Warehouse Clerk (Day)		8,000	
Warehouse Clerk (Eve)		8,000	
Subtotal		111,000	$0.22/ton
Plus fringes		39,960	
Subtotal		150,960	

NOTE: Hourly rate based on Magma-Union agreement applicable 7-1-76 with no cola roll in; fringes at 36%.

Table 3-5 (Contd)

Hourly Personnel	Pay Level	Base Annual Pay	Classification cost/yr
Underground Production Labor			
Tram Operator @ 5.673 (Day)	4	11,779.84	
Tram Operator @ 5.873 (Eve)	4	12,215.84	
Brakeman @ 5.462 (Day)	3	11,360.96	
Brakeman @ 5.662 (Eve)	3	11,776.96	
Truck Operator @ 5.673 (Day)	4	11,779.84	
Truck Operator @ 5.673 (Day)	4	11,779.84	
Truck Operator @ 5.873 (Eve)	4	12,215.84	
Truck Operator @ 5.873 (Eve)	4	12,215.84	
Scoop Operator @ 5.884 (Day)	5	12,238.72	
Scoop Operator @ 5.884 (Day)	5	12,238.72	
Scoop Operator @ 5.884 (Day)	5	12,238.72	
Scoop Operator @ 5.884 (Day)	5	12,238.72	
Scoop Operator @ 5.884 (Day)	5	12,238.72	
Scoop Operator @ 5.884 (Day)	5	12,238.72	
Scoop Operator @ 6.084 (Eve)	5	12,654.72	
Scoop Operator @ 6.084 (Eve)	5	12,654.72	
Scoop Operator @ 6.084 (Eve)	5	12,654.72	
Scoop Operator @ 6.084 (Eve)	5	12,654.72	
Scoop Operator @ 6.084 (Eve)	5	12,654.72	
Scoop Operator @ 6.084 (Eve)	5	12,654.72	
Miner @ 6.095 (Day)	6	12,677.60	
Miner @ 6.095 (Day)	6	12,677.60	
Miner @ 6.095 (Day)	6	12,677.60	
Miner @ 6.095 (Day)	6	12,677.60	
Miner @ 6.095 (Day)	6	12,677.60	
Miner @ 6.095 (Day)	6	12,677.60	
Miner @ 6.295 (Eve)	6	13,093.60	
Miner @ 6.295 (Eve)	6	13,093.60	
Miner @ 6.295 (Eve)	6	13,093.60	
Miner @ 6.295 (Eve)	6	13,093.60	
Miner @ 6.295 (Eve)	6	13,093.60	
Miner @ 6.295 (Eve)	6	13,093.60	
Longhole Driller @ 6.517 (Day)	8	13,555.36	
Longhole Driller @ 6.517 (Day)	8	13,555.36	
Longhole Driller @ 6.717 (Eve)	8	13,971.36	
Longhole Driller @ 6.717 (Eve)	8	13,971.36	
Drill Helper @ 5.673 (Day)	4	11,799.84	
Drill Helper @ 5.673 (Day)	4	11,799.84	
Drill Helper @ 5.873 (Eve)	4	12,215.84	
Drill Helper @ 5.873 (Eve)	4	12,215.84	

NOTE: Based on 40-hour week x 52-week year = 2,080 hours per year.

Table 3-5 (Contd)

Hourly Personnel	Pay Level	Base Annual Pay	Classification cost/yr
Underground Production Labor (Contd)			
Trackman @ 6.306 (Day)	7	13,116.48	
Powderman @ 6.306 (Day)	7	13,116.48	
Powderman @ 6.306 (Day)	7	13,116.48	
Powderman @ 6.306 (Day)	7	13,116.48	
Powderman @ 6.506 (Eve)	7	13,532.48	
Powderman @ 6.506 (Eve)	7	13,532.48	
Powderman @ 6.506 (Eve)	7	13,532.48	
Powder Helper @ 5.462 (Day)	3	11,360.96	
Powder Helper @ 5.462 (Day)	3	11,360.96	
Powder Helper @ 5.462 (Day)	3	11,360.96	
Powder Helper @ 5.662 (Eve)	3	11,776.96	
Powder Helper @ 5.662 (Eve)	3	11,776.96	
Powder Helper @ 5.662 (Eve)	3	11,776.96	
Grader Operator @ 5.884 (Day)	5	12,238.72	
Backfill @ 5.673 (Day)	4	11,799.84	
Backfill @ 5.873 (Eve)	4	12,215.84	
Laborer @ 5.251 (Day)	2	10,922.08	
Laborer @ 5.251 (Day)	2	10,922.08	
Laborer @ 5.251 (Day)	2	10,922.08	
Laborer @ 5.251 (Day)	2	10,922.08	
Laborer @ 5.251 (Day)	2	10,922.08	
Laborer @ 5.251 (Day)	2	10,922.08	
Laborer @ 5.451 (Eve)	2	11,338.08	
Laborer @ 5.451 (Eve)	2	11,338.08	
Laborer @ 5.451 (Eve)	2	11,338.08	
Laborer @ 5.451 (Eve)	2	11,338.08	
Laborer @ 5.451 (Eve)	2	11,338.08	
Laborer @ 5.451 (Eve)	2	11,338.08	
Subtotal		834,490.08	
Plus fringes		300,416.43	
Subtotal		1,134,906.51	$1.62/ton
Maintenance, Supply and Servicing Labor			
Lampman @ 5.462	3	11,360.96	
Drill Doctor @ 6.939	10	14,433.12	
Mechanic @ 6.728 (Day)	9	13,994.24	
Mechanic @ 6.728 (Day)	9	13,994.24	

Table 3-5 (Contd)

Hourly Personnel	Pay Level	Base Annual Pay	Classification cost/yr
Maintenance, Supply and Servicing Labor (Contd)			
Assistant @ 5.673 (Day)	4	11,799.84	
Assistant @ 5.673 (Day)	4	11,799.84	
Mechanic @ 6.928 (Eve)	9	14,410.24	
Mechanic @ 6.928 (Eve)	9	14,410.24	
Assistant @ 5.873 (Eve)	4	12,215.84	
Assistant @ 5.873 (Eve)	4	12,215.84	
Electrician @ 6.939 (Day)	10	14,433.12	
Whse. Driver @ 5.462	3	11,360.96	
Subtotal		156,428.48	
Plus fringes		56,314.25	
Subtotal		212,742.73	$0.30/ton
Total		1,375,918.56	
Plus total fringes		495,330.68	
Grand Total		1,871,249.24	$2.68/ton

EXPENDABLE MATERIALS AND SUPPLIES

Item	Cost
Air Compressors	
1% of purchase @ $266,000	$ 2,660
Ventilation Fans	
3% of purchase @ $100,000	3,000
Pumps	
3% of purchase @ $40,000	1,200
Slusher Cable	
4 units x 100 ft/unit/mo x 12 mo x $0.90	4,320
Communications	
$100/mo x 12 mo	1,200
Cap Lamps, Safety, and First Aid	
(rental, maintenance, and parts) 120 men x $2.00/man/mo x 12 mo	2,880
Shop Equipment and Small Tools	
$1,000/mo x 12 mo	12,000
Electrical Supplies	
$100/mo x 12 mo	1,200
Railroad Track, Switches, Ties etc	
5% of 20,000-ft 60-lb rail @ $0.25/lb	5,000
Ventilation Tubing	
3,000 linear ft @ $4.00/linear ft	12,000
Pipelines	
Air 8-in. dia	
3000 linear ft @ $12.45/linear ft	37,350

EXPENDABLE MATERIALS AND SUPPLIES (Contd)

Item		Cost
Water (2-in. dia)		
3000 linear ft @ $1.70/linear ft		$ 5,100
Pump (4-in. dia)		
3000 linear ft @ $4.90/linear ft		14,700
Backfill (6-in. dia)		
3000 linear ft @ $8.40/linear ft		25,200
Bits and Steel*		252,113
Explosives*		291,870
Diamond Drilling @ $0.10/ton*		70,000
	Subtotal	$741,793
	Contingency and Miscellaneous @ 15%	$111,269
	Total	$853,062
	Unit Cost	$1.22/Ton

*Details of these cost items are presented in Volume II.

SECTION 4

CRUSHING, GRINDING, AND CONCENTRATION

The plant design is based on the criteria for design capacity outlined in detail in this section. The crushing section operates two 8-hour shifts per day, 5 days per week, at a rate of 250 DST/day. The concentrator operates three 8-hour shifts per day, 7 days per week, 350 days per year, treating 2,105.3 DST/day.

CONCENTRATOR - METALLURGICAL DESIGN

METALLURGICAL INVESTIGATIONS

The metallurgy used in this study is based upon the results of the laboratory investigations conducted by Mr. G. W. Bossard at the facilities of Hazen Research Laboratory in Tucson. The report on this test work is included in Volume II. The basic flowsheet used by Continental Materials in its preliminary feasibility study has been expanded to include additional metallurgical data and design information.

The samples used in the test work were split from drill cores taken from seven areas in the ore body. These samples were tested both individually and in the form of a composite sample. Although the specific compositions of these samples may not be representative of the ore that will be processed over any specified period of time, Parsons considers the test results to be indicative of the ore's amenability to concentration; thus, the samples can be used as the basis for process design.

The investigations revealed the principal copper-bearing ore minerals to be chalcopyrite, bornite, and chalcocite, with lesser amounts of covellite. These sulfide minerals can be effectively recovered by conventional flotation procedures. Because of the high copper content of the bornite and chalcocite, the concentrate will be of a substantially higher grade than that produced from the more usual chalcopyrite concentrate containing the same relative amounts of gangue impurities.

The gangue minerals of the ore that affect process design include the talcose- and chloritic-type minerals, which tend to produce slimes on grinding. Because the process reuses the water, the tailing thickener design must be adequate to ensure a clear overflow. Also, the presence of these minerals requires special care in the design of the hydrocyclone classification system for the production of a tailing sand product. The design must provide an effective separation of slimes from sand to produce mine fill material that will drain when in place.

The investigations indicated that appreciable quantities of magnetite also occur in the ore. This mineral precludes the effective use of protective belt magnets for the removal of tramp iron in the crushing circuit, and necessitates the installation of a metal detector to stop the belts when damaging metal pieces are detected in the system.

The major part of the metallurgical investigations involved flotation test work. This work covered the effect of fineness of the primary grind on recovery, the fineness of grind required for upgrading the rougher concentrate, reagent selection, pulp retention times for flotation, and the pH of the pulp. A metallurgical balance has been prepared on the basis of these test results.

The investigations of fineness of the primary grind versus copper recovery in the rougher flotation operation indicated that a particle size of 65% minus 200 mesh is the minimum grind required for an acceptable copper recovery. The investigations, however, have not been extended to show the increase in recovery that might be expected at finer grinds. The grinding test results strongly indicate that a finer primary grind will be of economic benefit. Accordingly, Parsons has included enough grinding capacity in the equipment specified in this report to reduce the particle size of the rougher flotation feed approximately to 80% minus 200 mesh. This additional grinding capacity will help to assure that the copper recovery indicated in the metallurgical balance can be attained or improved without the production of enough fine tailing material to cause serious difficulties in drainage of the mine fill material.

Based upon the types of copper minerals occurring in the ore, calculations indicate that an ultimate grade of over 50% Cu in the final product could be attained, assuming a fine enough grind is used to liberate the gangue from the sulfide minerals. In normal concentrator operations it does not appear economical, however, to produce an exceptionally high-grade product because of the excessive copper losses likely to occur in the cleaning circuits. The laboratory investigation of rougher concentrate regrind and cleaning have shown approximate relationships of fineness of grind, grades of concentrate, and recoveries of copper in the cleaner circuits. From these investigations, the amount of copper recovered from the reground rougher concentrate by the cleaner circuits is estimated at 95% when a final concentrate containing 32% Cu is produced. These quantities, determined by Parsons to be conservative, have been used in the development of the metallurgical balance for the overall process.

Since substantially all sulfides present in the Oracle Ridge ore are copper-bearing minerals, the more active bulk flotation-type reagents can be effectively utilized. The reagent system determined by the laboratory test work is essentially a conventional combination of frothers, a strong collector (amyl xanthate), and lime for solution pH control. Because of the strong activation of the sulfides at the start of the float when amyl xanthate is used, difficulties were encountered with overflocculation and discharge of the mineral particles from the flotation equipment. This condition was avoided by the use of Aerofloat 238, a less active collector, during the first part of the rougher flotation operation. Using this procedure, the major part of the more easily floated material can be removed before the addition of the amyl xanthate to float the middling particles and other slower floating sulfides.

The laboratory investigations included the measurement of the work index of the Oracle Ridge ore. The index is used for the determination of power requirements to grind the ore to the fineness required for the liberation of the mineral values. This measurement made by Allis Chalmers Company on an ore composite sample indicated the work index to be 10.9. Since a composite sample was used, this index should be considered an average value and may not indicate power requirements for grinding mill feed at any specified time. Parsons used the work index of 10.9 in developing the grinding equipment sizes used in this study. However, since ore from various underground areas has compositions that vary greatly in proportions of hard and soft minerals, and since an acceptable copper recovery depends upon an adequate grind, it is recommended that work indexes be determined on individual samples representing these areas. The impact of any differences in these indexes on metallurgical results should be evaluated before the size and power requirements of the grinding equipment is finalized.

Investigations were also made on settling rates both for concentrate and tailing. We believe these data are not reliable enough for the determination of thickener sizes. Therefore, an area of 8 square feet per ton per day, which is roughly the average for Arizona concentrators, has been arbitrarily used in the calculations of the thickener sizes shown in the equipment list. A clear thickener overflow must be assured since this water is to be returned to the process for reuse.

METALLURGICAL BALANCE

The overall metallurgical balance for the Oracle Ridge concentration process shown in this subsection has been developed from the results of the metallurgical test work. This balance serves as the basis for the economic evaluation of the process and as the starting point for the development of the materials balance tabulated on the process flowsheet. The materials balance is required for the engineering design of the concentration plant.

Table 4-1 shows the annual designed feed rate of ore to the concentrator and the estimated weights of concentrate and tailing that will be produced by the operation. Estimates of copper, gold, and silver average analyses are shown for the feed and products, and a balance of the metal contents of these materials is tabulated. The distribution of gold and silver shown on the table has been estimated from assays of the cleaner products reported from the laboratory investigations.

On the basis of the laboratory test work, it is expected that the copper that will be recovered from the ore by the rougher flotation circuits will be between 90% and 91%. At a final concentrate grade of 32% copper, it is estimated the recovery of copper by the cleaner circuits will be 95%. Combining these recoveries, the net recovery of copper for the complete process will be approximately 86%. This figure has been used for the metallurgical balance.

Table 4-1 - Metallurgical Balance

	Annual Rate Dry Short Tons	Assays				Metal Content			Metal Distribution		
		% Wt	% Cu	Au $\frac{Oz}{T}$	Ag $\frac{Oz}{T}$	Cu-Tons	Au-Oz	Ag-Oz	% Cu	% Au	% Ag
Heading	700,000	100.0	1.70	0.007	0.40	11,900	5,202	280,000	100.0	100.0	100.0
Concentrate	31,981	4.6	32.0	0.10	7.30	10,234	3,198	233,800	86.0	61.5	83.5
Tailing	668,019	95.4	0.27	0.003	0.07	1,666	2,004	46,200	14.0	38.5	16.5

CRITERIA FOR DESIGN CAPACITY OF CONCENTRATION FACILITIES

The design capacities of the crushing plant and concentrator are based upon the following criteria specified by Continental.

Total Facilities

(1) The annual ore treatment capacity: 700,000 DST/day.

(2) The operating schedule per year: 350 days, with 15 days allotted to scheduled maintenance and vacations.

Concentrator

(1) The average percent operating time (availability): 95%.

(2) The operating schedule: 7 days per week with three 8-hour shifts per day.

Crushing Plant

(1) The operating schedule: 5 days per week with two 6-1/2-hour shifts per day.

(2) The allowance for unscheduled shutdown: 6 hours per week.

The designed ore treatment rate for the concentrator based upon the above criteria is 2,105.3 DST/day, or 14,737 DST/wk, as follows:

$$\frac{700{,}000 \text{ DST/yr}}{350 \text{ days} \times 0.95} = 2{,}105.3 \text{ DST/day}$$

$$2{,}105.3 \times 7 \text{ days} = 14{,}737 \text{ DST/wk}$$

The designed crushing rate based upon the above criteria is 250 DST/hr, as follows:

Operating Time - 5 days x 2 shifts x 6-1/2 hours = 65 hr/wk

Design Factor - To compensate for unscheduled shutdowns of 6 hours per week:

$$\frac{65 \text{ hr}}{65\text{-}6 \text{ hr}} = 1.10$$

Designed Crushing Rate - To maintain concentrator ore treatment rate:

$$\frac{14{,}737 \text{ DST/wk}}{65 \text{ hr}} \times 1.10 = 250 \text{ DST/hr}$$

CONCENTRATOR — PROCESS DESCRIPTION

CRUSHING AND SCREENING

The secondary crushing and screening facilities reduce the ore to the final particle size acceptable for feed to the grinding circuits. The principal equipment provided for these facilities includes a 7-foot short head cone crusher, a double-deck vibrating screen fitted with a 1-inch rod deck and 1/2-inch tyrod screen cloth, and a scissors conveyor. This equipment operates in closed circuit and is designed to handle a circulating load of approximately 200% weight of new feed to the system. The screen is located where the oversize product can be discharged directly into the crusher. The screen undersize is conveyed to the fine ore storage area, where it is distributed on two conical piles that supply the feed for the two grinding mills.

FINE ORE STORAGE

The fine ore storage facility must have about a 2-1/2-day-ore-supply surge capacity since the grinding circuits require an uninterrupted flow of ore on a 7-day-week, 24-hour-day schedule, and the crushing plant operates 5 days per week, 13 hours per day. The fine ore storage facility is designed to provide volume for a live load of ore to the feeders of 5,500 DST to accommodate these differences in operating times. The ore is metered to the grinding circuit by means of belt feeders fitted with variable speed drives controlled by belt weightometers.

PRIMARY GRINDING

The primary grinding and classification circuits are designed to reduce the crushed ore from 80% minus 1/2 inch to 80% minus 200 mesh particle size. At this size over 90% of the copper sulfide minerals can be collected by flotation in a low grade rougher concentrate. The grinding facilities are divided into two parallel circuits, each of which is provided with a 10-foot-diameter by 14-foot ball mill, together with a hydrocyclone classifier and a circulating slurry pump connected in closed circuit. Because of the high reduction ratio (i.e., 1/2-inch feed to 200 mesh product), a circulating load amounting to 400% weight of new feed is maintained in the closed circuit. The ball mill slurry contains 75% solids, and the final ground product is delivered to the rougher flotation circuits in a slurry containing 30% solids.

ROUGHER FLOTATION

The rougher flotation section of the plant is divided into two parallel circuits, each of which contains fourteen 60-cubic-foot flotation machines connected in series. The feed for these circuits is the combined flow of pulp from the two grinding mills. This pulp flow is redivided by means of a motor-driven pulp splitter into two equal streams feeding the two lines of rougher flotation cells. Twelve minutes of retention time is provided in the rougher circuits. The rougher concentrate product is pumped directly to a regrind circuit in preparation for the concentrate cleaning operations. The rougher tailing is sent either to a hydrocyclone classification system for the production of mine fill material when required, or directly to the tailing thickener for dewatering and disposal.

ROUGHER CONCENTRATE REGRIND

The concentrate regrind facility contains a hydrocyclone classifier, ball mill and slurry pump operating in a closed circuit system. The feed to the system is composed of the rougher concentrate and a cleaner-scavenger concentrate returning from the cleaning circuits for further grinding. The concentrates from both sources enter the regrind system at the slurry pump feed box where they are pumped to the hydrocyclone for classification before entering the regrind mill. With this arrangement, the tendency for overgrinding the finer particles is reduced. The regrind facility is designed to reduce the particle size of the concentrate from 80% minus 200 mesh to 90% minus 325 mesh.

CLEANER FLOTATION

The concentrate cleaning facility is divided into two identical lines operating in parallel. These two lines are fed from the regrind hydrocyclone overflow product that is divided into two equal streams by a motor-driven pulp splitter. Two stages of cleaning are provided for the production of a concentrate suitable for shipment and sale. A cleaner-scavenger circuit is also provided for the recovery of low grade middling particles and other copper bearing minerals that escape collection in the cleaning operation. The concentrate from the cleaner-scavenger circuits is returned for further grinding in the regrind facility. The tailing from this circuit is considered free from recoverable copper and is discharged to waste. Twenty-six 22.5-cubic-foot cells provide the flotation capacity for the two lines of the complete facility. A retention time of 8 minutes is provided for the first cleaning stage, 7-1/2 minutes for the recleaning stage, and 18 minutes for the cleaner-scavenger operation.

CONCENTRATE DEWATERING AND SHIPMENT

The final concentrate product discharged from the recleaner cells is pumped to a 35-foot-diameter concentrate thickener where it is settled to about 50% solids. The thickener underflow is then pumped to a 6-foot-diameter by 6-disc filter for final dewatering approximately to 12% moisture. The concentrate filter cake is discharged by gravity to a storage area directly below the filter. The concentrate is recovered from storage and loaded into trucks for shipment by use of a front end loader. The filtrate from the disc filter is returned to the thickener and the thickener overflow water is pumped to the head tank for reuse by the process.

TAILING DISPOSAL

The tailings derived from the rougher flotation and the cleaner-scavanger operations flow by gravity to two 100-foot diameter thickeners for partial dewatering to 50% solids in preparation for delivery by pipeline to the tailing disposal area. Arrangements are provided to bypass the rougher flotation tailing to hydrocyclone classification equipment for the production of mine sandfill material. The fine material removed by the classifiers is returned to the tailing thickener. The thickener overflow is returned to the head tank for reuse.

REAGENTS

The reagents used for the process consist of two collectors, a frother and lime for pH control of the flotation pulp. The first of the two collectors, Aerofloat No. 238, is used early in the rougher flotation operation to recover a large part of the more easily floatable minerals. The second collector, amyl xanthate, is used for the final collection and recovery of the remaining sulfides, particularly the slower floating minerals and middling particles. The test work has indicated that excessive flocculation of the sulfides with attending recovery problems occurs when only amyl xanthate is used. The frother used in the process is a 50:50 mixture of pine oil and Dowfroth No. 250. Clarkson feeders supplied with the above reagents (except lime) from small day tanks on the reagent gallery are provided to feed the various points indicated on the flowsheet. Reagent mixing and solution storage for a 30-hour supply of reagents are provided in a reagent preparation area.

The reagent preparation area also includes a facility for burned lime storage, slaking and 30-hour holding tanks for the 20% hydrated lime slurry used by the process. A separate flowsheet with a material balance for the slaking process is included in this section of the report. The lime slurry is recirculated continuously through a closed loop to the addition points in the primary and regrind mill circuits. The lime circulating in the loop is withdrawn as required by means of solenoid valves controlled by timers.

SAMPLING

Stations are provided for sampling the pulp streams representing the total feed to the concentrator, the total concentrate produced and the total tailing leaving the plant. The analyses of these samples will provide information on the overall metallurgical performance of the concentrator and the grade of the final concentrate product for shipment. For the control of the individual operations within the process, provisions have also been made for intermediate product sampling at the following points:

1. The rougher concentrate delivered to the regrind circuit.

2. The cleaner-scavenger tailing returned to the regrind circuit.

3. Feed to the cleaner circuits.

INSTRUMENTATION

Only limited provision has been made for the use of instrumentation in process control. It has been assumed that a manually operated pH meter can be utilized to determine the lime feed rate, since precision is not required in the rate of addition of this reagent. The percentage of solids in pulp may be determined at any point in the system by use of specific gravity balances. Flowrates may be estimated from the known feed rates of the ore, water, and these specific gravity measurements. The feed rate of the raw ore to the grinding circuits is regulated by belt weightometers with a feed-back circuit controlling the speed of the belt feeders. A truck scale is provided for the weighout of the final concentrate product.

An electrical interlock system is provided for the crushing plant area to ensure starting and stopping the equipment in a sequential order that will prevent spills or flooding. Also, in the event that a piece of equipment becomes inoperative, the interlock system will automatically shut down all equipment in the circuits ahead of it.

SAND PLANT FOR MINE FILL

Two stages of classification using hydrocyclones are provided for the preparation of mine fill material from the rougher flotation tailing. The two stages were chosen to ensure the removal of enough of the talc, serpentine, chlorite, and other slime-producing components of the tailing to obtain a porous sand product that will drain when in place underground. It is possible that one stage will be found adequate for the separation of a suitable mine fill material, but large-scale tests are needed for confirmation. In this regard, it is suggested that, for plant startup only one stage of classification be installed and tested for sand product characteristics under actual operating conditions. If the sand product is not found to be porous enough for use as mine fill, a second stage could then be provided. With these facilities, provision can also be made to relieve the load on the thickener mechanisms by continuing to operate one stage of classification even when mine fill material is not required. In this case, the sand product would bypass the thickener and be conducted directly to the tailing disposal line.

The flowsheet for the mine-fill facility is presented in this section. A rough estimate of the material balance, flowrates, and approximate screen analyses of the products are tabulated on the flowsheet.

PROCESS CRITERIA AND DESIGN DATA

- Run-of-Mine Ore

Bulk Density	136 lb/cu ft
Specific Gravity	3.13
Particle Size Distribution:	
Plus 12"	2% wt
Minus 12" plus 4"	48% wt
Minus 4" plus 3"	12% wt
Minus 3" plus 1-1/2"	16% wt
Minus 1-1/2" plus 1"	5% wt
Minus 1" plus 5/8"	4% wt
Minus 5/8" plus 1/2"	2% wt
Minus 1/2" plus 0"	11% wt

- Coarse Ore Storage

Minimum live-load capacity	1,250 DST
Feed rate from storage to crushing plant	250 DST/hr

- Crushing Plant Operating Schedule

5 days/wk
2 shifts/day
6-1/2 hr/shift

- Primary Crusher and Grizzly

Grizzly rod spacing	3"
Grizzly undersize product-% wt of feed	36%
Crusher setting-maximum	4" OSS
Combined crusher and grizzly products, particle size distribution:	
Plus 4"	2.0% wt
Minus 4" plus 3"	15.0% wt
Minus 3" plus 1-1/2"	34.0% wt
Minus 1-1/2" plus 1"	12.5% wt
Minus 1" plus 5/8"	10.0% wt
Minus 5/8" plus 1/2"	4.0% wt
Minus 1/2" plus 0"	22.5% wt

- Secondary Crusher and Screen

Circulating load through Crusher	220% of feed
Cone crusher setting	5/8" CSS
Reduction ratio - approximate	6:1
Screen cloth:	
Top deck, rod spacing	1" opening
Bottom deck, Tyrod spacing	1/2" slots
Final product - particle size distribution:	
Plus 5/8"	13% wt
Minus 5/8" plus 1/2"	7% wt
Minus 1/2" plus 0"	80% wt

- Fine Ore Storage

Minimum live-load capacity	5,500 DST
Bulk density of ore	136 lb/cu/ft
Moisture content	5%
Angle of repose	60 deg

- Grinding and Concentration Facilities

Operating Schedule	350 days/yr 7 days/wk 3 shift/day 8 hr/shift
Availability (operating time as percent of total available time)	95%

- Primary Grinding

Total ore feed rate	2,105.4 DST/day
Feed rate to each line	1,052.7 DST/day
Sp gr solids	3.13
Solids in pulp in mill charge	75%
Circulating load as percentage of new feed	400%
Work index of ore (Wi)	10.9
Feed - screen size passing 80% wt	1/2"
Product - screen size passing 80% wt	200 mesh
Screen analysis of ground product:	
Plus 100 mesh	3.6% wt
Minus 100 mesh - plus 150 mesh	6.4% wt
Minus 150 mesh - plus 200 mesh	10.0% wt
Minus 200 mesh - plus 270 mesh	11.5% wt
Minus 270 mesh - plus 325 mesh	6.5% wt
Minus 325 mesh - plus 400 mesh	5.0% wt
Minus 400 mesh - -	57.0% wt

- Rougher Flotation

Flotation retention time	12 min
Percentage of solids in pulp	30%
Estimated average percentage of solids in froth at lip of cells	30%

- Concentrate Regrind

Feed rate - combined rougher and cleaner-scavenger concentrate	232 DST/day
Sp gr solids	3.43
Solids in pulp in mill charge	60%
Circulating load as percentage of feed to regrind circuit	300%
Assumed Wi of solids	10.9
Product - screen size passing 80% wt	325 mesh

- Cleaner Flotation

Cleaner flotation retention time	8 min
Cleaner-scavenger retention time	18 min
Recleaner flotation retention time	7.5 min
Solids in pulp feed to cleaner circuits	20%

- Concentrate Thickener

Feed rate	96.2 DST/day
Area factor	8 sq ft/ton/day
Underflow - percentage of solids	50%

- Concentrate Filter

Feed rate	96.2 DST/day
Filter rate	30 lb/sq ft/hr
Moisture content of filter cake	12%

- Tailing Thickener

Feed rate	2,009.2 DST/day
Area factor	8 sq ft/ton/day
Underflow - percentage of solids	50%

- Sand Plant for Mine Fill

Rougher tailing feed rate	1,895.4 DST/day
Sand production rate	1,013.5 DST/day
Slimes to waste	881.9 DST/day
Sp gr solids	3.10

Sand Plant for Mine Fill (contd)

Particle size distribution:

Sands product -	plus 100 mesh	6.7% wt
	minus 100 plus 150 mesh	10.2% wt
	minus 150 plus 200 mesh	14.3% wt
	minus 200 plus 270 mesh	14.1% wt
	minus 270 plus 325 mesh	7.4% wt
	minus 325 plus 400 mesh	4.6% wt
	minus 400 mesh	42.7% wt

PROCESS WATER BALANCE

The process water makeup, or new water required to maintain the water balance for the concentration facility, is estimated at 211.2 gpm. The basis for this estimate is shown in Table 4-2. The water recovered from the tailing pond and returned to the process is estimated at 167.4 gpm; this recovered water is not considered as part of the process makeup.

At the full concentrator production rate, the water sent to the pond in the tailing slurry amounts to 334.9 gpm. In estimating the amount that can be returned from the pond, it is assumed that 40% of the volume of the settled and compacted material in the pond is water contained in the voids between particles of solids and is unrecoverable and is lost at the rate of 71.8 gpm. Of the remaining 263.1 gpm, the amount that can be recovered as clear, reusable water is estimated at about 64%, or 167.4 gpm; the remainder is lost by evaporation and seepage from the pond.

Table 4-2 - Water Balance

Item	Water In		Water Out	
	ST/day	gpm	ST/day	gpm
Water contained in ore (at 5% moisture)	110.8	18.5	-	-
Return from tailing pond	1,004.6	167.4	-	-
Process makeup	1,266.9	211.2	-	-
Water in concentrate product	-	-	13.1	2.2
Water in tailing slurry to pond	-	-	2,009.2	334.9
Assumed evaporation and spillage	-	-	120.0	20.0
Washdown and miscellaneous usage	-	-	240.0	40.0
Balance	2,382.3	397.1	2,382.3	397.1

PLANT DESCRIPTION

The location and general arrangement of the mill buildings were suggested by CMC to minimize excavation and to make optimum use of the terrain of the site. The plot plan was developed in detail by Parsons.

The coarse ore is delivered to its storage pile by trains of 10-ton Granby-type side-dump ore cars, the delivery point is located on a trestle extending over the coarse ore storage area at the 5,900-foot level. The location of the trestle, coarse ore storage, crushing plant, and concentrator are shown on Drawing 5382-1-4, Mill Site Plan, and Drawing 5382-1-10, Map of Mine and Mill.

Delivery of the coarse ore by dumping from the trestle over the reclaim feeder and conveyor, rather than down the hillside from the 5,900-foot level, has increased the live coarse ore storage by a factor of 2.6, with only about a 4% increase in total coarse ore storage. The coarse ore pile provides a total storage capacity of 7,300 tons, with 1,200 tons of live storage. In addition, the necessity for a 35- to 50-foot-high retaining wall along the 5,900-foot level is eliminated by use of the trestle.

The height of the scissors conveyor tower has been minimized by the use of the terrain, as has the total rise necessary to feed the double-deck screen above the 7-foot shorthead cone crusher.

The following ancillary equipment is included in the crushing plant:

(1) A 20-ton bridge crane for servicing equipment.

(2) Dust collection ducting to conveyor drop points, the vibrating screen, and cone crusher.

(3) A separate wet-type dust collection system for the coarse ore apron feeder. Recovered dust slurry is pumped to the head end of the mill circuit.

(4) Operating controls and instrumentation in the plant office immediately above the electrical equipment room. The control equipment includes an electrical interlock system for sequential equipment startup or shutdown. However, should any equipment become inoperative, the interlock system will also automatically shut down all equipment in the circuits ahead of it.

The fine ore storage building and the concentrator are at right angles to the crushing plant. Fitting the buildings to the terrain with minimal excavation permits almost horizontal conveying from the crushing plant to fine ore storage, and from there to the concentrator. This arrangement also permits in-line operation in the concentrator, and the concentrate discharges at the northeast end of the building adjacent to the main road.

The location of the reagent building and the mine fill plant are shown on both Drawings 5382-1-4 and 5382-1-10. The latter also shows the location of the point of delivery of the mine fill, at the 6,400-foot level, and the approximate location of the pipeline from the mine fill plant to the portal at 6,400 feet. Drawing 5382-1-10 also shows the relationship of the locations of the process water storage stope, the head tank, and the connecting piping. The details of the freshwater pumping system and the tailing water reclaim pumping system are presented in Section 5.

The principal equipment in the reagent building are the reagent mixing and 30-hour storage tanks, the burned lime storage, the 3-foot-2-inch-diameter by 5-foot slaking ball mill, and the 30-hour slaked-lime storage tank. The lime slurry is pumped continuously through a closed loop to the addition points in the primary and regrind mill circuits. The reagents are transferred by gravity flow to day tanks on the reagent platform in the plant at elevation 5,807 feet. Addition of reagents to the appropriate points in the circuits is by Clarkson feeders. The reagent supplies in the day tanks in the concentrator are maintained by float-controlled valves.

The mine fill plant is located adjacent to the two 100-foot tailing thickeners. A bypass line permits the entire plant tailing to be transferred either to the mine fill plant or to the thickeners. When the mine fill plant is in operation, the overflow from the hydrocyclones, primarily slimes and material unacceptable for fill, goes to the thickeners. The piping is also arranged so that the first tank and hydrocyclone can be used to lighten the load on the thickener mechanisms. In this case, the hydrocyclone underflow sand is conducted directly to the tailing disposal line.

CONCENTRATOR EQUIPMENT LIST

CONCENTRATOR EQUIPMENT LIST

Identification	Qty	Description	HP	Remarks
Crusher				
A	1	Coarse ore storage 1200 tons live 7300 tons total		
B	1	Apron feeder 42" x 17', 250 DST/hr	15	
C	1	Coarse ore conveyor 24" x 104'	15	
D	1	Vibrating Grizzly 3' x 8'	10	
E	1	Primary jaw crusher 32" x 42", 159.7 DST/hr	150	
F	1	Shorthead cone crusher 7'	300	
G	1	Double-deck screen, 6' x 16', 803.3 DST/hr	30	
H	1	Metal detector		
I	1	Scissors conveyor, 803.3 DST/hr 36" x 152' 36" x 130'	 40 50	
J	1	Belt scale		
K,L	1	Fine ore conveyor with tripper 24" x 141', 250 DST/hr	10	
		Dust Collection Equipment for Crushing Plant		
	1	20-ton bridge crane		
M	1	Fine ore storage 5,500 tons live		
N	6	Fine ore reclaim feeders 36" x 22', 14.6 DST/hr	20 each	
O	2	Mill feed conveyors 24" x 174', 43.8 DST/hr	5	

CONCENTRATOR EQUIPMENT LIST (Contd)

Identification	Qty	Description	HP	Remarks
P	1	Process water head tank 30' dia x 32', 150,000 gal		
Q	2	Ball mills, 10' ID liners x 14', 219 DST/hr each	750 each	
R	4	Hydrocyclones, 15" dia		
S	4	Slurry pumps with sump box, 924 gpm each	40 each	2 spares
T	2	Slurry pumps with sump box, 928 gpm each	30 each	1 spare
U	1	Splitter box, motor-driven	1	
V	1	Flotation feed sampler		
W	2 rows	14-cell rougher flotation machines, 60 cu ft/cell	105 each	
X	2	Submerged rougher concentrate pump with tank, 64 gpm	3 each	1 spare
Y	2	Submerged rougher concentrate pump with tank, 52 gpm	3 each	1 spare
Z	1	Rougher tails sampler		
AA	1	Rougher concentrate sampler		
AB	2	Regrind mill pump	10 each	1 spare
AC	1	Regrind ball mill 5'6"-ID liners x 12'	150	
AD	1	Hydrocyclone, 10" dia		
AE	1	Cleaner feed sampler		
AF	1	Splitter box, motor-driven	1	
AG	2 rows	13-cell cleaner-scavenger flotation machines, 22.5 cu ft/cell	48 each	

CONCENTRATOR EQUIPMENT LIST (Contd)

Identification	Qty	Description	HP	Remarks
	1	Blower for flotation machines	50	
AH	2	Submerged cleaner concentrate pump with tank, 66 gpm	2 each	1 spare
AI	2	Submerged final concentrate pump with tank, 95 gpm	3 each	
AJ	2	Submerged scavenger concentrate pump with tank, 19 gpm	1 each	1 spare
AK	1	Scavenger concentrate sampler		
AL	1	Scavenger tails sampler		
AM	2	Tailing thickener, 100' dia x 8' SWD	15 each	
AN	3	Tailing thickener overflow pumps with sump box, 474 gpm	15 each	1 spare
AO	1	Mill tailing sampler		
AP	1	Concentrate thickener 35' dia x 8' SWD	2	
AQ	1	Concentrate thickener underflow pump	5	
AR	2	Concentrate thickener overflow pump with sump box, 90 gpm	3 each	
AS	1	Concentrate disc filter, with accessories 6' dia x 6 discs	35	
AT	1	Tailing pipeline, 8" dia with 10 drop boxes		15,900 linear ft
AU	1	Tailing pond starter dam		40,320 cu yd
AV	1	Tailing water reclaim system with 8"-dia transite decant line, 2 booster stations, pipe and dam	180 total	
AW	2	Well water pumping stations 156 gpm @ 800'	150 total	

CONCENTRATOR EQUIPMENT LIST (Contd)

Identification	Qty	Description	HP	Remarks
AX	1	Fresh water supply system with 4 booster stations, pipe	775 total	
	2	Wells		
AY	1	Concentrate sampler		
AZ	1	Submerged pump for water storage stope, 378 gpm	20	
		Process water pipe, stope to head tank, 4"-dia steel		2,000 linear ft
	1	Bridge crane, 5-ton		
		Dust collection equipment		For fine ore storage
Reagent Preparation				
BA	1	Lime storage bin, 1,000 cu ft		
BB		Mill feed conveyor, 12" x 6'	1	
BC	1	Ball mill, 3-1/2' ID liners x 5'	30	
BD	2	Lime circulating pump, 20 gpm	2 each	1 spare
BE	1	Hydrocyclone, 8" dia		
BF	2	Agitated lime storage tanks, 8' dia x 8' SWD	10 each	
BG	2	Lime loop pump, 20 gpm	3	1 spare
	2	Agitated xanthate mix tanks 5' dia x 6'	1 each	
	2	Agitated Aeroflot mix tanks 4' dia x 5'	1 each	
	2	Agitated Frother mix tanks, 55-gal drum	1 each	

CONCENTRATOR EQUIPMENT LIST (Contd)

Identification	Qty	Description	HP	Remarks
Mine Fill Plant				
CA	2	Rubber-covered rougher tailing pump, with sump box, 839 gpm	25	1 spare
CB	1	Hydrocyclone, 20" dia		
CC	1	Agitated storage tank, 8' dia x 10' SWD	20	
CD	2	Rubber Covered Hydrocyclone Feed Pump, 644 gpm	20 each	1 spare
CE	1	Hydrocyclone, 20" dia		
CF	1	Agitated storage tank 8' dia x 10' SWD	20	
CG	1	Positive displacement pump 167 gpm at 750' head	75	
CH		Slurry pipeline to mine, 4" dia		3,400 linear ft
Other				
	1	Mine and ancillary facilities power substation, 3,200 kW		

Table 4-3 - Power Requirements

Facility	Connected hp	Peak kW	Demand kW-hr/day	Demand kW-hr/yr
Mine	4,220	3,148	32,110	8,027,500
Crushing	605	451	5867	1,466,823
Concentrator	33,632	2,508	60,192	21,067,200
Mill and Reagent Preparation	59	44	1,056	369,718
Sand Fill	155	116	2,784	487,200
Auxiliaries	-	250	3,000	1,950,000
	8,401	6,517	105,009	33,368,441

Operating Schedule:

Mine - 5 days, two 6-hr shifts, 250 days/yr

Crushing - 5 days, two 6.5-hr shifts, 250 days/yr

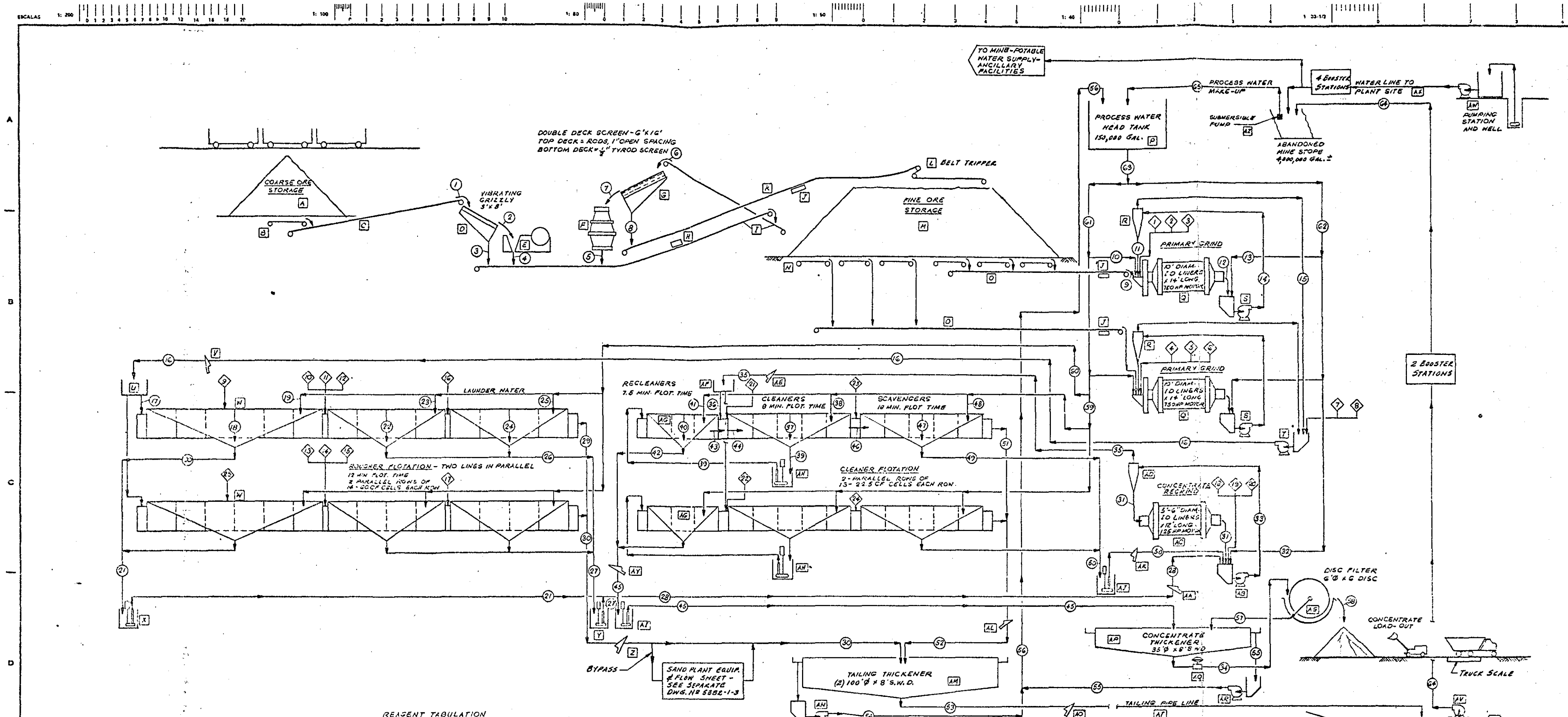
Mill and Reagent Preparation - 7 days, 3 shifts, 350 days/yr

Sandfill - 175 days/year, 3 shifts/day

Auxiliaries - 7 days, 12 hr/day, 365 days/yr

NOTE: Subsequent to the completion of this table, mine power requirements were lowered as a result of a charge in design criteria.

PROCESS AND DESIGN DRAWINGS



	1	2	3	4	5	6	7	8
SOLIDS - DST/HR	250.0	152.7	90.3	152.7	553.3	90.3	553.3	250.0
SCREEN ANALYSIS:								
PLUS 12"	2.0	2.1	-	-	-	-	-	-
MINUS 12" PLUS 6"	49.0	75.1	-	5.0	-	0.5	0.9	-
MINUS 6" PLUS 3"	12.0	12.8	-	37.5	-	4.1	6.6	-
MINUS 3" PLUS 1½"	16.0	1.3	42.0	47.0	-	10.6	15.4	-
MINUS 1½" PLUS 1"	5.0	0.4	12.0	19.4	6.0	3.0	11.1	-
MINUS 1" PLUS ¾"	4.0	0.3	11.0	15.5	34.0	26.5	32.0	13.0
MINUS ¾" PLUS ¼"	2.0	0.1	5.0	5.8	17.0	13.0	15.6	3.0
MINUS ¼"	11.0	0.9	29.0	30.1	43.0	35.6	17.0	80.0

REAGENT TABULATION

FEED POINT & ADDITION RATES	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	TOTAL ADDITION
LIME: (20% SOLIDS IN SLURRY)																										
GPM - HYDRATED LIME SLURRY	0.49			0.49														0.24								
LBS CaO/TON ORE	0.4			0.4														0.2								1.0
FROTHER (50 SOAP-50 OIL, CRESOL)																										
ML/MIN. (UNDILUTED)			25			25		40			5			5							5	5				
LBS/TON ORE			0.037			0.038		0.06			0.015			0.015							0.015	0.015				0.200
AEROFLOAT NO. 238																										
ML/MIN. (5% SOLUTION)		289			289					96			96						192							
LBS/TON ORE		0.022			0.023					0.007			0.008						0.015							0.075
POTASS - AMYL XANTHATE																										
ML/MIN. (5% SOLUTION)									132			132			132	513	513			132			66	66	132	
LBS/TON ORE									0.01			0.01			0.01	0.04	0.04			0.01			0.005	0.005	0.01	0.140

	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
SOLIDS - DST/D	[illegible]	-	4208	[illegible]	-	5763.5	1052.7	2054	[illegible]	620	-	550	116.0	210	-	200	-	47.0
WATER - ST/D	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	135.2	38.7	174.0	348.0	63.0	18.0	46.7	12.3	141.0
TOTAL FLOW - ST/D	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	193.5	38.7	232.0	464.0	90.0	12.0	66.7	13.3	159.0
% SOLIDS (BY WT.)	35.0	0	15.0	75.0	0	57.1	30.0	30.0	30.0	30.0	0	25.0	25.0	30.0	0	30.0	0	25.0
SP. GR. SOLIDS	3.11	-	3.13	3.13	-	3.13	3.13	3.13	3.13	3.11	-	3.71	3.71	3.21	-	3.17	-	3.19
SP. GR. SLURRY	-	1.00	2.34	2.04	1.00	1.65	1.25	1.25	1.26	1.28	1.00	1.22	1.22	1.26	1.00	1.26	1.00	1.21
US GPM	-	49	460	575	351	924	464	928	465	26	6	32	64	12	3	9	2	26
METAL CONTENT:																		
COPPER - ST/D	17.93							35.90	17.90	14.32	-	14.32	28.64	1.86	-	2.62	-	1.97
SILVER - Oz/D	421							842	421	322	-	322	644	10	-	14	-	44
ASSAYS % Cu	1.70							1.70	1.70	24.7	-	24.7	24.7	6.00	-	3.10	-	4.19
Oz/T Ag	0.40							0.40	0.40	5.55	-	5.55	5.55	1.11	-	0.70	-	0.94

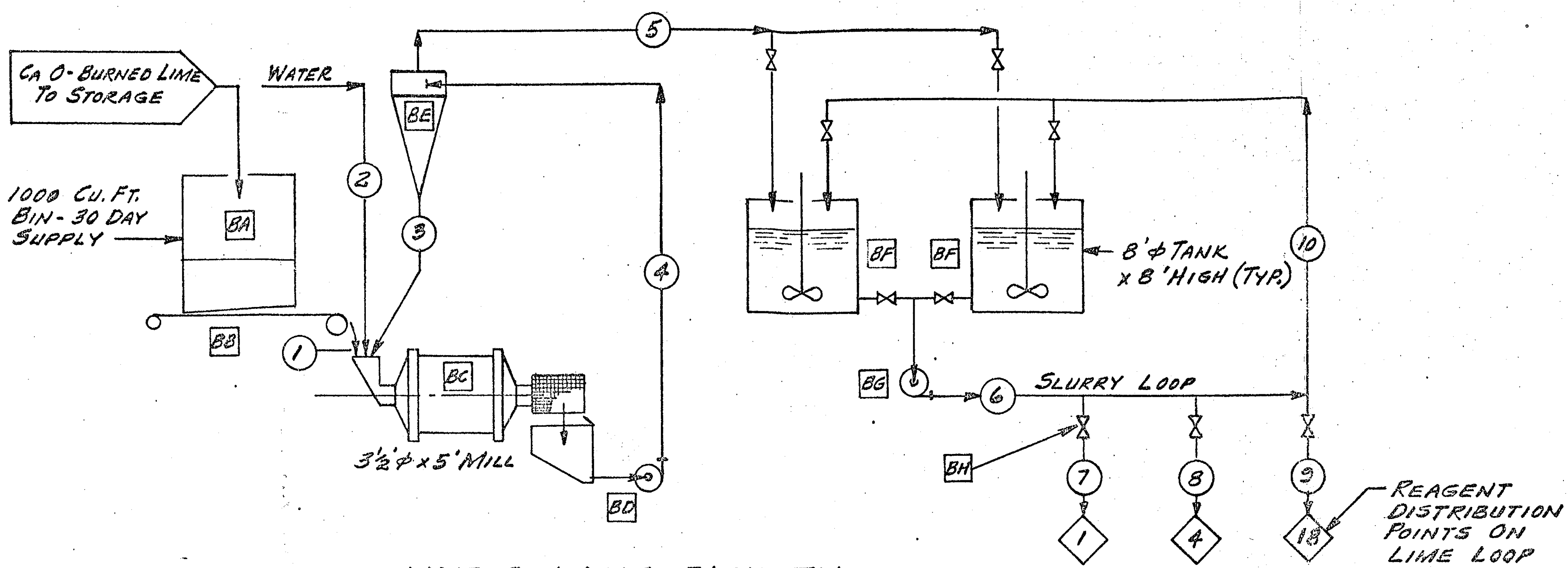
	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47
SOLIDS - DST/D	94.0	210.0	947.7	1895.4	69.50	-	228.0	96.2	232.0	116.0	72.0	-	72.0	48.1	-	48.1	23.9	139.9	96.2	67.9	11.0
WATER - ST/D	282.0	630.0	2211.3	4422.6	46.40	190.5	1292.0	96.2	928.0	464.0	328.0	50.0	378.0	252.5	20.1	272.6	125.5	589.5	545.2	261.5	42.4
TOTAL FLOW - ST/D	376.0	840.0	3159.0	6318.0	116.00	190.0	1520.0	192.4	1160.0	580.0	400.0	50.0	450.0	300.6	20.1	320.7	149.4	729.4	641.4	329.4	53.4
% SOLIDS (BY WT.)	25.0	25.0	30.0	30.0	60.0	0	15.0	50.0	20.0	20.0	18.0	0	16.0	16.0	0	15.0	16.0	19.2	15.0	20.6	20.6
SP. GR. SOLIDS	3.19	3.45	3.10	3.10	3.43	-	3.43	3.94	3.43	3.43	3.81	-	3.81	3.94	-	3.94	3.57	3.46	3.94	3.14	3.21
SP. GR. SLURRY	1.21	1.22	1.26	1.26	1.74	1.00	1.40	1.60	1.17	1.17	1.15	1.00	1.13	1.14	1.00	1.13	1.13	1.16	1.13	1.16	1.17
US GPM	52	115	418	835	111	32	277	19	166	83	58	8	66	44	3	47	22	105	95	47	8
METAL CONTENT:																					
COPPER - ST/D	3.94	32.58	1.61	3.22				30.8	33.68	16.84	20.16	-	20.16	15.4	-	15.4	4.76	21.60	30.8	1.44	0.55
SILVER - Oz/D	88	732	55	110				703	756	378	461	-	461	352	-	352	109	487	703	26	11
ASSAYS % Cu	4.19	15.5	0.17	0.17				32.0	14.5	14.5	28.0	-	28.0	32.0	-	32.0	19.9	15.4	32.0	2.12	5.00
Oz/T Ag	0.94	3.49	0.06	0.06				7.3	3.26	3.26	6.40	-	6.40	7.3	-	7.3	4.66	3.48	7.3	0.37	1.1

	48	49	50	51	52	53	54	55	56	57	58	59	60	61	62	63	64	65		
SOLIDS - DST/D	-	11.0	22.0	56.9	113.8	2003.2	-	-	-	-	96.2	-	-	-	-	-		-		
WATER - ST/D	11.3	53.7	107.4	219.1	438.2	2009.2	2851.6	532.1	3385.7	83.1	13.1	162.8	140.0	893.8	4401.4	5295.2	1004.6	906.9		
TOTAL FLOW - ST/D	11.3	64.7	129.4	276.0	552.0	4018.4	2851.6	532.1	3385.7	83.1	109.3	162.8	140.0	893.8	4401.4	5295.2	1004.6	906.9		
% SOLIDS (BY WT.)	0	17.0	17.0	20.6	20.6	50.0	0	0	0	0	88.0	0	0	0	0	0	0	0		
SP. GR. SOLIDS	-	3.21	3.21	3.13	3.13	3.11	-	-	-	-	3.94	-	-	-	-	-	-	-		
SP. GR. SLURRY	1.00	1.13	1.13	1.16	1.16	1.51	1.00	1.00	1.00	1.00	2.91	1.00	1.00	1.00	1.00	1.00	1.00	1.00		
US GPM	2	10	19	40	79	441	474	90	564	14	-	27	22	149	734	883	167	151		
METAL CONTENT:																				
COPPER - ST/D	-	0.55	1.10	0.89	1.78	5.00	-	-	-	-	30.8	-	-	-	-	-	-	-		
SILVER - Oz/D	-	11	24	15	29	139	-	-	-	-	703	-	-	-	-	-	-	-		
ASSAYS % Cu	-	5.00	5.00	1.56	1.56	0.25	-	-	-	-	32.0	-	-	-	-	-	-	-		
Oz/T Ag	-	1.1	1.1	0.25	0.25	0.07	-	-	-	-	7.3	-	-	-	-	-	-	-		

DWG. NO.	DESCRIPTION	DWG. NO.	DESCRIPTION
5382-1-2	MILK OF LIME FLOW SHEET	5382-1-6	FINE ORE STORAGE BLDG.
5382-1-3	SAND PLANT FLOW SHEET	5382-1-7	CONCENTRATOR BLDG.
5382-1-4	MILL SITE PLAN	5382-1-8	TAILINGS POND AREA SITE PLAN
5382-1-5	ORE RECLAIM, CRUSHING & SCREENING	5382-1-9	MAP OF MINE & MILL

DRAWN RAM 10/16/74
CHECKED T. H. J. 12/4/74

CONTINENTAL MATERIALS CORP.
ORACLE ARIZONA
ORACLE RIDGE PROJECT
PROCESS FLOW DIAGRAM
SCALE NONE
5382-1-1



LIME SLAKING FACILITY

4 HRS. PER DAY OPERATION	(1)	(2)	(3)	(4)	(5)
SOLIDS LBS/HR.	658.0	–	3308.8	4136.0	827.2
SOLUTION LBS/HR.	–	3478.0	4963.2	8272.0	3308.8
TOTAL FLOW LBS/HR.	658.0	3478.0	8272.0	12408.0	4136.0
% SOLIDS	100.0	0	40.0	33.3	20.0
SP. GR. SOLIDS	3.40	–	2.34	2.34	2.34
SP. GR. TOTAL FLOW	3.40	1.00	1.30	1.24	1.13
G.P.M.	–	6.95	12.7	20.1	7.3
CA O (AVAILABLE) LBS/HR.	526.4	–	–	–	526.4
INERTS LBS/HR.	131.6	–	–	–	131.6
ANALYSIS OF CAO AVAILABLE	80.0%	–	–	–	80.0%

24 HRS. PER DAY OPERATION	(6)	(7)	(8)	(9)	(10)
SOLIDS LBS/HR.	2260.4	55.4	55.4	27.1	2122.5
SOLUTION LBS/HR.	9041.6	221.5	221.5	108.5	8490.1
TOTAL FLOW LBS/HR.	11302.0	276.9	276.9	135.6	10612.6
% SOLIDS	20.0	20.0	20.0	20.0	20.0
SP. GR. SOLIDS	2.34	2.34	2.34	2.34	2.34
SP. GR. TOTAL FLOW	1.13	1.13	1.13	1.13	1.13
G.P.M.	20.0	0.49	0.49	0.24	18.78

NOTE: MATERIAL BALANCE IS APPROXIMATE

REFERENCES

DWG. NO.	DESCRIPTION

REVISIONS

NO.	BY	CH.	APP.	APP.	CLIENT	DATE	DESCRIPTION

DRAWN G.E. EHLE DATE 11/25/4
CHECKED E.H. Gator 12/9
SECT. HD.
PROJ. MGR. ENG M. Walker 2/7/5
CLIENT

PARSONS-JURDEN DIVISION OF THE RALPH M. PARSONS COMPANY LOS ANGELES, CALIFORNIA

CONTINENTAL MATERIALS
ORACLE RIDGE PROJECT

LIME SLAKING FLOW SHEET

JOB NO. 5382-1
ACCOUNT NO.
DWG. NO. 5382-1-2
REV.

RMP 315-3
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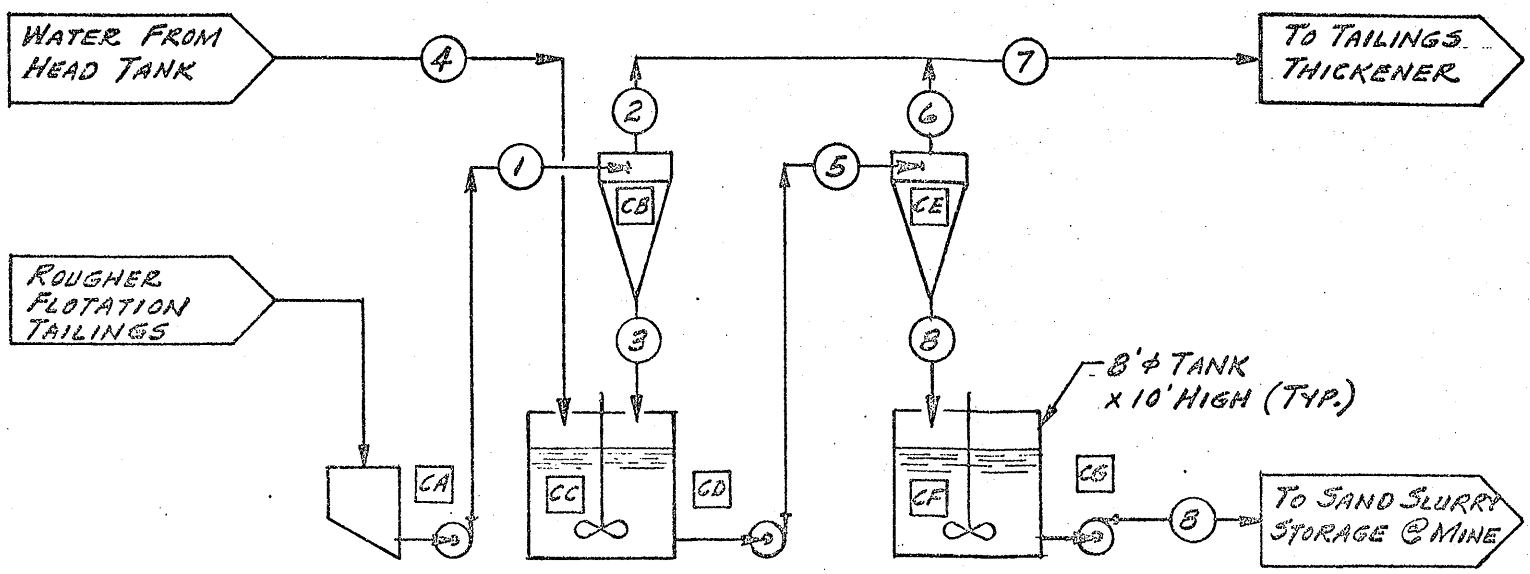

SAND PLANT FLOW & MATERIAL BALANCE FOR MINE FILL

	1	2	3	4	5	6	7	8
SOLIDS - DST/D	1895.4	441.0	1454.4	–	1454.4	440.9	881.9	1013.5
WATER - ST/D	4422.6	3453.0	969.6	2424.0	3393.6	2717.9	6170.9	675.7
TOTAL FLOW ST/D	6318.0	3894.0	2424.0	2424.0	4848.0	3158.8	7052.8	1689.2
% SOLIDS	30.0	11.3	60.0	0	30.0	14.0	12.5	60.0
SP. GR. SOLIDS	3.10	3.10	3.10	–	3.10	3.10	3.10	3.10
SP. GR. SLURRY	1.26	1.08	1.68	1.00	1.26	1.10	1.09	1.68
GPM	839	599	240	404	644	477	1076	167
SCREEN ANALYSES	% WT.						% WT.	% WT.
PLUS 100 MESH	3.6						–	6.7
MINUS 100 - PLUS 150	6.4						2.0	10.2
MINUS 150 - PLUS 200	10.0						5.0	14.3
MINUS 200 - PLUS 270	11.5						8.5	14.1
MINUS 270 - PLUS 325	6.5						5.5	7.4
MINUS 325 - PLUS 400	5.0						5.5	4.6
MINUS 400 MESH	57.0						73.5	42.7

REFERENCES	DWG. NO. / DESCRIPTION
REVISIONS	NO. / BY / CK. / APP. / APP. / CLIENT / DATE / DESCRIPTION

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CHECKED E H Gates 12/9
SECT. HD.
PROJ. MGR. ENGR. M Parker 2/7/5
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PARSONS-JURDEN DIVISION OF THE RALPH M. PARSONS COMPANY LOS ANGELES, CALIFORNIA

CONTINENTAL MATERIALS
ORACLE RIDGE PROJECT

SAND PLANT FLOW SHEET

JOB NO. 5382-1
ACCOUNT NO.
DWG. NO. 5382-1-3
REV.

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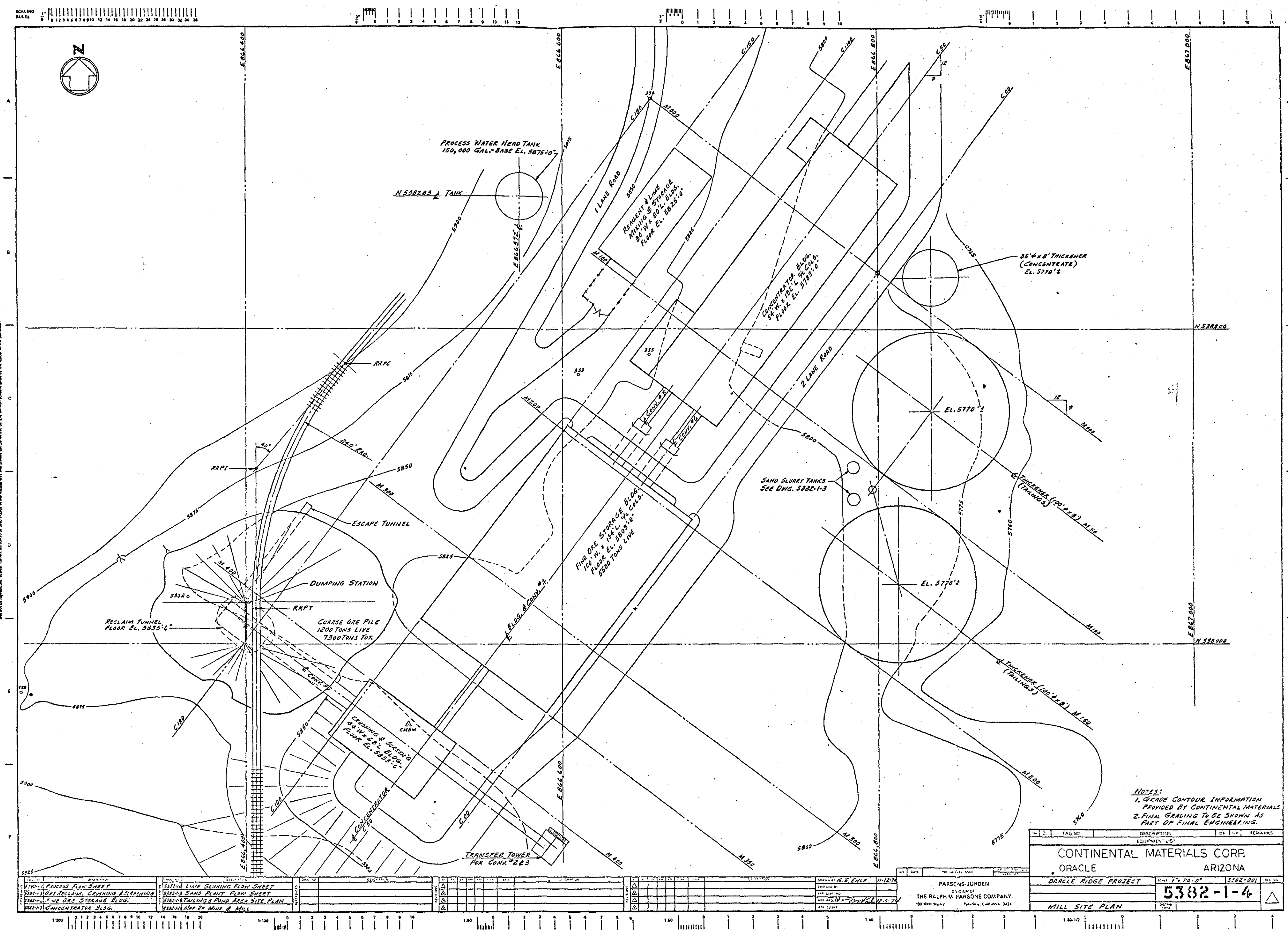
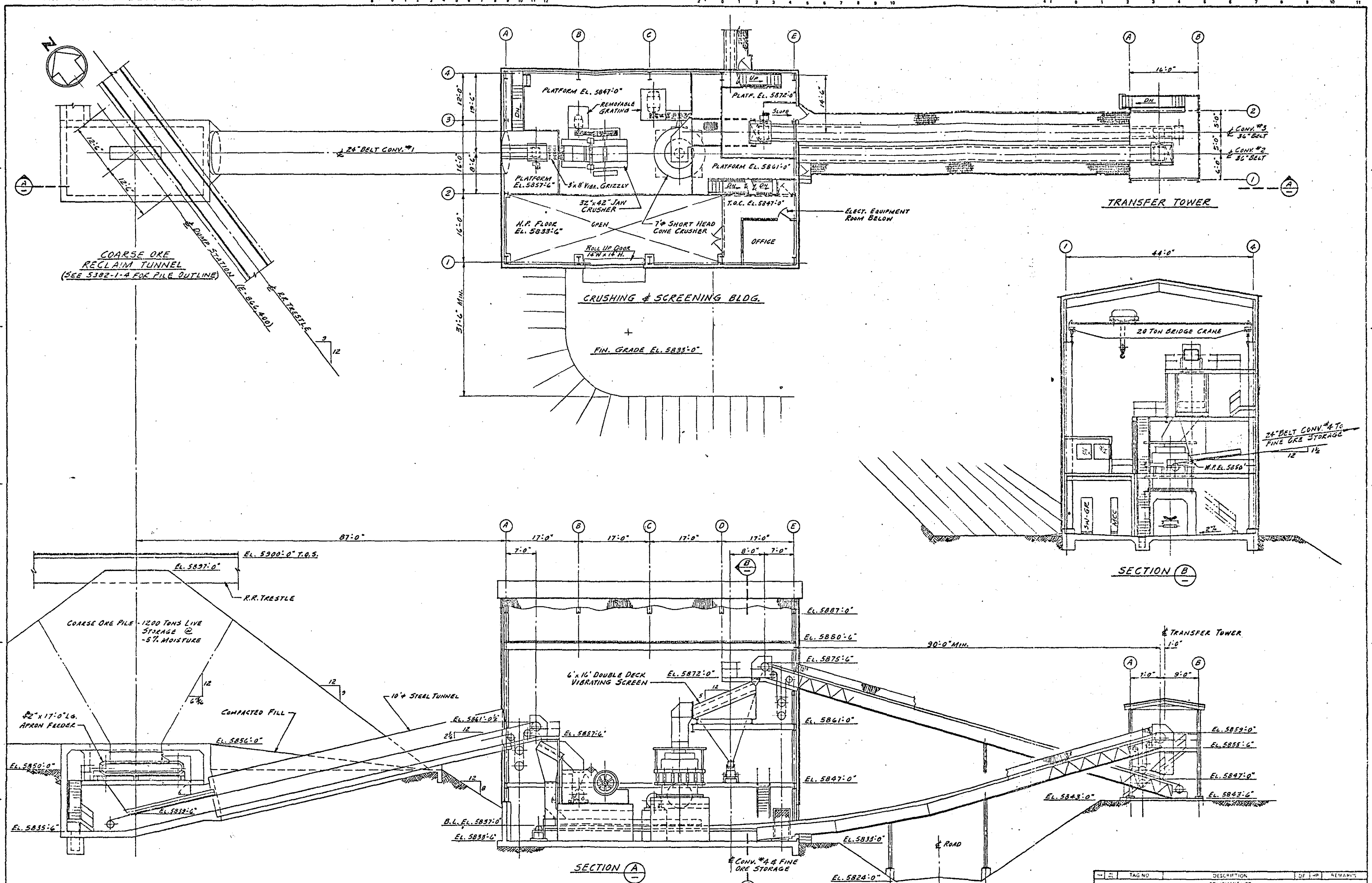

N
PROCESS WATER HEAD TANK
150,000 GAL.-BASE EL. 5875'-0"
N 538283 ℄ TANK
E 866 575' ℄
REAGENT & LIME MIXING & STORAGE 30'W x 80'L BLDG. FLOOR EL. 5825'-0"
1 LANE ROAD
CONCENTRATOR BLDG.
2 LANE ROAD
35'⌀ x 8' THICKENER (CONCENTRATE) EL. 5770'±
N 538200
EL. 5770'±
SAND SLURRY TANKS SEE DWG. 5382-1-3
℄ THICKENER (100'⌀ x 8') (TAILINGS)
FINE ORE STORAGE BLDG. FLOOR EL. 5803'-0" 5500 TONS LIVE
ESCAPE TUNNEL
DUMPING STATION
RRPT
RRPC
240' RAD.
RECLAIM TUNNEL FLOOR EL. 5835'-6"
COARSE ORE PILE 1200 TONS LIVE 7300 TONS TOT.
CRUSHING & SCREEN'G BLDG. FLOOR EL. 5835'-0"
℄ CONCENTRATOR C 50
TRANSFER TOWER FOR CONV. #2 & 3
N 538000
E 866 400
E 866 600
E 866 800
E 867 000
NOTES:
1. GRADE CONTOUR INFORMATION PROVIDED BY CONTINENTAL MATERIALS
2. FINAL GRADING TO BE SHOWN AS PART OF FINAL ENGINEERING.
5382-1-1 PROCESS FLOW SHEET
5382-1-5 ORE RECLAIM, CRUSHING & SCREENING
5382-1-6 FINE ORE STORAGE BLDG.
5382-1-7 CONCENTRATOR BLDG.
5382-1-2 LIME SLAKING FLOW SHEET
5382-1-3 SAND PLANT FLOW SHEET
5382-1-8 TAILINGS POND AREA SITE PLAN
5382-1-10 MAP OF MINE & MILL
DRAWN BY G.E. CHLE 11-12-74
PARSONS-JURDEN
DIVISION OF
THE RALPH M. PARSONS COMPANY
EQUIPMENT LIST
CONTINENTAL MATERIALS CORP.
ORACLE
ARIZONA
ORACLE RIDGE PROJECT
SCALE 1" = 20'-0"
5382-001
5382-1-4
MILL SITE PLAN



DWG. NO.	DESCRIPTION
5382-1-1	PROCESS FLOW SHEET
5382-1-6	MILL SITE PLAN
5382-1-2	FINE ORE STORAGE BLDG.
5382-1-7	CONCENTRATOR BLDG.

DRAWN BY G.E. EHLE 11/21/74

PARSONS-JURDEN
DIVISION OF
THE RALPH M. PARSONS COMPANY
100 West Walnut Pasadena California 91124

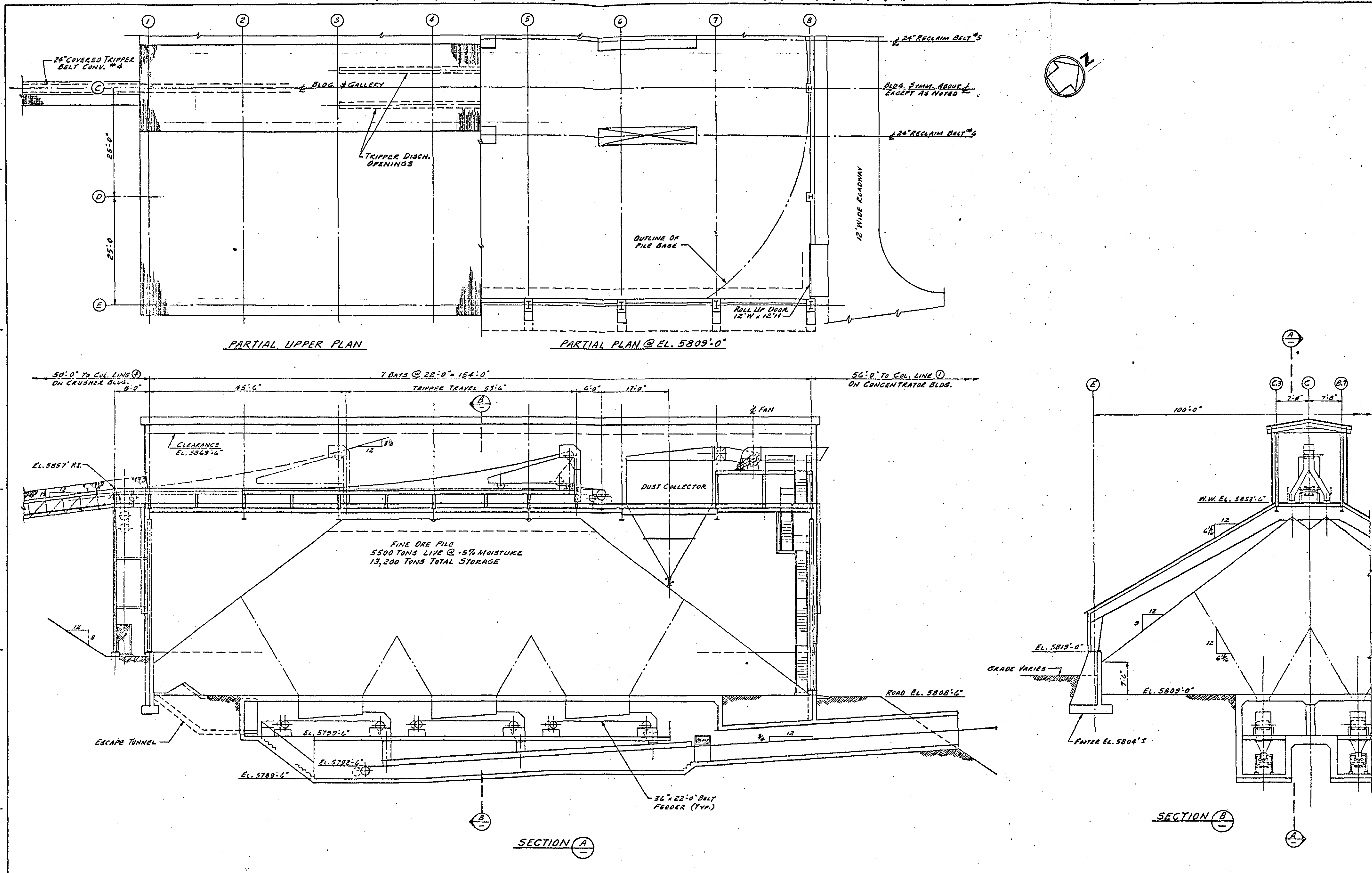
EQUIPMENT LIST

CONTINENTAL MATERIALS CORP.
ORACLE ARIZONA

ORACLE RIDGE PROJECT
COARSE ORE RECLAIM,
CRUSHING & SCREENING
GEN. ARRGT. PLAN & SECTIONS

SCALE 1/8" = 1'-0"
5382-001

5382-1-5



TAG NO	DESCRIPTION	DF	HP	REMARKS

EQUIPMENT LIST

DWG. NO.	DESCRIPTION
5382-1-1	PROCESS FLOW SHEET
5382-1-4	MILL SITE PLAN
5382-1-5	ORE RECLAIM, CRUSHING & SCREENING
5382-1-7	CONCENTRATOR BLDG.

DRAWN BY G. E. EHLE 11/18/74

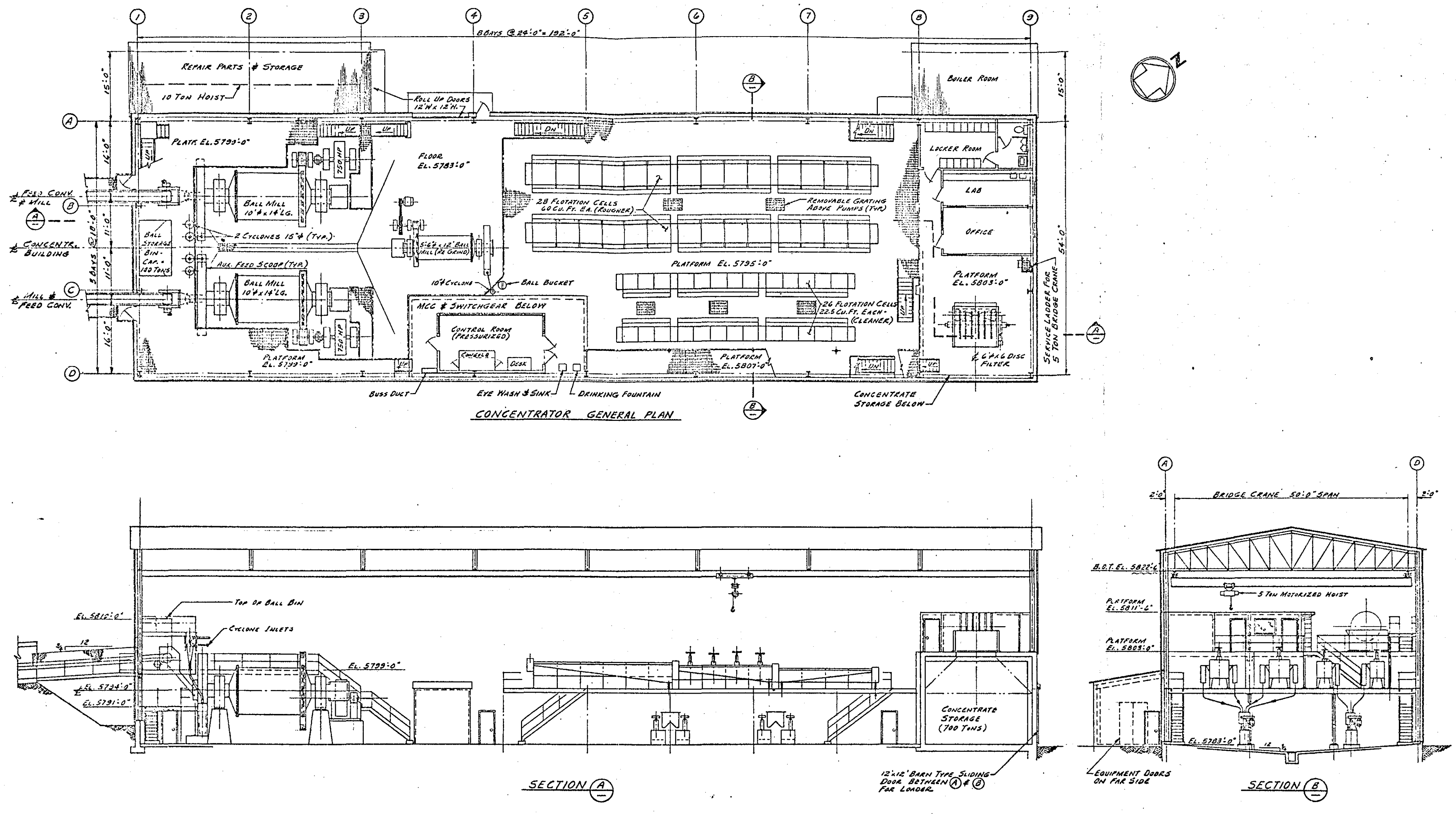
PARSONS-JURDEN
DIVISION OF
THE RALPH M. PARSONS COMPANY
100 West Walnut Pasadena, California 91124

CONTINENTAL MATERIALS CORP.
ORACLE ARIZONA

ORACLE RIDGE PROJECT
FINE ORE STORAGE
GENERAL ARRANGEMENT
PLAN & SECTIONS

SCALE 1/8" = 1'-0" 5382-001

5382-1-6



	TAG NO	DESCRIPTION	DF	HP	REMARKS

EQUIPMENT LIST

CONTINENTAL MATERIALS CORP.
ORACLE ARIZONA

PARSONS-JURDEN
DIVISION OF
THE RALPH M. PARSONS COMPANY
100 West Walnut Pasadena, California 91124

ORACLE RIDGE PROJECT
CONCENTRATOR BUILDING
GENERAL ARRANGEMENT
PLANS & SECTIONS

SCALE 1/8" = 1'-0"
5382-001
5382-1-7

SECTION 5

AUXILIARY FACILITIES

The mine and mill will require the following auxiliary facilities to support operation:

- Service buildings and roads
- Tailing disposal and water reclaim systems
- Freshwater supply
- Power
- Other facilities
    - Fuel oil storage
    - Concentrator compressed air
    - Fire protection
    - Townsite
    - Sanitary sewer system

These facilities are described in this section and their locations are shown on Drawings 5382-1-8, 5382-1-9, and 5382-1-10, presented at the end of the section.

SERVICE BUILDINGS AND ROADS

The site on which the service buildings will be located is shown on Drawing 5382-1-10. It is located at the 5,900-foot level, and will be fully developed during the preproduction period by dumping waste. Buildings on this site will include the general offices, laboratories, compressor building, changehouses, main shop, and warehouse.

GENERAL OFFICES

The general offices will be housed in a two-story 60- by 120-foot steel frame building. The engineering, geology, management staff, accounting, and other office personnel will be located on the second floor. The building will also contain a conference room/staff lunchroom and washroom and toilet facilities.

The ground floor will be divided into three areas:

(a) The first 30 feet will be the two-story portion with the offices.

(b) The next 45 feet will be the warehouse and toolroom with a balcony.

(c) The final 45 feet will be a shop facility, which will be full building height and have a 5-ton crane and the necessary equipment for heavy-duty repair, metal fabrication, and automotive repair.

The maintenance supervisor will have an office in the shop, and the supervisory staff for the concentrator will have offices in the mill building. Adequate toilet and washroom facilities will be provided throughout.

LABORATORIES

A general laboratory, suitable for servicing the mine and concentrator, will be located adjacent to the general offices in a 20- by 40-foot building. This laboratory will contain facilities for sample preparation, wet analyses, and analytical instrumentation. The building will be of the steel-frame type with insulated sandwich siding and an insulated roof.

The mill will have a simple laboratory for sample preparation and testing.

COMPRESSOR BUILDING

A 20- by 40-foot steel-framed building will be provided to house the three Atlas-Copco 700-hp compressors and the portable rotary-screw compressor. These units will provide the compressed air required by the mining operation.

CHANGEHOUSES

The changehouse for the mill operators, with lockers for about 50 men, will be in the northwest corner of the concentrator building and will contain toilet and washroom facilities. The building presently located at the 5,900-foot level will be remodeled as necessary to serve as the mine changehouse and will include offices for the mine superintendant, conference/training room, and a first aid facility.

MINE SHOP AND WAREHOUSE

Additional shop facilities will be located in an existing 25- by 40-foot building at the 6,400-foot level. This shop will provide a facility for routine maintenance and light repairs and a warehouse for mine supplies.

ROADS

Plant roads, service yards, and parking lots suitable for rubber-tired vehicles of up to 30 tons will be surfaced with gravel prepared from preproduction development waste.

A new direct access/service road will be constructed to connect the mine and plant area with a planned extension of State Highway 76. This road will be parallel to the power transmission line, power distribution line, and the freshwater and reclaim water supply lines, and will serve as both an access road and service road.

TAILING DISPOSAL AND WATER RECLAIM

TAILING DISPOSAL

The tailing area, shown on Drawing 5382-1-8, is located in Section 14, T11S, R16E. The tailing pipeline extends approximately 3-1/2 miles, with 10 drop boxes over the change in elevation from 5,790 to 4,600 feet. It will be Younstown Transol slurry pipe, 8-5/8 OD by 0.25 wall. This pipe is supplied in 40-foot lengths with single-groove Victaulic couplings.

The rate at which tailing is generated is 2,009.2 DST/day. This is equivalent to 1,278 cu yd/day, assuming final compaction at 60% (volume) solids. On this basis, the annual tailing volume is 447,300 cu yd.

Several alternatives were considered as a site for a dam for the west tailing area; these alternatives are described in the following paragraphs.

- Alternate 1 - This alternative would initially involve construction of a dam across the 4,600-foot contour, the east end of which would begin at the coordinates N541900,E878530 and proceed E42°15'S to coordinates N541720,E878740. The area within the 4,600-foot contour would impound 43,704 cu yd of tailing, or 34 days production.

- Alternate 2 - This alternative would involve construction of a dam to encompass the area enclosed by the 4,630-foot contour, extending from coordinate N542010,E878405 E42°15'S to N541670,E878,805. The total impound would be 404,444 cu yd, or 316 days tailing production, assuming no diversion of tailing to mine fill.

  Extension to the 4,650-foot level would provide an additional 410,000 cu yd (321 days) impound, providing for a total of 1.7 years operation.

  Construction of the dam at the 4,630-foot level would require a total of 67,400 cu yd of material, assuming a 2-1/2:1 downslope and 2:1 upslope. An additional 93,830 cu yd would be required for the 4,650-foot level.

  From a practical standpoint, while the volume impounded at the 4,650-foot level is being filled, it would be necessary to construct another dam farther down the valley to provide for continuing tailing disposal.

- Alternate 3 - This alternative would involve construction of a dam across the 4,600-foot level starting at the point defined by coordinates N542,550,E878,925 and proceeding E13°S to N542,315,E879,925, thence S9°15'W to the intersection with the 4,600-foot level at N541,735, E879,825.

  At the corner represented by coordinates N542,315,E879,925 the dam would be 10 feet deep. At its deepest point, N542,450,E879,340, it would be 82 feet deep. The total length would be 1,630 feet.

  When this dam is extended to the 4,650-foot level, the maximum depth will be 132 feet and the total length 2,525 feet.

  A total of 940,000 cu yd of material will be required for construction of the dam at the 4,600-foot level, with an additional 498,000 cu yd required to complete construction to the 4,650-foot level, assuming the same slopes as in Alternate 2.

  The tailing impound of the area at the 4,600-foot contour, excluding the area included as part of Alternate 2, will be 1,204,400 cu yd.

The tailing area at the full 4,650-foot level will contain a total impound volume of approximately 4,345,194 cu yd, exclusive of dam displacement, which will give approximately a 9.7-year tailing disposal period based on the annual tailing volume of 447,300 cy yd.

The drainage area for the west tailing area has been estimated at about 100 acres. The 1-year rainfall has been estimated at 1.5 inches per 24 hours, and the 25-year rainfall at 4 inches. This would result in a total rise in the ponds covered in Alternate 3 of 36 inches and 15 inches, respectively. For the 25-year program, a 3-foot freeboard should be incorporated for flood, wind runup, etc.

Since no soils data were available for this study, it was assumed that the soils are granular, overlaying a bed rock about 10 feet below the surface.

ASSUMPTIONS

The final selection for the location of the initial dam is based on the following assumptions, which are coordinated with Parsons proposed mine development program.

- Mill construction will begin at the same time as the mine development program, at month zero (0).

- Mill construction will be complete and the mill ready to operate at one-half capacity at the end of month 24. The crushers will be operable at the end of month 23.

- During month 24, the mine will deliver ore to the raw ore storage pile under the trestle. During the same period, the crushing plant will produce and deliver approximately 13,000 tons to fine ore storage.

- At the end of month 24, the mine will begin to deliver 1,470 TPD (5-day basis) to the crushing plant.

- The mill will begin operation at the daily rate of 1,050 TPD at the end of month 24, and continue at that rate during months 25, 26, and 27.

- At the end of month 27, the mill will begin operating at the full rate of 2,105 TPD, and the mine will deliver to coarse ore storage at the rate of 2,940 TPD (5-day basis).

- No sand fill will be required by the mine until the end of month 36. This will permit sand to be used for dam construction for 270 days after the mill goes on a full-production schedule.

During the mine development and mill construction period, a "starter" dam will be constructed with borrow material at the 4,630-foot level in the west tailing area, located as noted in the Alternate 2 discussion. An impervious core will be built and the downhill face of sand/rock placed. This will require placement of approximately 40,320 cu yd of material.

When the mill begins operating at half capacity, tailing production will be 1,004.6 DST/day or 639 cu yd/day.

For the purpose of this study it is assumed that the sands will be used to construct the balance of the tailing dam, and that approximately 53% (1,000 TPD, or 639 cu yd/day, at full production) of the tailing will be satisfactory for either mine fill or tailing dam construction. At half production, the mill will produce 320 cu yd/day settled sands. At these production rates, the volume of sands available for the dam will be as follows:

320 cu yd/day, 90 days	28,800 cu yd
639 cu yd/day, 270 days	172,530 cu yd
	201,330 cu yd

The total tails generated during the first 12 months of operation will be:

639 cu yd/day, 90 days	57,510 cu yd
1,278 cu yd/day, 270 days	345,060 cu yd
	402,570 cu yd

During the first 90 days, 26,880 cu yd of the sands produced will complete the upslope side of the dam at 4,630 feet, and the balance can be used to complete the dam to the 4,650-foot level (approximately 120,910 cu yd) and start the dam at the location indicated under Alternate 3.

At the end of month 36, the mine will start taking fill on a 50/50 basis. From that point on, only 50% of the sands will be available for dam construction. The tailing pond will continue to receive 100% of the fines generated at the reduced volume of 633 cu yd/day. The recoverable water from the tailing, 168 gpm (241,920 gpd), will be approximately 64% of the water remaining after the tailing has settled, and be equivalent to approximately 50% of the total contained water flowing to the pond in the tailing.

WATER RECLAIM

There are four alternate methods available for tailing water reclaim: (1) decant tower and buried drain line, (2) barge and pump, (3) siphon, and (4) spillway.

The method selected for tailing water reclaim is a modification of (1). The reclaim water storage will be within the area defined by the 4,450-foot level, with a dam located between coordinates N543,468,E879,855 and N543,525,E879,948. Total impound volume will be approximately 1,965,000 gallons. This volume represents storage capacity for recoverable tailing reclaim water for approximately 8 days.

During the period of 3 months when the plant is operating at one half capacity, reclaimed tailing water will be pumped up to the storage stope at the rate of 168 gpm for 24 hours every second day. This flow, together with well water pumped at the rate of 156 gpm for 32.4 hours out of every 48, will provide the mill makeup water. The following illustrates this:

Mill demand, 1/2 capacity over 2 days		547,200 gal
Reclaim pump-back 168 gpm x 1,440 min/day every second day	241,920	
Well 1 water (156 gpm x 60 x 32.6 hr out of every 48)	305,280	
	547,200	547,200 gal

When the plant operates at full capacity, the reclaimed water will be pumped back at the rate of 168 gpm on a 24-hr/day, 7-day/wk basis.

The reclaim water will be pumped to the stope through 4-inch-dia steel pipe from the reclaim water reservoir at el 4,435 feet to the first booster station at el 4,675 feet, located adjacent to the #3 well water booster station. The reclaim water is pumped through 6-inch-dia steel pipe to the second booster station at el 5,300 feet, and then to the storage stope at el 5,900 feet. Table 5-1 lists the location and pertinent details of the reclaim water pumping system.

Table 5-1 - Water Reclaim Pumping System

Sta	El	Lift	Pipe Size	Approximate Length	Pumping Rate (gpm)	HP
Dam	4,435'	240'	4"	3,120'	168	30
1	4,675'	625'	6"	10,480'	168	75
2	5,300'	600'	6"	8,000'	168	75

FRESHWATER SUPPLY

The water for the mill and ancillary facilities will be supplied from three wells. Well 1 is 6.7 miles away from the mill site at el 3,850 feet. Wells 2 and 3 are approximately 12 miles from the mill site at el 3,280 feet and 3,200 feet, respectively.

Well 1, which has been constructed under the direction of and tested by Willard Owens Associates, Inc., is in S8, T11S, R17E at el 3,850 feet. Their report states that this "...well is capable of producing 156 gpm on a sustained basis from an approximate pumping level of 712 feet." A copy of their report is included in Volume II of this study.

It is also their opinion that additional wells completed in the formation in that area could produce adequate water. However, they indicate the need for more than three wells to maintain the required yield based on an ultimate requirement of 500 gpm.

The mill demand for freshwater makeup at full capacity is outlined in Section 4. The mill makeup water is estimated at 211 gpm, plus 168 gpm reclaimed from the tailing pond.

The main water storage facility has an estimated capacity of 4 million gallons; it is an abandoned stope in the old Geesamen workings. The coordinates of the access to the stope are N538800, E865110. The access is at el 5,905 feet, and the top of the stope is at el 5,895 feet.

From the stope, water is pumped to a 150,000-gallon process water head tank located at the 5,875-foot level, coordinates N538,283,E866,572. The bottom 50,000 gallons are reserved for a constant-pressure loop-type fire protection system. The balance provides process makeup water. Because the concentrate thickener overflow also returns to this tank, provision is made for periodic return of any settled solids to the mill circuits.

The well water line is United Concrete Water Pipe, with ring seal field joints. This steel pipe, designed especially for surface installation, has an outside cement coating 1/2-inch thick and a 5/16-inch-thick inside coating; it is furnished in 40-foot lengths. Dresser style 38 couplings are installed at every 10 lengths to allow for expansion and contraction. The pipe will be installed with restrainers and anchored at each booster station.

As noted in the Tailing Disposal subsection, the mill will operate at one-half capacity for the first 3 months. On this basis, the makeup water required would be as follows:

	Makeup	Reclaim	Total
One-half capacity	106	84	109 gpm
Full capacity	211	168	379 gpm

Therefore, for the first 3 months a single well pumping 156 gpm for 32.6 hours out of every 48 hours, plus reclaim water from the tailing pond pumped at 168 gpm for 24 hours every second day would be sufficient to supply the makeup water demand for the mill. (See the tabulation in the Water Reclaim subsection.)

Since the combined capacity of the storage stope and the head tank, exclusive of the 50,000 gallons reserved for fire protection, is 4.1 million gallons, this would not result in excessive drawdown.

When the mill is operating at full capacity, one well pumping at 156 gpm operating 24 hr/day, plus one well operating at 156 gpm for 8.46 hr/day, plus tailing reclaim water at 168 gpm 24 hr/day, will provide the mill makeup water demand.

Mill daily demand, full capacity		545,760 gpd
Reclaim water (168 x 1,440)	241,920 gpd	
Well 2 water (156 x 1,440)	224,640	
Well 1 water (156 x 60 x 8.46 hr/day)	79,200	
	545,760 gpd	545,760 gpd

On the basis of this tabulation and the one in the Water Reclaim subsection, it is possible to install the pumps initially at Wells 1 and 2, which will provide the required makeup water for production at full capacity. Furthermore, installation of a 156-gpm deepwell pump at Well 3 will provide for standby capacity should problems occur at either Well 1 or 2. In the event that the reclaim water line were out of service completely, two pumps could supply approximately 82% of the mill makeup water requirements.

The well water line and booster pumps are sized for transmission of a maximum of 312 gpm, the full output of two well pumps.

Table 5-2 lists the locations of the booster stations along the water supply line.

Table 5-2 - Booster Pump System

Sta	El	Lift	Pipe Size	Approximate Length	Pumping Rate (gpm)	
					Max	Min
1	3,280'	570'	6"	20,000'	312	156
2	3,850'	825'	6"	22,600'	312	156
3	4,675'	625'	6"	10,480'	312	156
4	5,300'	600'	6"	8,000'	312	156
				61,080'		

The pumping system consists of one deepwell pump at each well rated at 156 gpm, and two pumps at each booster station - one for 156 gpm, and one for 312 gpm. During the period when only 156 gpm of freshwater is required, the smaller of the two pumps at each booster station will operate. When the second 156 gpm is required (8.46 hrs/day) the larger pump will pick up the full 312 gpm. A detailed evaluation of the two alternatives indicated that this system was more economical with regard to pump size, efficiency, and power requirements than a single pump capable of pumping at the rate of either 156 gpm or 312 gpm.

POWER

An estimated 8,400 connected hp will be required for the project. Electrical power required will be provided by a 14-mile 115-kV transmission line built by Trico Electric Cooperative, Inc. (TRICO) from its existing lines to the main substation at the concentrator. A second 3,200-kW substation will be provided at the 5,900-foot level for the mine. Electrical power will be distributed to the various plant load centers, where it will be stepped down to the desired voltages. The voltage levels are 4,160 for motors of 250 hp and over, and 480 for all other motors. The power for the mine area will be delivered to the 3,200-kW substation by overhead transmission lines. The substations will consist of main disconnect switches and oil-filled power transformers with provision for fan auxiliaries. The power feeders from the transformers to the indoor motor control centers will be bus ducts or cables. All motors and equipment feeders will be carried in rigid galvanized steel conduit.

Lighting will consist of incandescent fixtures for platform and low-ceiling areas, mercury-vapor fixtures for high bay and outdoor areas, and fluorescent fixtures for control centers, offices and laboratory areas.

REQUIREMENTS

The estimated power requirements and kilowatt-hours (kW-hr) are listed in Table 4-3 (Section 4).

RATES

TRICO has developed the following rates based on the load data shown in Table 4-3:

Charge/kW	$2.65
plus charge kW-hr	0.000525
plus charge for fuel adjustment	0.00763

TRICO will install a demand meter on the secondary side of the transformer. They estimate losses at about 7%, and would account for this by multiplying the monthly kW-hr demand by the factor 1.07.

Based on the various loads indicated in Table 4-3, TRICO estimates the average cost of power at 12 mils per kW-hr. A copy of the TRICO proposal is included in Volume III.

DISTRIBUTION

There are two methods for supplying power to the pumps at the wells and booster stations. One is to run a distribution line back down the hill from the plant substation; the other is to purchase distributed power from either Arizona Public Service or the San Carlos Project.

Arizona Public Service has a substation at San Manuel, approximately 13 miles from the location of the two wells farthest from the plant. Arizona Public Service also has a second substation equally distant to the south of the wells.

The San Carlos Project has a distribution line as far as the Forest Service Campground at Peppersauce Canyon, but this is only to provide domestic services. To provide sufficient power for the pumps and booster stations from their Oracle substation, it would be necessary for them to rehabilitate approximately 8 miles of existing line and to install 17 miles of new line.

Installation of a distribution line down the hill from the plant substation would appear to be the lowest first-cost method of providing the power to the freshwater and reclaim water pumping systems. However, this may require special certification from the Arizona Public Service Commission.

OTHER FACILITIES

FUEL OIL STORAGE

A storage tank containing 20 days supply (30,000 gallons) will be located at the 5,900-foot level.

CONCENTRATOR COMPRESSED AIR

An air compressor and air surge tank will be installed in the storage area adjacent to the concentrator, and a piping distribution system will be provided to supply 100-psig compressed air for general services throughout the plant. Instrument air at lower pressure for pneumatic controls will be provided by separate smaller compressors as parts of the instrumentation systems of the various facility units.

FIRE PROTECTION

Piping from the 150,000-gallon service water head tank will be arranged so that one-third (50,000 gallons) will always be held in reserve for emergency use.

A fire line will parallel the gravity line to the plant site. A pump on the line at the plant site will boost the water pressure of the plant protection system. This system will consist of a loop that will feed strategically located hydrants and hose installations in the plant area. This system will also feed sprinkler systems installed in the reagent storage building. Smoke detection facilities will be provided in areas with high concentrations of electrical cable trays, as well as in switchgear rooms.

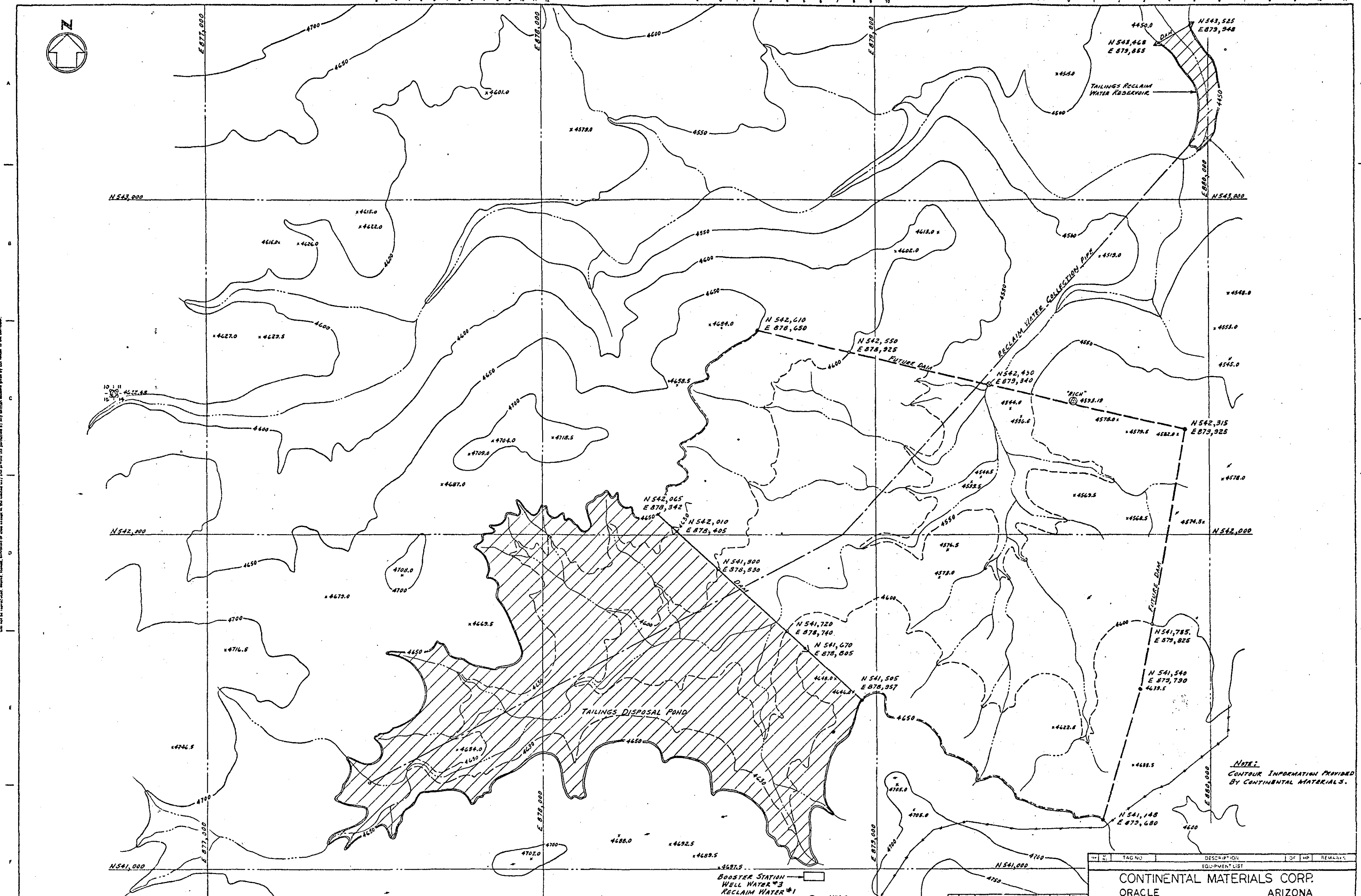
TOWNSITE

The townsite is located approximately 2-1/2 miles east of the plant site on Mt. Lemmon Road; it is shown on Drawing 5382-1-9 at the end of this section.

Power and potable water are available at the site. It is planned to use the site as a construction camp initially, and then as a townsite. The permanent facilities will include a bunkhouse for 30 men, pads for 10 trailers, and a cafeteria/recreation hall. A sewage disposal system will be provided for the permanent facilities.

SANITARY SEWER SYSTEM

A sanitary sewage disposal system will be provided for the various buildings at the 5,900-foot level and for the concentrator.



DWG. NO.	DESCRIPTION
5382-1-4	MILL SITE PLAN
5382-1-3	MAP OF MINE & MILL
5382-1-9	TAILINGS DISPOSAL & WATER SUPPLY

EQUIPMENT LIST

TAG NO	DESCRIPTION	DF	HP	REMARKS

DRAWN BY G.E. EHLE 12-2-74

PARSONS-JURDEN
DIVISION OF
THE RALPH M. PARSONS COMPANY
100 West Walnut Pasadena, California 91124

CONTINENTAL MATERIALS CORP.
ORACLE ARIZONA

ORACLE RIDGE PROJECT
TAILINGS DISPOSAL POND
AREA SITE PLAN

SCALE 1" = 100'-0"
5382-001
5382-1-8

1000

FEET

100'

ITEM	NO REQ.	TAG NO.	DESCRIPTION	DF	HP	REMARKS
			EQUIPMENT LIST			

CONTINENTAL MATERIALS CORP.

ORACLE ARIZONA

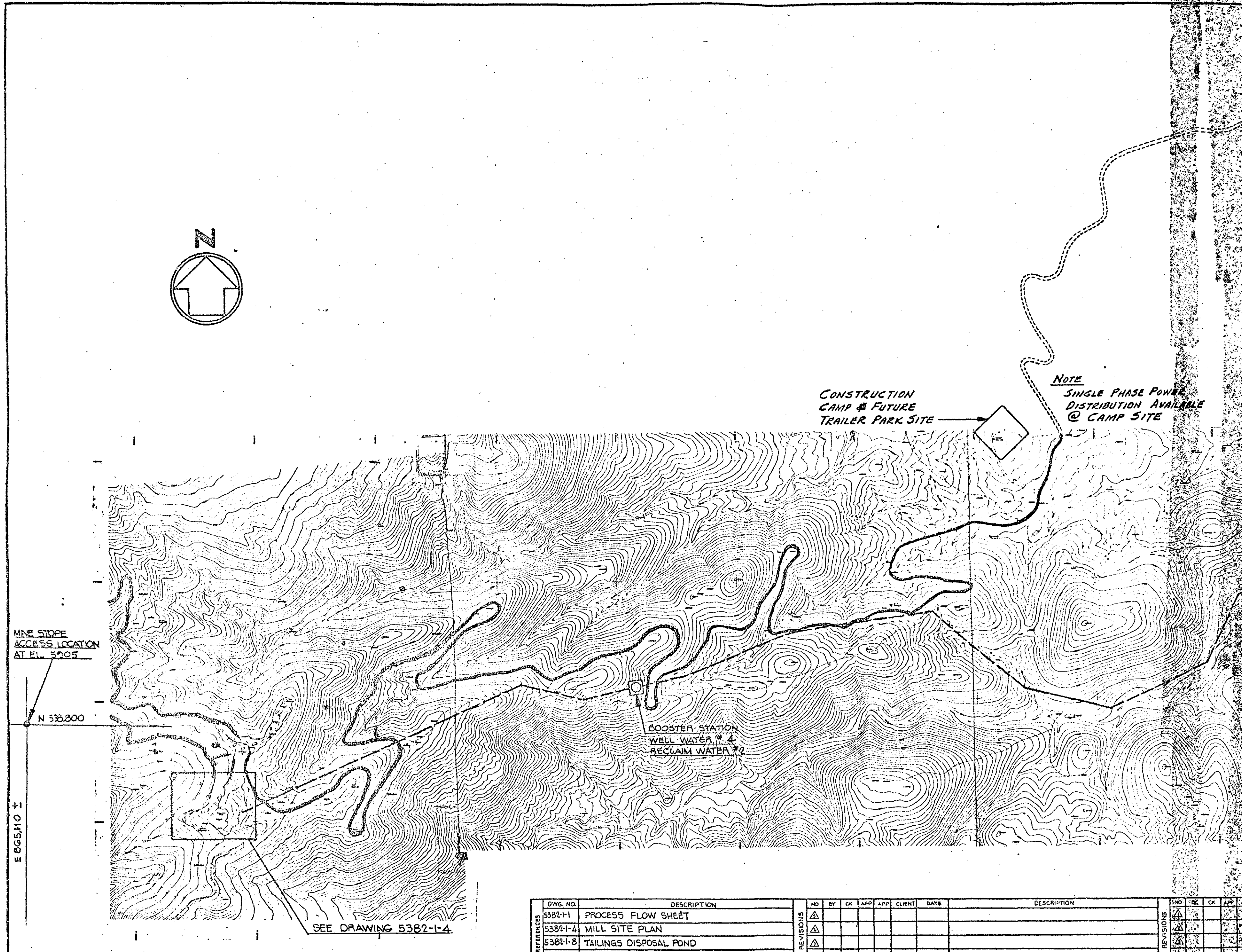
IMINARY ISSUE | DEPT HD | PROJ MGR | APPROVED

ONS-JURDEN
IVISION OF
M.PARSONS CO.
Pasadena, California 91124

ORACLE RIDGE PROJECT
TAILINGS DISPOSAL and
WATER SUPPLY

SCALE 1" = 500'-0" | PROJ. NO. 5382-001 | REV. NO.

5382-I-9



REFERENCES DWG. NO.	DESCRIPTION
5382-1-1	PROCESS FLOW SHEET
5382-1-4	MILL SITE PLAN
5382-1-8	TAILINGS DISPOSAL POND
5382-1-10	MAP OF MINE & MILL

REVISIONS NO	BY	CK	APP	APP	CLIENT	DATE	DESCRIPTION
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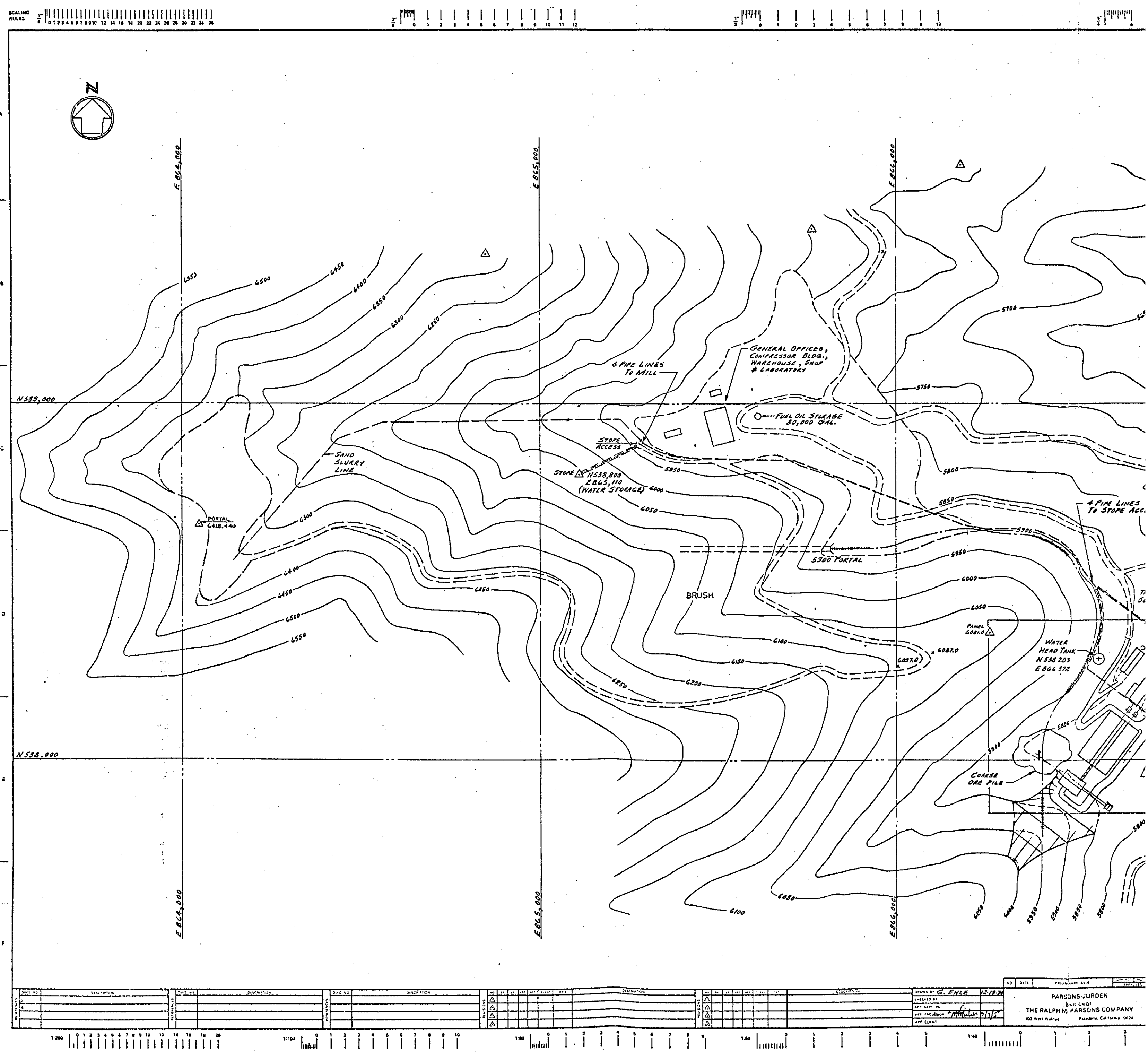

N
E 864,000
E 865,000
E 866,000
N 539,000
N 538,000
PORTAL 6418.440
SAND SLURRY LINE
4 PIPE LINES To MILL
GENERAL OFFICES, COMPRESSOR BLDG., WAREHOUSE, SHOP & LABORATORY
FUEL OIL STORAGE 50,000 GAL.
STOPE ACCESS
STOPE N538,800 E865,110 (WATER STORAGE)
5900 PORTAL
BRUSH
PANEL 6081.0
6087.0
WATER HEAD TANK N 538 203 E 866 572
4 PIPE LINES To STOPE ACC.
COARSE ORE PILE
6550
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DRAWN BY G. EHLE 12-19-74
PARSONS-JURDEN
DIVISION OF
THE RALPH M. PARSONS COMPANY
100 West Walnut Pasadena, California 91124
SCALING RULES

SECTION 6

CAPITAL COST ESTIMATE

The summaries of the capital costs presented in this section include the costs for the following:

(1) The equipment and facilities to mine the ore and transport it to the crushing plant.

(2) The concentrating plant, including facilities for crushing, grinding, flotation, thickening, and tailings disposal.

(3) The auxiliary facilities to provide water, power, and services to the plant and mine.

For the capital cost estimate, a copy of which is included in the Appendix III, the plant facilities were divided into the areas listed below:

Area 10 Crushing, including coarse ore storage

Area 15 Fine Ore Storage

Area 20 Concentrator

Area 30 Reagent Preparation

Area 40 Freshwater System

Area 50 Tailing Disposal and Water Reclaim

Area 60 Mine Fill

Area 70 Ancillary Facilities

Area 80 Town Site

Area 90 Railroad Trestle

BASIS OF ESTIMATE

In general, the capital cost estimate developed in the study has been based on the third quarter 1974 costs of the equipment and materials required, plus the estimated present labor costs required for their installation. The major equipment required is in accordance with the lists presented in Sections 3 and 4. The materials required for the various facilities and their installation labor costs have been estimated on the basis of costs for similar facilities in other plants. The sum of the foregoing cost items for each facility will constitute the direct field construction cost.

Quotations were obtained for those items of process equipment that constitute the bulk of the capital equipment cost. For those few items for which no quotations were obtained, in-house data were used.

Concrete, steel, piping, electrical power distribution within the plant area and similar construction materials were priced from in-house data.

The main 115-kV power transmission line, the lower voltage power distribution line to the freshwater system wells and booster stations, the tailing reclaim water system, and the long pipe lines were priced from quotations.

The labor rates listed in Table 6-1 are effective through June 1975.

The total estimated field constructed costs for the facilities will consist of the foregoing direct field costs plus the indirect field costs; the latter encompasses the following:

(1) Temporary construction facilities, including provision for power, water, fuel, and other utilities during construction.

(2) Rental or purchase charges, plus maintenance charges, for construction equipment and small tools.

(3) Construction supervision and field office staff.

(4) Consumable supplies.

(5) Subsistence, lodging, and transportation expenses for labor and supervisory staff.

(6) Warehousing, guards, cleanup, etc.

(7) Insurance and payroll taxes.

These indirect costs are generally estimated as a percentage of direct field labor costs.

Table 6-1 - Labor Rates

Classification	Excluding Fringes	With Fringes
Boiler Maker	9.950	2.17
Brick Layer	10.895	1.26
Carpenter	8.535	1.55
Electrician	11.130	1.17
Laborer	6.480	1.32
Insulators	9.890	1.33
Oper. Engineer	8.960	1.39
Millwright	8.930	1.55
Painter	9.930	0.96
Pipe Fitter	12.710	2.11
Rodman	10.080	2.21
Iron Worker	10.080	2.21
Teamster	8.000 (est)	1.00 (est)
Sheet Metal	11.290	2.33
Cement Mason	8.665	1.28

The total capital cost development for each facility consists of the foregoing field construction costs plus allowances for the following items:

(1) The cost of engineering, procurement, and construction management.

(2) Sales taxes, where applicable.

(3) Allowance for freight.

(4) Allowance for contractors' fees for profit; these are negotiated items between the owner and contractors, and generally amount to about 5% of the total field cost of the facilities.

EXCLUSIONS

The following costs were not included in these estimates:

(1) Site acquisition, pipeline, and other right-of-way costs.

(2) Allowance for escalation of costs from the present to the time the plant construction will start.

(3) Financing charges on borrowed funds, including interest on funds expended during construction.

(4) Costs for local permits, taxes (except as previously specified), and any special studies required by local, state, or federal agencies prior to construction of plant.

(5) Owner's expenses, such as exploration and process development costs.

(6) Allowance for contingencies.

(7) Raw materials, tools, and supplies for initial operation.

(8) Spare parts.

(9) Railroad work, other than trestle.

(10) Permanent mobile equipment, railroad cars, locomotives, and car dumping mechanism.

(11) Premium time costs.

(12) Soil investigations for plant areas or wells.

CAPITAL COST ESTIMATE SUMMARIES

The capital cost estimate has been reorganized in the following summaries (see Tables 6-2 and 6-3) to separate the costs directly applicable to the plant facilities, and support systems such as fresh water supply, tailing water reclaim, main power transmission, and distribution beyond the plant area, which are special for this facility.

Table 6-2 - Mine Capital Cost Estimate Summary*

Item	Year 1	Year 2	Year 3
Major Mine Equipment	917,569	1,049,223	369,948
Preproduction Preparation	657,456	1,561,705	987,999
Subtotal (Annual)	1,575,025	2,610,928	1,357,947
		TOTAL	$5,543,900

*For details see Section 3.

Table 6-3 - Concentrator and Support Facilities
Capital Cost Estimate Summary

A. Direct Costs			
Crushing, Grinding, and Concentration			
Crushing Plant	$ 768,720		
Mill	2,144,780		
Reagent Plant	94,270		
Concrete for Equipment	487,900		
Piping and Valves	481,780		
Structural Steel for Equipment	467,720		
Instrumentation and Controls	36,280		
Painting	87,120		
Electrical	353,500		
Buildings	845,270		
Site Preparation	105,320		
Subtotal		$5,872,660	
Support Facilities			
Sand Fill Plant	117,790		
Tailing Disposal and Water Reclaim	995,060		
Freshwater System	775,750		
Main Electrical Power Transmission and Distribution	1,255,100		
Ancillaries	429,580		
Townsite	341,360		
Railroad Trestle	221,980		
Subtotal		$4,136,820	
Subtotal			$10,009,480
B. Indirect Costs			2,622,400
C. Allowance for Engineering Process Construction			1,200,000
D. Allowance for Freight			391,300
E. Sales Tax			28,000
F. Fee @ 5% of A + B + C			691,620
Total Concentrator Capital Cost			$14,942,800
Plus Mine Capital Cost From Table 6-2			5,543,900
Grand Total			$20,486,700

SECTION 7

OPERATING COST ESTIMATES

CONCENTRATOR OPERATING AND MAINTENANCE COST

Direct operating and maintenance labor costs have been calculated from the concentrator manning table included in Volume II. The basic hourly rates for each job, as indicated in Continental Material Corporation's Feasibility Study, have been increased by approximately 10% to conform with the wage increases obtained in the strike settlement reached by Magma and the unions on August 13, 1974. Shift differentials of 20 cents for the second shift and 30 cents for the third shift have also been included in the hourly rate calculations.

The manning table presented in Section 3 has been expended from that of Continental's study to show an additional helper on each shift of the concentrator operation. Also, in view of building and maintaining the tailing dam on an operating cost basis, two additional tailing dam helpers have been provided on each of the three shifts.

Straight-time labor costs have been increased by an estimated 2% to allow for overtime wages required to ensure that the concentrator's on-stream operating time will be held at the designed maximum.

DIRECT SUPERVISION AND TECHNICAL AND CLERICAL PERSONNEL

Costs for concentrator supervision and supporting manpower have been estimated as annual salaries. A tabulation of the manpower in these categories and itemized costs are included in Volume II.

FRINGE BENEFITS

Fringe benefits for concentrator personnel have been estimated at 25% of the sum of the annual hourly labor cost and salaries. This percentage does not include holiday and vacation pay for the hourly labor because this cost has already been included in the direct operating and maintenance costs (see Section 3). Included in the fringe benefits are:

- FICA Tax - Employers
- State and Federal Unemployment Tax
- Workmen's Compensation Expense
- Nonindustrial Accident Payments

- Pension/Retirement Provisions
- Group Life Insurance Premiums
- Group Medical Insurance Premiums
- Bonuses
- Miscellaneous

OPERATING SUPPLIES

Wear Iron for Crushers	
Iron consumption	0.11 lb/ton ore
Percentage wear before replacement	70%
Wear Iron for Mill Liners	
Iron consumption	0.015 lb/kW-hr
Grinding energy - total for primary and regrind mills	11.94 kW-hr/DST ore
Percentage wear before replacement	70%
Grinding Media	
Primary grind - ball consumption	0.15 lb/kW-hr
Primary grind - energy consumption	11.57 kW-hr/DST ore
Primary grind - maximum ball size	3-in. dia
Regrind - ball consumption	0.15 lb/kW-hr
Regrind - energy consumption per ton rougher concentrate	8.07 kW-hr/DST ore
Regrind - maximum ball size	1-in. dia

The costs used for wear iron and grinding media delivered to the Tucson area are taken from Parsons in-house information. Flotation reagent costs are calculated from reagent consumption data supplied by laboratory test work and current (October 1974) telephone quotations. The cost of the unslaked lime is estimated and consumption is based upon an assumed value of 80% available CaO.

MAINTENANCE SUPPLIES

The cost of maintenance supplies, such as replacements, tools, and consumables, have been taken at 3% of the field cost of the concentrator facility.

UTILITIES

The principal cost of utilities required for operation is the cost of electrical energy. The cost of pumping both the makeup water and the water reclaimed from the tailing pond is included with the electrical energy requirements shown for the concentrator. The electrical energy requirements used for the operating cost estimate are listed in Section 5. The cost of $0.012 per kW-hr is calculated from the power rate schedule also shown in Section 5. Included with utilities cost is an allowance for seasonal heating of selected areas of the concentrator facility.

Table 7-1 - Concentrator Direct Operating Cost Estimate

Item	Annual Cost	Cost/DST Ore Processed
Labor		
Operating and Maintenance - Hourly Labor	$ 622,240	$0.889
Direct Supervision, Technical and Clerical	83,200	0.119
Fringe Benefits - 25% of above	176,360	0.252
Subtotal	881,800	1.260
Operating Supplies		
Wear Iron - Crushers	70,000	0.100
- Mill Liners	100,000	0.144
- Grinding Balls	202,800	0.290
Flotation Reagents	147,000	0.210
Maintenance Supplies		
Replacements, Tools, Consumables, etc. (3% of field cost of concentrator facility = $7,500,000 x 0.03)	225,000	0.321
Utilities		
Electrical		
Crushing (1,466,823 kW-hr/yr @ $0.012/kW-hr)	17,600	0.025
Concentrator (21,067,200 kW-hr/yr @ $0.012/kW-hr)	252,800	0.361
Reagent Preparation (369,718 kW-hr/yr @ $0.012/kW-hr)	4,440	0.006
Sand Fill Plant (487,200 kW-hr/yr @ $0.012/kW-hr)	5,850	0.008
Heating		
Seasonal Furnace Oil (120 bbl/yr x $15.00/bbl)	1,800	0.003
Total	$1,909,090	$2.727

Table 7-2 - Indirect General Staff* and Administrative Services

Indirect Costs	Annual Cost	Cost/DST
Staff*		
General Manager	$ 48,000	
Secretary	8,500	
Office Manager	27,500	
Secretary Clerk	7,000	
Accountant	22,000	
Payroll Clerk	9,000	
Utility Maintenance Foreman (2)	30,000	
Purchasing Agent	27,000	
Warehouse Clerk	9,000	
Pool Driver	9,000	
Personnel Engineer	22,000	
Subtotal	$219,000	$0.3129/DST
Administrative Services**		
Related Public Relations Costs and Benefits (@ 40% of staff)	$ 87,600	
Clerical and Laboratory Supplies, Transportation, Communications (@ 25%)	54,750	
Subtotal	$142,350	$0.2034
Total	$361,350	$0.5163/DST

*Follows General Staff table presented in Section 5, Appendix 1 of Continental Material Corporation's Feasibility Study. Salaries have been escalated approximately 10%.
**Follows same table; "corporate burden" has been omitted.

MINE OPERATING COST

The criteria and details of the mine operating cost are presented in Section 3.

Table 7-3 - Annual Mine Operating Cost

Function	Cost	Unit Cost/Ton
Supervision	$ 523,600	$0.76
Labor	1,347,650	1.92
Equipment Operation	762,302	1.09
Expendable Materials and Supplies	853,062	1.22
Power	79,111	0.11
Total	$3,565,725	$5.10

Table 7-4 - Summary of Concentrator and Mine Operating Cost Estimate

Item	Annual	Per DST
Mining	$3,565,725	$5.100
Milling	1,909,090	2.727
Indirect General Staff, Misc	361,350	0.5163
Total	$5,836,165	$8.3433

SECTION 8

FINANCIAL FEASIBILITY STUDY

An interim profitability analysis has been prepared to forecast net income, cash flow, and rates of return that may reasonably be expected to accrue from development of the copper ore bodies in the Oracle Ridge Project, held by Continental Materials Corporation. The forecasts are designed to indicate probable results based upon information available at the time of the study. The assumptions upon which the forecasts are based, exclusions from the cost estimates, and sources of data are presented in this section. While the assumptions are considered reasonable, significant deviations could necessitate adjustments to the forecasts. An analysis has been performed to investigate the sensitivity of results to the selling price of copper, a primary source of uncertainty in the forecasts. The price of copper was varied in 10-cent increments from $0.60 to $1.00 per pound.

PROFITABILITY ANALYSIS CONSIDERATIONS

The profitability study is based upon a 22-year period of construction, mine development, and operation. The construction period for the concentrator and ancillary facilities was considered to occur over the initial 2 years of the period, with premine development costs distributed over the first 3 years. Receipt of initial revenue is considered to occur during the third year at an annual rate of 87.5% of the normal production rate. Similarly, all third-year costs are assumed to be 87.5% of subsequent values, except insurance. The profitability analysis is based upon February 1975 dollars. By varying the head grade, three levels of production of concentrate were considered at constant tonnage: Case A - 30,207 TPY, Case B - 34,238 TPY, and Case C - 38,603 TPY. In all cases, annual production of ore was 700,000 tons. (See Volume II for metallurgical balances and calculation of net smelter value.) Discounted after-tax cash flow rates of return were computed for five prices of copper: $0.60, $0.70, $0.80, $0.90, and $1.00 per pound.

PROFITABILITY ANALYSIS RESULTS

A summary of the indicated after-tax rates of return for Cases A, B, and C vs variable copper prices is shown in Table 8-1. As may been seen, the range of expected returns is from 6.9% to 35.0%. This relationship is shown graphically in Figure 8-1.

The results are predicated upon the cash flow and internal rate of return for the total capital investment, with no financing charges included. These are the project's inherent internal rates of return exclusive of outside financing. The average net profits after tax considerations are for the

Table 8-1 - Summary of Discounted After-Tax Rates of Return vs Variation in Price of Copper

Case	Copper (¢/lb)				
	60	70	80	90	1.00
	Rates of Return (%)				
A	6.9	14.0	19.1	23.3	27.1
B	11.8	18.2	23.0	27.3	31.3
C	15.4	21.3	26.2	30.8	35.0

variable copper selling prices shown in Table 8-2. The range of expected net profits is from $0.346 million to $4.705 million. The average annual expected net cash flows during the 20-year operating life for variable copper selling prices are shown in Table 8-3, with the range being $1.009 million to $7.760 million.

Forecasts of payback periods versus variable selling copper prices are shown in Table 8-4. The table shows that the minimum payback period is 2.5 years and the maximum is 10.6 years.

The computer printouts generated during the study are presented in the Appendix to this section. The inputs, as previously noted, are inextricably related to the assumptions furnished for the financial analysis.

BASIS FOR PROFITABILITY ANALYSIS

The basis for the forecasts of net income, cash flow, the resultant rates of return, and payback periods are presented in the following subsections.

REVENUE

The profitability analysis for the Oracle Ridge Project was predicated upon the following quantities of concentrates produced and net smelter value based on $0.70 per pound of copper:

Case	Tons of Concentrate Produced	Net Smelter Value/Ton
A	30,207	$357.27
B	34,238	$357.57
C	38,063	$357.10

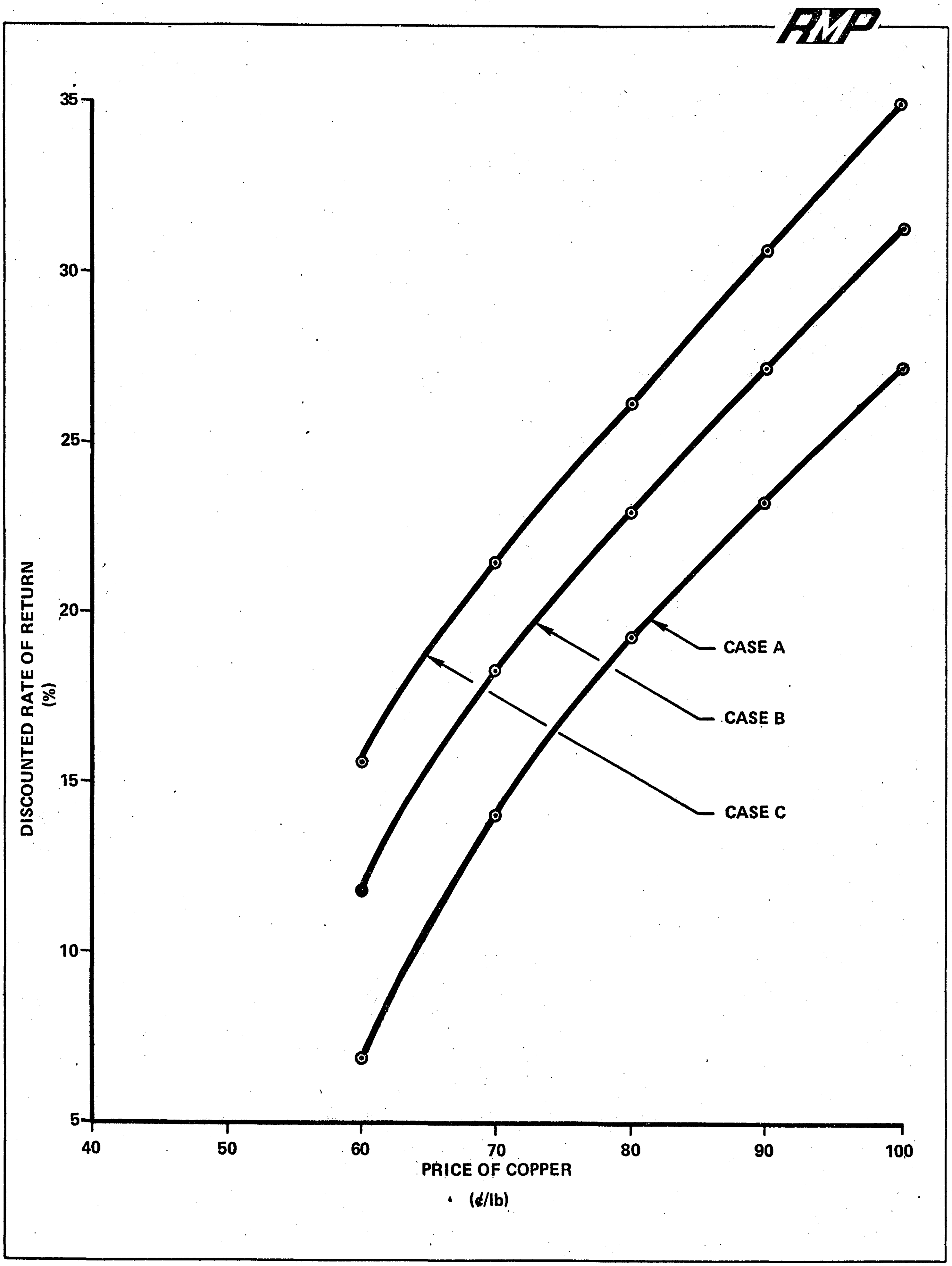

Figure 8-1 - Discounted Rate of Return
vs Price of Copper

Table 8-2 - Average After-Tax Annual Net Profits
vs Variation in Price of Copper

Case	Copper (¢/lb) 60	70	80	90	1.00
	Net Profit ($ million)				
A	0.346	0.850	1.531	2.268	3.009
B	0.658	1.374	2.214	3.054	3.895
C	1.011	1.906	2.839	3.772	4.705

Table 8-3 - Average Annual Net Cash Flow vs
Variation in Price of Copper

Case	Copper (¢/lb) 60	70	80	90	1.00
	Cash Flow ($ million)				
A	1.009	2.329	3.406	4.422	5.434
B	1.904	3.185	4.351	5.499	6.649
C	2.616	3.933	5.209	6.484	7.760

Table 8-4 - Payback Periods vs Variation in Price of Copper

	Copper (¢/1b)				
	60	70	80	90	1.00
Case	Payback Period (Years)				
A	10.6	6.1	4.6	3.8	3.3
B	7.1	4.8	3.8	3.2	2.8
C	5.6	4.1	3.4	2.9	2.5

Case B is the production of concentrate at the average grade of the ore reserve 1.82% Cu, Case A is 10% lower grade head feed or 1.64% Cu, and Case C is 10% higher than average head feed or 2.00% Cu.

Suitable modification was made for varying net smelter value per ton at the various prices of copper.

CAPITAL COSTS

The cost elements upon which the capital costs for the mill and premine development are based are shown in Section 6. Subsequent mine development costs (equipment replacement), starting with the third year of the project, are detailed in Section 3.

OPERATING COSTS

The detailed development of the annual operating costs is presented in Section 7.

TRUCK FREIGHT COSTS

Truck freight costs differ among the three cases. This expense does not vary with a change in assumed selling price of copper. See Volume II for tabulations of truck freight for each case.

WORKING CAPITAL

Working capital was assumed to be 1-1/2 month's net smelter value.

DEPRECIATION

Depreciation schedules were developed for the mill, premine development, and annual mine equipment replacement. The assets for the mill and premine development were depreciated over the project life, while annual mine equipment was depreciated over a 5 year period. The 200% declining balance method was used until it reached a point where straight-line depreciation was equal to the declining balance. This occurred in the tenth year for the mill and premine development assets and the fourth year for the mine replacement equipment. Three depreciation schedules are shown in Tables 8-5 and 8-6.

DEPLETION

The depletion allowances for copper, gold, and silver were taken to be 15%.

INSURANCE

Expense for insurance was estimated to be 1% of the fixed capital construction investment for the mill.

PROPERTY TAXES

Expenses for property taxes was estimated to be 1.75% of the fixed capital construction investment for the mill.

FEDERAL AND STATE INCOME TAXES

The income tax rate used in this study was assumed to be 52%. A 7% investment tax credit was taken in the cash flow conputation. It was estimated that 90% of the mill fixed capital investment cost is subject to the credit and 100% of the mine fixed capital investment.

Table 8-5 - Depreciation Schedules for Mill and Premine Development
(% million)

End of Year	Mill Asset Acquired	Mill Depreciation 200% Declining	Premine Development Assets	Premine Development Asset Depreciation - 200% Declining
1977	$14.943*	-	-	-
1978		$1.494	$3.207*	-
1979		1.345		$0.338
1980		1.210		0.302
1981		1.089		0.270
1982		0.980		0.242
1983		0.882		0.216
1984		0.794		0.194
1985		0.715		0.173
1986		0.643		0.155
1987		0.579		0.139
1988		0.522		0.124
1989		0.522		0.118
1990		0.521		0.117
1991		0.521		0.117
1992		0.521		0.117
1993		0.521		0.117
1994		0.521		0.117
1995		0.521		0.117
1996		0.521		0.117
1997		0.521		0.117

*Initial value of assets.

Table 8-6 - Depreciation Schedule for Mine Equipment Replacement ($ million)

End of the Year	Initial Equip- ment	Replace- ment Schedule	Equipment Depreciation 200% Declining and Straight Line for Following Assets														Total Equipment Depreciation
			2.337	0.213	0.223	0.475	0.240	0.567	0.236	0.223	0.484	0.219	0.588	0.232	0.231	0.491	
1977																	
1978	2.337*	0.213*															
1979		0.223	0.935	0.085													1.020
1980		0.475	0.561	0.051	0.089												0.701
1981		0.240	0.337	0.031	0.054	0.190											0.612
1982		0.567	0.252	0.023	0.032	0.115	0.096										0.518
1983		0.236	0.252	0.023	0.024	0.068	0.058	0.227									0.652
1984		0.223			0.024	0.051	0.035	0.136	0.094								0.340
1985		0.484				0.051	0.026	0.082	0.057	0.089							0.305
1986		0.219					0.025	0.061	0.034	0.054	0.194						0.368
1987		0.588						0.061	0.026	0.032	0.116	0.088					0.323
1988		0.232							0.025	0.024	0.070	0.053	0.235				0.407
1989		0.231								0.024	0.052	0.032	0.141	0.093			0.342
1990		0.491									0.052	0.023	0.085	0.056	0.092		0.308
1991												0.023	0.064	0.033	0.056	0.196	0.372
1992													0.063	0.025	0.033	0.118	0.239
1993														0.025	0.025	0.071	0.121
1994															0.025	0.053	0.078
1995																0.053	0.053
1996																	
1997																	

*Initial value of assets.

APPENDIX TO SECTION 8

APPENDIX TO SECTION 8

PARSONS CASH FLOW MODEL

Parsons cash flow computer evaluation model was used to evaluate the financial feasibility of the Continental Material Corporation's Oracle Ridge Project. This Appendix contains the computer printouts for Cases A, B, and C, together with the variations in the selling price of copper.

THE RALPH M. PARSONS COMPANY

CASH FLOW CASE EVALUATION

CONTINENTAL COPPER, INC.
JOB NO. 5302-001 DATE 02/12/75

CASE A - 60 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
(REVENUE)											
NET SMELTER RETURN	0.000	0.000	7.845	8.966	8.966	8.966	8.966	8.966	8.966	8.966	8.966
... T O T A L ...	0.000	0.000	7.845	8.966	8.966	8.966	8.966	8.965	8.966	8.966	8.966
(COSTS)											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.119	.136	.136	.136	.136	.136	.136	.136	.136
... T O T A L ...	0.000	0.000	5.603	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.215	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	.374	0.000	.185	.306	.421	.416	.627	.695	.708
COST + DEPR + DEPL	0.000	0.000	7.471	9.306	8.781	8.660	8.545	8.550	8.339	8.271	8.258
PROFIT BEFORE TAX	0.000	0.000	.374	-.120	.185	.306	.421	.416	.627	.695	.708
INCOME TAXES	0.000	0.000	.194	0.000	.034	.159	.219	.217	.326	.361	.368
TAX CREDIT	0.000	0.000	.194	0.000	.034	.159	.219	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	.374	-.120	.185	.306	.421	.200	.301	.334	.340
AVERAGE ANNUAL NET PROFIT		.346									
GROSS CF, OPERATIONS	0.000	0.000	2.242	2.563	2.563	2.563	2.582	2.366	2.257	2.222	2.215
ACCUM GROSS CASHFLOW	0.000	0.000	2.242	4.825	7.406	9.991	12.573	14.940	17.197	19.418	21.633
(INVESTMENT)											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	1.121	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-19.579	-17.219	-15.111	-12.768	-10.753	-8.622	-6.588	-4.851	-2.855
NET CASH FLOW	-10.541	-8.588	-.450	2.360	2.108	2.343	2.015	2.130	2.034	1.738	1.996
ACCUM NET CASHFLOW	-10.541	-19.129	-19.579	-17.219	-15.111	-12.768	-10.753	-8.622	-6.588	-4.851	-2.855

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
(REVENUE)											
NET SMELTER RETURN	8.966	8.966	8.966	8.966	8.966	8.966	8.966	8.966	8.966	8.966	8.966
... T O T A L ...	8.966	8.966	8.966	8.966	8.966	8.966	8.966	8.966	8.966	8.966	8.966
(COSTS)											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136
... T O T A L ...	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	.771	.765	.800	.818	.786	.852	.911	.933	.945	.972	.972
COST + DEPR + DEPL	8.195	8.201	8.166	8.148	8.180	8.114	8.055	8.033	8.021	7.994	7.994
PROFIT BEFORE TAX	.771	.765	.800	.818	.786	.852	.911	.933	.945	.972	.972
INCOME TAXES	.401	.398	.416	.425	.409	.443	.474	.485	.492	.505	.505
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	.370	.367	.384	.393	.377	.409	.438	.448	.454	.467	.467
AVERAGE ANNUAL NET PROFIT			.346								
GROSS CF, OPERATIONS	2.182	2.185	2.167	2.156	2.174	2.140	2.109	2.098	2.091	2.078	2.078
ACCUM GROSS CASHFLOW	23.815	26.000	28.167	30.324	32.499	34.636	36.747	38.845	40.937	43.014	45.092
(INVESTMENT)											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.121
ACCUM GROSS CF TOTAL	-1.261	.692	2.628	4.293	6.468	8.607	10.716	12.814	14.906	16.983	20.182
NET CASH FLOW	1.594	1.953	1.936	1.666	2.174	2.140	2.109	2.098	2.091	2.078	3.199
ACCUM NET CASHFLOW	-1.261	.692	2.628	4.293	6.468	8.607	10.716	12.814	14.906	16.983	20.182

----- NET CASH FLOW DISCOUNTED AT 6.894 / PERIOD -----

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
FACTORS	.936	.875	.819	.766	.717	.670	.627	.587	.549	.513	.480
PRESENT VALUE	-9.861	-7.516	-.368	1.808	1.510	1.571	1.264	1.250	1.116	.892	.959
CUMULATIVE VALUE	-9.861	-17.377	-17.746	-15.938	-14.428	-12.857	-11.593	-10.343	-9.227	-8.335	-7.377

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
FACTORS	.449	.420	.393	.368	.344	.322	.301	.282	.264	.247	.231
PRESENT VALUE	.716	.821	.761	.613	.748	.688	.635	.591	.551	.512	.738
CUMULATIVE VALUE	-6.660	-5.839	-5.078	-4.465	-3.717	-3.029	-2.393	-1.802	-1.250	-.738	-.000

PAY-OUT PERIOD FROM START OF OPERATION IS 10.646

8A-2

THE RALPH M. PARSONS COMPANY

CASH FLOW CASE EVALUATION

CONTINENTAL COPPER, INC.
JOB NO. 5382-001 DATE 02/12/75

CASE A - 70 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	9.443	10.792	10.792	10.792	10.792	10.792	10.792	10.792	10.792
... T O T A L ...	0.000	0.000	9.443	10.792	10.792	10.792	10.792	10.792	10.792	10.792	10.792
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.119	.136	.136	.136	.136	.136	.136	.136	.136
... T O T A L ...	0.000	0.000	5.603	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.215	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	1.173	.853	1.098	1.219	1.335	1.330	1.541	1.608	1.619
COST + DEPR + DEPL	0.000	0.000	8.270	9.939	9.694	9.573	9.458	9.463	9.252	9.184	9.168
PROFIT BEFORE TAX	0.000	0.000	1.173	.853	1.098	1.219	1.335	1.330	1.541	1.608	1.624
INCOME TAXES	0.000	0.000	.610	.444	.571	.634	.694	.691	.801	.836	.845
TAX CREDIT	0.000	0.000	.610	.444	.051	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	1.173	.853	.579	.585	.641	.638	.739	.772	.780
AVERAGE ANNUAL NET PROFIT .850											
GROSS CF, OPERATIONS	0.000	0.000	3.840	4.409	3.890	3.775	3.715	3.718	3.608	3.573	3.564
ACCUM GROSS CASHFLOW	0.000	0.000	3.840	8.249	12.139	15.914	19.629	23.346	26.954	30.527	34.092
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.956	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	1.349	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-18.209	-14.023	-10.608	-7.073	-3.925	-.444	2.941	6.030	9.376
NET CASH FLOW	-10.541	-8.588	.920	4.186	3.415	3.535	3.148	3.482	3.365	3.089	3.345
ACCUM NET CASHFLOW	-10.541	-19.129	-18.209	-14.023	-10.608	-7.073	-3.925	-.444	2.941	6.030	9.376
----- NET CASH FLOW DISCOUNTED AT 14.005 / PERIOD -----											
FACTORS	.877	.769	.675	.592	.519	.455	.400	.350	.307	.270	.236
PRESENT VALUE	-9.246	-6.608	.621	2.478	1.773	1.610	1.258	1.220	1.040	.833	.791
CUMULATIVE VALUE	-9.246	-15.854	-15.233	-12.755	-10.982	-9.372	-8.114	-6.894	-5.854	-5.021	-4.230

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	10.792	10.792	10.792	10.792	10.792	10.792	10.792	10.792	10.792	10.792	10.792
... T O T A L ...	10.792	10.792	10.792	10.792	10.792	10.792	10.792	10.792	10.792	10.792	10.792
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136
... T O T A L ...	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	1.619	1.619	1.619	1.619	1.619	1.619	1.619	1.619	1.619	1.619	1.619
COST + DEPR + DEPL	9.043	9.055	8.984	8.949	9.013	8.880	8.762	8.719	8.694	8.641	8.641
PROFIT BEFORE TAX	1.749	1.737	1.808	1.843	1.779	1.912	2.030	2.073	2.098	2.151	2.151
INCOME TAXES	.910	.903	.940	.958	.925	.994	1.056	1.078	1.091	1.119	1.119
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	.840	.834	.868	.885	.854	.918	.974	.995	1.007	1.033	1.033
GROSS CF, OPERATIONS	3.499	3.506	3.469	3.451	3.484	3.415	3.353	3.331	3.318	3.290	3.290
ACCUM GROSS CASHFLOW	37.591	41.097	44.565	48.016	51.500	54.914	58.268	61.599	64.917	68.207	71.497
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.349
ACCUM GROSS CF TOTAL	12.287	15.561	18.798	21.757	25.241	28.655	32.009	35.340	38.658	41.948	46.587
NET CASH FLOW	2.911	3.274	3.238	2.959	3.484	3.415	3.353	3.331	3.318	3.290	4.639
ACCUM NET CASHFLOW	12.287	15.561	18.798	21.757	25.241	28.655	32.009	35.340	38.658	41.948	46.587
FACTORS	.207	.182	.160	.140	.123	.108	.094	.083	.073	.064	.056
PRESENT VALUE	.604	.596	.517	.414	.428	.368	.317	.276	.241	.210	.259
CUMULATIVE VALUE	-3.626	-3.030	-2.513	-2.099	-1.671	-1.303	-.987	-.711	-.469	-.260	-.000

PAY-OUT PERIOD FROM START OF OPERATION IS 6.131

THE RALPH M. PARSONS COMPANY

CASH FLOW CASE EVALUATION

CONTINENTAL COPPER,INC.
JOB NO. 5382-001 DATE 02/12/75

CASE A - 80 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	11.040	12.618	12.618	12.618	12.618	12.618	12.618	12.618	12.618
... T O T A L ...	0.000	0.000	11.040	12.618	12.618	12.618	12.618	12.618	12.618	12.618	12.618
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.119	.136	.136	.136	.136	.136	.136	.136	.136
... T O T A L ...	0.000	0.000	5.603	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	1.656	1.766	1.893	1.893	1.893	1.893	1.893	1.893	1.893
COST + DEPR + DEPL	0.000	0.000	8.753	10.852	10.489	10.247	10.016	10.025	9.604	9.469	9.442
PROFIT BEFORE TAX	0.000	0.000	2.287	1.766	2.129	2.371	2.602	2.592	3.014	3.149	3.176
INCOME TAXES	0.000	0.000	1.189	.918	1.107	1.233	1.353	1.348	1.567	1.638	1.652
TAX CREDIT	0.000	0.000	1.105	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	2.203	.848	1.022	1.138	1.249	1.244	1.447	1.512	1.525
AVERAGE ANNUAL NET PROFIT		1.531									
GROSS CF, OPERATIONS	0.000	0.000	5.353	5.317	5.128	5.002	4.882	4.887	4.668	4.597	4.583
ACCUM GROSS CASHFLOW	0.000	0.000	5.353	10.669	15.797	20.799	25.681	30.568	35.236	39.833	44.416
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	1.577	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-16.924	-11.831	-7.178	-2.416	1.899	6.550	10.995	15.108	19.472
NET CASH FLOW	-10.541	-8.588	2.205	5.094	4.653	4.762	4.315	4.651	4.445	4.113	4.364
ACCUM NET CASHFLOW	-10.541	-19.129	-16.924	-11.831	-7.178	-2.416	1.899	6.550	10.995	15.108	19.472
----- NET CASH FLOW DISCOUNTED AT 19.140 / PERIOD -----											
FACTORS	.839	.705	.591	.496	.417	.350	.293	.246	.207	.174	.146
PRESENT VALUE	-8.848	-6.050	1.304	2.528	1.938	1.665	1.266	1.146	.919	.714	.636
CUMULATIVE VALUE	-8.848	-14.898	-13.594	-11.066	-9.128	-7.463	-6.196	-5.050	-4.131	-3.418	-2.782

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	12.618	12.618	12.618	12.618	12.618	12.618	12.618	12.618	12.618	12.618	12.618
... T O T A L ...	12.618	12.618	12.618	12.618	12.618	12.618	12.618	12.618	12.618	12.618	12.618
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136
... T O T A L ...	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.306	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	1.893	1.893	1.893	1.893	1.893	1.893	1.893	1.893	1.893	1.893	1.893
COST + DEPR + DEPL	9.317	9.329	9.258	9.223	9.287	9.154	9.036	8.993	8.968	8.915	8.915
PROFIT BEFORE TAX	3.301	3.289	3.360	3.395	3.331	3.464	3.582	3.625	3.650	3.703	3.703
INCOME TAXES	1.717	1.710	1.747	1.766	1.732	1.861	1.863	1.885	1.898	1.926	1.926
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	1.585	1.579	1.613	1.630	1.599	1.663	1.720	1.740	1.752	1.778	1.778
GROSS CF, OPERATIONS	4.518	4.525	4.488	4.469	4.503	4.434	4.372	4.350	4.337	4.309	4.309
ACCUM GROSS CASHFLOW	48.935	53.459	57.947	62.416	66.919	71.352	75.725	80.074	84.411	88.721	93.030
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.577
ACCUM GROSS CF TOTAL	23.403	27.695	31.952	35.929	40.432	44.865	49.238	53.588	57.924	62.234	68.120
NET CASH FLOW	3.930	4.293	4.257	3.977	4.503	4.434	4.372	4.350	4.337	4.309	5.886
ACCUM NET CASHFLOW	23.403	27.695	31.952	35.929	40.432	44.865	49.238	53.588	57.924	62.234	68.120
FACTORS	.122	.103	.086	.072	.061	.051	.043	.036	.030	.025	.021
PRESENT VALUE	.481	.441	.367	.256	.273	.226	.187	.156	.131	.109	.125
CUMULATIVE VALUE	-2.301	-1.861	-1.494	-1.207	-.933	-.707	-.521	-.364	-.234	-.125	.000

PAY-OUT PERIOD FROM START OF OPERATION IS 4.560

THE RALPH M. PARSONS COMPANY

CASH FLOW CASE EVALUATION

CONTINENTAL COPPER, INC.
JOB NO. 5382-001 DATE 02/12/75

CASE A - 90 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]																						
NET SMELTER RETURN	0.000	0.000	12.638	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444
... T O T A L ...	0.000	0.000	12.638	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444	14.444
[COSTS]																						
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.119	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136
... T O T A L ...	0.000	0.000	5.603	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	0.000	0.000	1.896	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167	2.167
COST + DEPR + DEPL	0.000	0.000	8.993	11.253	10.763	10.521	10.290	10.300	9.878	9.743	9.716	9.591	9.603	9.532	9.497	9.561	9.428	9.310	9.267	9.242	9.189	9.189
PROFIT BEFORE TAX	0.000	0.000	3.645	3.191	3.681	3.923	4.154	4.144	4.566	4.701	4.728	4.853	4.841	4.912	4.947	4.883	5.016	5.134	5.177	5.202	5.255	5.255
INCOME TAXES	0.000	0.000	1.896	1.660	1.914	2.040	2.160	2.155	2.375	2.445	2.459	2.524	2.518	2.554	2.573	2.539	2.609	2.670	2.692	2.705	2.733	2.733
TAX CREDIT	0.000	0.000	1.105	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	2.855	1.532	1.767	1.883	1.994	1.989	2.192	2.257	2.270	2.330	2.324	2.358	2.375	2.344	2.408	2.465	2.485	2.497	2.523	2.523
AVERAGE ANNUAL NET PROFIT		2.268																				
GROSS CF, OPERATIONS	0.000	0.000	6.244	6.401	6.147	6.021	5.901	5.906	5.686	5.616	5.602	5.537	5.543	5.507	5.488	5.522	5.452	5.391	5.369	5.356	5.328	5.328
ACCUM GROSS CASHFLOW	0.000	0.000	6.244	12.646	18.793	24.813	30.714	36.620	42.307	47.923	53.525	59.062	64.606	70.112	75.601	81.122	86.575	91.966	97.335	102.690	108.019	113.347
[INVESTMENT]																						
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	1.806	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.806
ACCUM GROSS CF TOTAL	-10.541	-19.129	-16.262	-10.083	-4.411	1.369	6.703	12.373	17.837	22.969	28.352	33.301	38.613	43.888	48.885	54.466	59.859	65.250	70.619	75.974	81.303	88.437
NET CASH FLOW	-10.541	-8.588	2.867	6.178	5.672	5.781	5.334	5.670	5.463	5.132	5.383	4.949	5.311	5.276	4.996	5.522*	5.452	5.391	5.369	5.356	5.328	7.134
ACCUM NET CASHFLOW	-10.541	-19.129	-16.262	-10.083	-4.411	1.369	6.703	12.373	17.837	22.969	28.352	33.301	38.613	43.889	48.885	54.466	59.859	65.250	70.619	75.974	81.303	88.437

----- NET CASH FLOW DISCOUNTED AT 23.255 / PERIOD -----

	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
FACTORS	.811	.658	.534	.433	.352	.285	.231	.188	.152	.124	.100	.081	.066	.054	.043	.035	.029	.023	.019	.015	.012	.010
PRESENT VALUE	-8.552	-5.653	1.531	2.677	1.994	1.649	1.234	1.065	.832	.634	.540	.403	.351	.283	.217	.195	.156	.125	.101	.082	.066	.072
CUMULATIVE VALUE	-8.552	-14.205	-12.674	-9.997	-8.003	-6.354	-5.120	-4.055	-3.223	-2.589	-2.049	-1.646	-1.296	-1.013	-.796	-.602	-.446	-.321	-.220	-.138	-.072	-.006

PAY-OUT PERIOD FROM START OF OPERATION IS 3.763

CASH FLOW CASE EVALUATION

THE RALPH M. PARSONS COMPANY

CONTINENTAL COPPER, INC.
JOB NO. 5382-001 DATE 02/12/75

CASE A 100 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	14.236	16.270	16.270	16.270	16.270	16.270	16.270	16.270	16.270
... T O T A L ...	0.000	0.000	14.236	16.270	16.270	16.270	16.270	16.270	16.270	16.270	16.270
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.119	.136	.136	.136	.136	.136	.136	.136	.136
... T O T A L ...	0.000	0.000	5.603	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.761	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	2.135	2.441	2.441	2.441	2.441	2.441	2.441	2.441	2.441
COST + DEPR + DEPL	0.000	0.000	9.232	11.527	11.037	10.795	10.564	10.574	10.152	10.017	9.990
PROFIT BEFORE TAX	0.000	0.000	5.004	4.743	5.234	5.476	5.707	5.697	6.119	6.254	6.281
INCOME TAXES	0.000	0.000	2.602	2.467	2.721	2.847	2.967	2.962	3.182	3.252	3.266
TAX CREDIT	0.000	0.000	1.105	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	3.507	2.277	2.512	2.628	2.739	2.734	2.937	3.002	3.015
AVERAGE ANNUAL NET PROFIT		3.009									
GROSS CF, OPERATIONS	0.000	0.000	7.136	7.420	7.166	7.040	6.920	6.925	6.705	6.635	6.621
ACCUM GROSS CASHFLOW	0.000	0.000	7.136	14.557	21.722	28.762	35.661	42.606	49.312	55.947	62.568
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	2.033	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-15.597	-8.399	-1.709	5.091	11.443	18.132	24.615	30.766	37.168
NET CASH FLOW	-10.541	-8.588	3.532	7.197	6.691	6.800	6.353	6.689	6.482	6.151	6.402
ACCUM NET CASHFLOW	-10.541	-19.129	-15.597	-8.399	-1.709	5.091	11.443	18.132	24.615	30.766	37.168
----- NET CASH FLOW DISCOUNTED AT 27.053 / PERIOD -----											
FACTORS	.787	.619	.488	.384	.302	.238	.187	.147	.116	.091	.072
PRESENT VALUE	-8.297	-5.320	1.722	2.762	2.021	1.617	1.189	.985	.751	.561	.460
CUMULATIVE VALUE	-8.297	-13.617	-11.894	-9.132	-7.112	-5.495	-4.306	-3.321	-2.570	-2.009	-1.549

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	16.270	16.270	16.270	16.270	16.270	16.270	16.270	16.270	16.270	16.270	16.270
... T O T A L ...	16.270	16.270	16.270	16.270	16.270	16.270	16.270	16.270	16.270	16.270	16.270
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136	.136
... T O T A L ...	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383	6.383
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.306	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	2.441	2.441	2.441	2.441	2.441	2.441	2.441	2.441	2.441	2.441	2.441
COST + DEPR + DEPL	9.865	9.877	9.806	9.771	9.835	9.702	9.584	9.541	9.516	9.463	9.463
PROFIT BEFORE TAX	6.406	6.394	6.465	6.500	6.436	6.569	6.687	6.730	6.755	6.808	6.808
INCOME TAXES	3.331	3.325	3.362	3.380	3.346	3.416	3.477	3.499	3.512	3.540	3.540
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	3.075	3.069	3.103	3.120	3.089	3.153	3.210	3.230	3.242	3.268	3.268
GROSS CF, OPERATIONS	6.556	6.562	6.525	6.507	6.541	6.471	6.410	6.388	6.375	6.347	6.347
ACCUM GROSS CASHFLOW	69.124	75.686	82.212	88.719	95.260	101.731	108.141	114.529	120.903	127.251	133.598
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-2.033
ACCUM GROSS CF TOTAL	43.136	49.466	55.761	61.776	68.317	74.788	81.198	87.586	93.960	100.308	108.688
NET CASH FLOW	5.968	6.330	6.294	6.015	6.541	6.471	6.410	6.388	6.375	6.347	8.380
ACCUM NET CASHFLOW	43.136	49.466	55.761	61.776	68.317	74.788	81.198	87.586	93.960	100.308	108.688
----- NET CASH FLOW DISCOUNTED AT 27.053 / PERIOD -----											
FACTORS	.057	.044	.035	.028	.022	.017	.013	.011	.008	.007	.005
PRESENT VALUE	.337	.282	.220	.166	.142	.110	.086	.068	.053	.042	.043
CUMULATIVE VALUE	-1.212	-.930	-.710	-.544	-.402	-.292	-.205	-.138	-.085	-.043	.000

PAY-OUT PERIOD FROM START OF OPERATION IS 3.251

THE RALPH M. PARSONS COMPANY

CASH FLOW CASE EVALUATION

CONTINENTAL COPPER,INC.
JOB NO. 5382-001 DATE 02/12/75

CASE B - 60 CENT COPPER

VARIATICNS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	8.901	10.173	10.173	10.173	10.173	10.173	10.173	10.173	10.173
... T O T A L ...	0.000	0.000	8.901	10.173	10.173	10.173	10.173	10.173	10.173	10.173	10.173
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.134	.154	.154	.154	.154	.154	.154	.154	.154
... T O T A L ...	0.000	0.000	5.618	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETICN	0.000	0.000	.894	.535	.780	.901	1.016	1.011	1.222	1.290	1.303
COST + DEPR + DEPL	0.000	0.000	8.007	9.639	9.394	9.273	9.157	9.162	8.951	8.884	8.870
PROFIT BEFORE TAX	0.000	0.000	.894	.535	.780	.901	1.016	1.011	1.222	1.290	1.303
INCOME TAXES	0.000	0.000	.465	.278	.405	.468	.528	.526	.635	.671	.678
TAX CREDIT	0.000	0.000	.465	.278	.362	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	.894	.535	.736	.432	.488	.485	.587	.619	.625
AVERAGE ANNUAL NET PROFIT		.658									
GROSS CF, OPERATIONS	0.000	0.000	3.283	3.772	3.729	3.304	3.244	3.246	3.137	3.101	3.094
ACCUM GROSS CASHFLOW	0.000	0.000	3.283	7.055	10.784	14.087	17.331	20.577	23.714	26.815	29.910
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.566	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	1.272	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-18.689	-15.140	-11.886	-8.823	-6.146	-3.136	-.222	2.395	5.271
NET CASH FLOW	-10.541	-8.588	.440	3.549	3.254	3.064	2.677	3.010	2.914	2.617	2.875
ACCUM NET CASHFLOW	-10.541	-19.129	-18.689	-15.140	-11.886	-8.823	-6.146	-3.136	-.222	2.395	5.271
----- NET CASH FLOW DISCOUNTED AT 11.813 % PERIOD -----											
FACTORS	.894	.800	.715	.640	.572	.512	.458	.409	.366	.327	.293
PRESENT VALUE	-9.427	-6.869	.315	2.271	1.862	1.568	1.225	1.232	1.067	.857	.842
CUMULATIVE VALUE	-9.427	-16.296	-15.982	-13.711	-11.850	-10.282	-9.057	-7.825	-6.758	-5.901	-5.059

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	10.173	10.173	10.173	10.173	10.173	10.173	10.173	10.173	10.173	10.173	10.173
... T O T A L ...	10.173	10.173	10.173	10.173	10.173	10.173	10.173	10.173	10.173	10.173	10.173
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.154	.154	.154	.154	.154	.154	.154	.154	.154	.154	.154
... T O T A L ...	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETICN	1.366	1.360	1.395	1.413	1.381	1.447	1.506	1.526	1.526	1.526	1.526
COST + DEPR + DEPL	8.808	8.814	8.778	8.761	8.793	8.726	8.667	8.644	8.619	8.566	8.566
PROFIT BEFORE TAX	1.366	1.360	1.395	1.413	1.381	1.447	1.506	1.529	1.554	1.607	1.607
INCOME TAXES	.710	.707	.725	.735	.718	.752	.783	.795	.808	.836	.836
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	.655	.653	.670	.678	.663	.695	.723	.734	.746	.771	.771
GROSS CF, OPERATIONS	3.062	3.065	3.047	3.038	3.054	3.020	2.989	2.977	2.964	2.936	2.936
ACCUM GROSS CASHFLOW	32.972	36.037	39.083	42.121	45.175	48.195	51.183	54.160	57.124	60.061	62.997
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.272
ACCUM GROSS CF TOTAL	7.745	10.578	13.393	15.939	18.993	22.013	25.001	27.978	30.942	33.879	38.087
NET CASH FLOW	2.474	2.833	2.816	2.546	3.054	3.020	2.989	2.977	2.964	2.936	4.208
ACCUM NET CASHFLOW	7.745	10.578	13.393	15.939	18.993	22.013	25.001	27.978	30.942	33.879	38.087
FACTORS	.262	.234	.209	.187	.168	.150	.134	.120	.107	.096	.086
PRESENT VALUE	.648	.663	.590	.477	.512	.452	.401	.357	.318	.281	.361
CUMULATIVE VALUE	-4.411	-3.748	-3.158	-2.681	-2.170	-1.717	-1.317	-.960	-.642	-.361	.000

PAY-OUT PERIOD FROM START OF OPERATION IS 7.085

THE RALPH M. PARSONS COMPANY

CASH FLOW CASE EVALUATION

CONTINENTAL COPPER, INC.
JOB NO. 5382-001 DATE 02/12/75

CASE 8 - 70 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	10.711	12.242	12.242	12.242	12.242	12.242	12.242	12.242	12.242
... T O T A L ...	0.000	0.000	10.711	12.242	12.242	12.242	12.242	12.242	12.242	12.242	12.242
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.134	.154	.154	.154	.154	.154	.154	.154	.154
... T O T A L ...	0.000	0.000	5.618	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.215	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	1.607	1.569	1.814	1.836	1.836	1.836	1.836	1.836	1.836
COST + DEPR + DEPL	0.000	0.000	8.719	10.673	10.428	10.208	9.977	9.987	9.565	9.430	9.403
PROFIT BEFORE TAX	0.000	0.000	1.992	1.569	1.814	2.034	2.265	2.255	2.677	2.812	2.839
INCOME TAXES	0.000	0.000	1.036	.816	.943	1.058	1.178	1.172	1.392	1.462	1.476
TAX CREDIT	0.000	0.000	1.036	.069	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	1.992	.822	.871	.976	1.087	1.082	1.285	1.350	1.363
AVERAGE ANNUAL NET PROFIT		1.374									
GROSS CF, OPERATIONS	0.000	0.000	5.093	5.094	4.898	4.783	4.663	4.669	4.449	4.379	4.365
ACCUM GROSS CASHFLOW	0.000	0.000	5.093	10.187	15.085	19.868	24.532	29.200	33.649	38.028	42.393
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	1.530	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-17.137	-12.266	-7.843	-3.300	.797	5.229	9.455	13.350	17.496
NET CASH FLOW	-10.541	-8.588	1.992	4.871	4.423	4.543	4.096	4.433	4.226	3.895	4.146
ACCUM NET CASHFLOW	-10.541	-19.129	-17.137	-12.266	-7.843	-3.300	.797	5.229	9.455	13.350	17.496
----- NET CASH FLOW DISCOUNTED AT 18.162 / PERIOD -----											
FACTORS	.846	.716	.606	.513	.434	.367	.311	.263	.223	.188	.160
PRESENT VALUE	-8.921	-6.151	1.207	2.499	1.920	1.669	1.274	1.166	.941	.734	.661
CUMULATIVE VALUE	-8.921	-15.072	-13.864	-11.366	-9.446	-7.776	-6.503	-5.336	-4.395	-3.661	-3.000

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	12.242	12.242	12.242	12.242	12.242	12.242	12.242	12.242	12.242	12.242	12.242
... T O T A L ...	12.242	12.242	12.242	12.242	12.242	12.242	12.242	12.242	12.242	12.242	12.242
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.154	.154	.154	.154	.154	.154	.154	.154	.154	.154	.154
... T O T A L ...	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	1.836	1.836	1.836	1.836	1.836	1.836	1.836	1.836	1.836	1.836	1.836
COST + DEPR + DEPL	9.278	9.290	9.219	9.184	9.248	9.115	8.997	8.954	8.929	8.876	8.876
PROFIT BEFORE TAX	2.964	2.952	3.023	3.058	2.994	3.127	3.245	3.288	3.313	3.366	3.366
INCOME TAXES	1.541	1.535	1.572	1.590	1.557	1.626	1.687	1.710	1.723	1.750	1.750
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	1.423	1.417	1.451	1.468	1.437	1.501	1.557	1.578	1.590	1.616	1.616
GROSS CF, OPERATIONS	4.300	4.306	4.269	4.251	4.284	4.215	4.154	4.131	4.118	4.091	4.091
ACCUM GROSS CASHFLOW	46.693	50.999	55.268	59.519	63.804	68.019	72.172	76.304	80.422	84.513	88.604
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.530
ACCUM GROSS CF TOTAL	21.208	25.282	29.320	33.079	37.364	41.579	45.732	49.864	53.982	58.073	63.694
NET CASH FLOW	3.712	4.074	4.038	3.759	4.284	4.215	4.154	4.131	4.118	4.091	5.621
ACCUM NET CASHFLOW	21.208	25.282	29.320	33.079	37.364	41.579	45.732	49.864	53.982	58.073	63.694
FACTORS	.135	.114	.097	.082	.069	.059	.050	.042	.036	.030	.025
PRESENT VALUE	.501	.465	.390	.308	.297	.247	.206	.173	.146	.123	.143
CUMULATIVE VALUE	-2.499	-2.033	-1.643	-1.335	-1.039	-.792	-.586	-.412	-.266	-.143	.000

PAY-OUT PERIOD FROM START OF OPERATION IS 4.806

THE RALPH M. PARSONS COMPANY

CASH FLOW CASE EVALUATION

CONTINENTAL COPPER, INC.
JOB NO. 5382-001 DATE 02/12/75

CASE B - 80 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	12.539	14.331	14.331	14.331	14.331	14.331	14.331	14.331	14.331
... T O T A L ...	0.000	0.000	12.539	14.331	14.331	14.331	14.331	14.331	14.331	14.331	14.331
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.134	.154	.154	.154	.154	.154	.154	.154	.154
... T O T A L ...	0.000	0.000	5.618	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	1.881	2.150	2.150	2.150	2.150	2.150	2.150	2.150	2.150
COST + DEPR + DEPL	0.000	0.000	8.993	11.254	10.764	10.522	10.291	10.301	9.879	9.744	9.717
PROFIT BEFORE TAX	0.000	0.000	3.546	3.077	3.567	3.809	4.040	4.030	4.452	4.587	4.614
INCOME TAXES	0.000	0.000	1.844	1.600	1.855	1.981	2.101	2.096	2.315	2.385	2.399
TAX CREDIT	0.000	0.000	1.105	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	2.807	1.477	1.712	1.828	1.939	1.935	2.137	2.202	2.215
AVERAGE ANNUAL NET PROFIT		2.214									
GROSS CF, OPERATIONS	0.000	0.000	6.182	6.330	6.075	5.949	5.829	5.834	5.615	5.545	5.531
ACCUM GROSS CASHFLOW	0.000	0.000	6.182	12.512	18.587	24.536	30.365	36.199	41.814	47.358	52.889
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	1.791	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-16.309	-10.202	-4.602	1.107	6.369	11.967	17.359	22.419	27.731
NET CASH FLOW	-10.541	-8.588	2.820	6.107	5.600	5.709	5.262	5.598	5.392	5.061	5.312
ACCUM NET CASHFLOW	-10.541	-19.129	-16.309	-10.202	-4.602	1.107	6.369	11.967	17.359	22.419	27.731
----- NET CASH FLOW DISCOUNTED AT 22.977 / PERIOD -----											
FACTORS	.813	.661	.538	.437	.356	.289	.235	.191	.155	.126	.103
PRESENT VALUE	-8.571	-5.679	1.516	2.670	1.991	1.651	1.237	1.070	.838	.640	.546
CUMULATIVE VALUE	-8.571	-14.250	-12.734	-10.064	-8.073	-6.422	-5.185	-4.115	-3.277	-2.637	-2.092

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	14.331	14.331	14.331	14.331	14.331	14.331	14.331	14.331	14.331	14.331	14.331
... T O T A L ...	14.331	14.331	14.331	14.331	14.331	14.331	14.331	14.331	14.331	14.331	14.331
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.154	.154	.154	.154	.154	.154	.154	.154	.154	.154	.154
... T O T A L ...	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	2.150	2.150	2.150	2.150	2.150	2.150	2.150	2.150	2.150	2.150	2.150
COST + DEPR + DEPL	9.592	9.604	9.533	9.498	9.562	9.429	9.311	9.268	9.243	9.190	9.190
PROFIT BEFORE TAX	4.739	4.727	4.798	4.833	4.769	4.902	5.020	5.063	5.088	5.141	5.141
INCOME TAXES	2.464	2.458	2.495	2.513	2.480	2.549	2.611	2.633	2.646	2.674	2.674
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	2.275	2.269	2.303	2.320	2.289	2.353	2.410	2.430	2.442	2.468	2.468
GROSS CF, OPERATIONS	5.466	5.472	5.435	5.417	5.450	5.381	5.319	5.297	5.284	5.256	5.256
ACCUM GROSS CASHFLOW	58.355	63.826	69.261	74.678	80.128	85.509	90.828	96.125	101.409	106.666	111.922
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.791
ACCUM GROSS CF TOTAL	32.609	37.848	43.052	47.977	53.427	58.808	64.127	69.424	74.708	79.965	87.012
NET CASH FLOW	4.878	5.240	5.204	4.925	5.450	5.381	5.319	5.297	5.284	5.256	7.047
ACCUM NET CASHFLOW	32.609	37.848	43.052	47.977	53.427	58.808	64.127	69.424	74.708	79.965	87.012
----- NET CASH FLOW DISCOUNTED AT 22.977 / PERIOD -----											
FACTORS	.084	.068	.055	.045	.037	.030	.024	.020	.016	.013	.011
PRESENT VALUE	.408	.356	.288	.221	.199	.160	.129	.104	.084	.068	.074
CUMULATIVE VALUE	-1.684	-1.328	-1.040	-.819	-.620	-.460	-.331	-.227	-.143	-.075	-.000

PAY-OUT PERIOD FROM START OF OPERATION IS 3.806

CASH FLOW CASE EVALUATION

THE RALPH M. PARSONS COMPANY

CONTINENTAL COPPER, INC.
JOB NO. 5382-001 DATE 02/12/75

CASE B - 90 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	14.352	16.403	16.403	16.403	16.403	16.403	16.403	16.403	16.403
... T O T A L ...	0.000	0.000	14.352	16.403	16.403	16.403	16.403	16.403	16.403	16.403	16.403
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.134	.154	.154	.154	.154	.154	.154	.154	.154
... T O T A L ...	0.000	0.000	5.618	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	2.153	2.460	2.460	2.460	2.460	2.460	2.460	2.460	2.460
COST + DEPR + DEPL	0.000	0.000	9.265	11.564	11.074	10.832	10.601	10.611	10.189	10.054	10.027
PROFIT BEFORE TAX	0.000	0.000	5.087	4.839	5.329	5.571	5.802	5.792	6.214	6.349	6.376
INCOME TAXES	0.000	0.000	2.645	2.516	2.771	2.897	3.017	3.012	3.231	3.301	3.315
TAX CREDIT	0.000	0.000	1.105	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	3.547	2.323	2.558	2.674	2.785	2.780	2.983	3.047	3.060
AVERAGE ANNUAL NET PROFIT		3.054									
GROSS CF, OPERATIONS	0.000	0.000	7.194	7.486	7.231	7.105	6.985	6.990	6.771	6.701	6.687
ACCUM GROSS CASHFLOW	0.000	0.000	7.194	14.680	21.911	29.016	36.001	42.992	49.763	56.463	63.150
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	2.050	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-15.556	-8.293	-1.537	5.328	11.746	18.501	25.049	31.265	37.733
NET CASH FLOW	-10.541	-8.588	3.573	7.263	6.756	6.865	6.418	6.754	6.548	6.217	6.468
ACCUM NET CASHFLOW	-10.541	-19.129	-15.556	-8.293	-1.537	5.328	11.746	18.501	25.049	31.265	37.733
----- NET CASH FLOW DISCOUNTED AT 27.288 / PERIOD -----											
FACTORS	.786	.617	.485	.381	.299	.235	.185	.145	.114	.090	.070
PRESENT VALUE	-8.281	-5.301	1.732	2.767	2.022	1.614	1.186	.980	.746	.557	.455
CUMULATIVE VALUE	-8.281	-13.582	-11.849	-9.083	-7.061	-5.447	-4.261	-3.281	-2.534	-1.978	-1.523

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	16.403	16.403	16.403	16.403	16.403	16.403	16.403	16.403	16.403	16.403	16.403
... T O T A L ...	16.403	16.403	16.403	16.403	16.403	16.403	16.403	16.403	16.403	16.403	16.403
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.154	.154	.154	.154	.154	.154	.154	.154	.154	.154	.154
... T O T A L ...	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	2.460	2.460	2.460	2.460	2.460	2.460	2.460	2.460	2.460	2.460	2.460
COST + DEPR + DEPL	9.902	9.914	9.843	9.808	9.872	9.739	9.621	9.578	9.553	9.500	9.500
PROFIT BEFORE TAX	6.501	6.489	6.560	6.595	6.531	6.664	6.782	6.825	6.850	6.903	6.903
INCOME TAXES	3.380	3.374	3.411	3.429	3.396	3.465	3.526	3.549	3.562	3.589	3.589
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	3.120	3.115	3.149	3.165	3.135	3.199	3.255	3.276	3.288	3.313	3.313
GROSS CF, OPERATIONS	6.622	6.628	6.591	6.573	6.606	6.537	6.476	6.453	6.440	6.413	6.413
ACCUM GROSS CASHFLOW	69.772	76.400	82.991	89.564	96.170	102.707	109.182	115.636	122.076	128.488	134.901
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-2.050
ACCUM GROSS CF TOTAL	43.767	50.163	56.523	62.604	69.210	75.747	82.222	88.676	95.116	101.528	109.991
NET CASH FLOW	6.034	6.396	6.360	6.081	6.606	6.537	6.476	6.453	6.440	6.413	8.463
ACCUM NET CASHFLOW	43.767	50.163	56.523	62.604	69.210	75.747	82.222	88.676	95.116	101.528	109.991
FACTORS	.055	.043	.034	.027	.021	.017	.013	.010	.008	.006	.005
PRESENT VALUE	.334	.278	.217	.163	.139	.108	.084	.066	.052	.040	.042
CUMULATIVE VALUE	-1.189	-.911	-.694	-.531	-.392	-.284	-.200	-.134	-.082	-.042	.000

PAY-OUT PERIOD FROM START OF OPERATION IS 3.224

THE RALPH M. PARSONS COMPANY

CASH FLOW CASE EVALUATION

CONTINENTAL COPPER, INC.
JOB NO. 5382-001 DATE 02/12/75

CASE B 100 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	16.166	18.476	18.476	18.476	18.476	18.476	18.476	18.476	18.476
... T O T A L ...	0.000	0.000	16.166	18.476	18.476	18.476	18.476	18.476	18.476	18.476	18.476
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.134	.154	.154	.154	.154	.154	.154	.154	.154
... T O T A L ...	0.000	0.000	5.618	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	2.425	2.771	2.771	2.771	2.771	2.771	2.771	2.771	2.771
COST + DEPR + DEPL	0.000	0.000	9.537	11.875	11.385	11.143	10.912	10.922	10.500	10.365	10.338
PROFIT BEFORE TAX	0.000	0.000	6.629	6.601	7.091	7.333	7.564	7.554	7.976	8.111	8.138
INCOME TAXES	0.000	0.000	3.447	3.432	3.687	3.813	3.933	3.928	4.147	4.218	4.232
TAX CREDIT	0.000	0.000	1.105	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	4.287	3.168	3.403	3.520	3.631	3.626	3.828	3.893	3.906
AVERAGE ANNUAL NET PROFIT		3.895									
GROSS CF, OPERATIONS	0.000	0.000	8.206	8.643	8.388	8.262	8.142	8.147	7.928	7.857	7.843
ACCUM GROSS CASHFLOW	0.000	0.000	8.206	16.849	25.236	33.498	41.640	49.788	57.715	65.573	73.416
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	2.309	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-14.803	-6.383	1.529	9.551	17.126	25.038	32.742	40.116	47.740
NET CASH FLOW	-10.541	-8.588	4.326	8.420	7.913	8.022	7.575	7.911	7.705	7.373	7.624
ACCUM NET CASHFLOW	-10.541	-19.129	-14.803	-6.383	1.529	9.551	17.126	25.038	32.742	40.116	47.740
----- NET CASH FLOW DISCOUNTED AT 31.334 / PERIOD -----											
FACTORS	.761	.580	.441	.336	.256	.195	.148	.113	.086	.066	.050
PRESENT VALUE	-8.026	-4.979	1.910	2.830	2.025	1.563	1.124	.894	.663	.483	.380
CUMULATIVE VALUE	-8.026	-13.005	-11.095	-8.265	-6.240	-4.677	-3.553	-2.659	-1.997	-1.514	-1.133

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	18.476	18.476	18.476	18.476	18.476	18.476	18.476	18.476	18.476	18.476	18.476
... T O T A L ...	18.476	18.476	18.476	18.476	18.476	18.476	18.476	18.476	18.476	18.476	18.476
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.154	.154	.154	.154	.154	.154	.154	.154	.154	.154	.154
... T O T A L ...	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401	6.401
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.306	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	2.771	2.771	2.771	2.771	2.771	2.771	2.771	2.771	2.771	2.771	2.771
COST + DEPR + DEPL	10.213	10.225	10.154	10.119	10.183	10.050	9.932	9.889	9.864	9.811	9.811
PROFIT BEFORE TAX	8.263	8.251	8.322	8.357	8.293	8.426	8.544	8.587	8.612	8.665	8.665
INCOME TAXES	4.297	4.290	4.327	4.345	4.312	4.381	4.443	4.465	4.478	4.506	4.506
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	3.966	3.960	3.994	4.011	3.980	4.044	4.101	4.122	4.134	4.159	4.159
GROSS CF, OPERATIONS	7.778	7.785	7.748	7.730	7.763	7.694	7.632	7.610	7.597	7.569	7.569
ACCUM GROSS CASHFLOW	81.195	88.979	96.727	104.457	112.219	119.913	127.546	135.155	142.752	150.322	157.891
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-2.339
ACCUM GROSS CF TOTAL	54.931	62.483	70.000	77.238	85.000	92.694	100.327	107.936	115.533	123.103	132.981
NET CASH FLOW	7.190	7.553	7.517	7.238	7.763	7.694	7.632	7.610	7.597	7.569	9.878
ACCUM NET CASHFLOW	54.931	62.483	70.000	77.238	85.000	92.694	100.327	107.936	115.533	123.103	132.981
FACTORS	.038	.029	.022	.017	.013	.010	.007	.006	.004	.003	.002
PRESENT VALUE	.273	.218	.165	.121	.099	.075	.056	.043	.033	.025	.025
CUMULATIVE VALUE	-.860	-.642	-.476	-.355	-.256	-.181	-.125	-.082	-.049	-.025	.000

PAY-OUT PERIOD FROM START OF OPERATION IS 2.807

CASH FLOW CASE EVALUATION

THE RALPH M. PARSONS COMPANY

CONTINENTAL COPPER, INC.
JOB NO. 5382-001 DATE 02/12/75

CASE C - 60 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	9.879	11.291	11.291	11.291	11.291	11.291	11.291	11.291	11.291
... T O T A L ...	0.000	0.000	9.879	11.291	11.291	11.291	11.291	11.291	11.291	11.291	11.291
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.149	.171	.171	.171	.171	.171	.171	.171	.171
... T O T A L ...	0.000	0.000	5.633	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	1.376	1.085	1.336	1.451	1.567	1.562	1.694	1.694	1.694
COST + DEPR + DEPL	0.000	0.000	8.503	10.206	9.961	9.840	9.725	9.730	9.440	9.305	9.278
PROFIT BEFORE TAX	0.000	0.000	1.376	1.085	1.330	1.451	1.567	1.562	1.851	1.986	2.013
INCOME TAXES	0.000	0.000	.716	.564	.692	.755	.815	.812	.963	1.033	1.047
TAX CREDIT	0.000	0.000	.716	.389	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	1.376	.910	.638	.696	.752	.750	.889	.953	.966
AVERAGE ANNUAL NET PROFIT		1.011									
GROSS CF, OPERATIONS	0.000	0.000	4.246	4.698	4.181	4.118	4.058	4.061	3.910	3.840	3.826
ACCUM GROSS CASHFLOW	0.000	0.000	4.246	8.944	13.126	17.244	21.303	25.364	29.274	33.114	36.940
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	1.411	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-17.865	-13.390	-9.683	-5.805	-2.313	1.512	5.199	8.555	12.162
NET CASH FLOW	-10.541	-8.588	1.264	4.475	3.706	3.878	3.491	3.825	3.687	3.356	3.607
ACCUM NET CASHFLOW	-10.541	-19.129	-17.865	-13.390	-9.683	-5.805	-2.313	1.512	5.199	8.555	12.162
----- NET CASH FLOW DISCOUNTED AT 15.446 / PERIOD -----											
FACTORS	.866	.750	.650	.563	.488	.422	.366	.317	.275	.238	.206
PRESENT VALUE	-9.131	-6.444	.822	2.519	1.807	1.638	1.277	1.212	1.012	.798	.743
CUMULATIVE VALUE	-9.131	-15.574	-14.753	-12.233	-10.426	-8.788	-7.510	-6.298	-5.286	-4.488	-3.745

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	11.291	11.291	11.291	11.291	11.291	11.291	11.291	11.291	11.291	11.291	11.291
... T O T A L ...	11.291	11.291	11.291	11.291	11.291	11.291	11.291	11.291	11.291	11.291	11.291
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.171	.171	.171	.171	.171	.171	.171	.171	.171	.171	.171
... T O T A L ...	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	1.694	1.694	1.694	1.694	1.694	1.694	1.694	1.694	1.694	1.694	1.694
COST + DEPR + DEPL	9.153	9.165	9.094	9.059	9.123	8.990	8.872	8.829	8.804	8.751	8.751
PROFIT BEFORE TAX	2.138	2.126	2.197	2.232	2.168	2.301	2.419	2.462	2.487	2.540	2.540
INCOME TAXES	1.112	1.106	1.143	1.161	1.128	1.197	1.258	1.280	1.293	1.321	1.321
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	1.026	1.021	1.055	1.072	1.041	1.105	1.161	1.182	1.194	1.219	1.219
AVERAGE ANNUAL NET PROFIT											
GROSS CF, OPERATIONS	3.761	3.767	3.730	3.712	3.745	3.676	3.615	3.593	3.580	3.552	3.552
ACCUM GROSS CASHFLOW	40.701	44.468	48.199	51.911	55.656	59.333	62.948	66.540	70.120	73.672	77.224
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.411
ACCUM GROSS CF TOTAL	15.335	18.870	22.370	25.590	29.335	33.012	36.627	40.219	43.799	47.351	52.314
NET CASH FLOW	3.173	3.535	3.499	3.220	3.745	3.676	3.615	3.593	3.580	3.552	4.963
ACCUM NET CASHFLOW	15.335	18.870	22.370	25.590	29.335	33.012	36.627	40.219	43.799	47.351	52.314
----- NET CASH FLOW DISCOUNTED AT 15.446 / PERIOD -----											
FACTORS	.178	.155	.134	.116	.100	.087	.075	.065	.057	.049	.042
PRESENT VALUE	.566	.546	.468	.373	.376	.320	.272	.235	.202	.174	.211
CUMULATIVE VALUE	-3.178	-2.632	-2.164	-1.790	-1.414	-1.094	-.822	-.587	-.385	-.211	-.000

PAY-OUT PERIOD FROM START OF OPERATION IS 5.605

THE RALPH M. PARSONS COMPANY

CASH FLOW CASE EVALUATION

CONTINENTAL COPPER, INC.
JOB NO. 5382-001 DATE 02/12/75

CASE C - 70 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	11.893	13.592	13.592	13.592	13.592	13.592	13.592	13.592	13.592
... T O T A L ...	0.000	0.000	11.893	13.592	13.592	13.592	13.592	13.592	13.592	13.592	13.592
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.149	.171	.171	.171	.171	.171	.171	.171	.171
... T O T A L ...	0.000	0.000	5.633	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	1.784	2.039	2.039	2.039	2.039	2.039	2.039	2.039	2.039
COST + DEPR + DEPL	0.000	0.000	8.911	11.160	10.670	10.428	10.197	10.207	9.785	9.650	9.623
PROFIT BEFORE TAX	0.000	0.000	2.982	2.432	2.922	3.164	3.395	3.385	3.807	3.942	3.969
INCOME TAXES	0.000	0.000	1.551	1.265	1.520	1.645	1.766	1.760	1.980	2.050	2.064
TAX CREDIT	0.000	0.000	1.105	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	2.536	1.167	1.403	1.519	1.630	1.625	1.827	1.892	1.905
AVERAGE ANNUAL NET PROFIT		1.906									
GROSS CF, OPERATIONS	0.000	0.000	5.814	5.909	5.654	5.529	5.408	5.414	5.194	5.124	5.110
ACCUM GROSS CASHFLOW	0.000	0.000	5.814	11.724	17.378	22.907	28.315	33.729	38.923	44.047	49.157
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	1.699	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-16.585	-10.898	-5.719	-.430	4.411	9.589	14.560	19.200	24.091
NET CASH FLOW	-10.541	-8.588	2.544	5.686	5.179	5.289	4.841	5.178	4.971	4.640	4.891
ACCUM NET CASHFLOW	-10.541	-19.129	-16.585	-10.898	-5.719	-.430	4.411	9.589	14.560	19.200	24.091
----- NET CASH FLOW DISCOUNTED AT 21.325 / PERIOD -----											
FACTORS	.824	.679	.560	.462	.381	.314	.258	.213	.176	.145	.119
PRESENT VALUE	-8.688	-5.834	1.425	2.624	1.970	1.658	1.251	1.103	.873	.671	.583
CUMULATIVE VALUE	-8.688	-14.523	-13.098	-10.474	-8.503	-6.845	-5.594	-4.491	-3.618	-2.946	-2.363

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	13.592	13.592	13.592	13.592	13.592	13.592	13.592	13.592	13.592	13.592	13.592
... T O T A L ...	13.592	13.592	13.592	13.592	13.592	13.592	13.592	13.592	13.592	13.592	13.592
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.171	.171	.171	.171	.171	.171	.171	.171	.171	.171	.171
... T O T A L ...	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	2.039	2.039	2.039	2.039	2.039	2.039	2.039	2.039	2.039	2.039	2.039
COST + DEPR + DEPL	9.498	9.510	9.439	9.404	9.468	9.335	9.217	9.174	9.149	9.096	9.096
PROFIT BEFORE TAX	4.094	4.082	4.153	4.188	4.124	4.257	4.375	4.418	4.443	4.496	4.496
INCOME TAXES	2.129	2.123	2.160	2.176	2.145	2.214	2.275	2.297	2.310	2.338	2.338
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	1.965	1.959	1.994	2.010	1.980	2.043	2.100	2.121	2.133	2.158	2.158
GROSS CF, OPERATIONS	5.045	5.051	5.014	4.996	5.029	4.960	4.899	4.877	4.864	4.836	4.836
ACCUM GROSS CASHFLOW	54.202	59.253	64.268	69.264	74.293	79.254	84.152	89.029	93.893	98.729	103.565
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.699
ACCUM GROSS CF TOTAL	28.548	33.367	38.151	42.655	47.684	52.645	57.543	62.420	67.284	72.120	78.655
NET CASH FLOW	4.457	4.819	4.783	4.504	5.029	4.960	4.899	4.877	4.864	4.836	6.535
ACCUM NET CASHFLOW	28.548	33.367	38.151	42.655	47.684	52.645	57.543	62.420	67.284	72.120	78.655
FACTORS	.098	.081	.067	.055	.045	.037	.031	.025	.021	.017	.014
PRESENT VALUE	.438	.391	.319	.246	.228	.186	.151	.124	.102	.083	.093
CUMULATIVE VALUE	-1.925	-1.534	-1.215	-.967	-.739	-.553	-.402	-.278	-.175	-.093	.000

PAY-OUT PERIOD FROM START OF OPERATION IS 4.089

THE RALPH M. PARSONS COMPANY

CASH FLOW CASE EVALUATION

CONTINENTAL COPPER,INC.
JOB NO. 5382-001 DATE 02/12/75

CASE C - 80 CENT COPPER

VARIATICNS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	13.906	15.893	15.893	15.893	15.893	15.893	15.893	15.893	15.893
... T O T A L ...	0.000	0.000	13.906	15.893	15.893	15.893	15.893	15.893	15.893	15.893	15.893
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.149	.171	.171	.171	.171	.171	.171	.171	.171
... T O T A L ...	0.000	0.000	5.633	6.418	6.416	6.418	6.418	6.418	6.418	6.418	6.418
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	2.086	2.384	2.384	2.384	2.384	2.384	2.384	2.384	2.384
COST + DEPR + DEPL	0.000	0.000	9.213	11.505	11.015	10.773	10.542	10.552	10.130	9.995	9.968
PROFIT BEFORE TAX	0.000	0.000	4.693	4.388	4.878	5.120	5.351	5.341	5.763	5.898	5.925
INCOME TAXES	0.000	0.000	2.440	2.282	2.537	2.662	2.783	2.777	2.997	3.067	3.081
TAX CREDIT	0.000	0.000	1.105	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	3.358	2.106	2.341	2.458	2.569	2.564	2.766	2.831	2.844
AVERAGE ANNUAL NET PROFIT		2.839									
GROSS CF, OPERATIONS	0.000	0.000	6.938	7.193	6.938	6.813	6.692	6.698	6.478	6.408	6.394
ACCUM GROSS CASHFLOW	0.000	0.000	6.938	14.131	21.069	27.882	34.574	41.272	47.750	54.158	60.552
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	1.987	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-15.749	-8.779	-2.316	4.257	10.382	16.844	23.099	29.023	35.198
NET CASH FLOW	-10.541	-8.588	3.380	6.970	6.463	6.573	6.125	6.462	6.255	5.924	6.175
ACCUM NET CASHFLOW	-10.541	-19.129	-15.749	-8.779	-2.316	4.257	10.382	16.844	23.099	29.023	35.198
----- NET CASH FLOW DISCOUNTED AT 26.223 / PERIOD -----											
FACTORS	.792	.628	.497	.394	.312	.247	.196	.155	.123	.097	.077
PRESENT VALUE	-8.351	-5.390	1.681	2.746	2.017	1.625	1.200	1.003	.769	.577	.477
CUMULATIVE VALUE	-8.351	-13.741	-12.061	-9.315	-7.298	-5.672	-4.472	-3.470	-2.700	-2.123	-1.647

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	15.893	15.893	15.893	15.893	15.893	15.893	15.893	15.893	15.893	15.893	15.893
... T O T A L ...	15.893	15.893	15.893	15.893	15.893	15.893	15.893	15.893	15.893	15.893	15.893
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.171	.171	.171	.171	.171	.171	.171	.171	.171	.171	.171
... T O T A L ...	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	2.384	2.384	2.384	2.384	2.384	2.384	2.384	2.384	2.384	2.384	2.384
COST + DEPR + DEPL	9.843	9.855	9.784	9.749	9.813	9.680	9.562	9.519	9.494	9.441	9.441
PROFIT BEFORE TAX	6.050	6.038	6.109	6.144	6.080	6.213	6.331	6.374	6.399	6.452	6.452
INCOME TAXES	3.146	3.140	3.177	3.195	3.162	3.231	3.292	3.315	3.328	3.355	3.355
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	2.904	2.898	2.932	2.949	2.918	2.982	3.039	3.060	3.072	3.097	3.097
AVERAGE ANNUAL NET PROFIT											
GROSS CF, OPERATIONS	6.329	6.335	6.298	6.280	6.313	6.244	6.183	6.160	6.147	6.120	6.120
ACCUM GROSS CASHFLOW	66.881	73.216	79.515	85.795	92.108	98.352	104.535	110.696	116.843	122.963	129.083
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.987
ACCUM GROSS CF TOTAL	40.939	47.042	53.110	58.898	65.211	71.455	77.638	83.799	89.946	96.066	104.173
NET CASH FLOW	5.741	6.103	6.067	5.788	6.313	6.244	6.183	6.160	6.147	6.120	8.107
ACCUM NET CASHFLOW	40.939	47.042	53.110	58.898	65.211	71.455	77.638	83.799	89.946	96.066	104.173
----- NET CASH FLOW DISCOUNTED AT 26.223 / PERIOD -----											
FACTORS	.061	.048	.038	.030	.024	.019	.015	.012	.009	.008	.006
PRESENT VALUE	.351	.296	.233	.176	.152	.119	.093	.074	.058	.046	.048
CUMULATIVE VALUE	-1.296	-1.000	-.767	-.591	-.439	-.320	-.226	-.153	-.094	-.048	-.000

PAY-OUT PERIOD FROM START OF OPERATION IS 3.352

THE RALPH M. PARSONS COMPANY

CASH FLOW CASE EVALUATION

CONTINENTAL COPPER, INC.
JOB NO. 5382-001 DATE 02/12/75

CASE C - 90 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	15.919	18.194	18.194	18.194	18.194	18.194	18.194	18.194	18.194
... T O T A L ...	0.000	0.000	15.919	18.194	18.194	18.194	18.194	18.194	18.194	18.194	18.194
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.149	.171	.171	.171	.171	.171	.171	.171	.171
... T O T A L ...	0.000	0.000	5.633	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	2.388	2.729	2.729	2.729	2.729	2.729	2.729	2.729	2.729
COST + DEPR + DEPL	0.000	0.000	9.515	11.850	11.360	11.118	10.887	10.897	10.475	10.340	10.313
PROFIT BEFORE TAX	0.000	0.000	6.404	6.344	6.834	7.076	7.307	7.297	7.719	7.854	7.881
INCOME TAXES	0.000	0.000	3.330	3.299	3.554	3.679	3.800	3.794	4.014	4.084	4.098
TAX CREDIT	0.000	0.000	1.105	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	4.179	3.045	3.280	3.396	3.507	3.503	3.705	3.770	3.783
AVERAGE ANNUAL NET PROFIT		3.772									
GROSS CF, OPERATIONS	0.000	0.000	8.061	8.477	8.222	8.097	7.976	7.982	7.762	7.692	7.678
ACCUM GROSS CASHFLOW	0.000	0.000	8.061	16.538	24.760	32.857	40.833	48.815	56.577	64.269	71.947
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	2.274	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-14.913	-6.659	1.088	8.945	16.354	24.100	31.639	38.847	46.306
NET CASH FLOW	-10.541	-8.588	4.216	8.254	7.747	7.857	7.409	7.746	7.539	7.208	7.459
ACCUM NET CASHFLOW	-10.541	-19.129	-14.913	-6.659	1.088	8.945	16.354	24.100	31.639	38.847	46.306
----- NET CASH FLOW DISCOUNTED AT 30.767 / PERIOD -----											
FACTORS	.765	.585	.447	.342	.262	.200	.153	.117	.089	.068	.052
PRESENT VALUE	-8.061	-5.022	1.885	2.823	2.026	1.571	1.133	.906	.674	.493	.390
CUMULATIVE VALUE	-8.061	-13.083	-11.198	-8.375	-6.349	-4.778	-3.644	-2.738	-2.064	-1.571	-1.181

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	18.194	18.194	18.194	18.194	18.194	18.194	18.194	18.194	18.194	18.194	18.194
... T O T A L ...	18.194	18.194	18.194	18.194	18.194	18.194	18.194	18.194	18.194	18.194	18.194
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.171	.171	.171	.171	.171	.171	.171	.171	.171	.171	.171
... T O T A L ...	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	2.729	2.729	2.729	2.729	2.729	2.729	2.729	2.729	2.729	2.729	2.729
COST + DEPR + DEPL	10.188	10.200	10.129	10.094	10.158	10.025	9.907	9.864	9.839	9.786	9.786
PROFIT BEFORE TAX	8.006	7.994	8.065	8.100	8.036	8.169	8.287	8.330	8.355	8.408	8.408
INCOME TAXES	4.163	4.157	4.194	4.212	4.179	4.248	4.309	4.332	4.345	4.372	4.372
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	3.843	3.837	3.871	3.888	3.857	3.921	3.978	3.998	4.010	4.036	4.036
GROSS CF, OPERATIONS	7.613	7.619	7.582	7.564	7.597	7.528	7.467	7.444	7.431	7.404	7.404
ACCUM GROSS CASHFLOW	79.560	87.179	94.761	102.325	109.923	117.451	124.918	132.362	139.794	147.198	154.601
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-2.274
ACCUM GROSS CF TOTAL	53.331	60.718	68.069	75.141	82.739	90.267	97.734	105.178	112.610	120.014	129.691
NET CASH FLOW	7.025	7.387	7.351	7.072	7.597	7.528	7.467	7.444	7.431	7.404	9.678
ACCUM NET CASHFLOW	53.331	60.718	68.069	75.141	82.739	90.267	97.734	105.178	112.610	120.014	129.691
FACTORS	.040	.031	.023	.018	.014	.010	.008	.006	.005	.004	.003
PRESENT VALUE	.281	.226	.172	.126	.104	.079	.060	.046	.035	.026	.026
CUMULATIVE VALUE	-.900	-.674	-.502	-.376	-.272	-.193	-.133	-.088	-.053	-.026	-.000

PAY-OUT PERIOD FROM START OF OPERATION IS 2.860

CASH FLOW CASE EVALUATION

THE RALPH M. PARSONS COMPANY

CONTINENTAL COPPER, INC.
JOB NO. 5382-001 DATE 02/12/75

CASE C 100 CENT COPPER

VARIATIONS EVALUATED IN THIS CASE ARE%
COST 0.0 REVENUE 0.0 INVESTMENT 0.0

PERIOD	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986
[REVENUE]											
NET SMELTER RETURN	0.000	0.000	17.933	20.495	20.495	20.495	20.495	20.495	20.495	20.495	20.495
... T O T A L ...	0.000	0.000	17.933	20.495	20.495	20.495	20.495	20.495	20.495	20.495	20.495
[COSTS]											
MINE, MILL, G AND A	0.000	0.000	5.106	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	0.000	0.000	.229	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	0.000	0.000	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	0.000	0.000	.149	.171	.171	.171	.171	.171	.171	.171	.171
... T O T A L ...	0.000	0.000	5.633	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418
DEPREC. PRE-MINE DEV	0.000	0.000	0.000	.338	.302	.270	.242	.216	.194	.173	.155
DEPREC. MINE EQUIP.	0.000	0.000	0.000	1.020	.701	.612	.518	.652	.340	.305	.368
DEPREC. PLANT	0.000	0.000	1.494	1.345	1.210	1.089	.980	.882	.794	.715	.643
DEPLETION	0.000	0.000	2.690	3.074	3.074	3.074	3.074	3.074	3.074	3.074	3.074
COST + DEPR + DEPL	0.000	0.000	9.817	12.195	11.705	11.463	11.232	11.242	10.820	10.685	10.658
PROFIT BEFORE TAX	0.000	0.000	8.116	8.300	8.790	9.032	9.263	9.253	9.675	9.810	9.837
INCOME TAXES	0.000	0.000	4.220	4.316	4.571	4.697	4.817	4.811	5.031	5.101	5.115
TAX CREDIT	0.000	0.000	1.105	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	0.000	0.000	5.001	3.984	4.219	4.335	4.446	4.441	4.644	4.709	4.722
AVERAGE ANNUAL NET PROFIT		4.705									
GROSS CF, OPERATIONS	0.000	0.000	9.185	9.761	9.506	9.380	9.260	9.266	9.046	8.976	8.962
ACCUM GROSS CASHFLOW	0.000	0.000	9.185	18.946	28.452	37.833	47.093	56.359	65.405	74.381	83.342
[INVESTMENT]											
PRE-MINE DEVELOPMENT	.657	1.562	.988	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	.918	1.049	.370	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	0.000	0.000	.213	.223	.475	.240	.567	.236	.223	.484	.219
MILL CONSTRUCTION	8.966	5.977	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	2.562	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
ACCUM GROSS CF TOTAL	-10.541	-19.129	-14.077	-4.539	4.492	13.633	22.326	31.356	40.179	48.671	57.413
NET CASH FLOW	-10.541	-8.588	5.052	9.538	9.031	9.140	8.693	9.030	8.823	8.492	8.743
ACCUM NET CASHFLOW	-10.541	-19.129	-14.077	-4.539	4.492	13.633	22.326	31.356	40.179	48.671	57.413
----- NET CASH FLOW DISCOUNTED AT 35.045 / PERIOD -----											
FACTORS	.740	.548	.406	.301	.223	.165	.122	.090	.067	.050	.037
PRESENT VALUE	-7.606	-4.709	2.051	2.068	2.011	1.507	1.061	.816	.591	.421	.321
CUMULATIVE VALUE	-7.806	-12.515	-10.463	-7.596	-5.585	-4.078	-3.017	-2.203	-1.610	-1.189	-.868

PERIOD	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
[REVENUE]											
NET SMELTER RETURN	20.495	20.495	20.495	20.495	20.495	20.495	20.495	20.495	20.495	20.495	20.495
... T O T A L ...	20.495	20.495	20.495	20.495	20.495	20.495	20.495	20.495	20.495	20.495	20.495
[COSTS]											
MINE, MILL, G AND A	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836	5.836
PROPERTY TAX	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262	.262
INSURANCE	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149	.149
TRUCK FREIGHT	.171	.171	.171	.171	.171	.171	.171	.171	.171	.171	.171
... T O T A L ...	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418	6.418
DEPREC. PRE-MINE DEV	.139	.124	.118	.117	.117	.117	.117	.117	.117	.117	.117
DEPREC. MINE EQUIP.	.323	.407	.342	.308	.372	.239	.121	.078	.053	0.000	0.000
DEPREC. PLANT	.579	.522	.522	.522	.522	.522	.522	.522	.522	.522	.522
DEPLETION	3.074	3.074	3.074	3.074	3.074	3.074	3.074	3.074	3.074	3.074	3.074
COST + DEPR + DEPL	10.533	10.545	10.474	10.439	10.503	10.370	10.252	10.209	10.184	10.131	10.131
PROFIT BEFORE TAX	9.962	9.950	10.021	10.056	9.992	10.125	10.243	10.286	10.311	10.364	10.364
INCOME TAXES	5.180	5.174	5.211	5.229	5.196	5.265	5.326	5.349	5.362	5.389	5.389
TAX CREDIT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PROFIT AFTER TAX	4.782	4.776	4.810	4.827	4.796	4.860	4.917	4.937	4.949	4.975	4.975
GROSS CF, OPERATIONS	8.897	8.903	8.866	8.848	8.881	8.812	8.751	8.728	8.715	8.688	8.688
ACCUM GROSS CASHFLOW	92.239	101.143	110.009	118.857	127.738	136.550	145.301	154.029	162.745	171.433	180.120
[INVESTMENT]											
PRE-MINE DEVELOPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIPMENT	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MINE EQUIP. REPLAC.	.588	.232	.231	.492	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MILL CONSTRUCTION	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
WORKING CAPITAL	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-2.562
ACCUM GROSS CF TOTAL	65.722	74.394	83.029	91.385	100.266	109.078	117.829	126.557	135.273	143.961	155.210
NET CASH FLOW	8.309	8.671	8.635	8.356	8.881	8.812	8.751	8.728	8.715	8.688	11.250
ACCUM NET CASHFLOW	65.722	74.394	83.029	91.385	100.266	109.078	117.829	126.557	135.273	143.961	155.210
FACTORS	.027	.020	.015	.011	.008	.006	.004	.003	.002	.002	.001
PRESENT VALUE	.226	.175	.129	.092	.073	.053	.039	.029	.021	.016	.015
CUMULATIVE VALUE	-.642	-.467	-.339	-.247	-.174	-.121	-.081	-.052	-.031	-.015	-.000

PAY-OUT PERIOD FROM START OF OPERATION IS 2.503