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McINTYRE PORCUPINE MINES LIMITED

A Report on

THE LAKESHORE MINE

Pinal County

Arizona

December, 1968

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INTRODUCTION

The El Paso Natural Gas Company of El Paso, Texas, is the owner of the Lakeshore Mine property on which a sizable tonnage of copper mineralization has been outlined by diamond drilling over the last two years. El Paso has invited the Superior Oil Company along with other oil and mining companies to bid on the property. Superior Oil in turn invited McIntyre to assess the merits of the Lakeshore from the point of view of mutual interest.

Messrs. Parfitt and Plaxton of McIntyre visited Tucson during the period November 18th to December 2nd, 1968 for the purpose of examining and collecting available data on the property. Mr. John Hite, McIntyre resident geologist, was of assistance through his knowledge of the area geology. Superior Oil's minerals division office in Tucson was helpful in providing the necessary contacts and information.

El Paso have indicated that the type of deal which they expect is an initial cash payment of $2,000,000*(approximately their expenditure on the property) and a share of the net profits after return of capital investment. Latest information on this subject is given in their letter of December 24th to Superior Oil, which is included in the Appendix.

*Note - All monies in this report are in U.S. funds.

SUMMARY

The Lakeshore Mine property lies 60 miles west northwest of Tucson, on the south boundary of Pinal County, Arizona, and within the Papago Indian Reservation. The property comprising 4,180 acres is held under lease from the Indians and will be subject to royalty payments.

History:

Oxide copper showings on the Lakeshore property have been known for many years. Transcontinental Resources Limited, a Canadian company, attempted a small open-pit oxide operation and segregation process metallurgy. This proved uneconomic and the property became available and remained so for a number of years. Naragansett Wire, an American company, acquired the property and plant and finally El Paso, with its takeover of Naragansett, became owner and has continued to operate the plant on a 500 to 600 tons per day basis. Exploration drilling by El Paso over the last two years has resulted in the situation now under review.

Work Done:

The current open-pit and plant operation is not considered significant therefore is not covered in this report. El Paso carried out a geophysical survey which resulted in anomalous readings in the area to the west of the oxide pit. At the time of our examination some 78,000 feet of drilling in 51 holes had been completed in the anomalous area, establishing the existence of deep-seated copper mineralization. Drilling was continuing with three coring rigs and one rotary drill.

Metallurgical bench tests had been run on the core and chips from several of the holes, of both the oxide and sulphide sections.

All assaying has been done by El Paso on the property. Only occasional spot checks by a Tucson custom lab are noted.

With the limited rock exposure apparently no surface geological mapping has been attempted.

Geology:

Southern Arizona lies within the Basin and Range complex with its typical topography of abrupt rocky ridges and wide flat valleys filled with sand and gravels. The Cretaceous to Precambrian have been intruded by plutonic rocks of the Laramide Disturbance. With few exceptions, presently known "porphyry" copper deposits bear a close genetic relationship to these monzonitic acid intrusives.

The Lakeshore property lies on a pediment along the south-west side of the Slate Mountains which are comprised of Older Precambrian schists unconformably overlain by younger volcanic and sedimentary rocks. Precambrian granite intrudes the schists along the westerly slope of the range.

The mine area is underlain by a late Precambrian sedimentary sequence of limestones (tactites) and quartzites which strike generally north-south and dip to the west at from 25° to 30°. Quartz-feldspar porphyry (monzonite) intrudes all rocks of the sequence in the form of dikes and sills with a considerable thickening in the central portion of the drilled area. The source of the intrusion would appear to lie to the west or north-west. Diabase dikes and sills intrude the tactite and underlying quartzites. Granite is exposed along the east wall of the oxide pit. This is assumed to be part of the Precambrian granite stock. All rock types examined in the core showed a high degree of random fracturing. Normal faulting occurs along the west limits of the drilling, dropping the sequence down some 700 feet.

Primary copper sulphides, chalcopyrite and bornite, occur in all rock types but more selectively in the tactite. The mineralized zone conforms generally in strike and dip to the geologic sequence. Values below the tactite at present drilling limits are still in the 0.3 to 0.4% range. Above the sulphides is a blanket of copper oxides, primarily chrysocola. There is no significant secondary enrichment in the area east of the fault but two holes west of and close by the fault show sizable thicknesses of chalcocite mineralization. The mineralized zone extends 2,800 feet north-south and 2,000 feet east-west and is open in all directions except the east.

Potential mineral additions which could effect economics are:

1) Mineralization of the same tenor extending in the three open directions;

2) An extensive chalcocite blanket in the area west of the fault;

3) Mineralization below the limits of drilling to date;

4) A close-by mineralized monzonite stock.

Mining:

As considerable metallurgical work is required to determine if economic extraction of copper from the oxide zone is possible, mining plans were made for the sulphide zone only. Both open pit and underground mining were investigated. Mining by open pit can only be considered for the ore east of the fault, whereas most of the known and probable ore could be mined from underground. Underground mining presents some problems due to the sloping base of the orebody and its limited thickness.

Open Pit:

A pit was designed to mine 168,454,000 tons of sulphide ore averaging 0.69% copper after dilution. This pit would be 1,700 feet deep with a stripping ratio of 6.92:1. Pre-production capital, including $91,000,000 for stripping, would total $191,000,000. With copper at 42¢ (U.S.) per lb. and a production rate of 25,000 tons per day, operating profit after paying Indian royalties would total $241,000,000 over the 19.2 production years. This would be insufficient to repay the investment.

If it were found that the oxides could be leached economically and could be considered as ore, the waste ore ratio would be reduced to 3.71:1. Present information, however indicates that the oxides are not economic.

Underground Mine:

The intense fracturing of the mineralized zone indicates that block caving would be a suitable mining method. Rock conditions appear to be similar to those at San Manuel where this method is being used successfully. Mine layouts and cost estimates were therefore based on San Manuel practice with modifications made necessary by the attitude and shape of the orebody.

Ore reserves for block caving, using a cut-off of 0.5% copper and with allowances for dilution and ore lost in pillars, are as follows:

	Tons	% Cu
Reasonably proven	145,334,000	0.716
Probable	21,706,000	1.078
Possible	41,133,000	0.730
	208,162,000	0.756

This total has been used for the economic appraisals.

It is estimated that, after two years of preliminary testing, it would take an additional three years to bring the mine into production.

Two production rates, 25,000 and 40,000 tons per day, and situations with and without a smelter were considered.

Pre-production capital requirements (excluding pre-production interest) are estimated as follows:

	25,000 TPD	40,000 TPD
Test program	$ 2,382,000	$ 2,382,000
Underground development	18,531,000	24,250,000
	$ 20,913,000	$ 26,632,000
Plant	60,759,000	70,885,000
Working Capital	8,000,000	12,000,000
	$ 89,672,000	$109,517,000
Smelter	24,200,000	26,600,000
Add Working Capital	2,000,000	4,000,000
	$115,872,000	$140,117,000

Mine operating costs at 25,000 and 40,000 tons per day are estimated at $2.65 and $2.53 per ton respectively; and operating profits with 42¢ copper but before payment of Indian royalties, are as follows:

25,000 TPD			40,000 TPD		
	Operating Profits			Operating Profits	
Years	With smelter	Without smelter	Years	With smelter	Without smelter
1-16	$1.99	$1.80	1-3	$1.99	$1.80
			4-12	$2.11	$1.92
17-22	$2.86	$2.63	13-15	$2.98	$2.75
23-24	$3.33	$3.10	16-17	$3.45	$3.22

The increased profit per ton shown in the later years is due to the better grade ore, which is west of the fault and at depth, being unavailable until much of the ore above and to the east of the fault has been mined out. Profit in the last two years is again increased due to cessation of development.

Joint Operation - Lakeshore and Vekol

The Vekol property situated about 10 miles west of Lakeshore, is controlled jointly by the Superior Oil and Newmont companies. Here a sulphide deposit containing 74.7 million tons averaging 0.61% copper, and which could be mined by open pit, has been outlined by diamond drilling. No further work has been done on the property due to a decision in October, 1967 that higher prices for copper were required to bring it into production. Newmont suggested that significant savings would be possible by combining the two operations. We have therefore studied a joint operation of Vekol and Lakeshore at rates of 15,000 and 25,000 tons per day respectively, both with and without a smelter. We have assumed that Vekol will start production two years before Lakeshore reaches full capacity.

The proposed mining schedule is as follows:

Years	Vekol	Lakeshore
1-4	(Pre-production)	
5	25,000 TPD	Nil
6	25,000 TPD	10,000 TPD
7-17	15,000 TPD	25,000 TPD
18-25	Nil	40,000 TPD

Capital requirements are estimated as follows:

Testing and mine development	$ 31,271,000
Plant	77,487,000
	$108,758,000
Smelter	26,600,000
	$135,358,000

These amounts are exclusive of working capital and pre-production interest.

Estimated operating profits for the joint operation are as follows:

Years	Profit per ton With smelter	Profit per ton Without smelter
5	$1.99	$1.83
6	$1.99	$1.82
7-17	$2.04	$1.86
18-20	$2.11	$1.92
21-24	$2.98	$2.75
25	$3.45	$3.22

Financial Analysis

Analyses of the various situations @ 42¢ copper give the following:

Situation	Pre-production Capital (millions) (1)	R.O.I. (2)	Years to Pay Back @ 7% (3)	Mean Life Years	D.C.F. (millions) (4)
a) Without Smelter:					
Lakeshore alone:					
@ 25,000 TPD	$88.8	9.0%	16.1	24	$18.1
@ 40,000 TPD	$91.3	11.6%	10.9	17	$37.8
Lakeshore + Vekol:					
Lakeshore	$62.8	10.7%	13.8	20	$26.9
Vekol	$37.8	16.6%	6.3	13	$21.1
Total (40,000 TPD)	$100.6	12.0%		21	$48.0
b) With Smelter					
Lakeshore alone:					
@ 25,000 TPD	$114.3	7.7%	18.5	24	$14.6
@ 40,000 TPD	$116.9	9.9%	12.6	17	$32.3
Lakeshore + Vekol:					
Lakeshore	$83.5	9.4%	15.3	20	$24.9
Vekol	$45.1	14.4%	7.1	13	$20.1
Total (40,000 TPD)	$128.6	10.4%		21	$45.0

Notes: (1) Includes pre-production interest
(2) Rate of return on pre-production capital
(3) From start of production
(4) Discounted net cash flow after debt repayment (@ 10%)

The above were considered on the basis of 100% debt financing (@ 7%).

If the above are considered on the basis of 75% debt and 25% equity financing, return on equity is as follows:

Situation	Equity Financing Required (millions)	Return on Equity	Total Cash Return to Investor (millions)
a) Without Smelter:			
Lakeshore alone:			
@ 25,000 TPD	$24.5	12.1%	$169.7
@ 40,000 TPD	$25.2	17.1%	$193.9
Lakeshore + Vekol:			
Lakeshore	$21.6	15.6%	$189.5
Vekol	$ 9.9	26.2%	$ 69.6
Total (40,000 TPD	$27.4	17.9%	$255.0
b) With Smelter:			
Lakeshore alone:			
@ 25,000 TPD	$30.9	9.7%	$157.1
@ 40,000 TPD	$31.8	14.4%	$194.3
Lakeshore + Vekol:			
Lakeshore	$28.0	13.2%	$188.5
Vekol	$11.1	23.3%	$ 69.0
Total (40,000 TPD)	$34.8	15.0%	$253.6

The effect of sharing earnings with El Paso, on a formula which probably would apply, would be:

El Paso's Share	Rate of Return on Investment: Lakeshore + Vekol	Rate of Return on Investment: Lakeshore @ 40,000 TPD
0	12.0%	11.6%
10%	11.4%	10.9%
20%	11.1%	10.2%

0.7% 0.7%

CONCLUSIONS

It has reasonably been proven that an important body of "porphyry" copper mineralization exists on the Lakeshore property. The possibility of finding additional tonnage is good.

A sizeable extension of the secondary enrichment intersected in the drill holes west of the fault could decidedly improve the economics.

Present information indicates that the copper in oxide form cannot be recovered profitably. For this reason and because of the depth of the sulphide zone, mining by open pit is not feasible.

Rock characteristics are such that underground block-caving appears practical and that the system used at San Manuel can be applied with modifications.

The financial analyses show that, at 42¢ copper and after paying Indian royalties, Lakeshore by itself provides at best an 11.6% rate of return on capital invested. Return on investment for a combined operation, Lakehsore and Vekol, is 12.0%. With leverage, i.e. 75% debt-25% equity, return on equity for the combined operation reaches 17.9%. In all cases results are more favourable using a custom smelter.

The joint operation of the two properties appears the more feasible situation. Although we have not considered the economics of Vekol by itself, our figures indicate that at 42¢ copper it should be a viable operation.

On the basis of our present information, return on investment is marginal and the necessity of sharing profits with El Paso would make the situation even less favourable. However, we consider that the Lakeshore property has possibilities for improvement and that further exploration is warranted.

RECOMMENDATIONS

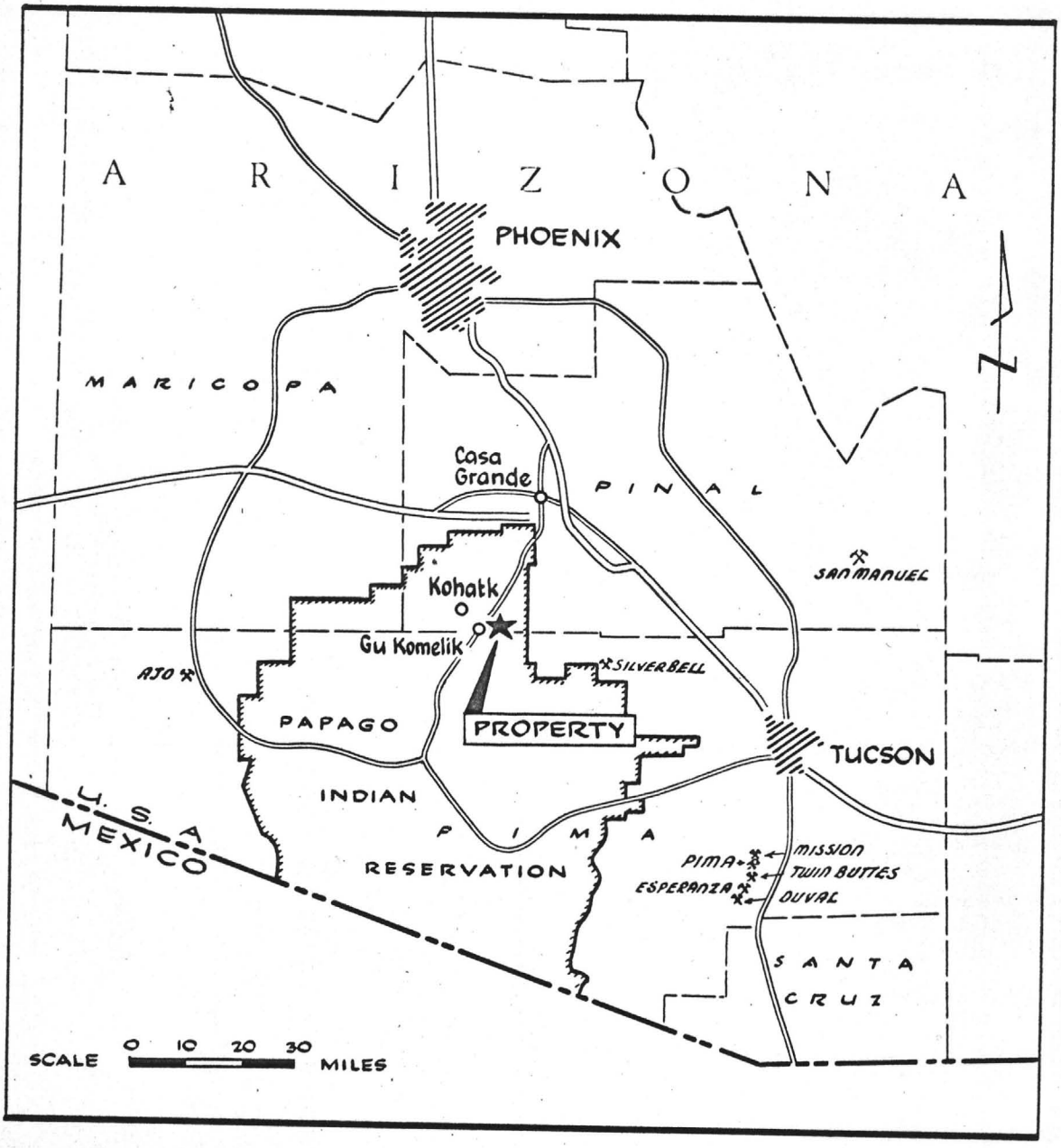
We feel that this mineral deposit will eventually be mined; but our calculations at this time do not justify a recommendation to invest in the property, particularly on El Paso's terms as we understand them.

However, as the full potential of the property has yet to be explored we recommend that the door be left open to negotiations. The objective would be a working option with no initial cash payment and all monies spent in the form of work done.

J. A. Plaxton J. A. Plaxton

P. O. Parfitt P. O. Parfitt

T. L. Bedard T. L. Bedard



EL PASO NATURAL GAS COMPANY

LAKESHORE MINE

LOCATION MAP

PINAL COUNTY PROPERTY - ARIZONA

(NEWMONT-SUPERIOR)

VEKOL MINE

Vekol Mountains

KOHATK

Kohatk

wash

Road

wash

Hwy

Road

Road

Rosa

Santa

Slate Mountains

LAKESHORE MINE

PAPAGO INDIAN RESERVATION

PINAL COUNTY
PIMA COUNTY

Tucson Rosa 15 miles

GU KOMELIK

LAKESHORE MINE & VEKOL MINE
PROPERTY MAP
PINAL COUNTY-ARIZONA
SCALE: 1inch=1mile

R.G.

PROPERTY

The Lakeshore Mine property lies 60 air miles west north-west of Tucson, on the south boundary of Pinal County, Arizona, and within the Papago Indian Reservation. The property is reached by driving some 90 miles from Tucson, all on good black-topped road surface.

The property, comprising 4,180 acres, is held for the greater part under lease from the Papago Indians and is subject to royalty agreement. A number of patented claims are contained within the leased area and these, we understand would not be subject to royalties. Their locations, however, are not within Lakeshore's present area of interest.

HISTORY

Some eleven years ago the Lakeshore Mine property was held by Transcontinental Resources Limited, a Canadian company. Transcontinental attempted a small open-pit copper oxide operation and, due to the high acid consuming gangue, treated the material by a segregation process (heat and a reducing atmosphere) producing metallic copper in concentrates. The operation proved uneconomic and the property became available through default and remained so for a number of years.

Naragansett Wire, an American company, acquired the property and plant and attempted to continue the operation using the custom smelted product in its fabricating business. Finally El Paso Natural Gas Company acquired the property through its take-over of Naragansett and has continued the reduction plant operation.

Exploratory drilling by El Paso over the last two years has resulted in the situation now under review.

WORK DONE TO DATE

The current open-pit and segregation plant operation is not considered significant therefore is not covered in this report. El Paso, in their exploration, carried out an IP survey which resulted in anomalous readings in an area just west of the oxide pit. Diamond drill testing of the anomaly established the existence of deep-seated copper mineralization and at the time of our examination some 78,000 feet of drilling in 51 holes had been completed in the area of interest. The drilling pattern to date forms roughly a 300 foot grid spacing but has not been systematic.

The hole numbering sequence indicates that 23 other holes have been drilled. Some of these were lost holes and have been re-drilled. The remainder are in other areas of the property.

Drilling was continuing with three coring rigs and one rotary drill. The latter has been used with few exceptions from the collar down to significant copper values and then has been replaced with a coring drill.

Metallurgical bench tests have been carried out on the core and chips from several of the drill holes, both in the oxide and sulphide sections.

All assaying has been done by El Paso at the lab on the property. Only occasional spot assay checks by a Tucson custom lab are noted.

Rock exposure on the property is limited and apparently no effort has been made on geological mapping.

GEOLOGY

General

In the Basin and Range area of southern Arizona, Cretaceous to Precambrian have been intruded by plutonic rocks of the late Cretaceous-Tertiary period known as the Laramide Disturbance. It is significant that, with few exceptions, the presently known "porphyry" copper deposits bear a close genetic relationship to these monzonitic acid intrusions.

Late Cenozoic normal faulting and a subsequent period of erosion and sedimentation have provided the typical topography of abrupt rocky ridges and wide flat valleys filled with sand and gravels to depths in the thousands of feet. Some of the gravels have become partially re-consolidated and now are known as conglomerates or fanglomerates (Gila Conglomerate). In some mineralized areas the conglomerates are important as locale of supergene or secondary enrichment; in others they are significant only from the point of view of their required stripping from above ore deposits at the old erosion surface.

The Lakeshore Mine property lies on a pediment along the south-west side of the Slate Mountains. Little detailed geology is available but in general the range is shown to be comprised of

Older Precambrian schists and gneisses unconformably overlain by younger volcanic and sedimentary rocks. A roughly oval shaped mass of Precambrian granite of several miles long axis dimension intrudes the schists and outcrops along the west slope of the range.

Geology of the Lakeshore Mine

The mine area is covered with a veneer of unconsolidated alluvium and conglomerate varying in depth from 50 to more than 1,000 feet. The nearest rock exposures are in the present pit where the Precambrian granite and a small amount of limestone altered to tactite and overlain by "andesite" are uncovered. Bedding in the tactite generally strikes north-south and dips 50°-85° to the west. All rocks in the pit show a high degree of fracturing and several fault structures are apparent.

The geologic section on the property based on the drill logs has not been clearly defined by El Paso. This is partly due to the fact that rotary drilling has been used to considerable depths and the geology of these sections has been largely ignored. On the opposite slope of the valley some 10 miles to the west and north lies the Vekol copper property (Newmont - Superior Oil controlled). There, the rock sequence has been identified as the late Precambrian Apache series. Superior report that a geologist familiar with the Vekol has identified the Lakeshore series as Apache (the tactite as the Mescal Limestone) and that the Lakeshore drilling for the greater part, by his interpretation, has stopped in the transition zone between the Mescal and the underlying Dripping Springs Quartzite. This assumes importance as on the Vekol the better mineralization occurs in diabase sills intruding the Quartzite and a similar occurrence could exist on the Lakeshore below present drilling limits.

Our interpolation of the Lakeshore geology is shown on the accompanying sections and is based on the El Paso logging and the several drill cores which we checked. The section from surface to the depth drilled reads:

Aluvium	-	50 to 100 feet in depth where noted.
Conglomerate	-	With interbeds of basalt flows. Could be the Gila. Greatest depth noted is 1,800 feet.

Andesite, Silicified Andesite, Andesite Breccia - As logged do not offer any obvious correlation. Our brief examination indicates at least two and probably three different rock types, some sedimentary. Newmont consider these as possibly Cretaceous sediments.

Tactite - Silicified limey sediments and limestones, some interbeds of quartzite near base.

Quartzite - Interbeds of tactite near top, grading to quartzite.

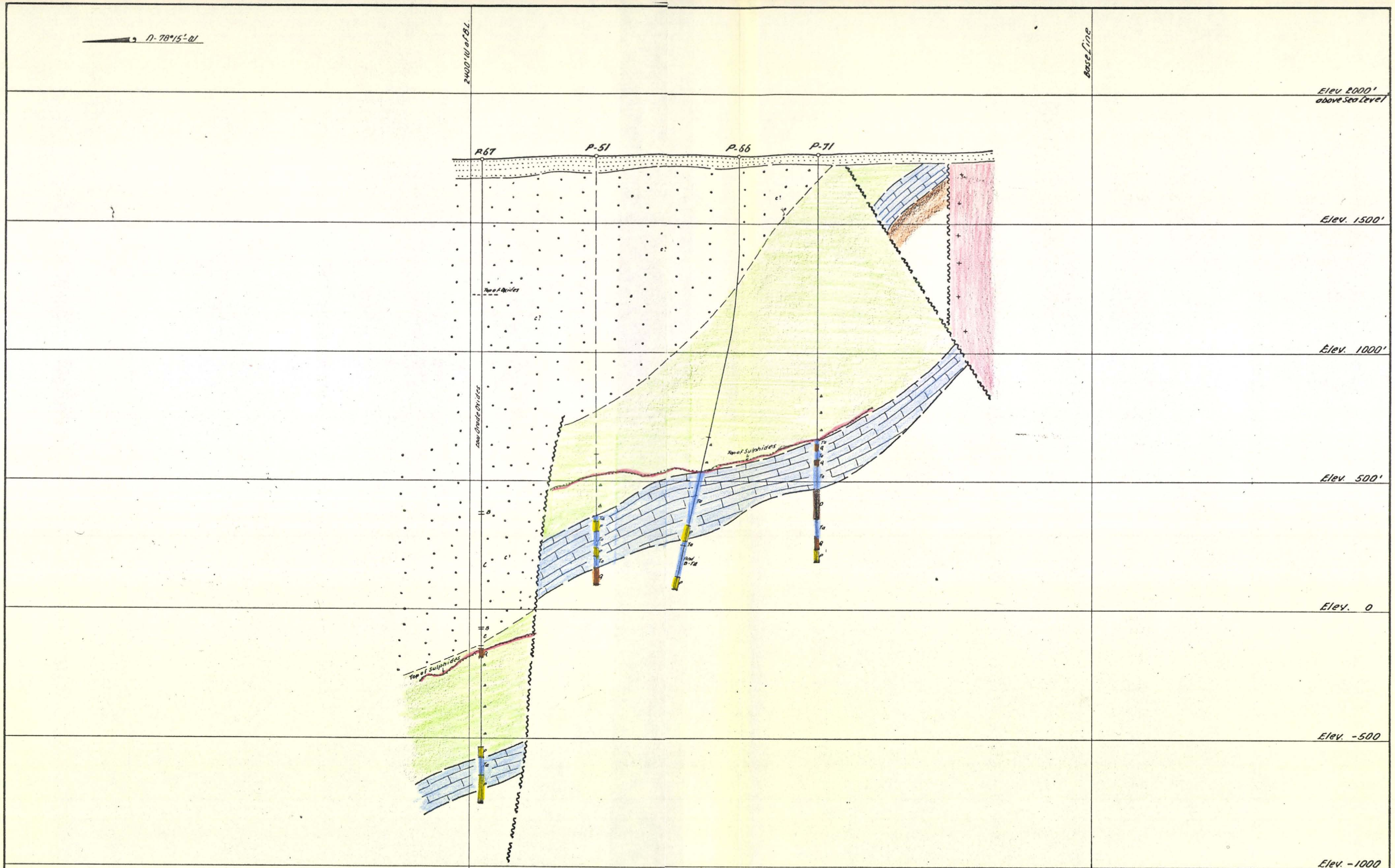
Porphyry - Quartz-feldspar with biotite (secondary?), light to dark grey in colour. Probably an altered quartz monzonite intrudes all rock sequence except granite.

Diabase - Intrudes the tactite and quartzite sequence.

Granite - Precambrian forms the east wall of the pit. Light grey in colour, unmineralized.

The tactite-quartzite sequence strikes roughly north-south and dips to the west at from 25°-30° with local flattenings. The porphyry intrusion appears to have entered from a westerly or north-westerly direction in the form of numerous dikes and sill-like bodies. In the central portion of the drilled area there is a considerable thickening of the porphyry where it has stoped up into the "andesite" complex.

All drill cores examined showed a high degree of random fracturing and brecciation similar to that observed in the surface pit. Near the west limits of the drilled area, normal faulting is indicated with the west side dropped down approximately 700 feet. This is evident on Sections 0 and 300 N and can be interpolated on Section 1800 N from the results in DDH P-73. The fault strikes roughly north-south and probably dips steeply to the west. This strike direction, lying approximately parallel to the pediment, is common. Major structural lineaments are probably present, but their attitudes or importance are unknown at this time.



LEGEND:

CENOZOIC

Conglomerate (Gila?)

CRETACEOUS-TERTIARY (LARAMIDE?)

Biotite-Feldspar-Quartz Porphyry

LATE PRECAMBRIAN (APACHE?)

Andesite, Andesite Breccia

Diabase

Tactite (Mescal?)

Quartzite (Dripping Springs?)

PRECAMBRIAN

Granite

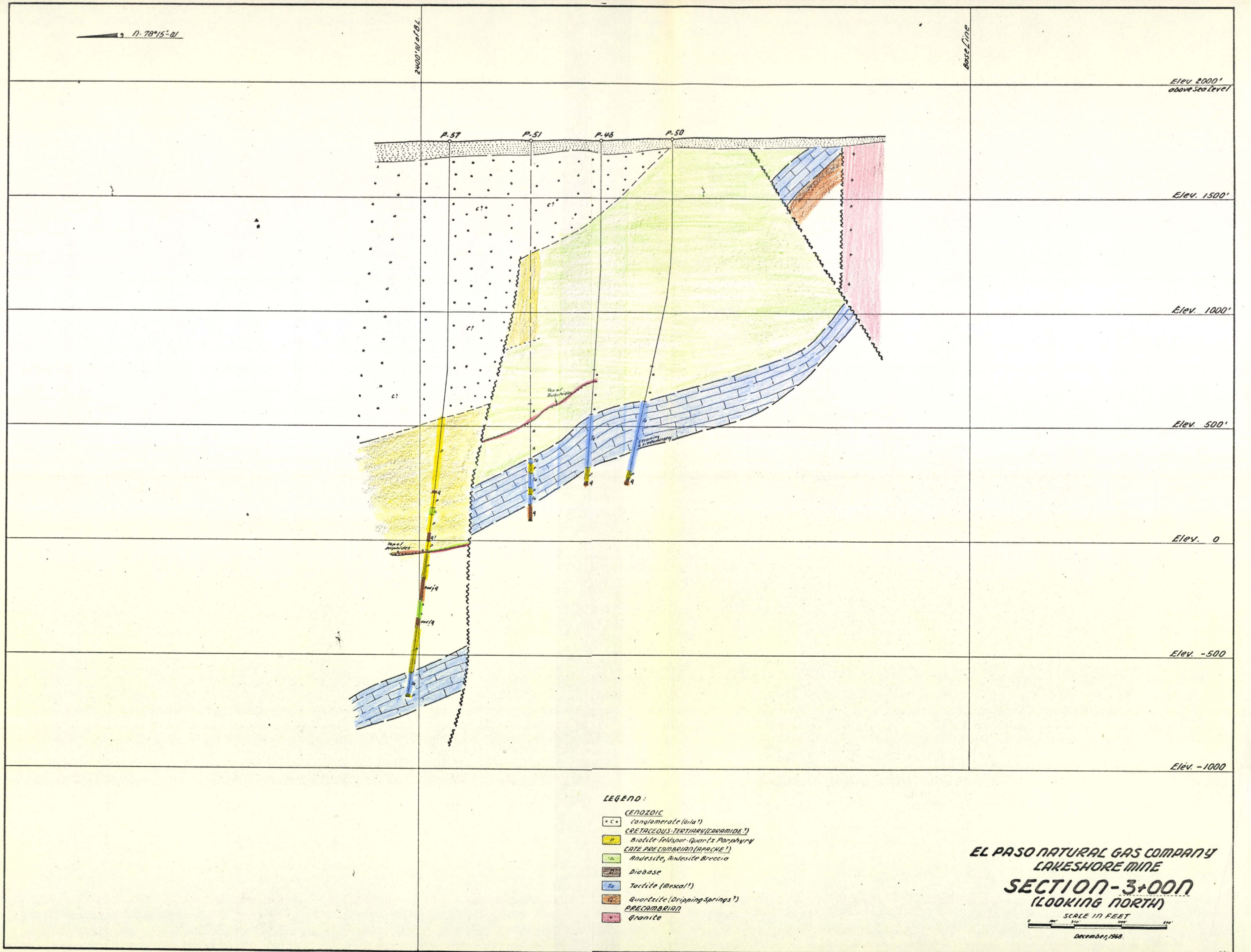
EL PASO NATURAL GAS COMPANY
LAKESHORE MINE

SECTION - 0+00
(LOOKING NORTH)

SCALE IN FEET

0 100' 200' 400' 600'

December, 1968.


N-78°15'-01
2400' W of BL
Base Line
Elev 2000' above Sea Level
Elev. 1500'
Elev. 1000'
Elev. 500'
Elev. 0
Elev. -500
Elev. -1000
P-57
P-51
P-46
P-50
LEGEND:
CENOZOIC
Conglomerate (Gila?)
CRETACEOUS-TERTIARY (LARAMIDE?)
Biotite-Feldspar-Quartz Porphyry
LATE PRECAMBRIAN (APACHE?)
Andesite, Andesite Breccia
Diabase
Tactite (Mescal?)
Quartzite (Dripping Springs?)
PRECAMBRIAN
Granite
EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION-3+00N
(LOOKING NORTH)
SCALE IN FEET
December, 1968.

N-78°15'-W

2400'W of B.L.

Base Line

Elev. 2000' above Sea Level

Elev. 1500'

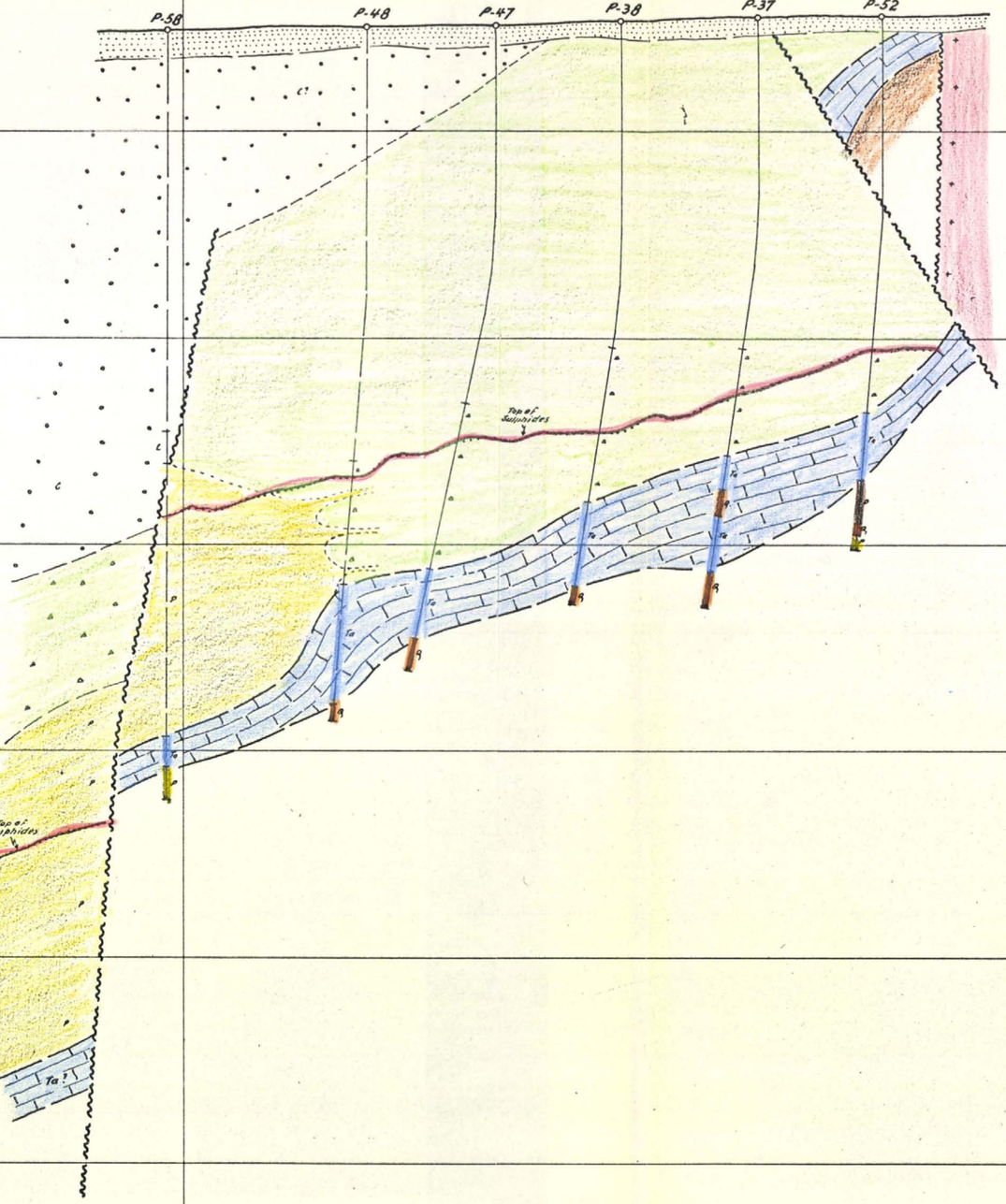
Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000



LEGEND:

CENOZOIC
- Conglomerate (Gila?)

CRETACEOUS-TERTIARY (LARAMIDE?)
- Biotite-feldspar-Quartz Porphyry

LATE PRECAMBRIAN (APACHE?)
- Andesite, Andesite Breccia
- Diabase
- Tactite (Mescal?)
- Quartzite (Dripping Springs?)

PRECAMBRIAN
- Granite

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION - 6+00N
(LOOKING NORTH)

SCALE IN FEET

0 100' 200' 400' 600'

December, 1968.

N-78°15'-W

2400' W of B.L.

Base Line

Elev. 2000' above Sea Level

Elev. 1500'

Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000

P-63 P-58 P-45 P-39 P-29 P-8 P-36

Silic'd. A

Chipe A

Top of Sulphides

Top of Sulphides ?

Fetsite

LEGEND:

CENOZOIC
- C — Conglomerate (Gila?)

CRETACEOUS-TERTIARY (LARAMIDE?)
- P — Biotite-feldspar-Quartz Porphyry

LATE PRECAMBRIAN (APACHE?)
- A — Andesite, Andesite Breccia
- D — Diabase
- Ta — Tactite (Mescal?)
- Q — Quartzite (Dripping Springs?)

PRECAMBRIAN
- + — Granite

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION - 9+00N
(LOOKING NORTH)

SCALE IN FEET

0 100' 200' 300' 400'

December, 1948.

N-78°15'-W

2400' W of B.L.

Base Line

Elev. 2000' above sea level

Elev. 1500'

Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000

P-63 P-56 P-40 P-3 P-5 P-28A

Top of Sulphides

Granite Porphyry

Top of Sulphides

Ta

LEGEND:

CENOZOIC
- C — Conglomerate (Gila?)

CRETACEOUS-TERTIARY (LARAMIDE?)
- P — Biotite-feldspar-Quartz Porphyry

LATE PRECAMBRIAN (APACHE?)
- A — Andesite, Andesite Breccia
- D — Diabase
- Ta — Tactite (Mescal?)
- Q — Quartzite (Dripping Springs?)

PRECAMBRIAN
- + — Granite

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION - 12+00N
(LOOKING NORTH)

SCALE IN FEET

0 100' 200' 300' 400'

December, 1968.

N. 78°15'-W

2400'W of BL

Base Line

Elev 2000' above Sea level

Elev. 1500'

Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000

LEGEND:

CENOZOIC
- C. Conglomerate (Gila?)

CRETACEOUS-TERTIARY (LARAMIDE?)
- P Biotite feldspar-Quartz Porphyry

LATE PRECAMBRIAN (APACHE?)
- Andesite, Andesite Breccia
- Diabase
- Ta Tactite (Mescal?)
- Quartzite (Dripping Springs?)

PRECAMBRIAN
- Granite

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION -15+00N
(LOOKING NORTH)

SCALE IN FEET
0 100' 200' 400'

December, 1968.

N-78°15'-W

Base Line

Elev. 2000' above Sea Level

Elev. 1500'

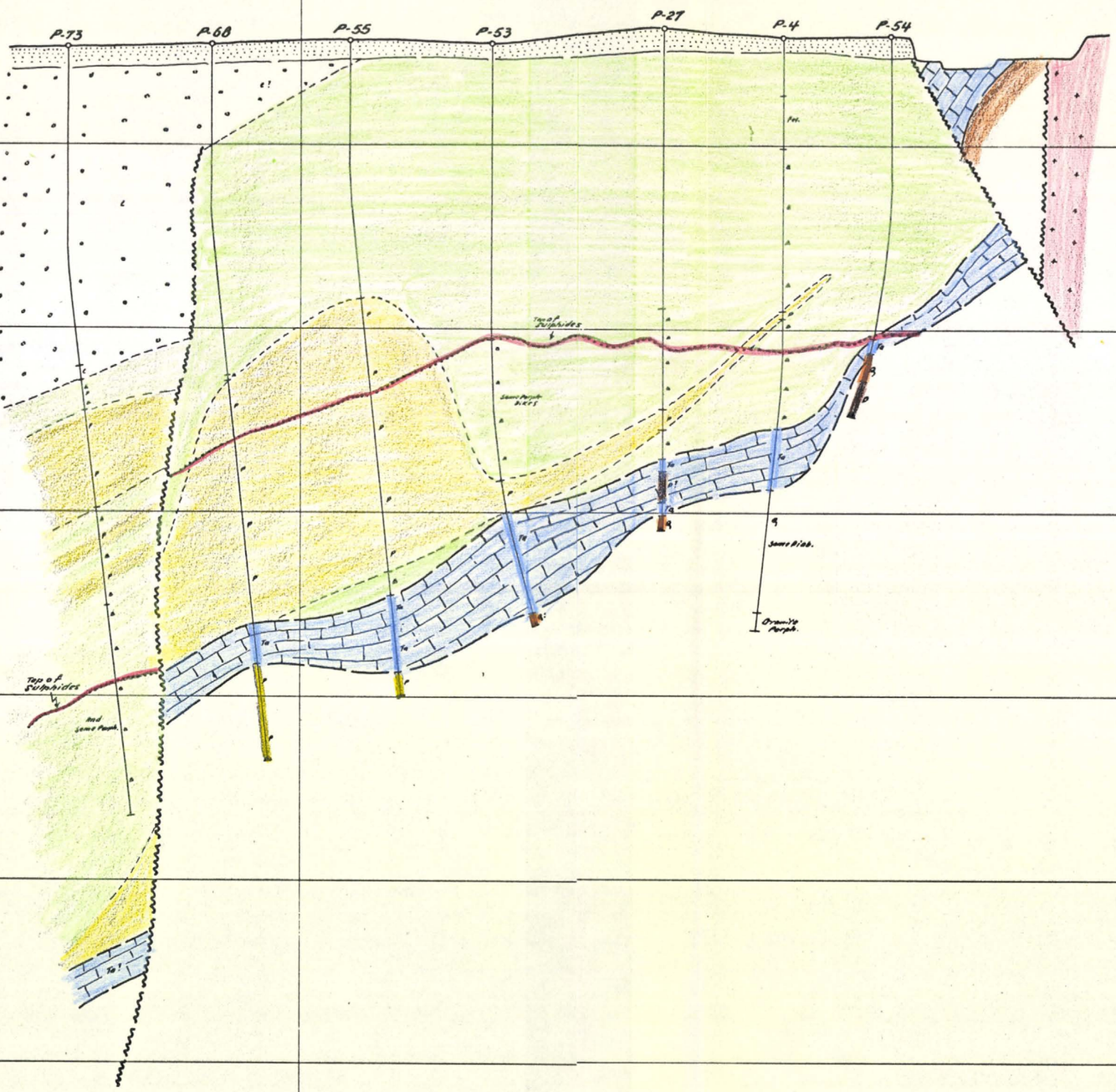
Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000



LEGEND:

CENOZOIC
- C — Conglomerate (Gila?)

CRETACEOUS-TERTIARY (LARAMIDE?)
- P — Biotite-feldspar-Quartz Porphyry

LATE PRECAMBRIAN (APACHE?)
- A — Andesite, Andesite Breccia
- D — Diabase
- Ta — Tactite (Mescal?)
- Q — Quartzite (Dripping Springs?)

PRECAMBRIAN
- + — Granite

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION - 18+00N
(LOOKING NORTH)

SCALE IN FEET

0 100' 200' 400' 600'

December, 1968.

N-78°15'-W

2+10'-W of 21+00 E

Base Line

Elev 2000' above Sea Level

Elev. 1500'

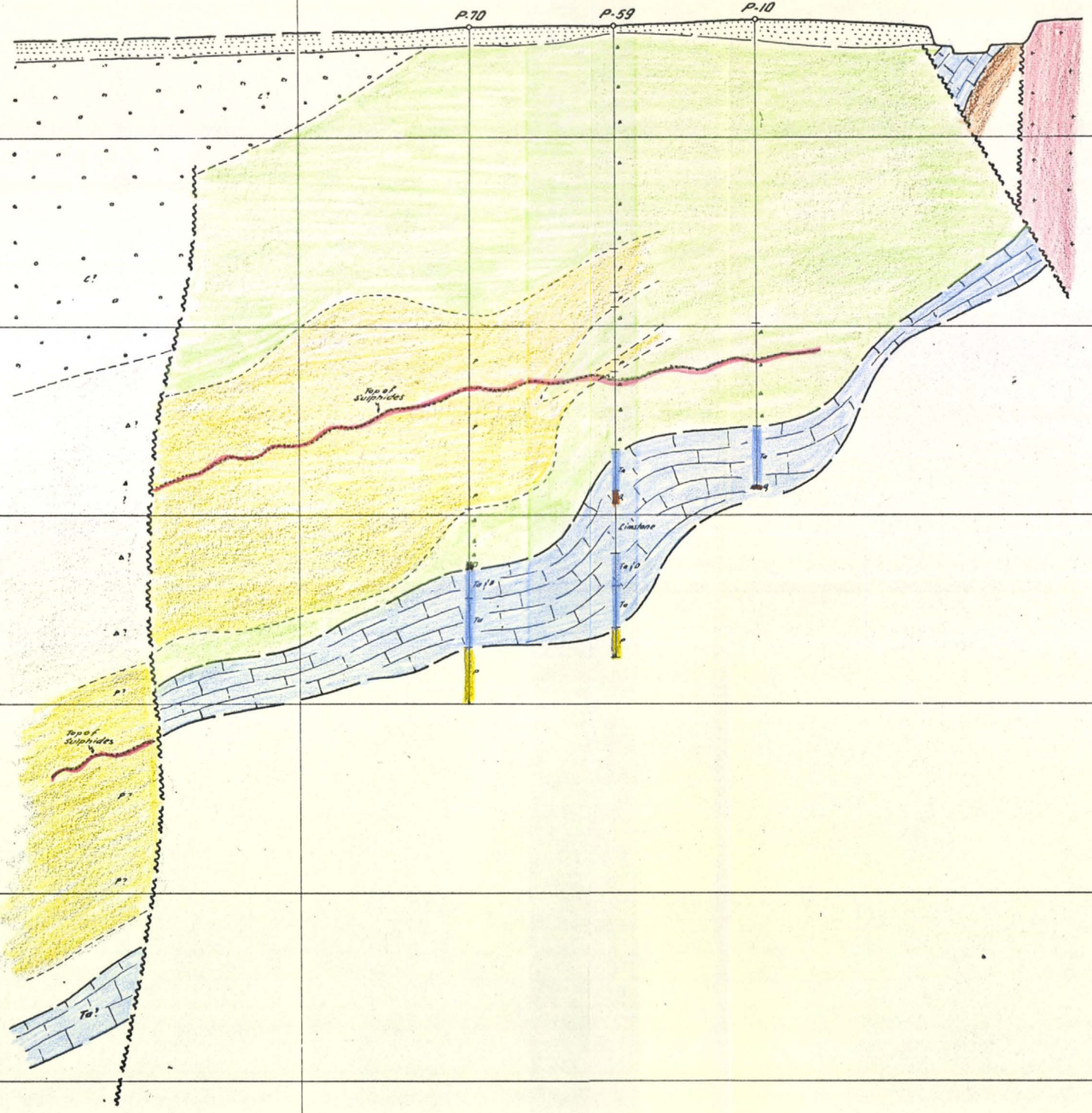
Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000



LEGEND:

CENOZOIC

C — Conglomerate (Gila?)

CRETACEOUS-TERTIARY (LARAMIDE?)

P — Biotite-Feldspar-Quartz Porphyry

LATE PRECAMBRIAN (APACHE?)

A — Andesite, Andesite Breccia

D — Diabase

Ta — Tactite (Mescal?)

Q — Quartzite (Dripping Springs?)

PRECAMBRIAN

G — Granite

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION-21+00N
(LOOKING NORTH)

SCALE IN FEET

0 100' 200' 400' 600'

December, 1968.

N-78°15'-W

2400' W of ℄

Base Line

Elev 2000' above Sea Level

Elev. 1500'

Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000

P-73

P-69

Alluvium

C

Top of Sulphides ?

A & Q Breccia

P

A

Ta

Nor. 28°

Top of Sulphides ?

Ta ?

LEGEND:

CENOZOIC

C — Conglomerate (Gila ?)

CRETACEOUS-TERTIARY (LARAMIDE ?)

P — Biotite feldspar-Quartz Porphyry

LATE PRECAMBRIAN (APACHE ?)

A — Andesite, Andesite Breccia

D — Diabase

Ta — Tactite (Mescal ?)

Q — Quartzite (Dripping Springs ?)

PRECAMBRIAN

+ — Granite

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION - 24+00N
(LOOKING NORTH)

SCALE IN FEET

0 100' 200' 400' 600'

December, 1968.

N-78°15'-W

2400'10 of BL

Base Line

Elev. 2000' above sea level

Elev. 1500'

Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000

P.76 (Proposed)

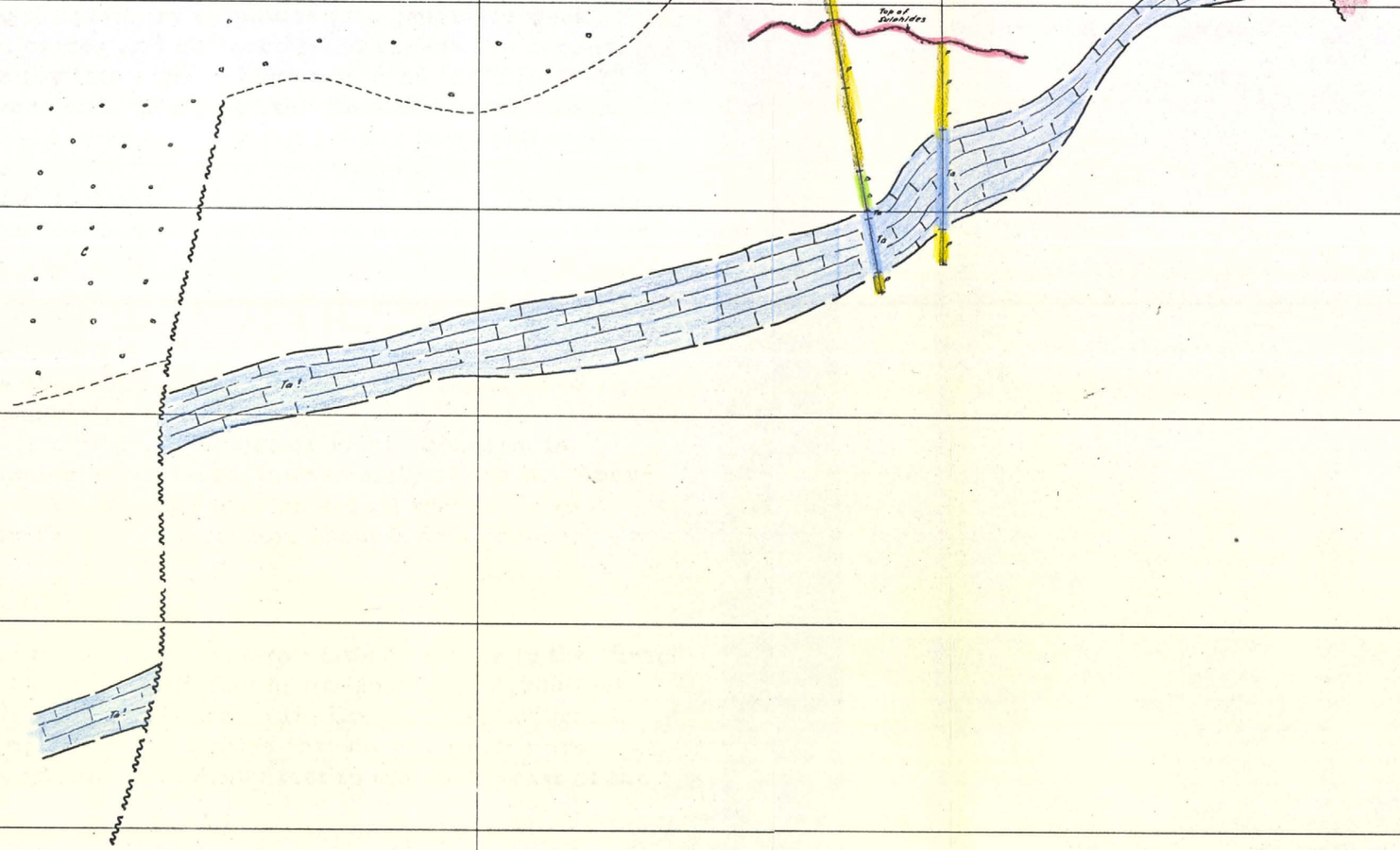
P.77 (Proposed)

P.44

P.33

605

Top of Sulphides



LEGEND:

CENOZOIC

C — Conglomerate (Gila?)

CRETACEOUS-TERTIARY (LARAMIDE?)

Biotite-Feldspar-Quartz Porphyry

LATE PRECAMBRIAN (APACHE?)

Andesite, Andesite Breccia

Diabase

Ta — Tactite (Mescal?)

Quartzite (Dripping Springs?)

PRECAMBRIAN

Granite

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE

SECTION - 27+00N
(LOOKING NORTH)

SCALE IN FEET

0 100' 200' 400'

December, 1968.

Mineralization

Primary sulphide mineralization in the form of chalcopyrite and bornite occurs in all rock types, in attitude generally conforming with the strike and dip of the formation, and extending in significant amounts to the base of the tactite. Many of the drill holes bottomed in mineralization grading 0.3 - 0.4% copper. The top of the sulphides is on the average around 900 feet below surface, although one section at the apex of the porphyry thickening shows them up to within 650 feet. Here the sulphide zone reaches a maximum thickness of over 1,000 feet.

Above the primary sulphides is a generally weak supergene zone of oxides and chalcocite-bornite-native copper. Exceptions to this (by late report) have occurred in DDHs P-73 and P-74 which were still drilling at the time of our visit to the property. DDH P-73 is reported to have a 444 foot section of primarily chalcocite mineralization which averages 1.7% copper. DDH P-74 has a 200 foot chalcocite section grading 1.28% copper. Both holes are considered to be drilling in close proximity to the interpolated fault, which could explain the thickness of secondary enrichment.

Above the supergene zone, wherever this can be defined, the oxide zone mineralization consists primarily of chrysacolla with malchite, azurite, cuprite, etc. and is relatively erratic in outline. The better values conform in general to the sulphide zone and lie immediately above it. Above this higher grade oxide zone there is an abrupt transition to a large blanket of oxide material grading about 0.25% copper.

Potential Mineral Extensions

Except to the east, mineralization extends to the limits of the area drilled, some 2,800 feet north-south and 2,000 feet east-west. At this time there are no limiting factors evident in the other directions. It may be noted that copper oxides are reported in outcroppings some 2,000 feet to the south-east of the present pit.

There is no appreciable zone of secondary enrichment in the area to the east of the fault, however just west of the fault two holes, P-73 and P-74, apparently have intersected important thicknesses of supergene chalcocite mineralization. The proximity of these holes to the fault would be a logical explanation, however the possibility exists of an extensive blanket of higher grade values west of the fault.

In the area to the east of the fault, drilling with few exceptions was bottomed just below the tactite. There were few holes that did not have 0.3-0.4% Cu values at final depths. The possibility exists of repeating the Vekol section with its mineralized diabase sills in quartzite.

As the porphyry (monzonite) intrustion from its configuration appears to have originated from a source to the west or north-west of the drilled area, there is the possibility of a mineralized parent stock lying close by.

Summing up, the possibilities of significent mineral extensions which could effect mining economics are:

1) Mineralization of the same tenor extending to the north, south and west;

2) An extensive chalcocite blanket in the area to the west of the fault;

3) Mineralized diabase sills in the quartzite below presently drilled depths;

4) The near proximity of a mineralized monzonite stock.

FEASIBILITY

This report covers the potentials of open pit and underground mining on the Lakeshore and in addition looks at the possibility of a joint operation of the Lakeshore and Vekol properties.

General Considerations

Factors which effect the various considerations are the following:

Assay Correction:

Many operators through experience find it necessary to apply assay correction factors on diamond drill hole sampling results. These commonly range from 5 - 10%.

In our examination, we obtained 10 of the El Paso diamond drill-hole sample pulp rejects, 5 each of oxide and sulphide sections. These were run at a Tucson custom assay

lab as a check on the El Paso assaying. The results without exception were lower than El Paso's by an average of 10.1% on the oxides and 3.9% on the sulphides. While the checks were insufficiently comprehensive to permit a conclusion, some doubt is cast on the El Paso assaying.

For the reasons above, we have applied assay corrections to the El Paso drilling results of -10% and -4% to oxides and sulphides respectively.

Metallurgy:

From the very limited metallurgical test work done on the oxides (results from test work on cores and chips of two holes, P-1 and P-53) and the necessity of a further very comprehensive test program, this material cannot be considered economic at this time. For the open pit mining study the oxides would have to be economic for the method to be feasible. We have had metallurgical consultants, A. H Ross & Associates of Toronto, make some estimates on the basis of the test data in order that we could complete the exercise on open pitting.

On the sulphides, we have assumed from the test work that a 90% copper recovery and a concentrate grading 28.5% Cu, 0.11 ozs. Au and 1.70 ozs. Ag will be obtained.

Copies of the test data and the Ross report are attached in the Appendix.

Indian Royalties:

We initially understood that these royalties were set at 10% of the net smelter return (NSR), however on the Vekol lease Newmont have negotiated a graduated royalty agreement which reads:

Royalty		NSR/Ton Ore
5%	up to	$3.99
6%	up to	4.24
7%	up to	4.49
8%	up to	4.74
9%	up to	4.99
10%	up to	5.00 and on

For the purpose of the valuations we have assumed that Lakeshore could negotiate a similar royalty schedule.

Smelter:

As custom smelter capacity within economic distance of the Lakeshore mine might not be available we have considered both custom smelter and Lakeshore having its own smelter.

Railroad:

A 30 mile rail link would be required from the Southern Pacific at Casa Grande. Capital allowances have been made for this construction and for an additional spur line to the Vekol for joint operation.

Power:

Power is currently supplied to Lakeshore over a line from Silver Bell. We did not investigate a source for mine requirements but understand that this would be a major problem. The going rate is about 1¢/Kwh.

Housing:

The labour force could be housed at Casa Grande, eliminating the cost of setting up a new townsite. A capital allowance has been made for housing construction.

Water:

Water from wells in the valley to the west of the mine should be adequate for the mine requirements. El Paso presently have a permit to obtain water from points up to 3 miles from the leased area.

PRELIMINARY TEST PROGRAM

Prior to any major capital expenditure consideration a preliminary program of surface diamond drilling and underground exploration development and drilling would be required. The program would be for the purpose of confirming the El Paso results, obtaining additional detail, exploring for ore extensions, and providing bulk samples of both the oxide and sulphide zones for metallurgical testing.

The program as recommended would comprise,

Surface Diamond Drilling - 100,000' @ $7.00/ft.			$. 700,000
Underground Development -			
Shaft	- 1,250' @ $500/ft.	$625,000	
Drifting (oxides)	- 1,000' @ $80/ft.	80,000	
Drifting (sulphides)	- 2,000' @ $80/ft.	160,000	
Core drilling	- 20,000' @ $5/ft.	100,000	965,000
Metallurgical Test Work			500,000
			$2,165,000
Contingencies @ 10%			217,000
			$2,382,000

The program should be completed by the end of the second year in order to permit decision and subsequent engineering for a go ahead.

In order to carry out such a program before major commitment, an option period of at least 3 years would have to be built into any agreement with El Paso.

OPEN PIT MINING

A feasibility study was made for mining the ore east of the fault by open pit. No ore west of the fault was considered, as this is at much too great a depth. As the value of the oxide zone will remain in doubt until comprehensive metallurgical test work has been done, the pit was designed solely for mining the sulphides. Production at 25,000 tons per day was assumed, and that Lakeshore would have its own smelter.

Pit limits were fixed at the point where the cost of additional stripping exceeded the value of the additional ore uncovered. The pit outlined in this manner measured approximately 6,000' by 4,000' across the top and was 1,700' deep. The amount of sulphide ore uncovered amounted to 168,454,000 tons averaging 0.691% copper. Tonnages of overburden and oxide material to be stripped totalled 1,166 million tons, for a stripping ratio of 6.92:1. Owing to the configuration of the sulphide zone, this ratio cannot be improved by any reduction in pit size.

If the oxide material were found to be amenable to leaching and could be considered as ore, the waste: ore ratio would be improved to 3.71:1.

Pit Design:

For the purposes of pit design it was assumed that all overburden would require drilling and blasting and that a final pit stope of 45° was possible. It is quite probable that the more or less unconsolidated conglomerate which overlies the west side of the orebody can be removed by ripping. This would reduce the cost per ton, but, at the same time, the final pit slope would have to be flattened to 32°. The economic position of the bottom of the final pit slope would thus remain almost unchanged. The overall stripping ratio of 6.92:1 allows for the additional material required to be removed by the flatter slope in this section.

For this study a copper price of 37¢/lb. was used, and the assumed costs were taken from the approximate averages for a number of pits. Haulage costs had to be extrapolated, as no figures were available for pits of this depth.

Costs and Minimum Stripping Ratios:

a) Estimated Costs without Haulage

Drilling	2¢/ton
Blasting	3¢/ton
Loading	4¢/ton
General	7¢/ton
	16¢/ton

b) Haulage Costs (assuming 100 ton capacity trucks)

Material	Av. Depth	Haulage Distance			Cost
		Slope @ 10%	Level	Total	¢/ton
Waste	300'	3,000'	5,000'	8,000'	9¢
Oxides	700'	7,000'	3,000'	10,000'	12¢
Sulphides	1,300'	13,000'	3,000'	16,000'	19¢

c) Total Costs per Ton

Material	Haulage	Balance	Total	
Waste	9¢	16¢	25¢	Av. 26.5¢
Oxides	12¢	16¢	28¢	
Sulphides	19¢	16¢	35¢	

d) Maximum Stripping Ratios at Pit Limits (Cu @ 37¢/lb)

Head % Cu		0.8	0.7	0.6	0.5	0.4
Recovery lbs. (@ 90%)		14.4	12.6	10.8	9.0	7.2
Costs:	Mining[1]	$0.35				
	Milling	0.85				
	Overheads	0.50				
		$1.70	$1.70	$1.70	$1.70	$1.70
	Treatment[2]	1.35	1.19	1.03	0.87	0.71
		$3.05	$2.89	$2.73	$2.57	$2.41
Value @ 37¢ Cu		5.33	4.67	4.00	3.33	2.66
Operating Profit		$2.28	$1.78	$1.27	$0.76	$0.25
Max. Stripping Ratio[3]		8.6:1	6.7:1	4.8:1	2.9:1	0.9:1

Notes: 1 Excluding stripping cost
2 Smelting, refining and freight assuming custom smelter
3 Assumed average stripping cost of 26.5¢/ton

Pre-production stripping:

The initial pit would be located on the north-east side of the mineralized area, as this would expose the largest tonnage of sulphide ore at the shallowest depth. Before production started it would be carried down to an elevation of 900 feet above sea level, or approximately 900 feet below surface. The floor at the bottom level would be 340' by 680'. This would be large enough to accommodate large equipment and give sufficient room to begin cutting another level while equipment worked on the floor. For safety and efficient operation of equipment, the sides of the pit would have a 150' berm for each 50' cut. As the sides of the initial pit approached the final limits, the slope would change from 3:1 to 1:1 in rock and 1.5:1 in unconsolidated material.

Tonnages stripped would be as follows:

Alluvium	67.0	million tons
Conglomerate	100.0	" "
Rock	202.5	" "
Oxides	20.3	" "
	389.8	million tons

At a stripping rate of 300,000 tons per day this would take four years to complete.

Pit Reserves:

The pit limits and the diamond drill holes were plotted on a series of cross sections at 300' intervals. On these the ore was outlined using a cut-off grade of 0.4% copper. Sulphides and oxides were outlined separately, and some areas of oxides above 0.4% were excluded as there was evidence that they were high in lime and probably unsuitable for leaching. Areas of sulphide and possible oxide ore were measured from the sections and values from the diamond drill holes were weighted according to their areas of influence. The volumes of the overlying "andesite", conglomerate and alluvium were also estimated from the sections. Tonnage conversion factors were as follows:

Sulphide ore	12.5	cu ft/ton
Possible oxide ore	13.5	" "
Rock ("Andesite")	13.5	" "
Conglomerate	15.9	" "
Alluvium	15.9	" "

Assay corrections of minus 4% and minus 10% were used on the sulphide and oxide material respectively. Dilution allowances were 5% at 0.4% copper with the sulphides and 10% at 0.3% copper with the oxides.

A summary of reserves and stripping ratios is as follows:

Reserves	Tons	Grade % Cu
Sulphides	168,454,000	0.691
Oxides	114,747,000	0.639

Stripping Ratios

Sulphides alone	6.92:1
Sulphides + Oxides	3.71:1

Details of the above are given in the appendix.

Pit Economics (Sulphides only):

PIT OPERATING COSTS (LESS STRIPPING):

Drilling	2¢/ton
Blasting	3¢/ton
Loading	4¢/ton
Haulage (av. depth 1,300')	19¢/ton
General	7¢/ton
Total per ton of ore	35¢/ton

STRIPPING COSTS:

a) Total Stripping:

Material	Tons (millions)	Average Depth	Broken by	Unit Cost ¢/ton	Total Cost (millions)
Alluvium	92.6	40'	Ripping	13¢	$ 12.0
Conglomerate	573.6	700'	Ripping	21¢	$120.5
Rock	395.5	300'	Blasting	25¢	$ 98.9
Oxides	104.3	700'	Blasting	28¢	$ 29.2
	1,166.0				$260.6

At the stripping ratio of 6.92:1, the average stripping cost is $1.55 per ton of ore.

b) Pre-production stripping:

Material	Tons (millions)	Average Depth	Broken by	Unit Cost ¢/ton	Total Cost (millions)
Alluvium	67.0	40'	Ripping	13¢	$ 8.7
Conglomerate	100.0	300'	Ripping	18¢	18.0
Rock	202.5	300'	Blasting	25¢	50.6
Oxides	20.3	700'	Blasting	28¢	5.7
	390.0				$83.0
Contingencies @ 10%					8.3
Total Capital required					$91.3

c) Production stripping:

Cost of stripping during production $177.6
Cost per ton = $1.05

TOTAL OPERATING COST:

With the $91,300,000 pre-production stripping capitalized the operating cost is as follows:

	Per ton
Total mining cost without stripping	$0.35
Stripping cost	1.05
Total mining cost	$1.40
Milling cost	0.85
General overheads	0.50
Operating cost before royalties	$2.75

OPERATING PROFIT (@ 42¢ Copper)

Head grade = 0.691% Copper = 13.82 lbs./ton
Recovered 90% = 12.44 lbs./ton

Concentrate grade Cu - 28.5%
Au - 0.11 oz/ton
Ag - 1.70 oz/ton

Ratio of concentration: 45.8:1

Value of metals recovered per ton of conc.

Cu	570 lbs. @ $ 0.42	=	$239.40
Au	(0.90 x 0.11) @ $35.00	=	3.47
Ag	(1.70 - 0.50) @ $ 2.00	=	2.40
			$245.27
Treatment Charges @ 7.0¢/lb. Cu		=	39.90
Net Smelter Return per ton of conc.		=	$205.37
		=	$4.48/ton ore
Operating cost before royalties		=	2.75/ton ore
Operating profit before royalties		=	$1.73/ton ore

Calculation of Indian Royalty:

Assumed treatment charge @ 8.5¢/lb.	=	$ 48.45
Net smelter return per ton of conc.	=	$196.82
	=	$ 4.30/ton ore
Indian royalty assumed 7% of N.S.R.	=	$ 0.30/ton ore
Operating profit after royalty	=	$ 1.43/ton ore

Capital Expenditures - Open Pit Operation

Test Program	$ 2,382,000	
Pre-production stripping	91,300,000	$93,682,000
Mine Equipment	$10,500,000	
Mill	35,000,000	
Shops	4,000,000	
Office and Housing	3,000,000	
Water Supply	1,000,000	
Smelter	22,000,000	
Railroad	2,000,000	
	$77,500,000	
Contingencies @ 10%	7,750,000	85,250,000
		$178,932,000
Working Capital		12,000,000
		$190,932,000

Summary

Operating profit per ton after payment of Indian Royalty	=	$1.43
Total operating profit - 19.2 years' life	=	$240,889,000
Operating profit per year	=	$12,546,000
Value at start of production, discounted @ 7%	=	$130,000,000
Capital expenditures required prior to production (excluding pre-production interest)	=	$190,932,000

The open pit operation for mining sulphides alone will thus not repay the investment.

ECONOMICS OF TREATING OXIDES

The value of the oxides cannot be properly assessed at this time as considerable metallurgical testing on representative samples of several hundred tons would be required to determine the economics of leaching. Production from oxide leaching would be in addition to the mining of sulphides at 25,000 tons per day and could start 1 1/2 years after stripping commenced. Using the estimates provided for us by A. H. Ross and Associates, metallurgical consultants, based on the limited information available from El Paso, the economics are as follows:

Total tons available	=	114,747,000
Head grade	=	0.639% Cu/ton
Recovered grade @ 65%	=	0.415% Cu/ton
	=	8.30 lbs/ton
Gross value: 8.30 lbs. @ 42¢/lb.	=	$3.49/ton

Operating costs:

Mining	$0.28/ton	
Leaching	1.72/ton	
Smelting	0.59/ton	
Overheads	0.50/ton	$3.09/ton
Operating profit, before royalty		$0.40/ton
Estimated Indian royalty @ 5% NSR		0.15/ton
Operating profit after royalty		$0.25/ton
5,250,000 tons/year	=	$1,315,000

The capital cost of the leaching plant is estimated at $22,100,000.

The operating cost above does not include stripping which it is assumed will be carried by the sulphides. Stripping cost for the sulphides will be reduced by the mining cost of the oxides, or $32,129,000. Of this amount $5.7 million will be a reduction in the pre-production stripping cost and the balance will result in a reduction of operating cost of $0.16 per ton.

Total costs and revenue would then be as follows:

Operating profits (@ 42¢ Cu)

Sulphide ore:	168,454,000 tons	@ $1.59	= $267.8 million
Oxide ore:	114,747,000 tons	@ $0.25	= $ 28.7 million
	283,201,000		$296.5 million
Present value @ 7% discount			= $162 million

Capital Costs:

Cost for sulphide operation	= $190,932,000
Reduction in stripping cost	= 6,270,000
	$184,662,000
Cost of leaching plant	= 22,100,000
Total for sulphide + oxide operation	= $206,762,000

Unless recoveries from the oxides are considerably better than estimated above, pit operation is still uneconomic.

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE

PINAL COUNTY
ARIZONA

SCALE: 1 inch = 400 feet

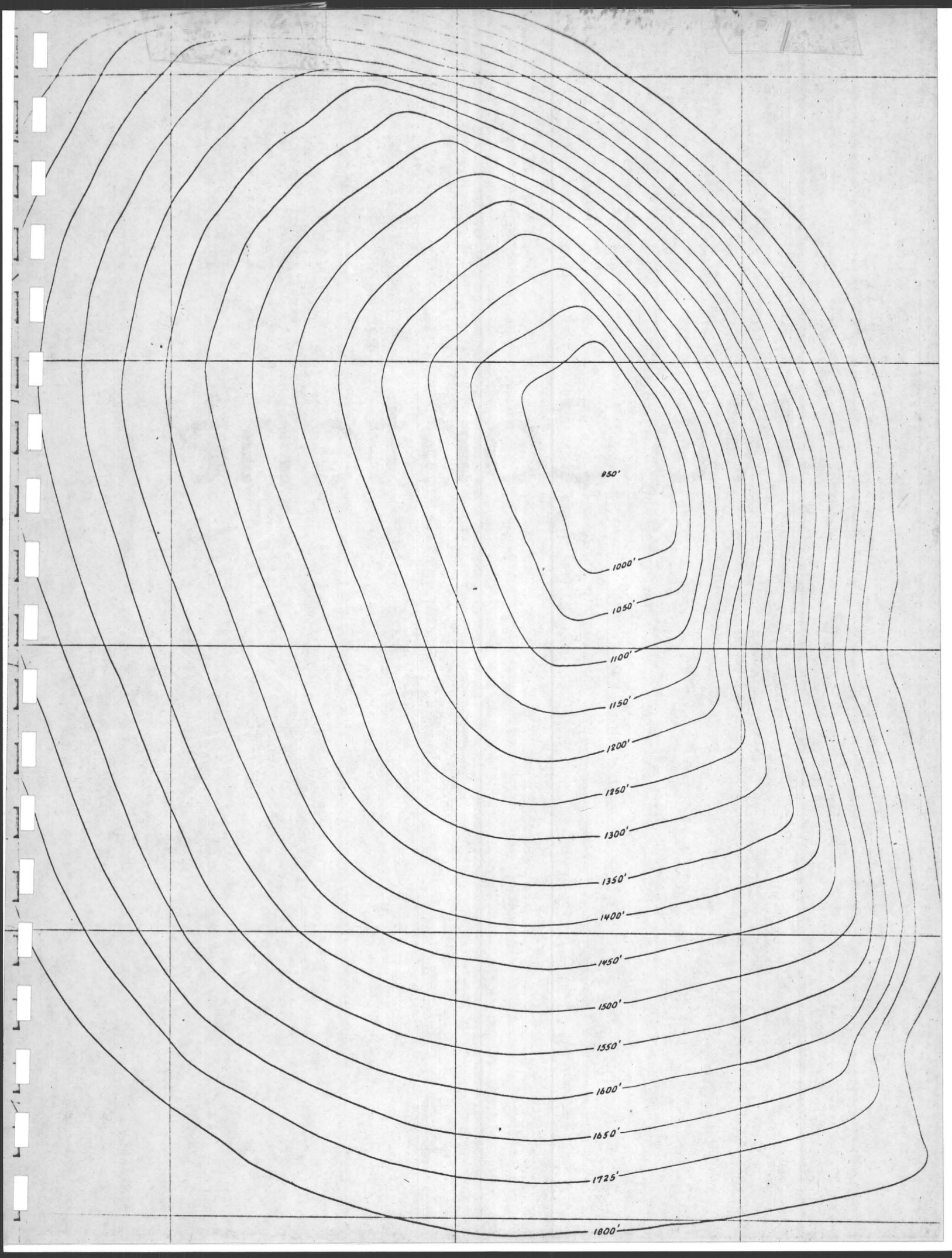
LEGEND

Base Map: Preliminary Pit Layout	Shown in black
Upper Surface Sulphides Overlay	Shown in red
Upper Surface Oxides Overlay	Shown in green
Pit when floor reaches 950' El. Overlay	Shown in blue

NOTE:

Elevation Contours above Sea Level.

A-611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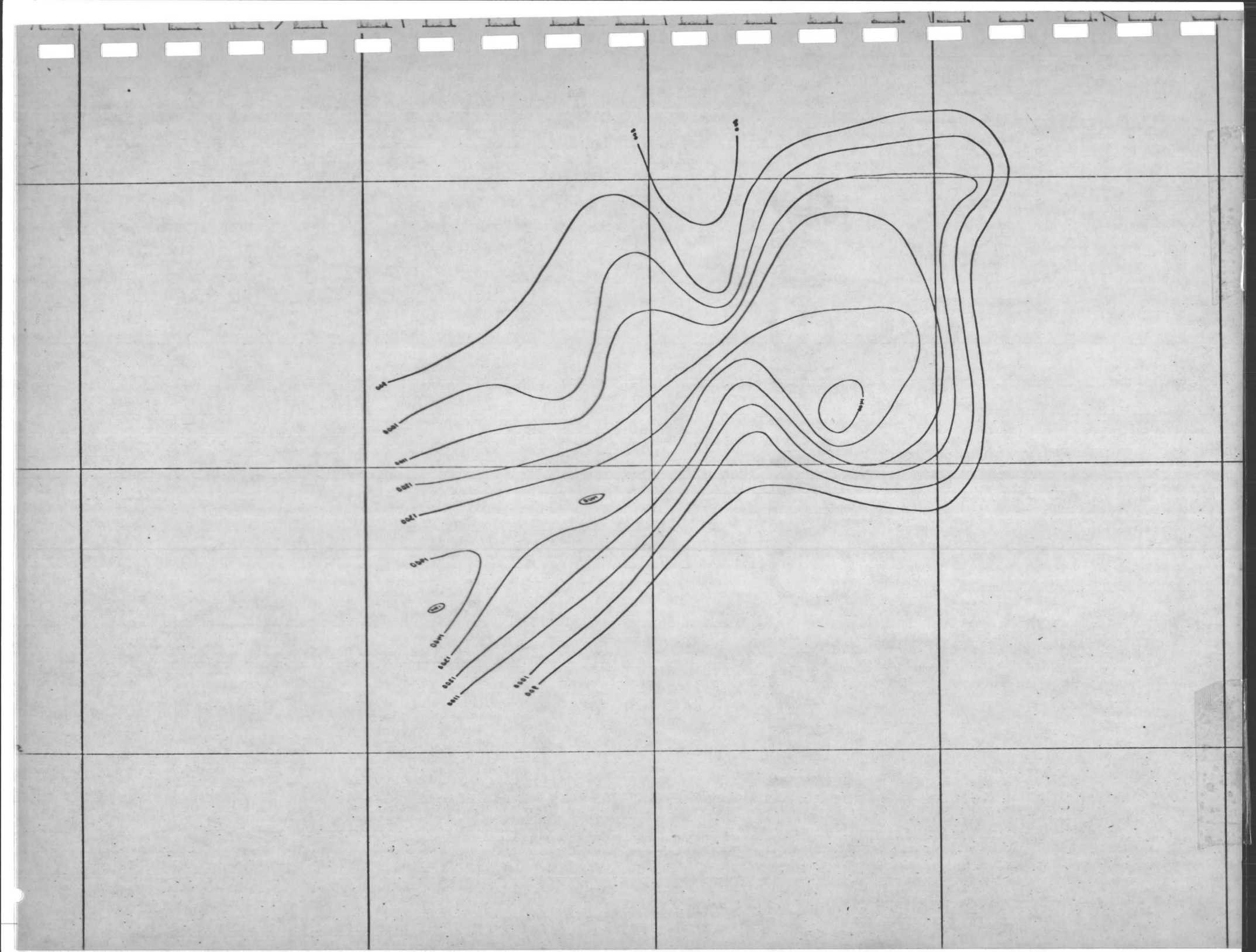
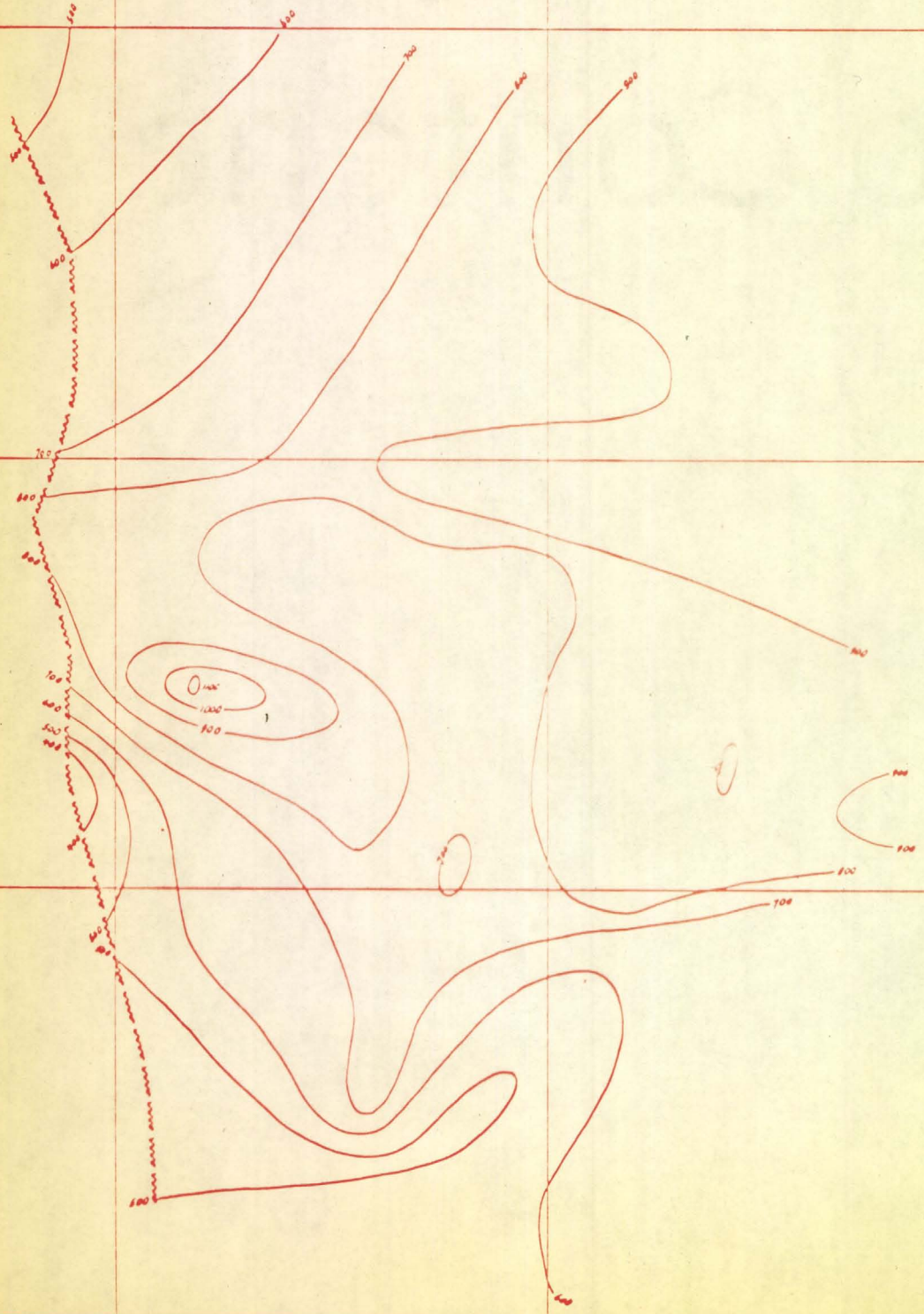
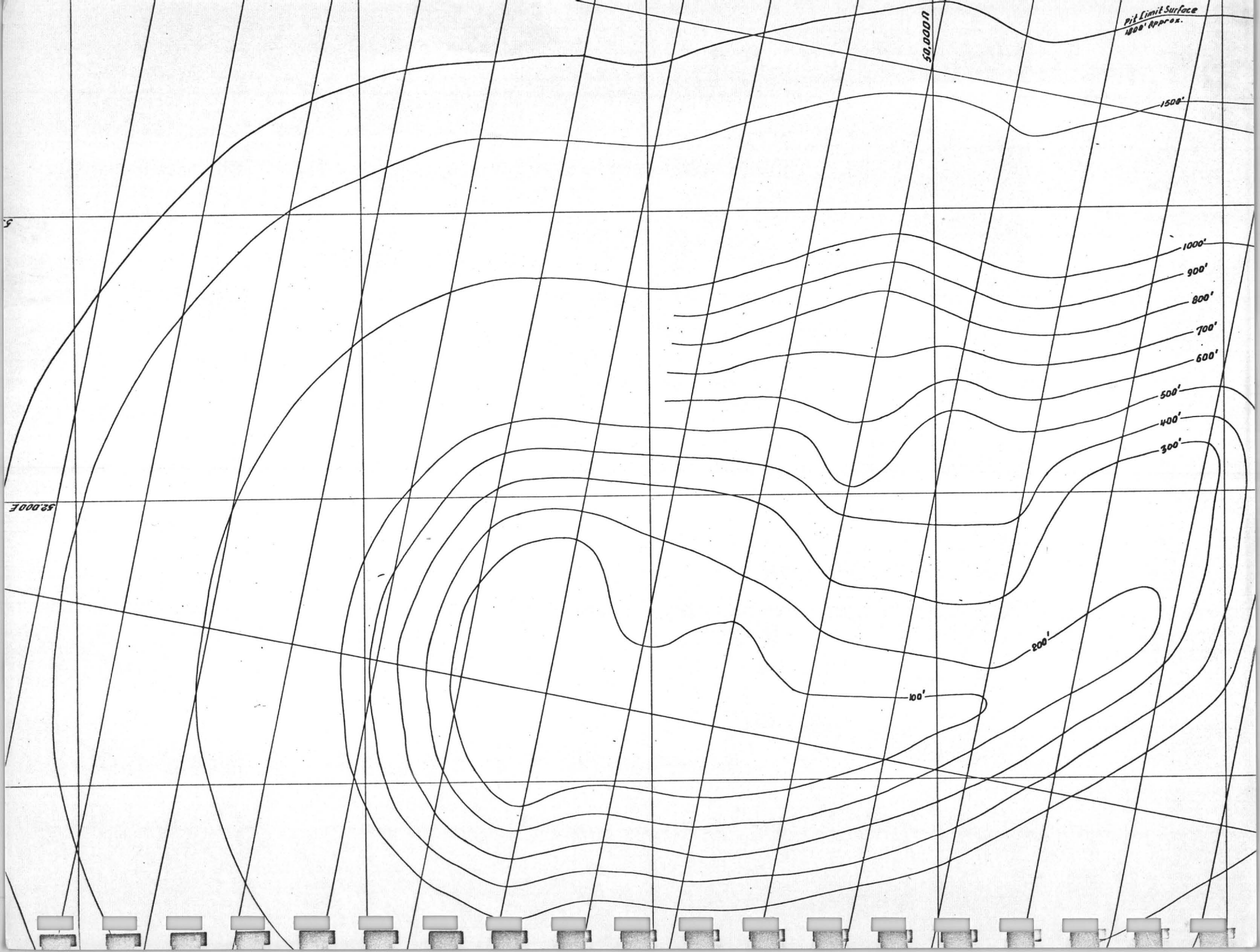





Pit Limit Surface
1800' Approx.
50,000N
1500'
1000'
900'
800'
700'
600'
500'
400'
300'
200'
100'
52,000E

4000' N. of B.L.

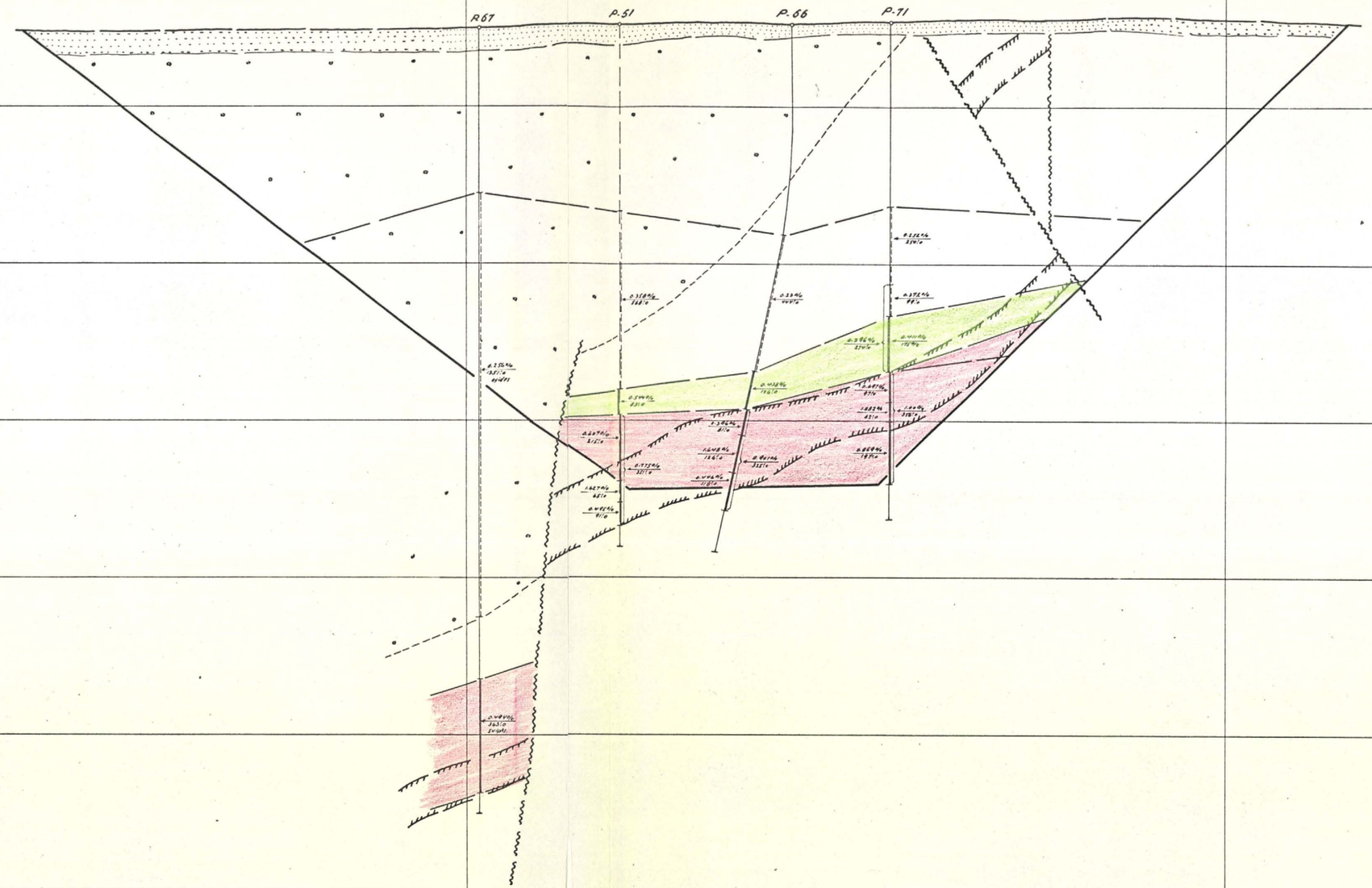
N-78°15'-W

2400' W. of B.L.

Baseline

Elev. 2000' Above Sea Level

R67 P.51 P.66 P.71



Elev. 1500'

Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000

LEGEND:

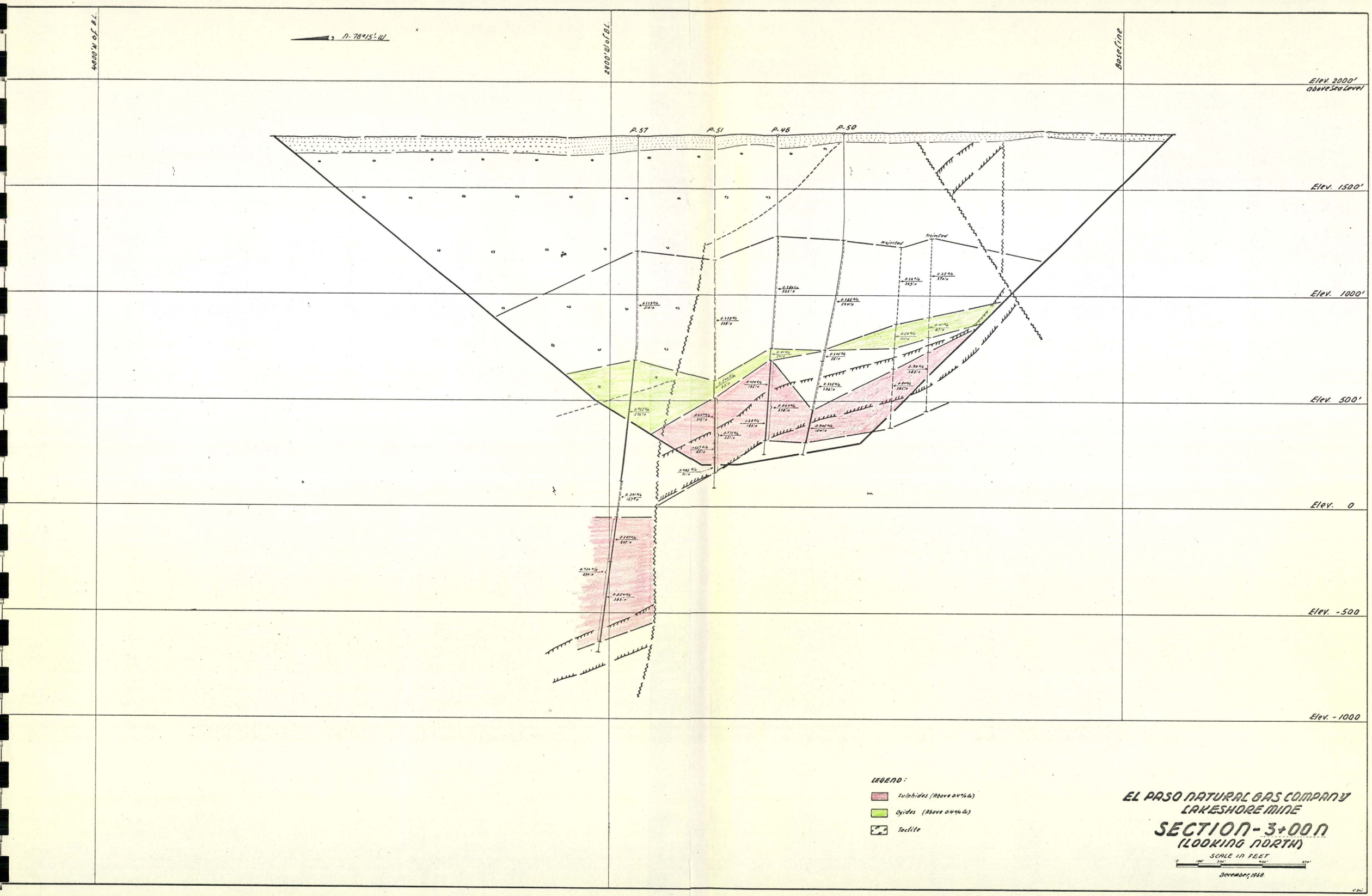
- Sulphides (Above 0.4% Cu)
- Oxides (Above 0.4% Cu)
- Faults

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE

SECTION - 0+00
(LOOKING NORTH)

SCALE IN FEET
0 100' 200' 300' 400'

December, 1968.


4000'W of B.L.
2900'W of B.L.
Baseline
N. 78°15' W
P.37
P.51
P.46
P.50
Elev. 2000'
Above Sea Level
Elev. 1500'
Elev. 1000'
Elev. 500'
Elev. 0
Elev. -500
Elev. -1000
LEGEND:
Sulphides (Above 0.4% Cu)
Oxides (Above 0.4% Cu)
Tactite
EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION - 3+00 N
(LOOKING NORTH)
SCALE IN FEET
December, 1968.

4800' W of B.L.

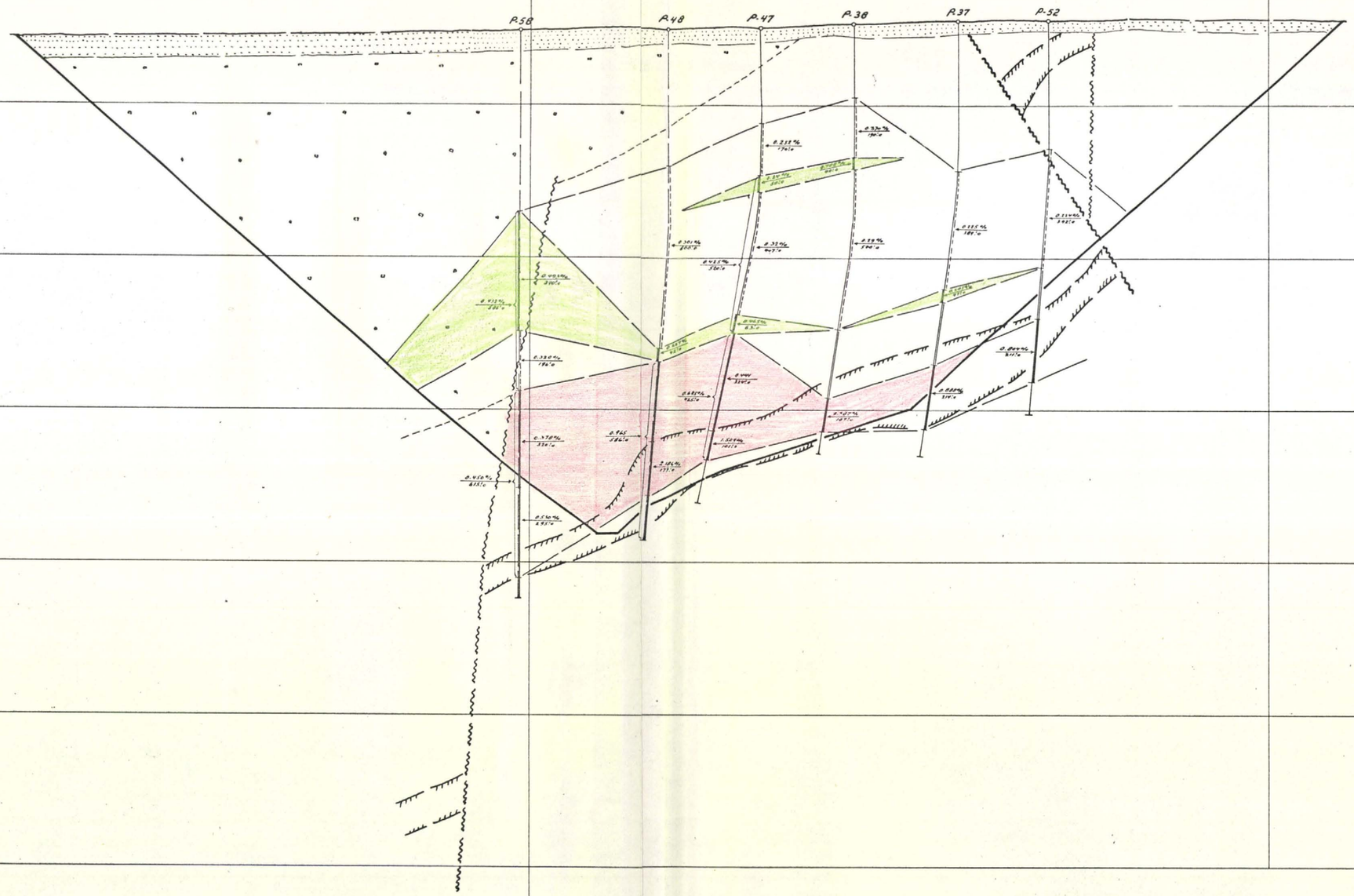
N-78°15'-W

2800' W of B.L.

Baseline

Elev. 2000' above Sea Level

P-58 P-48 P-47 P-38 P-37 P-52



Elev. 1500'

Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000

LEGEND:

- Sulphides (Above 0.4% Cu)
- Oxides (Above 0.4% Cu)
- Faults

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE

SECTION - 6+00N
(LOOKING NORTH)

SCALE IN FEET
0 100' 200' 400' 600'

December, 1968.

7800' N of 0+00

7800' W of 0+00

Baseline

N-78°15'-W

Elev. 2000' above Sea Level

Elev. 1500'

Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000

P-63

P-58

P-45

P-39

P-28

P-8

P-36

0.328% oxides 960'±

0.715% oxides 1790'±

0.687% Sulph. Ppt. 360'±

LEGEND:

- Sulphides (Above 0.4% Cu)
- Oxides (Above 0.8% Cu)
- Faults

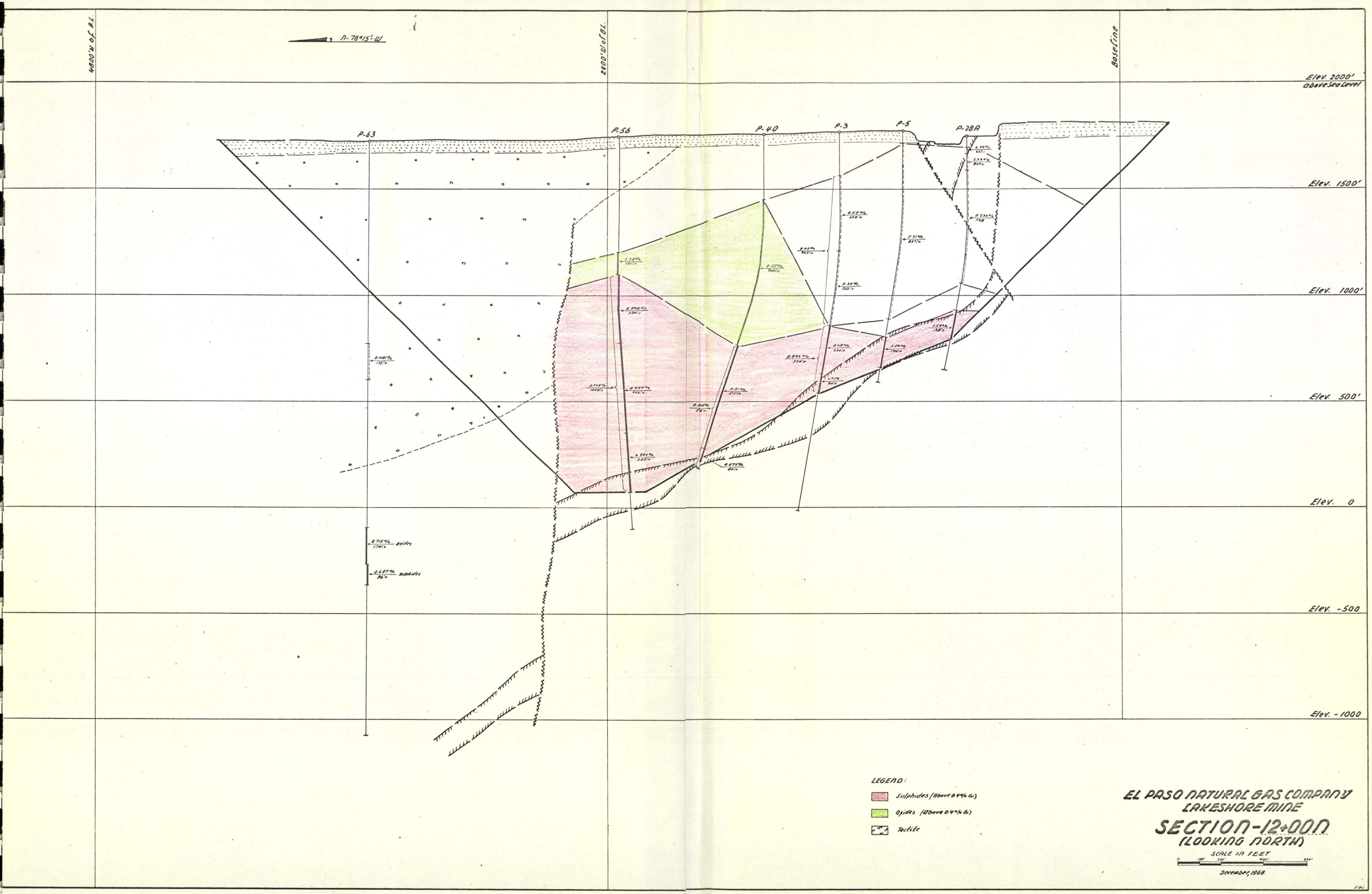
EL PASO NATURAL GAS COMPANY
LAKESHORE MINE

SECTION - 9+00N
(LOOKING NORTH)

SCALE IN FEET

0 100' 200' 400' 600'

December, 1968.


N-78°15'-W
4800' W of B.L.
2400' W of B.L.
Baseline
Elev. 2000' above Sea Level
Elev. 1500'
Elev. 1000'
Elev. 500'
Elev. 0
Elev. -500
Elev. -1000
P-63
P-56
P-40
P-3
P-5
P-28A
LEGEND:
Sulphides (Above 0.4% Cu)
Oxides (Above 0.4% Cu)
Tactite
EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION-12+00N
(LOOKING NORTH)
SCALE IN FEET
0 100' 200' 400' 600'
December, 1968.

N-78°15'-W

4800' N of 78

78 of M.004

Baseline

Elev. 2000' above Sea Level

Elev. 1500'

Elev. 1000'

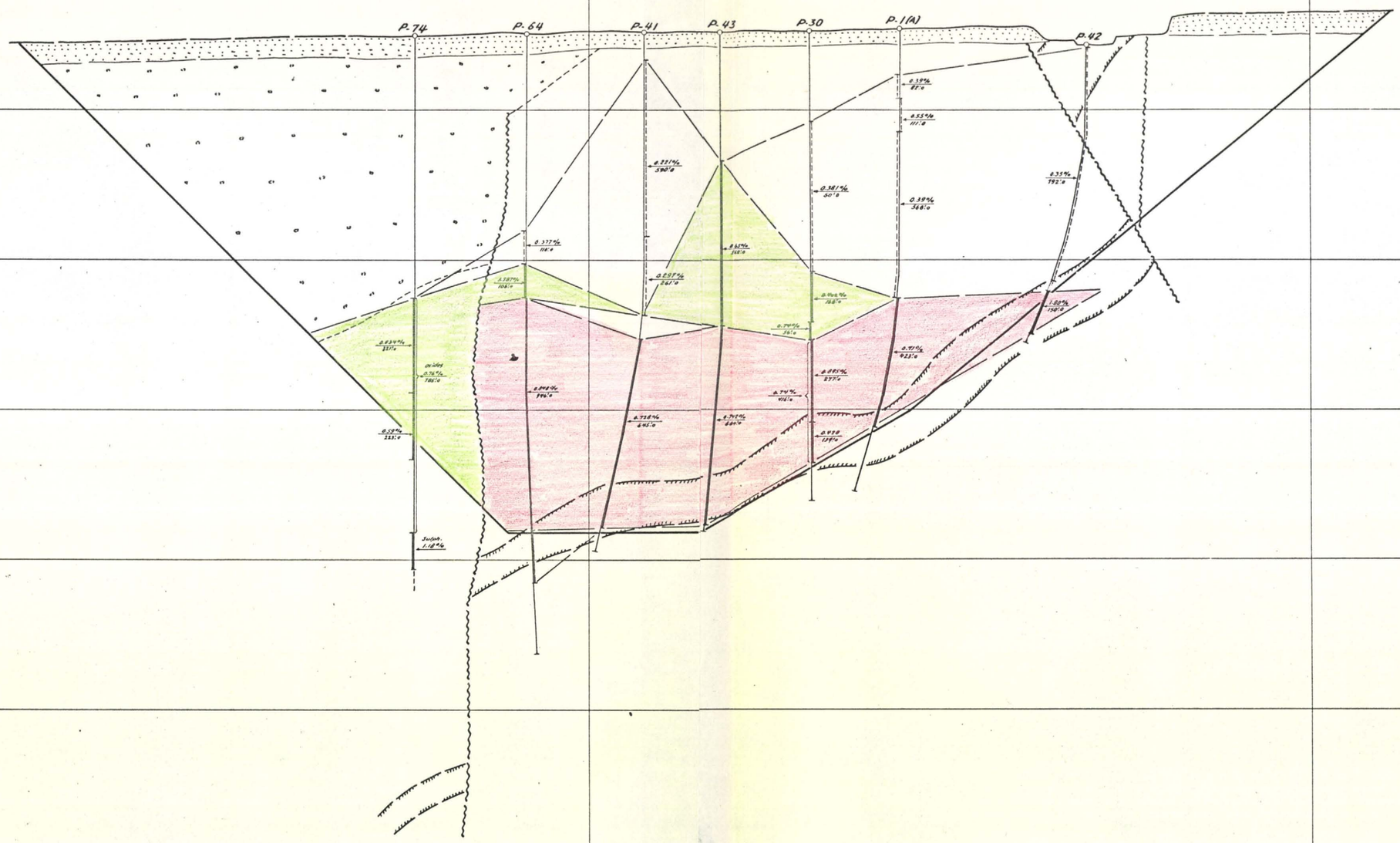
Elev. 500'

Elev. 0

Elev. -500

Elev. -1000

P-74 P-64 P-41 P-43 P-30 P-1(A) P-42



LEGEND:

- Sulphides (above 0.4% Cu)
- Oxides (above 0.4% Cu)
- Tactite

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE

SECTION - 15+00 N
(LOOKING NORTH)

SCALE IN FEET

0 100' 200' 400' 600'

December, 1968.

4800' N of 18+00E

N-78°15'-W

18+00N 0042

Baseline

Elev. 2000' Above Sea Level

Elev. 1500'

Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000

P.73 P.68 P.55 P.53 P.27 P.4 P.54

LEGEND:

- Sulphides (Above 0.4% Cu)
- Oxides (Above 0.8% Cu)
- Porclite

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE

SECTION-18+00N
(LOOKING NORTH)

SCALE IN FEET
0 100' 200' 300' 400'

December, 1968.

400' W of 21

N-78°15'-W

2400' W of 21

Baseline

Elev. 2000'
Above Sea Level

Elev. 1500'

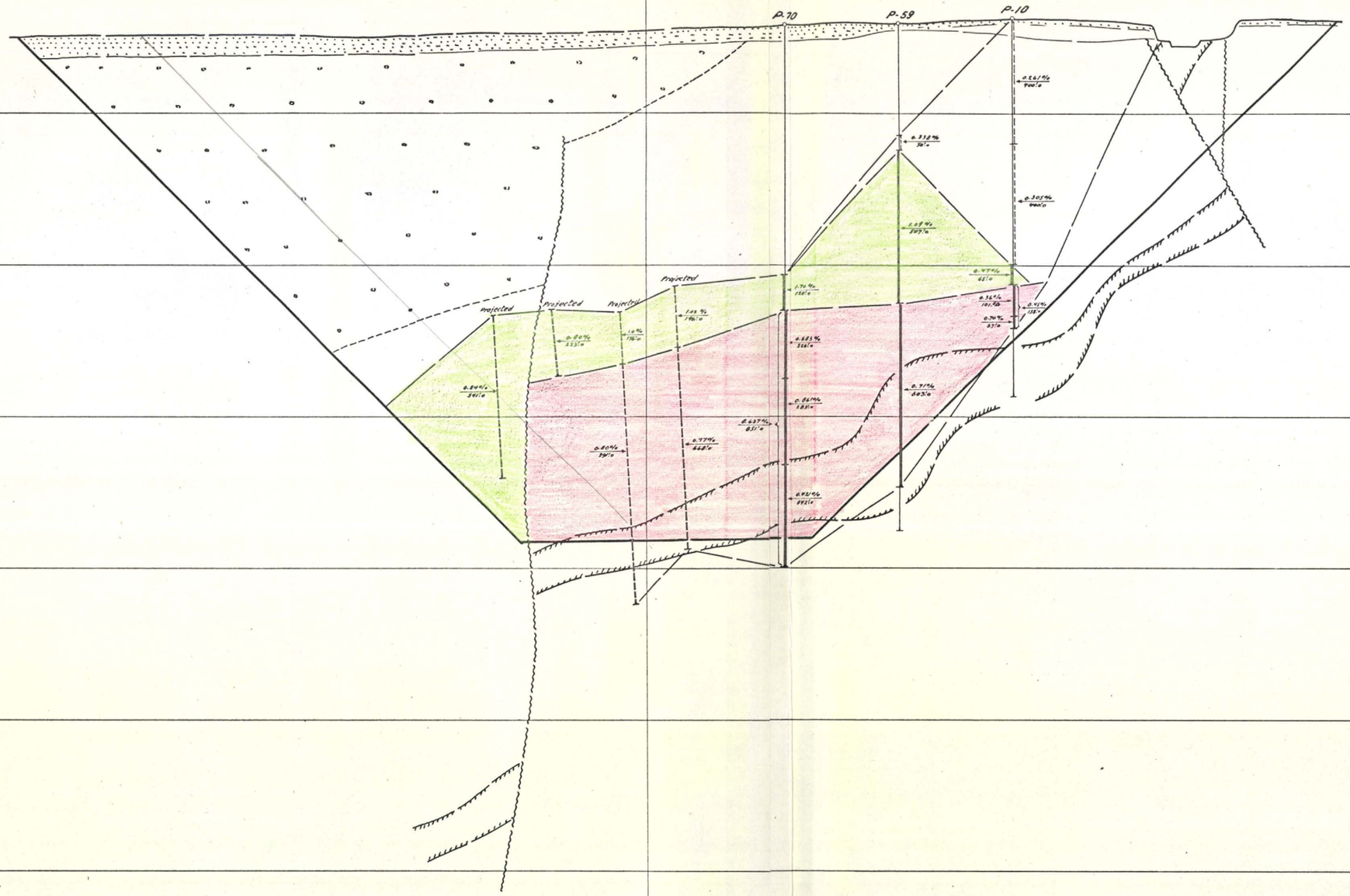
Elev. 1000'

Elev. 500'

Elev. 0

Elev. -500

Elev. -1000



LEGEND:

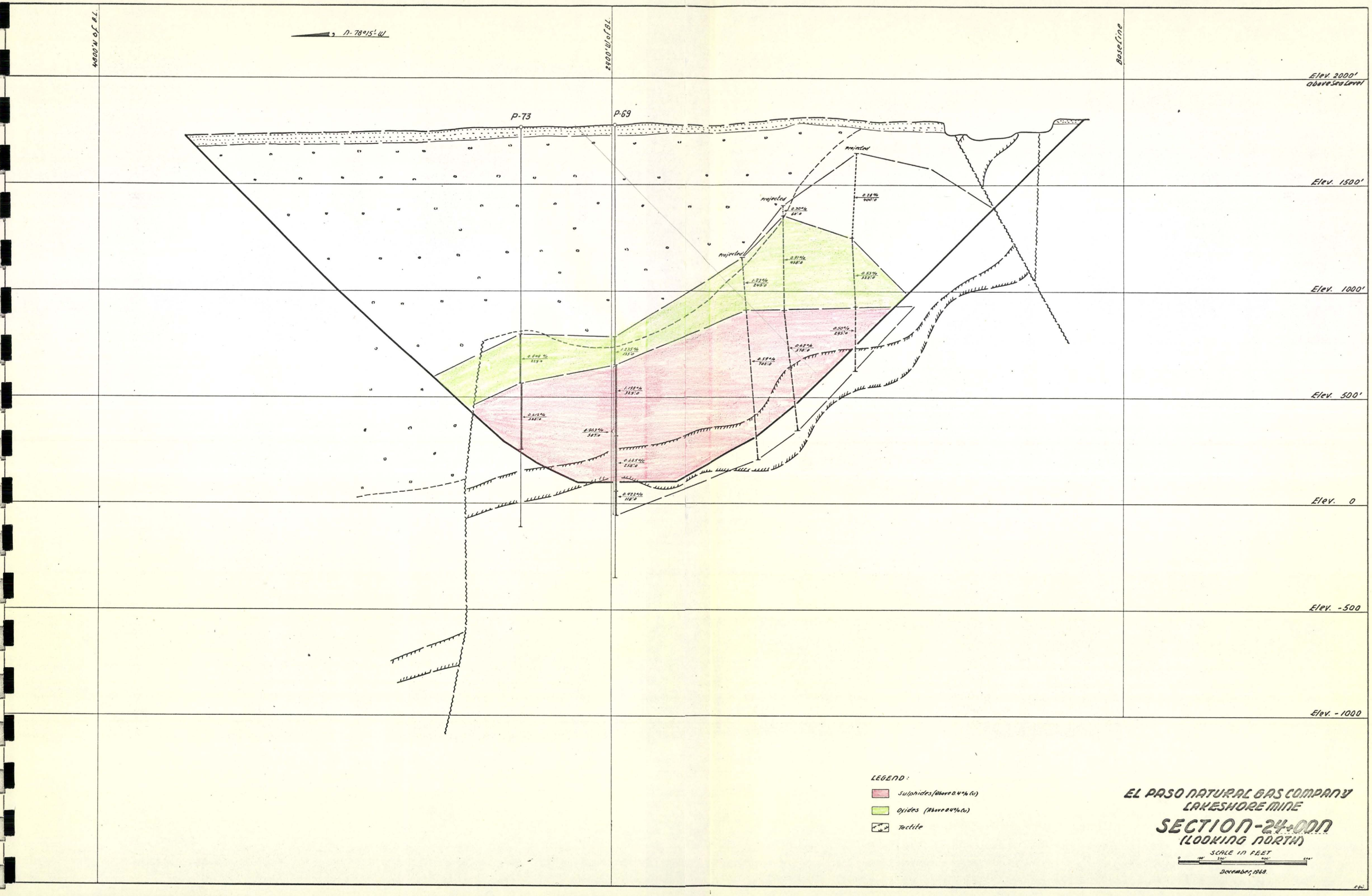
- Sulphides (Above 0.4% Cu)
- Oxides (Above 0.4% Cu)
- Faclite

EL PASO NATURAL GAS COMPANY
LAKESHORE MINE

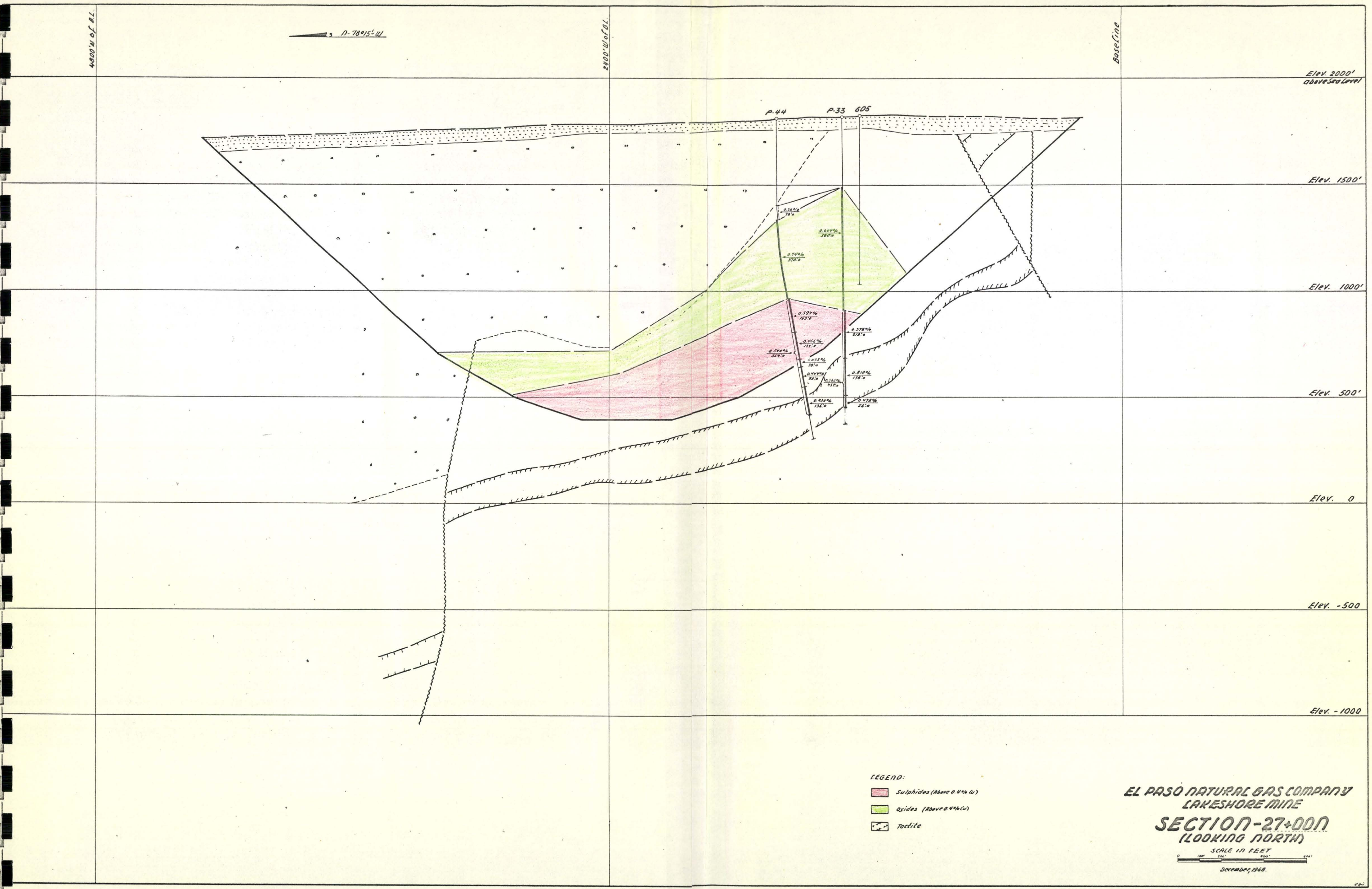
SECTION - 21+00N
(LOOKING NORTH)

SCALE IN FEET
0 100' 200' 400' 600'

December, 1968.



P-73
P-69
Projected
Baseline
Elev. 2000'
Above Sea Level
Elev. 1500'
Elev. 1000'
Elev. 500'
Elev. 0
Elev. -500
Elev. -1000
LEGEND:
Sulphides (Above 0.4% Cu)
Oxides (Above 0.4% Cu)
Tactite
EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION-24+00N
(LOOKING NORTH)
SCALE IN FEET
December, 1968.



N-78°15'-W
4800'W of B.L.
2400'W of B.L.
Baseline
Elev. 2000' above Sea Level
Elev. 1500'
Elev. 1000'
Elev. 500'
Elev. 0
Elev. -500
Elev. -1000
P-44
P-33
6D5
0.26%/70'
0.609%/580'
0.74%/370'
0.594%/163'
0.378%/218'
0.466%/122'
0.540%/554'
1.073%/39'
0.444%/95'
0.565%/457'
0.818%/178'
0.438%/136'
0.478%/56'
LEGEND:
Sulphides (Above 0.4% Cu)
Oxides (Above 0.4% Cu)
Tactite
EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
SECTION-27+00N
(LOOKING NORTH)
SCALE IN FEET
0 100' 200' 400' 600'
December, 1968.

UNDERGROUND MINING

The sulphide zone which is to be mined is a roughly tabular deposit of varying thickness and a generally westerly dip. On the east side, where the orebody is relatively thin, the base is at an elevation of 800' above sea level, or approximately 1,000 feet below surface. From here it slopes westward at an average grade of 1 in 2 to slightly below sea level where it is cut by a normal fault which displaces the ore downwards by about 700 feet. As the orebody approaches the fault, thicknesses tend to increase and one hole shows over 1,000 feet of ore-grade material. Approximate contours of the base of the orebody are shown on the accompanying plan.

Ore outlined with a fair degree of certainty by diamond drilling east of the fault, and capable of being mined, is estimated at 145,334,000 tons averaging 0.716% copper. This has an average vertical height of about 425 feet. Values are not evenly distributed throughout but tend to be concentrated in the tactite horizon which is about 100 feet in thickness and is normally situated at the base of the ore zone.

Three scattered diamond drill holes west of the fault show ore with an average grade of 1.078% copper. Allowing for ore in this area and for extensions north and south in areas where the ore zone has not been delimited by drilling, an additional 62,828,000 tons averaging 0.850% copper has been added to the reserve above to cover possible extensions.

Mining Method

Examination of diamond drill cores showed the ore zone, as well as the overlying rock, to be highly fractured. Some system of block caving is therefore probably the best means of extracting this ore; and the expected poor ground conditions make other methods difficult if not impossible.

Rock conditions appear similar to those at San Manuel where block caving is now being done successfully. The proposed test program will provide valuable information as to cavability. In the meantime it is assumed that the San Manuel method can be adapted to this deposit, and costs are based on their experience.

We estimate that the direct cost for stope development and undercutting below a 180' x 160' block of ore by the San Manuel method will be in excess of $363,000. This is independent of the height of the ore column to be caved. As the average height of the reserves is only 425', the stope development cost per ton will be comparatively high.

The sloping bottom of the orebody also poses a problem. Using the San Manuel system, blocks are undercut horizontally. A reasonable number of blocks must also be undercut at the same elevation so as to avoid having to open up too many levels. This means that the orebody will have to be mined in a series of steps, which will result in some low grade material being mined below the orebody and, in other cases, some ore being left below the undercut. As the better grade material tends to be at the base of the ore zone, this lost ore is liable to be of better than average grade.

It is proposed to mine the ore east of the fault in three steps, undercutting the orebody at elevations approximately 500' and 220' above and 40' below sea level as shown in the accompanying sections. This will require opening up haulage levels at +425', +145' and -115', and service and ventilation levels at +485', +205' and -55' elevation. One or more additional pairs of levels will be required for mining the down-faulted ore block to the west of the fault.

Three shafts would be required. These would be circular and concreted throughout. For a milling rate of 25,000 tons per day, the hoisting shaft would be equipped with twin skip hoists with a total daily capacity of 35,000 tons. For any increase in daily capacity an additional hoisting shaft would be needed. The service shaft and the hoisting shaft would be located south-east of the orebody, and probably could be sunk in granite. The ventilation shaft would be put down approximately 2,000 feet to the north. This would place it roughly opposite the north end of the presently known orebody and would facilitate ventilation from the service and hoisting shafts through the ore zone. All shafts would be sited east of the 45° plane carried up to the surface from the east end of the orebody so as not to be affected by subsidence following caving.

Proposed layouts for the levels required to mine the first two level blocks are shown in the accompanying plans. The haulage level in each case is doubled with frequent cross overs so as to provide separate headings for outcoming and ingoing traffic. The disposition of the ore on several levels makes it impractical to place these two headings on different sides of the orebody.

Stoping blocks would be 180' long by 140' wide with 20' pillars between blocks. The haulage crosscuts and finger raises would be developed from the haulage level while the grizzly crosscuts and draw raises would be developed from the service level.

Stoping could start on the first and second level blocks simultaneously and would proceed from the centre of the west side, in each case, advancing to the north-east and south-east. With this sequence, by the time mining on the second level had reached the east boundary of the level block, mining in the adjacent area on the level above would be finished and no interference would result. On the second level, where the layout shows the west side of the block close to the fault zone, mining from this level should probably be carried over to the fault. The third level in this area would thus mine only the ore below the second. This would increase development costs but might reduce dilution from the fault zone.

The problem of sideways draw into the caving areas puts a serious limit on mining sequence. It is assumed that no caving can be started until all work within a plane drawn upward at 45° from the caving area has been finished. This restraint is particularly severe on the down-faulted ore to the west of the fault, and means that very little of the ore so far assumed in this area can be mined until mining on the levels above is well advanced. As drilling indicates that this is better than average grade ore, it has the serious economic effect of postponing the mining of some of the most potentially profitable material until late in the life of the operation. If further drilling to the west showed that this block extended westward and was of similar grade, mining of some of the higher grade ore might be started sooner with considerable improvement to the cash flow.

Ore Reserves

The ore reserve east of the fault was calculated by plotting the diamond drill intersections on plan and assigning an area of influence to each, as shown on the accompanying plan. As the development cost for block caving is a constant for each block regardless of height, there is a limiting height, dependent on grade, below which there is insufficient ore to pay for the development. The reserve was therefore segregated into two categories: that over 200' in thickness and that between 100 and 200 feet thick. No section less than 100 feet thick was considered. Using a cut off grade of 0.5% copper, the average uncorrected grade of all the reserve was found to be 0.85%, while that of the blocks between 100' and 200' thick was 1.19%. The latter is sufficient to cover the increased unit development cost and therefore this tonnage can be included in the reserve.

In addition to the reserve as calculated above, a probable reserve was calculated adjacent to three holes drilled just west of the fault, and possible reserves extending an additional 300 feet were added to the north, south and west where the boundaries of the ore zone are not yet delimited.

To arrive at an estimate of the ore expected to be delivered to the mill, the crude reserves calculated from the drill hole averages were adjusted as follows:

1) Assays were reduced by 4%. This correction was indicated by the check assays that were made;

2) A correction for ore lost and low grade material added by undercutting on a hoizontal plane;

3) A grade correction due to an assumed 10% loss of ore not recovered and an addition of 10% low grade dilution.

Estimated reserves are summarized as follows:

	Tons (x 1000)	Copper %	Copper Tons
Indicated by drilling	145,334	0.716	1,039,799
Probable	21,706	1.078	234,088
	167,040	0.763	1,273,887
Possible additional	41,122	0.730	300,267
	208,162	0.756	1,574,154

Details of the blocks included are shown in the appendix.

Economics

Two production rates were considered:

1) 25,000 tons per milling day;

2) 40,000 tons per milling day.

The disposition of the ore as presently outlined and assumed makes it necessary to mine most of the ore east of the fault before mining west of the fault can be started. For the purposes of evaluation it is assumed that all the reasonably proved ore east of the fault is mined first, followed by the probable and possible reserves. Since these latter reserves are estimated to be of better than average grade, earnings show an increase when mining reaches this stage. A further increase in earnings occurs in the last two years, as it is assumed that there will be no development charges, these being covered by the pre-production development cost.

Operating costs at the two production rates are estimated as follows:

	At 25,000 TPD	At 40,000 TPD
Development	$0.465	$0.465
Mining Direct	0.850	0.850
Milling Direct	0.850	0.800
Overhead	0.484	0.410
	$2,649	$2.525

CAPITAL EXPENDITURE - 25,000 T/D

a) Preliminary Test Program

Surface Core Drilling: 100,000' @ 7.00/ft.		$ 700,000
Underground Development:		
Shaft - 1250' @ $500/ft.	$625,000	
Drifting (Oxides) - 1000' @ $80/ft.	80,000	
Drifting (Sulphides) - 2000' @ $80/ft.	160,000	
Core Drilling - 20,000' @ $5/ft.	100,000	965,000
Metallurgical Test Work:		500,000
		$2,165,000
Contingencies - 10%		217,000
Total Preliminary Test Program		$2,382,000

b) Mine Development

1st Level:	
Main Drives - 11,250' @ $110/ft.	$1,195,000
X-Cuts to Stoping Areas - 7,550'	810,000
Stope Development (9 stopes - 4260'/stope)	3,270,000
Slashing for Double Tracking, Shops etc.	100,000
	$5,375,000
2nd Level:	
Main Drives - 16,560' @ $110/ft.	$1,760,000
X-Cuts to Stoping Areas - 6000' @ $107	640,000
Stope Development (4 stopes - 4260'/stope)	1,450,000
Slashing -	100,000
	$3,950,000
Carried forward	$9,325,000

	Brought forward	$9,325,000
Shaft Sinking:		
Hoisting Shaft - 2,780' @ $1,100/ft.		$3,058,000
Service " - 2,500' @ $ 950/ft.		2,423,000
Ventilation " - 2,550' @ $ 800/ft.		2,040,000
		$7,521,000
		$16,846,000
Contingencies - 10%		1,685,000
Total Mine Development		$18,531,000
Total Testing and Pre-production Development		$20,913,000

c) Plant

Mine: Headframes, bins, hoists, etc.		$5,615,000
Mine equipment		3,500,000
Compressors		800,000
		$9,915,000
Mill - 25,000 T/D @ $1,400/T.D. capacity		35,000,000
Shops		1,320,000
Offices and Housing		6,000,000
Water Supply		1,000,000
Railroad - 30 miles to Casa Grande		2,000,000
Total		$55,235,000
Contingencies @ 10%		5,524,000
Total Plant Without Smelter		$60,759,000

Summary - Without Smelter:		
Pre-production Testing and Development		$20,913,000
Plant		60,759,000
Working Capital		8,000,000
		$89,672,000
Additional for Smelter		
Smelter (55,000 tons Cu/Yr.)	$24,200,000	
Additional Working Capital	2,000,000	26,200,000
Total With Smelter		$115,872,000

SCHEDULE OF CAPITAL EXPENDITURES - 25,000 Tpd

($ x 1,000)

	TOTAL	YEARS 1	2	3	4	5	6
Test Period							
Surface drilling	770	770					
Shaft sinking	688	688					
Cross-cutting	264	-	264				
U.G. drilling	110	-	110				
Metallurgical tests	550	-	550				
	2,382						
Shaft Sinking							
Service shaft	2,665			2,665			
Hoisting shaft	3,364			3,364			
Ventilation shaft	2,244	-	-	-	2,244		
	8,273						
Development							
Lateral & Stope	10,258	-	-	-	3,410	6,848	
Mine Plant							
Shaft Equipment	6,177			2,000	3,177	1,000	
Compressors	880			880			
Mine Equipment	3,850				1,000	2,850	
	10,907						
Mill	38,500			500	9,500	28,500	
Surface Plant							
Shops	1,452			452	1,000	-	
Office & Housing	6,600				2,000	4,600	
Water Supply	1,100			1,100	-	-	
	9,152						
Railway	2,200			1,100	1,100		
TOTAL	81,672	1,458	924	12,061	23,431	43,798	
Working Capital	8,000						8,000
Total without smelter	89,672	1,458	924	12,061	23,431	43,798	8,000
Smelter	24,200			500	6,000	17,700	
Add working capital	2,000						2,000
Total with smelter	115,872	1,458	924	12,561	29,431	61,498	10,000

CAPITAL EXPENDITURE - 40,000 T/D

Preliminary Testing (with 10% contingencies)			$2,382,000
Mine Development:			
1st level	$5,375,000		
2nd level	6,091,000	$11,466,000	
Shafts			
Hoisting shafts (2)	$6,116,000		
Service shaft	2,423,000		
Ventilation shaft	2,040,000	10,579,000	
		$22,045,000	
Contingencies @ 10%		2,205,000	24,250,000
Mine Plant			
Headframes, bins, hoists, etc.		$ 7,615,000	
Mine Equipment		4,800,000	
Compressors		1,000,000	
		$13,415,000	
Contingencies @ 10%		1,342,000	14,757,000
Mill (including contingencies)			42,828,000
Shops			1,500,000
Office and buildings			8,000,000
Water supply			1,600,000
Railway			2,200,000
			$97,517,000
Working Capital			12,000,000
Total Without Smelter			$109,517,000
Additional for Smelter		$26,600,000	
Additional Working Capital		4,000,000	30,600,000
Total With Smelter			$140,117,000

SCHEDULE OF CAPITAL EXPENDITURES - 40,000 Tpd.

($ x 1,000)

ITEM	TOTAL	YEARS 1	2	3	4	5	6	7	8	9
Test Period	2,382	1,458	924							
Shaft Sinking	11,637	-	-	6,029	2,244			1,682	1,682	
Development	12,613	-	-	-	3,410	6,848	800	800	755	
Mine Plant	14,757	-	-	2,880	4,177	3,850	-	1,850	2,000	
Mill	42,828	-	-	500	9,500	30,000	-	-	2,828	
Surface Plant	1,500	-	-	500	1,000	-	-	-	-	
Office & Heating	8,000	-	-	-	2,000	5,000	-	1,000	-	
Water Supply	1,600	-	-	1,600	-	-	-	-	-	
Railway	2,200	-	-	1,100	1,100	-	-	-	-	
	97,517	1,458	924	12,609	23,431	45,698	800	5,332	7,265	
Working Capital	12,000						3,500	4,500		4,000
TOTAL	109,517	1,458	924	12,609	23,431	45,698	4,300	9,832	7,265	4,000
Add:-										
Smelter	26,600			500	6,000	17,700			2,400	
Working Capital	4,000						1,000	1,500		1,500
TOTAL With Smelter	140,117	1,458	924	13,109	29,431	63,398	5,300	11,332	9665	5,500

Mining Schedules

a) For 25,000 TPD rate:

Years	TPD	Grade
1	10,000	0.716% Cu
2-16	25,000	0.716% Cu
17-24	25,000	0.850% Cu

b) For 40,000 TPD rate:

Years	TPD	Grade
1	10,000	0.716% Cu
2-3	25,000	0.716% Cu
4-12	40,000	0.716% Cu
13-17	40,000	0.850% Cu

(No development in last two years)

Operating Profit - 25,000 TPD (Copper @ 42¢/lb.)

a) Years 1-16

Head grade = 0.716% Cu — 14.32 lbs./ton
Recovered 90% — 12.89 lbs./ton

Concentrate grade — Cu 28.5%; Au 0.11 ox./ton; Ag 1.70 ox./ton

Ratio of concentration = 44.2:1

Value of metals recovered per ton of conc.

Cu 570 lbs. @ $0.42	$239.40	
Au (0.90x 0.11) @ $35.00	3.47	
Ag (1.70-0.50) @ $2.00	2.40	
	$245.27	
Treatment charges (@ 7.0¢/lb. Cu)[1]	39.90 (@ 8.5¢/lb)[1]	$ 48.45
Net Smelter Return:-per ton of conc.	$205.37	$196.82
-per ton of ore	$ 4.64	$ 4.45
Operating cost per ton[2]	2.65	2.65
Operating profit per ton[2]	$ 1.99	$ 1.80

b) Years 17-22

Head grade	= 0.850% Cu	17.00 lbs./ton	
Recovered @ 90%		15.30 lbs./ton	
Concentrate grade 28.5%			
Ratio of concentration = 37.3:1			
Net smelter return: per ton of conc.		$205.37	$196.82
per ton of ore		$ 5.51	$ 5.28
Operating cost per ton(2)		$ 2.65	$ 2.65
Operating profit per ton(2)		$ 2.86	$ 2.63

(c) Years 23-24

Head grade	= 0.850% Cu		
Net smelter return per ton		$ 5.51	$ 5.28
Operating cost per ton(2)		2.18	2.18
Operating profit per ton(2)		$ 3.33	$ 3.10

Notes

(1) Treatment charges (smelting, refining and marketing):
7.0¢/lb. copper recovered with own smelter
8.5¢/lb. copper recovered with custom smelter

(2) Operating costs and profits before payment of Indian royalties

Operating Profit - 40,000 TPD (Copper @ 42¢/lb.)

	Custom Smelter	Own Smelter
a) Years 1-3		
Net smelter return per ton of ore	$4.45	$4.64
Operating cost per ton	2.65	2.65
Operating profit per ton	$1.80	$1.99
b) Years 4-12		
Net smelter return per ton of ore	$4.45	$4.64
Operating cost per ton	2.53	2.53
Operating profit per ton	$1.92	$2.11
c) Years 13-15		
Net smelter return per ton of ore	$5.28	$5.51
Operating cost per ton	2.53	2.53
Operating profit per ton	$2.75	$2.98
d) Years 16-17		
Net smelter return per ton	$5.28	$5.51
Operating cost per ton	2.06	2.06
Operating profit per ton	$3.22	$3.45

Note - Operating costs and profits before payment of Indian royalties.

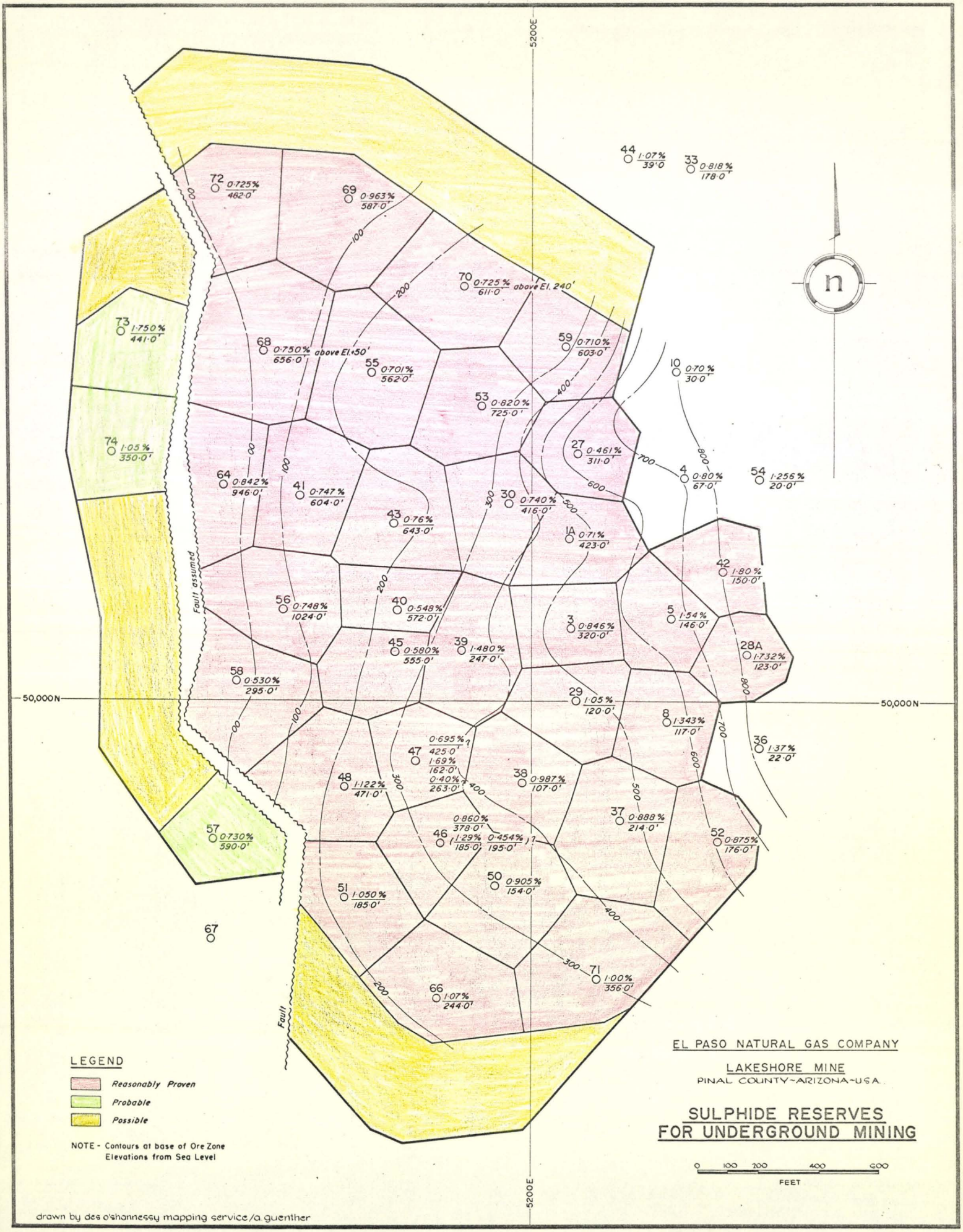
SUMMARY CAPITAL EXPENDITURES - UNDERGROUND MINE

	For 25,000 TPD	For 40,000 TPD
Test program	$ 2,382,000	$ 2,382,000
Underground development	18,531,000	24,250,000
	$20,913,000	$26,632,000
Plant	60,759,000	70,885,000
Working Capital	8,000,000	12,000,000
Total Without Smelter	$89,672,000	$109,517,000
Smelter	24,200,000	26,600,000
Extra working capital	2,000,000	4,000,000
Total With Smelter	$115,872,000	$140,117,000

SUMMARY OPERATING PROFITS (Copper @ 42¢)

At 25,000 TPD			At 40,000 TPD		
Years	Custom Smelter	Own Smelter	Years	Custom Smelter	Own Smelter
1-16	$1.80	$1.99	1-3	$1.80	$1.99
17-22	$2.63	$2.86	4-12	$1.92	$2.11
23-24	$3.10	$3.33	13-15	$2.75	$2.98
			16-17	$3.22	$3.45

Note - Operating Profits before Indian Royalty.



LEGEND

- Reasonably Proven
- Probable
- Possible

NOTE - Contours at base of Ore Zone
Elevations from Sea Level

EL PASO NATURAL GAS COMPANY

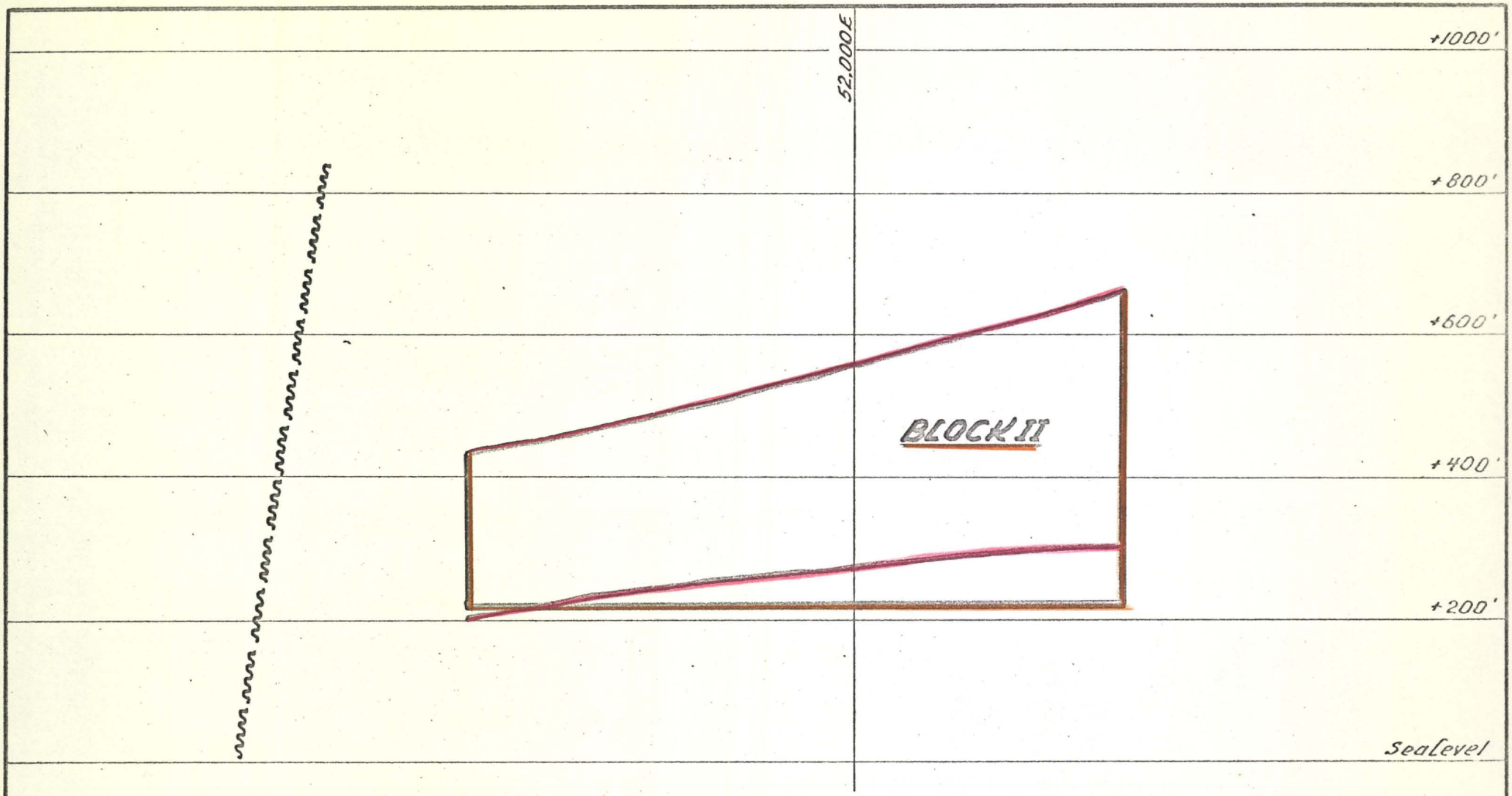
LAKESHORE MINE

PINAL COUNTY-ARIZONA-U.S.A.

SULPHIDE RESERVES FOR UNDERGROUND MINING

0 100 200 400 600
FEET

drawn by des o'shannessy mapping service/a guenther

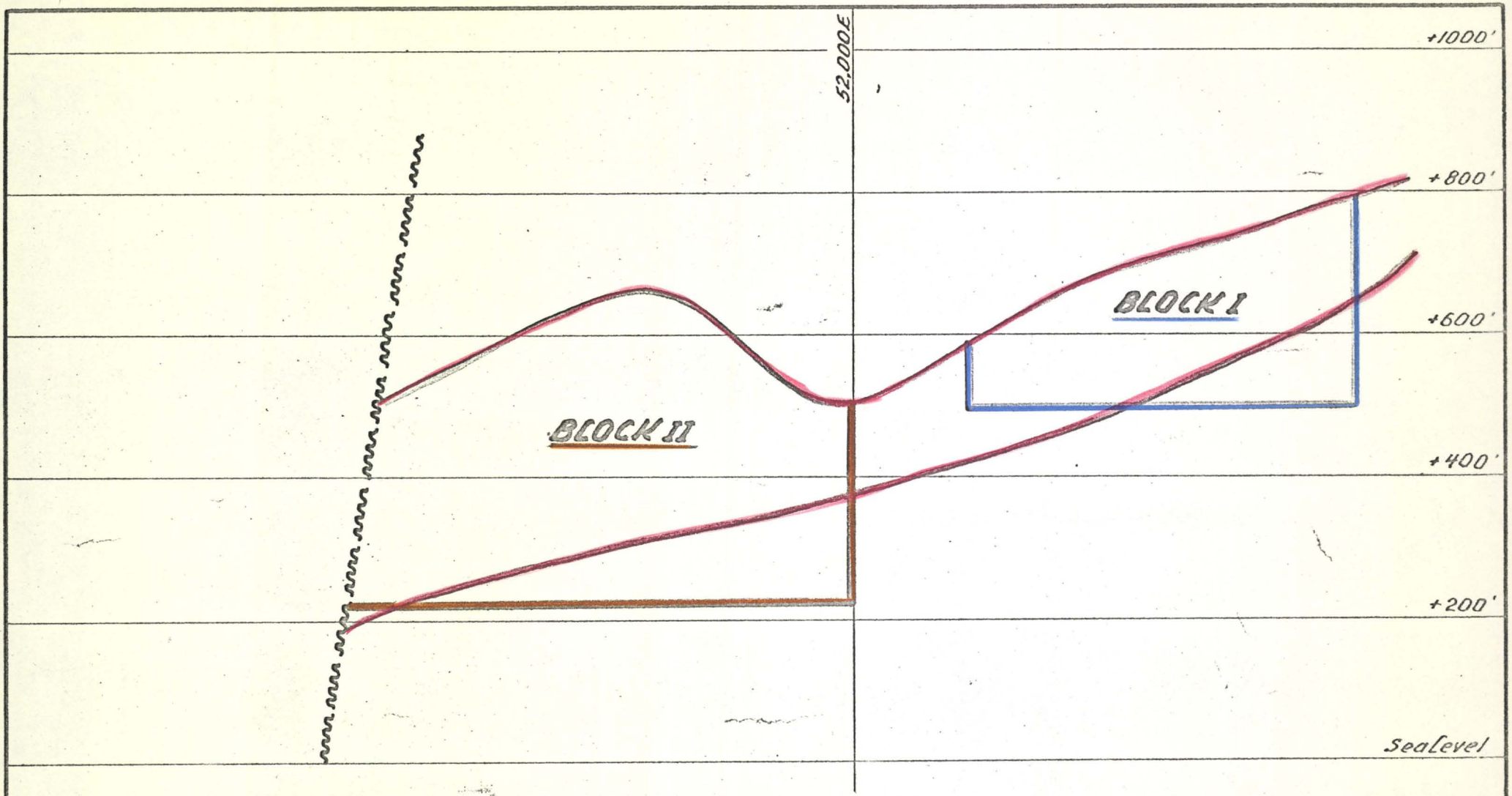


EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
PINAL COUNTY-ARIZONA

PRELIMINARY MINE LAYOUT
SECTION-49,000 N

SCALE: 1" = 200' DATE: Dec., 1968.

R.G.

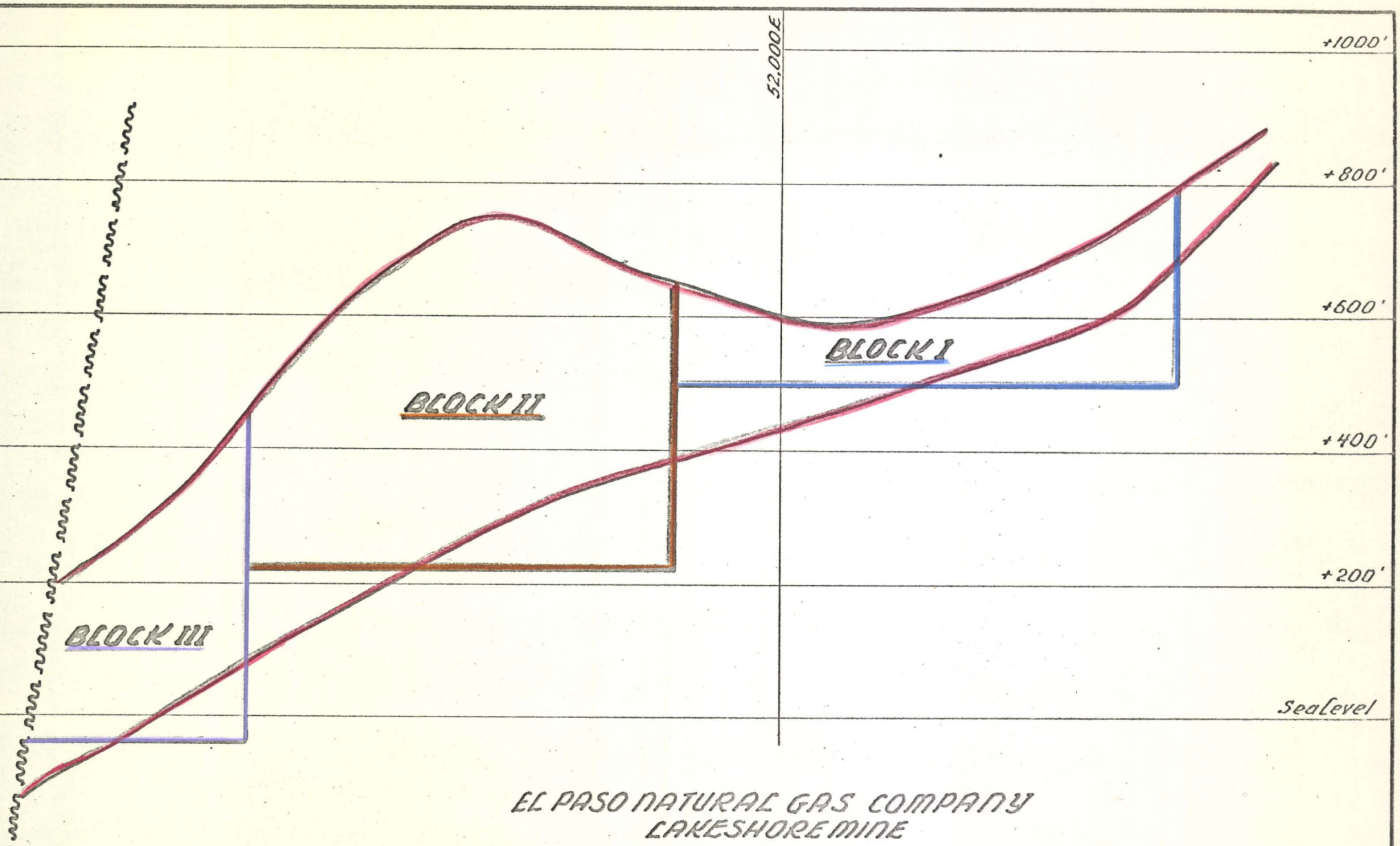


EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
PINAL COUNTY-ARIZONA

PRELIMINARY MINE LAYOUT
SECTION-49,500N

SCALE: 1" = 200' DATE: Dec., 1968.

R.G.W.

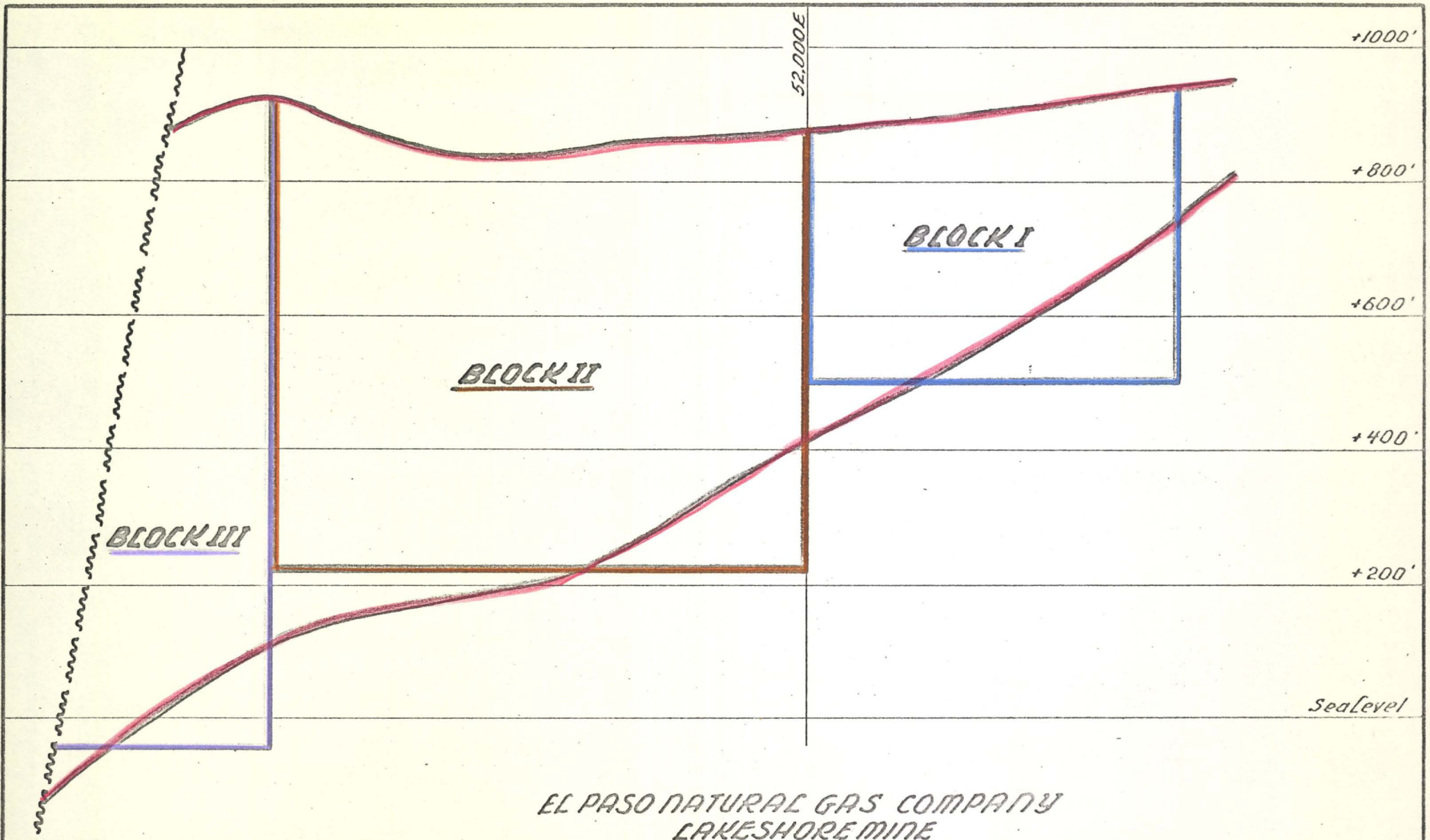


EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
PINAL COUNTY-ARIZONA

PRELIMINARY MINE LAYOUT
SECTION-50,000 N

SCALE: 1" = 200' DATE: Dec, 1968.

R.G.H.

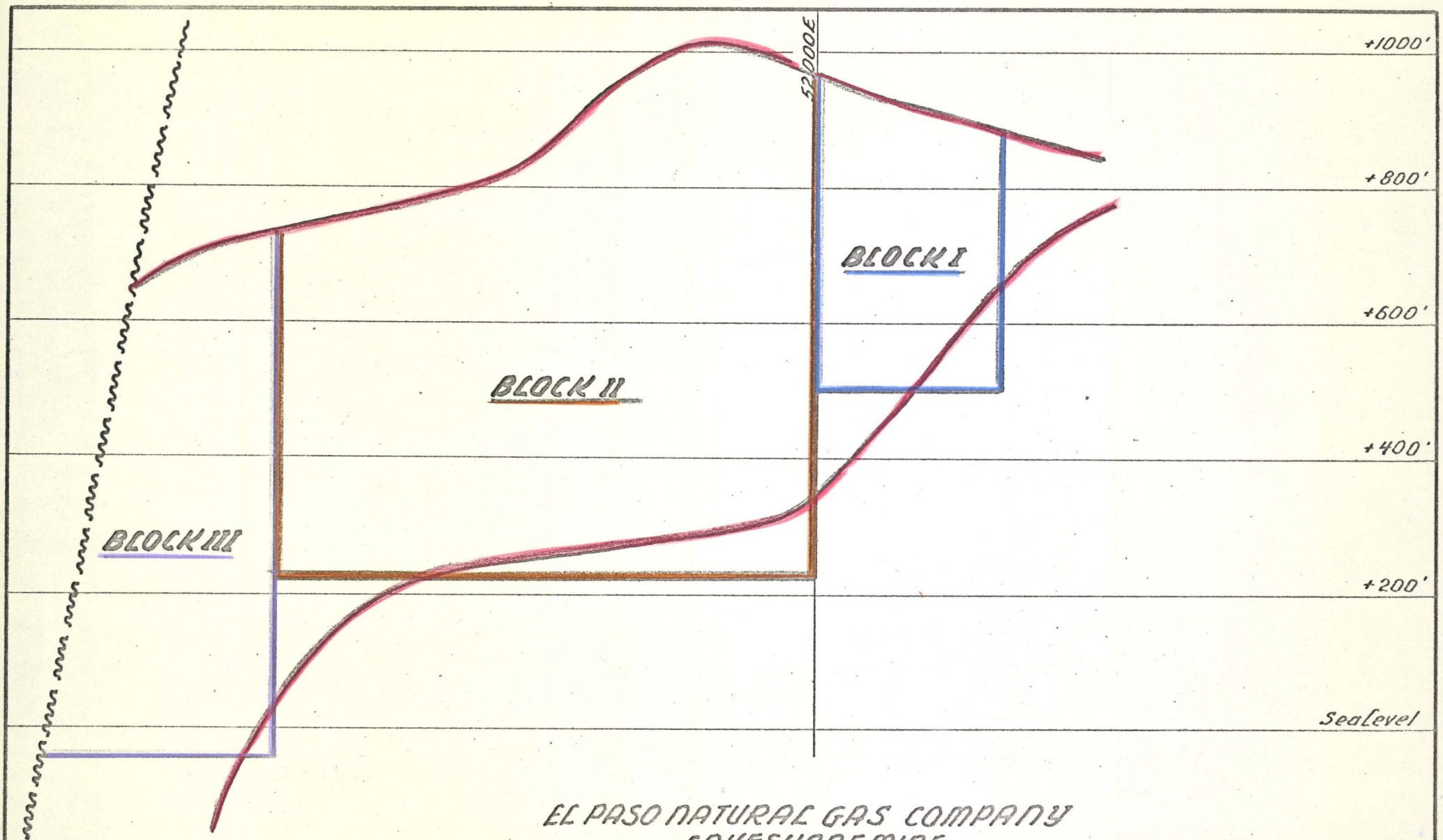


EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
PINAL COUNTY-ARIZONA

PRELIMINARY MINE LAYOUT
SECTION-50,500 N

SCALE: 1" = 200' DATE: Dec., 1968.

R.G.

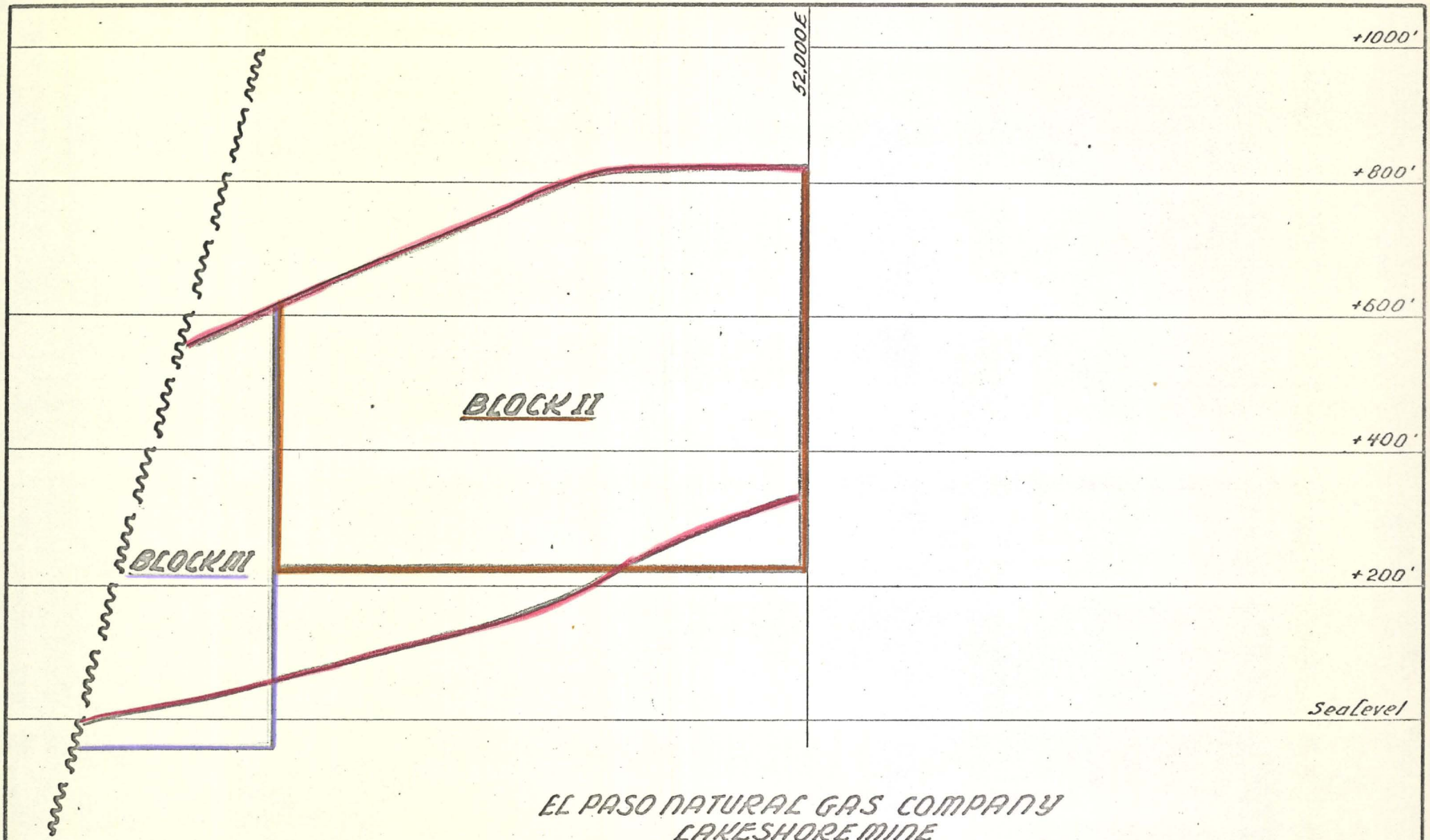


EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
PINAL COUNTY-ARIZONA

PRELIMINARY MINE LAYOUT
SECTION-51,000N

SCALE: 1" = 200' DATE: Dec., 1968.

RGH.

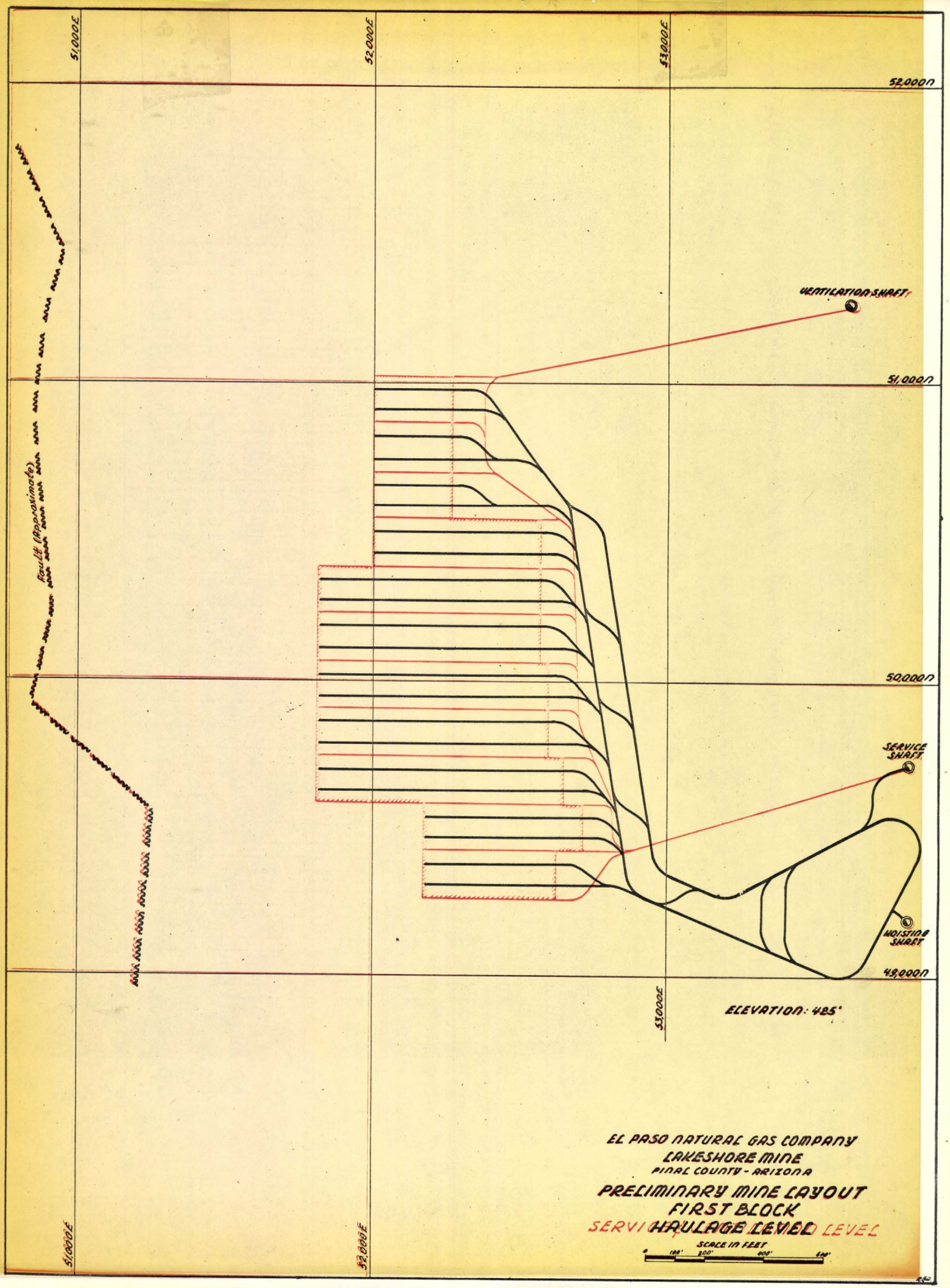


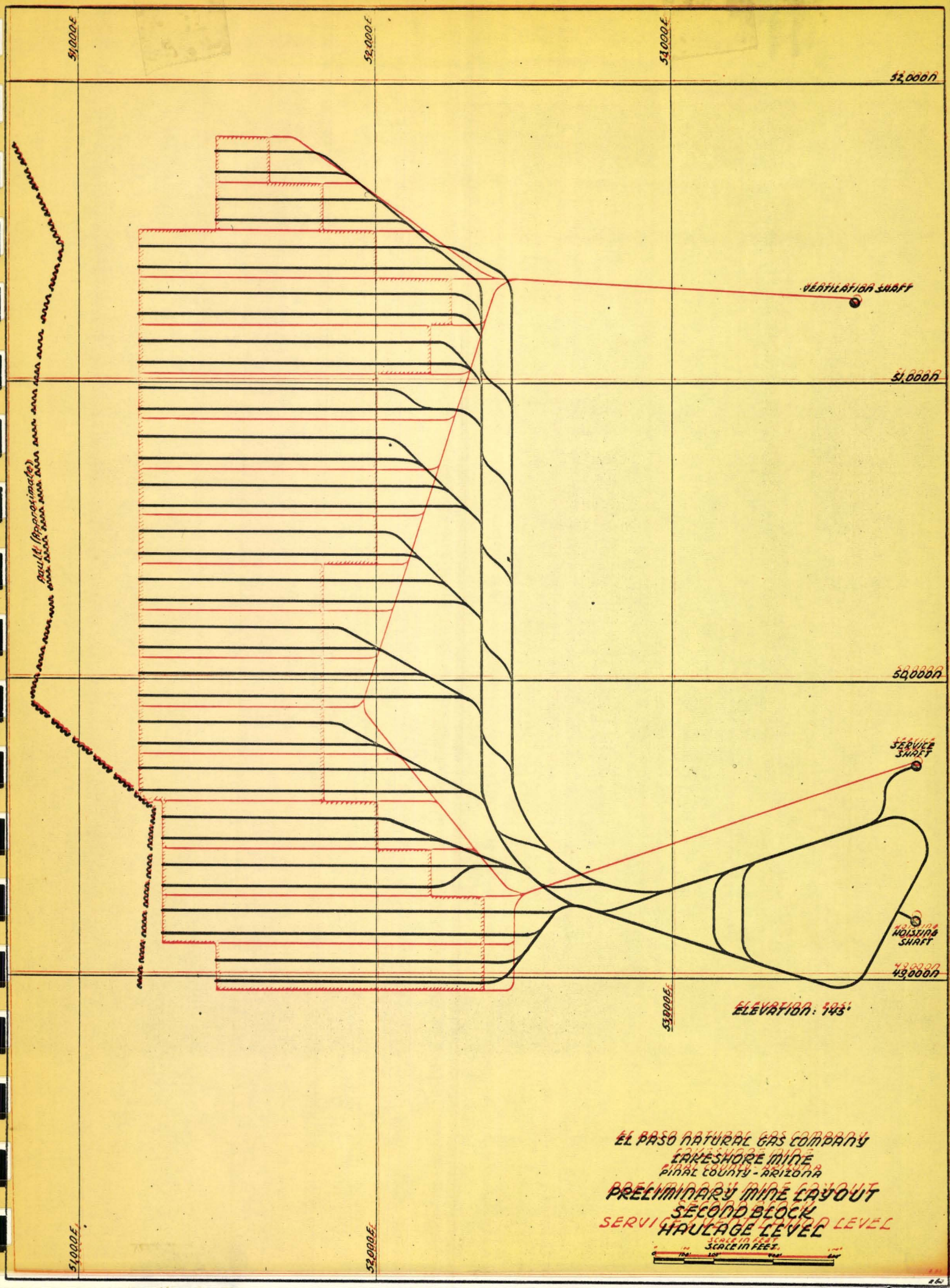
EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
PINAL COUNTY-ARIZONA

PRELIMINARY MINE LAYOUT
SECTION-51,500N

SCALE: 1" = 200' DATE: Dec., 1968.

R.G.W.


51,000E
52,000E
53,000E
52,000N
51,000N
50,000N
49,000N
VENTILATION SHAFT
SERVICE SHAFT
HOISTING SHAFT
Fault (Approximate)
ELEVATION: 485'
SCALE IN FEET
0
100'
300'
600'
EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
PINAL COUNTY - ARIZONA
PRELIMINARY MINE LAYOUT
FIRST BLOCK
SERVICE & HAULAGE LEVEL



EL PASO NATURAL GAS COMPANY

LAKESHORE MINE

PINAL COUNTY - ARIZONA

PRELIMINARY MINE LAYOUT

SECOND BLOCK

HAULAGE LEVEL

SCALE IN FEET

0 100' 200' 400' 600'

LAKESHORE - VEKOL JOINT OPERATION

The Vekol property, situated about 10 miles west of Lakeshore on the opposite side of the Santa Rosa Valley, is controlled jointly by the Superior Oil and Newmont companies each of which now holds approximately 38.2% of the stock. A sulphide deposit containing 74.7 million tons averaging 0.61% copper after allowing for dilution has been outlined by diamond drilling, but no further work has been done on the property. This deposit could be mined by open pit. Preliminary stripping would amount to 60 million tons and the subsequent waste ore ratio would be about 2:1.

An evaluation by Newmont in October 1967 assuming a 38¢/lb. price for copper concluded that higher prices for copper were needed to provide a suitable return on this investment. Newmont now suggest that by combining the Vekol and Lakeshore operations significant savings could be made to the benefit of both. We have therefore attempted to assess the economics of this situation. As we do not have first-hand information on this property and due to lack of time the following evaluation is based on data supplied by the Tucson office of Superior Oil.

The proposed combined operation would be for a production of 15,000 tons per day from the Vekol open pit and 25,000 tons per day from the Lakeshore underground mine. Production could however start at Vekol two years before Lakeshore reached full production and a rate of 25,000 tons per day would be planned for this period. On the basis of present ore reserves Lakeshore would outlast Vekol and after Vekol had ceased production, the rate at Lakeshore would be increased to 40,000 tons per day to make use of the mill capacity. A second hoisting shaft would be required at this time. Operating data would be as follows:

Mining Schedule

Years	Vekol	Lakeshore
1-4	(Pre-production)	
5	25,000 TPD @ 0.61%	-
6	25,000 TPD @ 0.61%	10,000 TPD @ 0.716%
7-17	15,000 TPD @ 0.61%	25,000 TPD @ 0.716%
18-20	-	40,000 TPD @ 0.716%
21-25		40,000 TPD @ 0.850%

Vekol Operating Costs
(Based on costs estimated by D. J. Pope in report to Newmont Apr.25/-68)

	Years 5-6	Years 7-17
Mining ore	$0.288/ton	$0.288/ton
Stripping waste	0.519/ton ore	0.519/ton ore
	$0.81/ton	$0.81/ton
Milling	0.85/ton	0.80/ton
Overheads	0.50/ton	0.50/ton
	$2.16/ton	$2.11/ton

Treatment Charges

Custom smelter @ 8.5¢/lb.	=	$0.93/ton
Own smelter @ 7.0¢/lb.	=	$0.77/ton

Total Costs

	Years 5-6	Years 7-17
With Custom Smelter	$3.09	$3.04
With Own Smelter	$2.93	$2.88

Vekol Operating Profits

Head grade	=	0.61% Cu	0.015% Mo
Recoveries	=	90% Cu	60% Mo

Gross metal value:	Per ton
Cu - 10.98 lbs. @ .42¢	$4.61
Mo	$0.18
P.M.s	$0.13
	$4.92

	Custom Smelter	Own Smelter
Years 5-6		
Operating costs	$3.09	$2.93
Operating Profit before royalty	$1.83	$1.99
Indian Royalty @ 5% of $3.98	0.20	0.20
Profit:	$1.63	$1.79
Years 7-17		
Operating Costs	$3.04	$2.88
Operating Profit before royalty	$1.88	$2.04
Indian Royalty	0.20	0.20
	$1.68	$1.84

Lakeshore Operating Costs Per Ton (with joint operation)

	Year 6	Years 7-17	Years 18-24	Year 25
Development	$0.465	$0.465	$0.465	Nil
Mining	0.850	0.850	0.850	$0.850
Milling	0.850	0.800	0.800	0.800
Overheads	0.484	0.484	0.410	0.410
	$2.649	$2.599	$2.525	$2.060

Lakeshore Operating Profits (with joint operation)

	Custom Smelter	Own Smelter
a) Year 6		
Net smelter return per ton ore	$4.45	$4.64
Operating cost per ton	2.65	2.65
Operating profit per ton	$1.80	$1.99
b) Years 7-17		
Net smelter return per ton ore	$4.45	$4.64
Operating cost per ton	2.60	2.60
Operating profit per ton	$1.85	$2.04
c) Years 18-20		
Net smelter return per ton ore	$4.45	$4.64
Operating cost per ton	2.53	2.53
Operating profit per ton	$1.92	$2.11
d) Years 21-24		
Net smelter return per ton ore	$5.28	$5.51
Operating cost per ton	2.53	2.53
Operating profit per ton	$2.75	$2.98
e) Year 25		
Net smelter return per ton ore	$5.28	$5.51
Operating cost per ton	2.06	2.06
Operating profit per ton	$3.22	$3.45

Note: All costs and profits before royalty payments.

Summary

Years	VEKOL Tons/ Day	VEKOL Heads % Cu	VEKOL Op. Profit[1] per ton	LAKESHORE Tons/ Day	LAKESHORE Heads % Cu	LAKESHORE Op. Profit[1] per ton
1) With Custom Smelter:						
1-4	-	-	-	—	—	—
5	25,000	0.61%	$1.83	—	—	—
6	25,000	0.61%	$1.83	10,000	0.716%	$1.80
7-17	15,000	0.61%	$1.88	25,000	0.716%	$1.85
18-20	—	—	—	40,000	0.716%	$1.92
21-24	—	—	—	40,000	0.850%	$2.75
25	—	—	—	40,000	0.850%	$3.22
2) With Own Smelter						
1-4	—	—	—	—	—	—
5	25,000	0.61%	$1.99	—	—	—
6	25,000	0.61%	$1.99	10,000	0.716%	$1.99
7-17	15,000	0.61%	$2.04	25,000	0.716%	$2.04
18-20	—	—	—	40,000	0.716%	$2.11
21-24	—	—	—	40,000	0.850%	$2.98
25	—	—	—	40,000	0.850%	$3.45

Note: (1) All operating profits before royalty payments.

CAPITAL EXPENDITURES - LAKESHORE - VEKOL JOINT OPERATION

($ x 1,000)

Item	Total	Distribution	
		Lakeshore	Vekol
Preliminary testing	2,882	2,382	500
Mine development	18,531	18,531	-
Stripping	9,858	-	9,858
Mine Equipment	19,107	10,907	8,200
Mill	42,828	31,521	11,307
Shops	3,452	1,452	2,000
Office & Buildings	7,600	5,590	2,010
Water Supply	1,600	1,180	420
Railway	2,900	1,600	1,300
Total without Smelter	108,758	73,163	35,595
Smelter	26,600	19,600	7,000
Total with Smelter	135,358	92,763	42,595

Added capital required to bring Lakeshore to 40,000 tpd. rate at start of year 18 is:

Hoisting shaft	=	3,364
Mine equipment	=	3,850
Total	=	7,214 (3,000 in yr. 16; 4,214 in year 17)

NOTE: Working capital is not included above, but is shown in the financial analysis.

SCHEDULE OF CAPITAL EXPENDITURE - JOINT OPERATION

Item	Total	YEARS 1	2	3	4	5
Lakeshore						
Preliminary testing	2,382	1,458	924	-	-	-
Mine development	18,531	-	-	6,029	5,654	6,848
Mine equipment	10,907	-	-	2,880	4,177	3,850
Mill	31,521	-	-	7,300	24,221	-
Shops	1,452	-	-	452	1,000	-
Office & Buildings	5,590	-	-	-	2,000	3,590
Water Supply	1,180	-	-	1,180	-	-
Railway	1,600	-	-	1,600	-	-
	73,163	1,458	924	19,441	37,052	14,288
Smelter	19,600	-	-	4,800	14,800	-
	92,763	1,458	924	24,241	51,852	14,288
VEKOL						
Preliminary testing	500	200	300	-	-	-
Stripping	9,858	-	-	5,000	4,858	-
Mine Equipment	8,200	-	-	-	8,200	-
Mill	11,307	-	-	2,700	8,607	-
Shops	2,000	-	-	2,000	-	-
Office & Buildings	2,010	-	-	-	2,010	-
Water Supply	420	-	-	420	-	-
Railway	1,300	-	-	1,300	-	-
	35,595	200	300	11,420	23,675	-
Smelter	7,000	-	-	1,700	5,300	-
	42,595	200	300	13,120	28,975	-

ECONOMICS

Results

The following results were obtained from a financial analysis of El Paso's Lakeshore Mine property to determine if a large scale operation would be viable. All rates of return and discounted cash flows are calculated from the day production begins.

Situation	Capital Required ($ millions) Pre-production	Maximum	ROI	Payback years at 7% debt	Mine life Years	DCF @ 10% after tax after debt
a) Custom Smelter						
Lakeshore alone						
@ 25,000 TPD	$88.8	$91.8	9.0%	16.1	24	$18.1
@ 40,000 TPD	$91.3	$94.6	11.6%	10.9	17	37.8
Lakeshore and Vekol						
Lakeshore portion	$62.8	$85.1	10.7%	13.8	20	26.9
Vekol portion	$37.8	$44.1	16.6%	6.3	13	21.1
Total @ 40,000 TPD	$100.6	N.A.	12.0%	N.A.	21	48.0
b) With Smelter						
Lakeshore alone						
@ 25,000 TPD	$114.3	$118.4	7.7%	18.5	24	14.6
@ 40,000 TPD	$116.9	$121.4	9.9%	12.6	17	32.3
Lakeshore and Vekol						
Lakeshore portion	$83.5	$108.3	9.4%	15.3	20	24.9
Vekol portion	$45.1	$53.7	11.4%	7.1	13	20.1
Total @ 40,000 TPD	$128.6	N.A.	10.4%	N.A.	21	45.0

The following results on the more favourable situations were obtained from an analysis to indicate the return on equity that could be earned on a low risk, low return project by financing. This assumes a debt-equity ratio of approximately 3:1.

Situation	Equity financing Req. ($ millions)	ROE	Total cash return to Investor ($ millions)	Mine life
Custom Smelting				
Lakeshore @ 25,000 TPD	$24.5	12.1%	$169.7	24
Lakeshore @ 40,000 TPD	25.2	17.1%	193.9	17
*Lakeshore and Vekol				
Lakeshore portion	21.6	15.6%	189.5	20
Vekol portion	9.9	26.2%	69.5	13
Total	27.4	17.9%	255.0	21
With Smelter				
Lakeshore @ 25,000 TPD	30.9	9.7%	157.1	24
Lakeshore @ 40,000 TPD	31.8	14.4%	194.3	17
*Lakeshore and Vekol				
Lakeshore portion	28.0	13.2%	188.5	20
Vekol portion	11.1	23.3%	69.0	13
Total	34.8	15.0%	253.6	21

*The sums of each portion will not equal the total results because of different starting times.

The above analyses were done for the total project before sharing profits with El Paso. The effect of sharing earnings with El Paso, on a formula which probably would apply, would be:

El Paso's Share	R.C.I.	
	Lakeshore & Vekol	Lakeshore @ 40,000 TPD
0	12.0%	11.6%
10%	11.4%	10.9%
20%	11.1%	10.2%

Costs and Production Schedule

The capital costs, operating costs and net metal revenues used in the analysis were developed in previous sections in this report dealing with mining methods. Pre-production interest, calculated from the schedule of capital expenditures, was added to the capital costs. It was assumed that capital expenditures in each year were spread equally over the year.

The maximum capital required equalled capital expenditures plus interest to the beginning of production, 4 months operating costs and any necessary amounts of capital expenditures during production.

The operating costs were decreased near the end of the life of the mine to allow for the completion of development before all the ore had been drawn from the stopes.

Conditions

1) Taxes Considered

Arizona State Taxes were taken as 8% of net income after allowing for operating costs, Indian Royalties, depreciation, development write offs and interest.

U.S. Federal Taxes were taken on the same taxable income as State Taxes, less the State Taxes and a depletion allowance. The Federal Tax rate taken was 48%. The 10% surcharge was not considered as this is supposed to be a temporary tax to dampen inflation.

The 7% investment credit for new equipment purchases was used to decrease the Federal Taxes payable. This credit was taken as a maximum of 50% of the Federal Taxes and could be carried forward for a maximum of 7 years.

2) Indian Royalties

The Indian Royalties were calculated as an escalating percentage of net smelter return (using custom tariffs), the percentage dependent on the net smelter return per ton of ore mined.

3) Depreciation - Equipment and Buildings

The equipment and buildings were written off by an accelerated method after allowing 10% of the value for salvage. The depreciable value was divided by the years remaining in the mine and the resulting value was doubled and then taken as the depreciation for the year in question. This depreciation was subtracted from the depreciable value and the new value was used for the next depreciation calculation.

Capital expenditures during the year were added to the depreciable total after subtracting 10% for salvage.

4) Development Write Offs

Expenditures for preliminary testing, mine development, and pre-production interest were written off straight-line over the life of the mine.

5) Depletion

Depletion allowance for Federal Taxes was 50% of the taxable income after all other write offs or a maximum of 15% of the net smelter return. Depletion was applicable only on mine revenue and not on smelter revenues.

6) Salvage

Salvage was assumed to equal 10% of the value of all buildings and equipment purchased for the mine. This salvage was allowed for in calculating depreciation.

LAKESHORE MINE

EARNINGS PROJECTION

@ 25,000 T. P. D. - CUSTOM SMELTING

($000's Omitted)

Cu @ 42¢/lb.

YEAR	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
OPERATIONS																									
TONS MILLED /day	10,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,060	25,000	25,000	SALVAGE
OPERATING PROFIT	6,300	15,800	15,800	15,800	15,800	15,800	15,800	15,800	15,800	15,800	15,800	15,800	15,800	15,800	15,800	15,800	23,000	23,000	23,000	23,000	23,000	25,060	27,100	27,100	
DEPRECIATION	4,010	4,840	4,250	4,120	4,020	3,900	3,780	3,680	3,550	3,430	3,320	3,200	3,080	2,970	2,850	2,730	2,610	2,500	2,380	2,260	2,150	2,030	510	510	
DEVELOPMENT WRITE-OFFS	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	
DEPLETION	0	3,250	3,510	3,560	3,610	3,670	3,720	3,760	3,830	3,880	3,930	3,990	4,040	4,090	4,150	4,200	6,700	6,750	6,810	6,850	6,920	6,940	6,940	6,940	
INDIAN ROYALTIES	1,090	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	4,620	4,620	4,620	4,620	4,620	4,620	4,620	4,620	
TOTAL TAXES	0	1,340	1,440	1,470	2,100	2,320	2,350	2,390	2,430	2,460	2,490	2,530	2,570	2,600	2,640	2,670	4,300	4,330	4,370	4,410	4,450	5,570	7,510	7,510	
DEFERRED INCOME	5,200	8,800															2,400								15,400
CAPITAL EXPENDITURES	0	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	0	0	8,070
NET PROFIT	0	2,440	2,670	2,720	2,140	-,980	2,020	2,040	2,060	2,100	2,130	2,150	2,180	2,210	2,230	2,270	3,570	3,600	3,620	3,660	3,660	4,730	6,320	6,320	
NET CASH AVAILABLE	10	1,930	10,630	10,600	9,970	9,750	9,720	9,680	9,640	9,610	9,580	9,540	9,500	9,470	9,430	9,400	10,680	13,050	13,010	12,970	12,930	13,870	14,970	14,970	23,470
CAPITAL REPAYMENT																									
TAX SAVINGS DUE TO INTEREST WRITE-OFFS	0	1,340	1,440	1,470	2,100	2,320	1,820	1,760	1,440	1,150	1,030	880	720	560	390	210	20								
PRINCIPAL	0	0	0	1,010	5,930	6,340	6,260	6,600	6,690	6,850	7,180	7,490	7,820	8,180	8,530	8,920	930								
INTEREST	10	3,270	12,070	11,060	6,140	5,730	5,280	4,840	4,390	3,910	3,430	2,930	2,400	1,850	1,290	690	65								

RETURN ON INVESTMENT - 100% EQUITY = 9.0% AFTER TAXES

PAYBACK PERIOD AT 7% DEBT INTEREST = 16.1 YEARS

D.C.F. AT 10% AFTER DEBT REPAYMENT = $18.1 MILLION

A CHANGE OF 1¢ IN COPPER PRICE WILL CHANGE NET CASH AVAILABLE YR. 2-24- $0.7 MILLION /YR.

LAKESHORE MINE

EARNINGS PROJECTION

@ 40,000 T.P.D. -CUSTOM SMELTER

($000's Omitted) Cu @ 42¢/lb.

YEAR	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
OPERATIONS																		
TONS MILLED /day	10,000	25,000	25,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	SALVAGE
OPERATING PROFIT	6,300	16,100	16,100	27,000	27,000	27,000	27,000	27,000	27,000	27,000	27,000	27,000	38,600	38,600	38,600	45,200	45,200	
DEPRECIATION	3,560	10,170	7,360	7,040	6,700	6,370	6,030	5,710	5,370	5,030	4,700	4,350	4,020	3,690	3,360	845	845	
DEVELOPMENT WRITE-OFF	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	
DEPLETION	0	710	2,000	3,430	6,580	6,730	6,880	7,030	7,190	7,340	7,490	7,650	11,100	11,100	11,100	11,100	11,100	
INDIAN ROYALTIES	1,090	2,730	2,730	4,360	4,360	4,360	4,360	4,360	4,360	4,360	4,360	4,360	7,710	7,710	7,710	7,710	7,710	
TOTAL TAXES	0	295	830	2,650	2,710	3,875	4,385	4,485	4,585	4,685	4,785	4,885	7,725	7,885	8,085	12,930	12,930	
DEFERRED INCOME	5,200	7,800		7,800									3,900					24,700
CAPITAL EXPENDITURES	800	5,335	7,265	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	0	0	9,460
NET PROFIT	0	545	1,530	4,870	5,000	4,015	3,695	3,765	3,845	3,935	4,015	4,105	6,395	6,565	6,695	10,965	10,965	
NET CASH AVAILABLE	- 790	- 60	5,275	10,690	18,430	17,265	16,755	16,655	16,555	16,455	16,355	16,255	17,765	21,505	21,305	24,560	24,560	34,160
CAPITAL REPAYMENT																		
TAX SAVINGS DUE TO INTEREST WRITE-OFFS	0	295	830	2,650	2,710	2,465	2,515	2,190	915	620	305							
PRINCIPAL	- 790	0	0	0	5,550	13,680	14,180	14,645	14,390	15,005	14,610							
INTEREST	0	235	6,105	13,340	15,590	6,050	5,090	4,200	3,080	2,070	1,020							

RETURN ON INVESTMENT - 100% EQUITY = 11.6% AFTER TAXES

PAYBACK AT 7% DEBT INTEREST = 10.9 YEARS

D.C.F. AT 10% AFTER DEBT REPAYMENT = $37.8 MILLION

A 1¢ CHANGE IN THE PRICE OF COPPER WOULD CHANGE NET CASH AVAILABLE YR. 4-17 —$1.1 MILLION / YR.

LAKESHORE AND VEKOL COMBINED

EARNINGS PROJECTION

LAKESHORE PORTION - CUSTOM SMELTER

($000's Omitted)

Cu @ 42¢/lb.

YEAR	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
OPERATIONS																						
TONS MILLED /day		10,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	SALVAGE
OPERATING PROFIT		6,310	16,200	16,200	16,200	16,200	16,200	16,200	16,200	16,200	16,200	16,200	16,200	27,000	27,000	27,000	38,600	38,600	38,600	41,450	45,200	
DEPRECIATION		3,740	4,650	4,490	4,340	4,180	4,020	3,860	3,700	3,540	3,380	3,680	4,010	3,720	3,460	3,180	2,910	2,630	2,350	590	590	
DEVELOPMENT WRITE-OFFS		1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	1,480	
DEPLETION		0	3,380	3,460	3,520	3,600	3,670	3,740	3,820	3,890	3,960	3,830	3,680	8,000	8,150	8,270	11,100	11,100	11,100	11,100	11,100	
INDIAN ROYALTIES		1,090	2,720	2,720	2,720	2,720	2,720	2,720	2,720	2,720	2,720	2,720	2,720	4,370	4,370	4,370	7,400	7,400	7,400	7,400	7,400	
TOTAL TAXES		0	1,400	1,425	1,655	2,270	2,320	2,370	2,420	2,470	2,520	2,390	2,235	5,150	5,240	5,330	8,590	8,740	8,870	11,850	13,830	
DEFERRED INCOME		5,190	7,790											7,820			3,870					24,670
CAPITAL EXPENDITURES		0	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	3,000	4,215	1,000	1,000	1,000	1,000	1,000	1,000	0	0	7,000
NET PROFIT		0	2,570	2,605	2,485	1,950	1,990	2,030	2,060	2,100	2,140	2,100	2,075	4,280	4,360	4,370	7,120	7,250	7,380	9,030		
NET CASH AVAILABLE		30	3,290	11,055	10,825	10,210	10,160	10,110	10,060	10,010	9,960	8,090	7,030	8,660	16,360	16,300	17,740	21,460	21,330	22,200	23,970	31,670
CAPITAL REPAYMENT																						
TAX SAVINGS DUE TO INTEREST WRITE-OFFS		0	1,400	1,425	1,655	2,090	1,720	1,650	1,580	1,010	830	660	520	400	240							
PRINCIPAL	-14,290	0	0	0	710	6,960	7,030	7,400	7,790	7,660	8,010	6,545	5,810	7,720	11,475							
INTEREST		30	4,690	12,480	11,770	5,340	4,850	4,360	3,850	3,360	2,780	2,205	1,740	1,340	800							

RETURN ON INVESTMENT - 100% EQUITY - 10.7% AFTER TAXES

PAYBACK PERIOD AT 7% DEBT INTEREST - 13.8 YEARS

D.C.F. AT 10% AFTER DEBT REPAYMENT = $26.9 MILLION

A CHANGE OF 1¢ IN COPPER PRICE WILL CHANGE NET CASH AVAILABLE
YRS. 3-13 - $0.7 MILLION /YR.
YRS. 14-27 - $1.1 MILLION/YR.

LAKESHORE AND VEKOL COMBINED

EARNINGS PROJECTION

($000's Omitted)

VEKOL PORTION - CUSTOM SMELTER

Cu @ 42¢/lb.

YEAR	1	2	3	4	5	6	7	8	9	10	11	12	13	14
OPERATIONS														
TONS MILLED /day	25,000	25,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	SALVAGE
OPERATING PROFIT	16,050	16,050	9,900	9,900	9,900	9,900	9,900	9,900	9,900	11,590	12,620	12,620	12,620	
DEPRECIATION	3,465	3,285	3,070	2,850	2,630	2,420	2,200	1,980	1,760	1,550	1,330	290	290	
DEVELOPMENT WRITE-OFFS	960	960	960	960	960	960	960	960	960	960	960	960	960	
DEPLETION	4,540	4,660	2,220	2,310	2,410	2,510	2,610	2,720	2,820	3,130	3,130	3,130	3,130	
INDIAN ROYALTIES	1,750	1,750	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	
TOTAL TAXES	1,930	2,925	1,415	1,475	1,545	1,595	1,665	1,735	1,795	2,645	3,305	3,885	3,885	
DEFERRED INCOME	11,650		+4,650											7,000
CAPITAL EXPENDITURES	0	500	500	500	500	500	500	500	500	500	500	0	0	3,020
NET PROFIT	3,405	2,470	1,185	1,255	1,305	1,365	1,415	1,455	1,515	2,255	2,845	3,305	3,305	
NET CASH AVAILABLE	720	10,875	11,585	6,875	6,805	6,755	6,685	6,615	6,555	7,395	7,765	7,685	7,685	10,020
CAPITAL REPAYMENT														
TAX SAVINGS DUE TO INTEREST WRITE-OFFS	550	825	640	425	300	175	40							
PRINCIPAL	0	7,360	10,105	5,880	6,095	6,345	1,985							
INTEREST	1,270	4,340	2,120	1,420	1,010	585	140							

RETURN ON INVESTMENT - 100% EQUITY - 16.6% AFTER TAXES

PAYBACK PERIOD AT 7% DEBT INTEREST - 6.3 YEARS

D.C.F. AT 10% AFTER DEBT REPAYMENT = $21.1 MILLION

A CHANGE OF 1¢ IN COPPER PRICE WILL CHANGE NET CASH AVAILABLE YRS. 3-13 - $0.35 MILLION / YR.

LAKESHORE MINE

EARNINGS PROJECTION

@ 25,000 T.P.D. - WITH SMELTER

($000's Omitted)

Cu @ 42¢/lb.

YEAR	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
OPERATIONS																									
TONS MILLED/day	10,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000
OPERATING PROFIT	6,900	17,400	17,400	17,400	17,400	17,400	17,400	17,400	17,400	17,400	17,400	17,400	17,400	17,400	17,400	17,400	25,000	25,000	25,000	25,000	25,000	27,060	29,100	29,100	
DEPRECIATION	4,590	7,940	5,970	5,780	5,580	5,380	5,180	4,980	4,780	4,580	4,380	4,180	3,990	3,790	3,590	3,390	3,190	2,990	2,790	2,600	2,400	2,200	550	550	
DEVELOPMENT WRITE-OFFS	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	1,220	
DEPLETION	0	3,230	3,510	3,560	3,610	3,670	3,720	3,760	3,830	3,880	3,930	3,990	4,040	4,090	4,150	4,200	6,700	6,750	6,810	6,850	6,920	6,940	6,940	6,940	
INDIAN ROYALTIES	1,090	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	2,730	4,620	4,620	4,620	4,620	4,620	4,620	4,620	4,620	
TOTAL TAXES	0	880	1,400	1,450	1,500	1,540	2,300	2,540	2,610	2,690	2,770	2,850	2,930	3,000	3,080	3,160	5,040	5,120	5,190	5,270	5,340	6,430	8,520	8,520	
DEFERRED INCOME	6,500	9,800															3,000								19,300
CAPITAL EXPENDITURES	0	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	10,600
NET PROFIT	0	1,400	2,570	2,660	2,760	2,860	2,250	2,170	2,230	2,300	2,370	2,430	2,490	2,570	2,630	2,700	4,230	4,300	4,370	4,440	4,500	5,650	7,250	7,250	
NET CASH AVAILABLE	- 690	2,990	12,270	12,220	12,170	12,130	11,370	11,130	11,060	10,980	10,900	10,820	10,740	10,670	10,590	10,510	11,340	14,260	14,190	14,110	14,040	15,010	15,960	15,960	29,900
CAPITAL REPAYMENT																									
TAX SAVINGS DUE TO INTEREST WRITE-OFFS	0	880	1,400	1,450	1,500	1,540	2,300	2,540	2,610	2,690	2,770	2,690	1,800	1,230	1,040	860	660	440	140						
PRINCIPAL	- 690	0	0	0	2,240	5,780	6,180	6,610	7,070	7,570	8,110	8,510	8,140	8,070	8,370	8,690	9,930	13,330	6,360						
INTEREST	0	3,870	13,670	13,670	11,430	7,890	7,490	7,060	6,600	6,100	5,560	5,000	4,400	3,830	3,260	2,680	2,070	1,370	440						

RETURN ON INVESTMENT - 100% EQUITY = 7.7% AFTER TAX

PAYBACK PERIOD AT 7% DEBT INTEREST - 18.5 YEARS

D.C.F. AT 10% AFTER DEBT REPAYMENT = $14.6 MILLION

A CHANGE OF 1¢ IN COPPER PRICE WILL CHANGE NET CASH AVAILABLE YRS. 2-24 - $0.7 MILLION/ YR.

LAKESHORE MINE

EARNINGS PROJECTION

@ 40,000 T. P. D. - WITH SMELTER

Cu @ 42¢/lb.

($000's Omitted)

YEAR	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
OPERATIONS																		
TONS MILLED /day	10,000	25,000	25,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	SALVAGE
OPERATING PROFIT	6,900	17,700	17,700	29,600	29,600	29,600	29,600	29,600	29,600	29,600	29,600	29,600	41,700	41,700	41,700	48,300	48,300	
DEPRECIATION	4,080	13,240	11,240	9,380	8,890	8,350	7,830	7,300	6,790	6,290	5,780	5,260	4,740	4,230	3,710	940	940	
DEVELOPMENT WRITE-OFFS	1,730	1,730	1,730	1,730	1,730	1,730	1,730	1,730	1,730	1,730	1,730	1,730	1,730	1,730	1,730	1,730	1,730	
DEPLETION	0	0	920	6,430	6,580	6,730	6,880	7,030	7,190	7,340	7,490	7,650	11,100	11,100	11,100	11,100	11,100	
INDIAN ROYALTIES	1,090	2,730	2,730	4,360	4,360	4,360	4,360	4,360	4,360	4,360	4,360	4,360	7,710	7,710	7,710	7,710	7,710	
TOTAL TAXES	0	0	380	2,705	2,820	3,650	4,765	4,975	5,155	5,345	5,545	5,765	8,915	9,185	9,465	14,430	14,430	
DEFERRED INCOME	6,500	9,700	0	9,800									4,900					30,900
CAPITAL EXPENDITURES	800	5,330	9,665	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	0	0	12,120
NET PROFIT	0	0	700	4,995	5,220	4,780	4,035	4,205	4,375	4,535	4,695	4,835	7,505	7,745	7,985	12,390	12,390	
NET CASH AVAILABLE	-1,490	- 60	4,925	11,235	20,920	20,090	18,975	18,765	18,585	18,395	18,195	17,975	18,675	23,305	23,025	26,160	26,160	43,020
CAPITAL REPAYMENT																		
TAX SAVINGS DUE TO INTEREST WRITE-OFFS			380	2,705	2,820	3,650	4,765	2,895	1,565	1,250	990	625	255					
PRINCIPAL	-1,490	- 60	0	0	0	12,165	16,330	15,360	14,930	15,475	16,095	16,640	11,455					
INTEREST	0	0	5,305	13,940	23,740	11,575	7,410	6,300	5,220	4,170	3,090	1,960	800					

RETURN ON INVESTMENT - 100% EQUITY = 9.9% AFTER TAXES

PAYBACK AT 7% DEBT INTEREST = 12.6 YEARS

D. C. F. AT 10% AFTER DEBT REPAYMENT = $32.3 MILLION

A 1¢ CHANGE IN THE PRICE OF COPPER WOULD CHANGE NET CASH AVAILABLE
YRS. 4-17 -$1.1 MILLION/YR.

LAKESHORE AND VEKOL COMBINED

LAKESHORE PORTION - WITH SMELTER

EARNINGS PROJECTION

($000's Omitted) Cu @ 42¢/lb.

YEAR	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
OPERATIONS																						
TONS MILLED /day		10,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000	SALVAGE
OPERATING PROFIT		6,960	17,750	17,750	17,750	17,750	17,750	17,750	17,750	17,750	17,750	17,750	17,750	29,600	29,600	29,600	41,600	41,600	41,600	44,450	48,200	
DEPRECIATION		4,260	6,450	6,200	5,930	5,670	5,420	5,160	4,910	4,650	4,390	4,690	5,000	4,600	4,210	3,810	3,400	3,000	2,600	660	650	
DEVELOPMENT WRITE-OFFS		1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	1,610	
DEPLETION		0	3,380	3,460	3,520	3,600	3,670	3,740	3,820	3,890	3,960	3,830	3,680	8,000	8,150	8,270	11,100	11,100	11,100	11,100	11,100	
INDIAN ROYALTIES		1,090	2,720	2,720	2,720	2,720	2,720	2,720	2,720	2,720	2,720	2,720	2,720	4,370	4,370	4,370	7,400	7,400	7,400	7,400	7,400	
TOTAL TAXES		0	1,285	1,345	1,410	1,465	2,105	2,440	2,530	2,630	2,740	2,615	2,455	6,010	6,140	6,290	9,820	10,040	10,250	12,830	14,760	
DEFERRED INCOME		6,500	9,700											9,800			4,800					30,000
CAPITAL EXPENDITURES		0	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	3,000	4,215	1,000	1,000	1,000	1,000	1,000	1,000	0	0	9,070
NET PROFIT		0	2,305	2,415	2,560	2,685	2,225	2,080	2,160	2,250	2,330	2,285	2,285	5,010	5,120	5,250	8,270	8,450	8,640	10,850	12,680	
NET CASH AVAILABLE		- 630	3,045	12,685	12,620	12,565	11,925	11,590	11,500	11,400	11,290	9,415	8,360	8,420	18,090	17,940	18,580	23,160	22,950	24,220	26,040	30,070
CAPITAL REPAYMENT																						
TAX SAVINGS DUE TO INTEREST WRITE-OFFS		0	1,285	1,345	1,410	1,465	2,105	2,440	2,530	1,790	1,410	1,240	1,090	970	850	480	110					
PRINCIPAL	-14,288	- 630	0	0	0	1,810	7,340	7,780	8,330	8,140	8,150	6,675	5,940	6,290	16,300	16,910	4,705					
INTEREST	0	0	4,330	14,030	14,030	12,220	6,690	6,250	5,700	5,050	4,550	3,980	3,510	3,100	2,640	1,510	330					

RETURN ON INVESTMENT - 100% EQUITY - 9.4% AFTER TAXES

PAYBACK PERIOD AT 7% DEBT INTEREST - 15.3 YEARS

D.C.F. AT 10% AFTER DEBT REPAYMENT - $24.9 MILLION

A 1¢ CHANGE IN THE PRICE OF COPPER WOULD CHANGE NET CASH AVAILABLE
YRS. 3-13 - $0.7 MILLION / YR.
YRS. 14-21 - $1.1 MILLION / YR.

LAKESHORE AND VEKOL COMBINED

EARNINGS PROJECTION

($000's Omitted)

VEKOL PORTION - WITH SMELTER

Cu @ 42¢/lb.

YEAR	1	2	3	4	5	6	7	8	9	10	11	12	13	14
OPERATIONS														
TONS MILLED /day	25,000	25,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	SALVAGE
OPERATING PROFIT	17,400	17,400	10,700	10,700	10,700	10,700	10,700	10,700	10,700	12,390	13,420	13,420	13,420	
DEPRECIATION	4,450	4,160	3,880	3,570	3,280	2,990	2,680	2,390	2,090	1,790	1,490	375	375	
DEVELOPMENT WRITE-OFFS	990	990	990	990	990	990	990	990	990	990	990	990	990	
DEPLETION	4,330	4,450	2,010	2,100	2,200	2,300	2,400	2,510	2,610	3,130	3,130	3,130	3,130	
INDIAN ROYALTIES	1,750	1,750	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	
TOTAL TAXES	2,030	3,060	1,495	1,615	1,715	1,815	1,935	2,025	2,145	2,935	3,625	4,240	4,240	
DEFERRED INCOME	14,300		+5,700											8,600
CAPITAL EXPENDITURES	0	500	500	500	500	500	500	500	500	500	500	0	0	3,720
NET PROFIT	3,850	2,990	1,275	1,375	1,465	1,555	1,645	1,735	1,815	2,495	3,135	3,635	3,635	
NET CASH AVAILABLE	- 680	12,090	13,355	7,535	7,435	7,335	7,215	7,125	7,005	7,905	8,245	8,130	8,130	12,320
CAPITAL REPAYMENT														
TAX SAVINGS DUE TO INTEREST WRITE-OFFS	610	1,545	860	600	470	320	175	15						
PRINCIPAL	- 70	6,775	11,525	6,255	6,455	6,665	6,865	660						
INTEREST		6,860	2,690	1,880	1,450	990	525	45						

RETURN ON INVESTMENT - 100% EQUITY - 14.4% AFTER TAXES

PAYBACK AT 7% DEBT INTEREST = 7.1 YEARS

D.C.F. AT 10% AFTER DEBT REPAYMENT = $20.1 MILLION

A 1¢ CHANGE IN THE PRICE OF COPPER WOULD CHANGE NET CASH AVAILABLE YRS. 3-13 - $0.35 MILLION/YR.

APPENDICES

APPENDIX I

OPEN PIT RESERVES

SUMMARY

RESERVES

	Tons	Grade % Cu
Sulphides -	168,454,000	0.691
Oxides -	114,747,000	0.639

STRIPPING RATIOS

Sulphides alone	-	6.92:1
Sulphides + Oxides	-	3.71:1

SULPHIDES (Factor 12.5 Cu. Ft/Ton) Cut-off @ 0.4% Cu

Section	Tons (000)	Grade % Cu	Tons Cu
900 S	-	-	-
600 S	-	-	-
300 S	264	0.600	1,580
0-0	7,560	0.892	67,450
300 N	7,452	0.861	64,170
600 N	9,732	0.798	77,630
900 N	15,372	0.667	102,490
1200 N	23,844	0.770	183,490
1500 N	17,616	0.826	145,520
1800 N	23,304	0.686	159,780
2100 N	22,104	0.714	157,810
2400 N	22,948	0.680	156,060
2700 N	9,684	0.627	60,680
3000 N	552	0.500	2,760
	160,432	0.735	1,179,420
Uncorrected -	160,432	0.735	1,179,420
Assay Correction 4% -	-	-	47,177
	160,432	0.706	1,132,243
Dilution - 5% of 0.4% -	8,022	0.400	32,088
Expected Mill Heads	168,454	0.691	1,164,331

OXIDES (Factor 13.5 Cu. Ft/Ton Cut-off @ 0.4% Cu

Section	Tons (000)	Grade % Cu	Tons Cu (000)
900 S	-	-	-
600 S	-	-	-
300 S	1,511	0.43	6,500
0-0	4,222	0.438	18,510
300 N	5,445	0.610	33,200
600 N	5,877	0.561	32,990
900 N	14,199	0.510	72,410
1200 N	9,477	0.789	70,000
1500 N	10,066	0.759	76,430
1800 N	12,054	0.847	102,070
2100 N	12,499	1.01	125,800
2400 N	11,344	0.932	105,710
2700 N	11,665	0.738	86,040
3000 N	5,956	0.850	50,620
	104,315	0.748	780,280
Uncorrected -	104,315	0.748	780,280
Assay Correction 10% -	-	-	78,028
	104,315	0.673	702,252
Dilution 10% of 0.3%	10,432	0.300	31,296
	114,747	0.639	733,548

LOW GRADE OXIDES (Factor 13.5 Cu. Ft./Ton)

Section	Tons (000)	Grade % Cu	Tons Cu (000)
0-0	22,576	0.29	65,470
300 N	23,775	0.28	66,570
600 N	22,842	0.28	63,960
900 N	19,887	0.30	59,660
1200 N	13,221	0.28	37,020
1500 N	28,352	0.35	99,280
1800 N	7,510	0.33	24,800
2100 N	9,266	0.28	25,940
2400 N	8,599	0.28	29,080
2700 N	260	0.26	680
	156,288	0.30	472,540

Oxides above 0.4% Cut-off (with expected high acid consumption)

1200 N	4,933	0.43	21,210
1800 N	10,799	0.49	53,210
	15,732	0.47	74,420
TOTAL	172,020	0.318	546,960
Uncorrected	172,020	0.318	546,960
Assay Correction 10%	-	-	54,696
	172,020	0.286	492,264

WASTE TO BE MOVED

Pit was de signed for a 45° slope. Additional tonnage of semi-consolidated sediments ("conglomerate") and alluvium, allowing for a 32° slope, has been estimated.

Alluvium	92,575,000 tons
Low Grade Oxides	172,020,000 "
"Conglomerate"	423,559,000 "
Rock	212,790,000 "
Waste winin Oxides-Sulphides	18,721,000 "
Added Conglomerate - Alluvium	150,000,000 "
	1,069,665,000 tons

Total tons to be moved - 1,334,412,000

Stripping Ratio: Sulphides only (1,165,958/168,454 = 6.92:1

Sulphides + High Grade Oxides
(1,051,211/283,201) = 3.71:1

APPENDIX II

RESERVES FOR BLOCK CAVING

SUMMARY OF EXPECTED ORE DELIVERED TO MILL

	Tons (x'1000)	Copper %	Copper Tons
Indicated by drilling	145,334	0.716	1,039,799
Probable	21,706	1.078	234,088
	167,040	0.763	1,273,887
Possible additional	41,122	0.730	300,267
	208,162	0.756	1,574,154

Note -

Ore indicated by drilling covers only the section east of the fault which has been drilled off by holes spaced roughly 300 feet apart and includes ore within 150 feet of the holes. Intersections less than 100 feet thick not included.

Probable ore reserve includes ore indicated and within 150 feet of three holes drilled west of the fault at depth.

Possible additional reserves include extensions 300 feet beyond the reserves indicated above to the north, south and west in areas not yet drilled.

UNCORRECTED RESERVES FOR BLOCK CAVING E. OF FAULT

D.D.Hole	Area (x 1000)	Thickness	Tons (x 1000) (12.5 c.f./t)	Percent Cu	Tons Cu (x 1000)
(a) Blocks over 200' thick					
# 72	118	482'	4550	0.725	32.99
69	170	587'	7983	0.963	76.88
70	150.5	611'	7356	0.725	53.33
59	114.5	603'	5523	0.710	39.21
68	200.5	656'	10522	0.750	78.92
55	166	562'	7463	0.701	52.32
53	125	725'	7250	0.820	59.45
27	102	311'	2538	0.461	11.70
64	117	946'	8855	0.842	74.56
41	124	604'	5992	0.747	44.76
43	132.5	643'	6816	0.760	51.80
30	110	416'	3661	0.740	27.09
1A	99	423'	3350	0.710	23.79
56	132	1024'	10813	0.748	80.88
40	75	572'	3432	0.548	18.81
45	90	555'	3996	0.580	23.18
39	124	247'	2450	1.480	36.26
3	94	320'	2406	0.846	20.35
58	145	295'	3422	0.530	18.14
48	139	471'	5238	1.122	58.77
47	92.5	425'	3145	0.695	21.86
46	90	378'	2722	0.860	23.41
37	145.5	214'	2491	0.888	22.12
66	142	244'	2772	1.070	29.66
71	173	356'	4927	1.000	49.27
Total (a)	3171		129,673		102,951
(b) Blocks from 100' to 200' thick					
42	81	150'	972	1.800	17.50
5	96	146'	1121	1.540	17.26
28 A	72	123'	708	1.732	12.26
38	116	107'	993	0.987	9.80
29	100.5	120'	965	1.050	10.13

Cont'd.....

D.D. Hole	Area (x 1000)	Thickness	Tons (x 1000)	Percent Cu	Tons Cu (x 1000)
# 8	104.5	117'	978	1.340	13.11
51	118	185'	1746	1.050	18.33
50	133	154'	1639	0.905	14.83
52	121.5	176'	1107	0.875	9.69
Total (b)	943		10,229		122.91
Total (a & b)	4114		139,902		

Overall Average Height - $\frac{139.902 \times 12.5}{4114} = 425'$

SUMMARY - UNDERGROUND SULPHIDES

Blocks Over 200':

Tons	129,673,000
Grade	.79% Cu
Height	511'

Blocks Under 200':

Tons	10,229,000
Grade	1.19% Cu
Height	136'

ORE RESERVES FOR BLOCK CAVING

SECTION E. OF FAULT

	Tons (x 1000)	Copper %	Copper Tons
Uncorrected reserve from d. drilling	139,900	0.824	1,152,420
Assay correction - 4%			- 46,097
	139,900	0.791	1,106,323
For caving on San Manuel system in three lifts:			
Ore lost in pillars	7,846	0.791	62,062
	132,054	0.791	1,044,261
Low grade stoped	13,280	0.350	46,480
Ore to be mined	145,334	0.751	1,090,741
Ore not recovered @ 10%	14,533	0.751	109,074
	130,801	0.751	981,667
Dilution 10% of ore to be mined	14,533	0.400	58,132
Expected mill heads	145,334	0.716	1,039,799

ORE INDICATED WEST OF FAULT (PROBABLE RESERVE)

(D.D. HOLES #57, 73 and 74)

D.D.H. #57: 590' @ 0.730% Cu
73: 444' @ 1.75% Cu
74: 350' @ 1.05% Cu (partly est.)

Probable Reserves	Tons (x 1000)	Copper %	Copper Tons
D.D.H. #73 & 74	16,318	1.441	235,140
D.D.H. #57	4,564	0.730	33,317
	20,882	1.286	268,457
Deduct assay correction 4%			10,738
	20,882	1.234	257,719
Est. ore lost in pillars 5 1/2%	1,149	1.234	14,175
	19,733	1.234	243,544
Low grade stoped 10%	1,973	0.350	6,905
	21,706	1.154	250,449
Ore not recovered @ 10%	2,171	1.154	25,045
	19,535	1.154	225,404
Dilution @ 10%	2,171	0.400	8,684
Expected mill heads	21,706	1.078	234,088

POSSIBLE RESERVES

	Tons (x 1000)	Copper %	Copper Tons
South End	6,860	1.040	71,344
North End	18,600	0.780	145,080
West (South)	12,200	0.764	93,208
West (North)	1,900	1.240	23,560
	39,560	0.842	333,192
Deduct assay correction 4%			13,328
	39,560	0.809	319,864
Est. ore left in pillars 5 1/2%	2,176	0.809	17,593
	37,384	0.809	302,271
Low grade stoped 10%	3,738	0.350	13,083
	41,122	0.767	315,354
Ore not recovered 10%	4,112	0.767	31,535
	37,010	0.767	283,819
Dilution @ 10%	4,112	0.400	16,448
Expected mill heads	41,122	0.730	300,267

APPENDIX III

UNDERGROUND DEVELOPMENT COSTS

PRIMARY UNDERGROUND DEVELOPMENT COSTS (25,000 Tpd.)

FIRST BLOCK

Haulage Level				
Main Drives	6.960'	@$110	765,600	
Crosscuts	4.140'	@$110	455,400	
				$1,221,000
Service Level				
Main Drives	4.290'	@$100	$ 429,000	
Crosscuts	2.030'	@$100	203,000	
				$ 632,000
				$1,853,000

SECOND BLOCK

Haulage Level				
Main Drives	10.450'	@$110	$1,149,500	
Crosscuts	10.780'	@$110	1,185,800	
				$2,335,300
Service Level				
Main Drives	6.110'	@$100	$ 611,000	
Crosscuts	4.220	@$100	422,000	
				$1,033,000
				$3,368,300

APPROXIMATE TONS DEVELOPED

First Block = 21,000,000 TONS
Second Block= 89,000,000 TONS

(TONNAGES INCLUDE DILUTION)

NO. OF 160' x 180' STOPING BLOCKS DEVELOPED

First Block = 33.7
Second Block = 73.6

NOTE: Development outside ore blocks for access, ventilations & Haulage.

COST OF STOPE DEVELOPMENT PER STOPE BLOCK

Cost of developing a stoping block 160' x 180', as at San Manuel including haulage drifts and all work above haulage level.

ESTIMATED COSTS

Raising			
Drawpoints	96 @ 15' = 1.440'	@$ 75 =	$108,000
Finger raises	24 @ 60' = 1.440'	@$ 75 =	108,000
Crosscutting			
Haulage crosscuts	2 @180' = 360'	@$110 =	$ 39,600
Access drift	1 @180' = 180'	@$100 =	18,000
Grizzly drifts	6 @140' = 840'	@$ 90 =	75,600
Undercutting		@$ 1 =	$114,000
Slashing 14,000 tons (break only)		@$ 1 =	$ 14,000
TOTAL PER BLOCK			$363,200

Tons developed per vertical foot = 2,300 tons

Stope Height	Stope dev. cost/ton
100'	$1.580
200'	0.790
300'	0.525
400'	0.395
500'	0.315
600'	0.263
700'	0.225
800'	0.197
900'	0.175
1000'	0.158

Average height of reserves = 425' Cost = 37.2¢/ton

Average height of first block = 257'
Average height of second block = 500'

DEVELOPMENT COSTS - FIRST & SECOND BLOCKS (FOR 25,000 T./D.)

FIRST BLOCK

Tons developed = 21,000,000
Stoping blocks developed = 33.7

		Cost/Ton
Main drives, haulage & service levels	$ 1,194,600	5.7¢
Crosscuts to stoping area	658,400	3.1¢
	1,853,000	8.8¢
Stope development 33.7 x $363,200	12,240,000	58.3¢
	$14,093,000	67.1¢

SECOND BLOCK

Tons developed = 89,000,000
Stoping blocks developed = 73.6

Main drives, haulage & service levels	$ 1,760,500	2.0¢
Crosscuts to stoping area	1,607,800	1.8¢
	$ 3,368,300	3.8¢
Stope development 73.6 x $363,200	26,731,500	30.0¢
	$30,099,800	33.8¢

AVERAGE

Tons developed = 110,000,000
Stoping blocks developed = 107.3

Main drives, haulage & service levels	$ 2,955,100	2.7¢
Crosscuts to stoping area	2,266,200	2.1¢
	$ 5,221,300	4.8¢
Stope development 107.3 blocks	38,971,500	35.4¢
	$44,192,800	40.2¢

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CODE 416-366-1653
CABLE "ROSSONTO"

December 9, 1968

Mr. E. Johnson
McIntyre - Porcupine Mines Ltd.
Suite 1200
55 Yonge Street
Toronto, Ontario

Re: Arizona Copper Project

Dear Eric:

This letter will confirm our telephone conversation of December 6, 1968, in which we forwarded to you a very preliminary order-of magnitude estimate of capital and operating costs for vat-leaching and cementation of an oxide copper deposit in Arizona.

The terms of reference for the estimates, as discussed at our meeting on December 5, 1968, were as follows:

- Treatment rates: 10,000 and 15,000 tons per day
- Ore grade: 0.67 percent copper
- Flowsheet: Vat-leaching followed by cementation of the copper from pregnant liquor
- Recovery of copper: as indicated in limited test data supplied by McIntyre - Porcupine Mines Ltd.

1. Flowsheet

As shown in the attached flowsheet, we have detailed a slime treatment circuit which in our opinion should be included in the estimates to guarantee satisfactory operation of the vats, until test work can prove it to be unnecessary.

. . . /2

We have estimated the cementation of copper on the basis of Kennecott type precipitation cones, in view of the reported improvements in costs and metallurgy.

2. Metallurgy

The overall recovery of copper in the operation has been estimated at 65 percent, from ore containing 0.67 percent copper. This estimate is based on the agitation-leaching data for test hole P-53, which was the only applicable data available, and makes an allowance for the lower grade of ore and the possible inefficiencies of vat leaching.

The major reagent consumptions were estimated at -

6.0 lb 93% sulphuric acid per lb copper recovered
(based on the test data)

1.5 lb iron scrap per lb copper recovered
(industry experience)

3. Unit Costs

The following unit costs were used as being applicable in Arizona:

Labour	-	$3.50 per hour including overhead
Acid	-	$30.00 per ton (93%) f.o.b. mine-site
Scrap iron	-	$50.00 per ton f.o.b. mine-site
Power	-	1¢ per KWH
Water	-	20¢ per 1000 gallons

4. Capital Cost Estimate

The capital estimates cover the following basic plant:

- three stage crushing
- storage of crushed ore in a stockpile with a tunnel reclaim system
- agglomeration, tripper and distribution conveyor system for loading vats
- complete vat system

. . . /3

- clam-shell unloading system for leached ore
- slime separation and leaching system
- storage tanks for various grades of copper liquor
- precipitation cones for copper recovery
- conveying of tailing to disposal area.

Not included in the estimates are:

- offices, service buildings
- roads
- tailing dam

The capital estimates for the two treatment rates are tabulated:

	10,000 TPD	15000 TPD
Crushing and Vat Leaching	$15,000,000	$19,100,000
Slime Leaching	1,500,000	1,800,000
Cementation	900,000	1,200,000
Total	$17,400,000	$22,100,000

5. Operating Costs

The estimated direct cost of operation for the two treatment rates being considered is detailed as follows:

	Cost per ton of Ore	
	10,000 TPD	15,000 TPD
Supervision	$0.023	$0.015
Assaying and Sampling	0.016	0.010
Crushing and Conveying	0.073	0.060
Leaching	0.090	0.067
Cementation	0.020	0.014

. . . /4

	Cost per ton or Ore (Continued) 10,000 TPD	15,000 TPD
Maintenance	$0.223	$0.193
Reagents	1.319	1.319
Water	0.011	0.011
Tailing Dam	0.035	0.035
Total cost per ton of ore	$1.810	$1.724

Further costs applicable to the operation are:

- smelter charges at approximately 7¢ per lb copper in cement
- freight to the smelter
- Arizona state tax at 1.5 percent of net smelter return
- normal corporation taxes and royalties
- administration, insurance and sales expenses.

6. Qualifications

These estimates of capital and operating costs have been prepared at short notice from readily available data on the basis of limited metallurgical information, and should be used only in the context of a very preliminary order-of-magnitude appraisal of a leaching operation. A program of metallurgical testing would be required and more detailed cost data would have to be developed before a reliable feasibility study could be made.

7. Operation

The amenability of this ore to an acid vat-leaching operation has not been established. Certain Arizona ores have been found to be unsuitable for a straight vat operation, due to the blinding of the ore during leaching. Such a condition decreases further the normally inefficient flow of liquor through a bed of ore.

In regard to these factors, a separate slime circuit has been included in the estimate.

. . . /5

The recovery of copper in a vat operation is dependent on the efficiency of the liquor flow as well as the leachability of the ore. Consequently, the recovery of copper from such an operation can only be estimated with reservation, in the absence of pilot plant data. If a pilot plant study of vat leaching is not possible, laboratory scale test work would permit confident scale-up to a commercial operation of only those flowsheets employing agitation leaching of the ground ore.

We appreciate the opportunity to be of service to your company in this matter, and we would be pleased to have you contact us, if we can be of any further assistance.

Yours very truly,

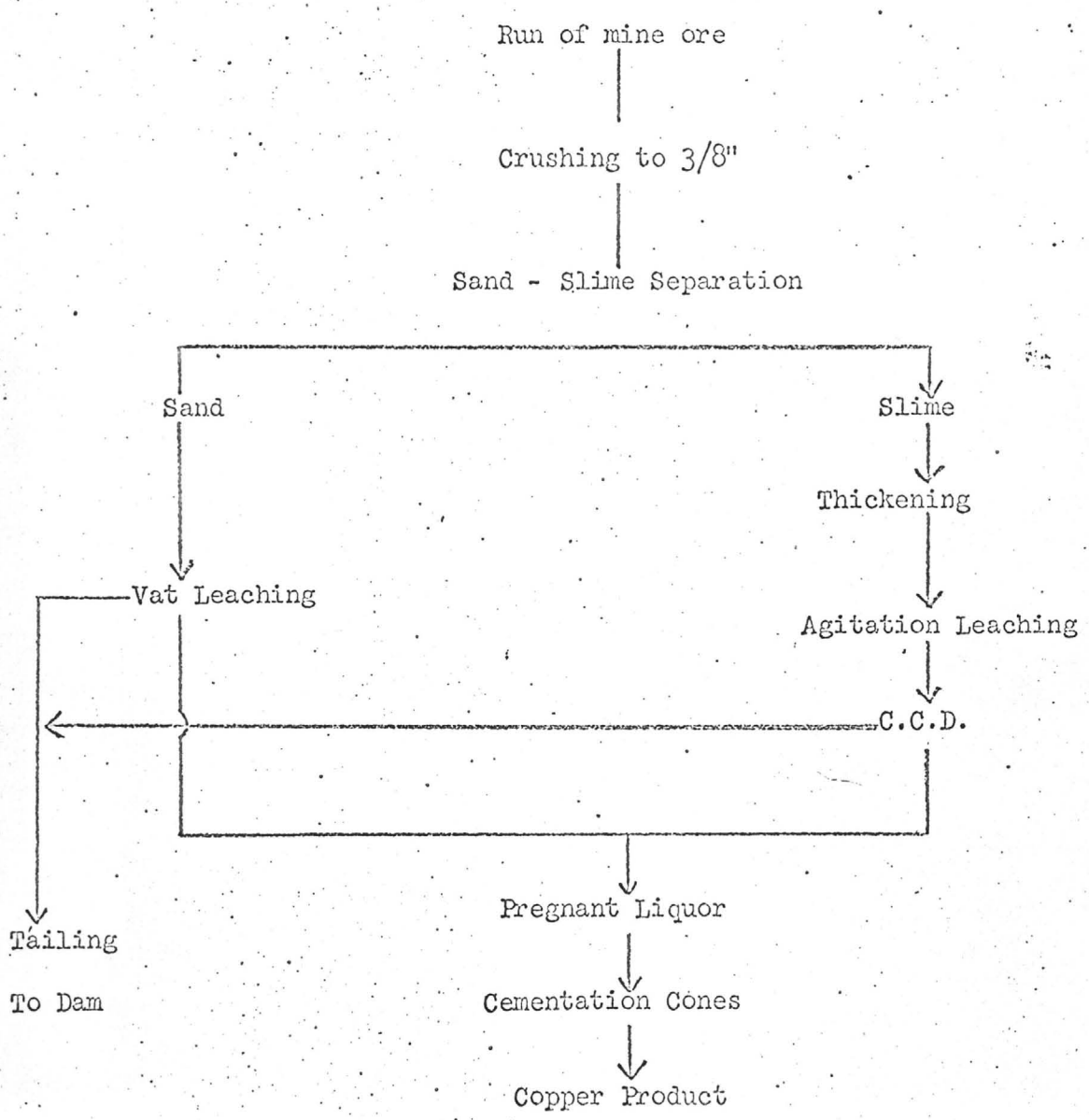
K. R. Rawling

K. R. Rawling, P. Eng.

KRR:mm
Enclosure

FLOWSHEET

Leaching of Oxide Copper Ore



COPPER LEACHING

Copper minerals occurring in oxidized deposits are not amenable to concentration by flotation as are sulphides. Provided that the deposit does not contain excessive amounts of carbonate or clay or slime, the metal con be recovered by leaching with sulphuric acid. Oxide ores not amenable to sulphuric acid leaching are in some instances being treated by the segregation process. This process employs calcining at 1500 - 1700° F in the presence of salt and coke to produce metallic copper which is recovered by conventional flotation.

Detailed cost information on leaching operations is not readily available. Each operation becomes an individual problem because in each case amenability to the process is governed by the nature of the mineral constituents associated with the copper and the reaction of those minerals to the acid leach. These in turn, govern acid consumption and metal recovery. The solution to each problem can be arrived at only by careful and complete testing of bulk samples and by sound planning towards efficient operations.

In order to point out these facts in somewhat greater detail, a number of applications of the sulphuric acid leaching process will be outlined.

COPPER DISSOLUTION

Sulphuric acid leaching is applied to the leaching of oxide copper contained in dump or strip waste material and as well to material which can be profitably mined and treated as copper ore. The criteria which govern the profitability of a leaching operation are acid consumption and metal recovery.

The process has been applied in (a) dump leaching
(b) heap leaching
(c) vat and tank leaching
(d) L-P-F (leaching-precipitation -flotation)
(e) microbiological leaching.

continued . . . . . .

Copper Dissolution

(a) Dump Leaching is used to recover copper from various types of waste. It requires low capital investment, low labor charges and little close supervision. It is simple and continuous in operation. Recoveries are low. Chemical reactions in dump leaching are complex and recently, studies have been undertaken to learn more about them and to try to reduce reaction time and increase recovery.

(b) Heap Leaching is applied to ores which have been mined for the purpose of leaching. Metal production time is measured in months. Recoveries can never be accurately forecast and will never approach that of either conventionally milled sulphide ore or vat-leached sulphide ore. In heap leaching the operator accepts low recovery in exchange for low capital cost. A thorough test program must be carried out beforehand to ensure that high-acid consuming materials are not present and that the heap does not degrade and blind. Sites for heaps must be carefully selected, to minimize acid loss.

Ideally, recovery is best forecast by leaching a test heap of the ultimate height. Frequently this is impossible or too costly and as an alternative, tests are frequently made in columns 16-20 ft. high by 4 - 5 ft. in diameter. In either case, the amount of material required necessitates bulk sampling from the ore body.

(c) Vat-Leaching is applied to ores which can support the higher capital and operating costs incurred by the installation of large vats and ancillary equipment for loading and unloading them and hauling the residues to the waste dumps. The ore is crushed in conventional equipment to 7/16 inch size, prior to leaching.

Anaconda's Weed Heights plant in Nevada is a recent vat leaching installation. Here, the crushed ore is loaded in to one of eight vats, each holding a charge of 12300 dry tons. The tanks measure 120 ft. by 135 ft. by 19 feet and are constructed of reinforced concrete and lined with asphaltic mastic. The ore is dropped into the tanks through a grid of wooden slats. It is bedded in six layers, each about 3 ft. thick, over only one third of the tank area at a time, to ensure that there will be no segregation in the tanks, no complete blanket of fines and therefore no channeling of solution nor complete blockage of circulation.

continued . . . . . .

Copper Dissolution

(c) Acid of 93% strength is fed to the vats along with air-lifted streams of process leach solution. Leach solutions progress from low grade to high grade tanks by gravity and by pumping. Reaction time is 120 hours and recovery is reported at 92%. Bedding time for a 12300 ton charge is 15 hours and unloading time by 8 ton clamshell, is 14 hours. Leached material is hauled to the dumps in 55 ton trucks. Acid consumption is 67 lbs. $H_2 SO_4$ per ton of ore bedded or 4.80 lbs. per lb. of copper precipitated.

In other operations copper extraction is reported in the 60-70% range. Steps taken to improve extraction in vat leaching include de-sliming to prevent sealing of the bed followed by treatment of the slimes in agitators or pachuca tanks.

Tank Leaching employs conventional agitation circuits for treating oxide ores. The capital cost is relatively high which is offset by maximum recoveries. It is generally used as an ancillary to other operations.

(d) L-P-F- (Leaching-precipitation-flotation) utilizes a leaching step as one phase of a sulphide flotation operation in which oxide ores are present. At Kennecot's Hayden plant, the ore is ground and passed through leaching drums, in which the pulp is made acid with sulphuric to 1.5 - 1.7 pH. Treatment time is 10 minutes and acid consumption is 8 lbs. per ton of ore. Dissolved copper is precipitated with sponge iron and then recovered by regular flotation. Sponge iron is made from pyrite which is recovered in milling operations.

(e) Microbiological Leaching utilizes the bacterium, Thiobacillus ferrooxidans (T. Ferrooxidans) for the oxidation of sulphide minerals. The oxidation process is accelerated by the bacteria, which use the energy for growth and reproduction. Environmental conditions for the bacteria are more restricted than for a chemical process. Reaction rates increase as the particle size decreases to 400 mesh. Finer size is not rate-controlling. Environmental controls include pH, oxygen, carbon dioxide and nutrient supply and temperature. The best pH range is between 1.2 and 3.0.

continued . . . . . .

Copper Dissolution

(e) Oxygen is best supplied by intermittent application of leach solution and by creating air currents through the dump or mine. Carbon dioxide supplies carbon which is essential for bacteria growth. Nutrients include phosphate and ammonia which are two most critical components. The optimum temperature is 95° F. Above 104° F bacterial action is inhibited. Bacterial action has been recorded at 37 - 43° F, but at a slow rate.

Bacterial action requires the presence of sulphur to produce the sulphuric acid leaching agent. Rate of copper release is slow and percentage extraction low, but the over-all operation is kept at a profitable level due to the large scale of operations.

COPPER RECOVERY

Having extracted the copper from the ore, it becomes necessary to recover it in a marketable form. This is usually done either by precipitating it as cement copper using iron as a precipitant or by the electrowinning processes. The production of cement copper requires the less costly plant, but this is offset by higher shipping and smelting charges (including recovery losses) and less favourable marketing possibilities. The electrowinning process requires a high capital investment but eliminates all smelting and some shipping charges and commands a preferential market for the product.

Precipitation at Weed Heights is carried out in large launders in cycles of four steps each. In the first, a charge of 80,000 - 90,000 lbs. of scrap iron is bedded uniformly into the launder sections. In the second, pregnant copper solution is pumped through pipes set in the floor, percolating upwards through the iron until level with the upper surface of the iron bed. To assure maximum contact with the iron, the solution is never allowed to flow above the scrap. In the third the section is flooded with fresh water to remove the acid and then drained and excavated. The recovered cement copper is trommeled for further washing and removal of unconsumed iron, tramp material and salts. Finally it is once more drained and dried on gas-fired hot plates. The production of high-grade cement copper, in a coarse, granular product, is best obtained by close control over solution purity, strength, flow rate and distribtion. Of equal importance, is the control of the type, gauge, uniformity and cleanliness of the iron precipitant.

continued . . . . . .

Copper Recovery

Drying is carried out to 15% moisture, on gas-fired hot plates. The fine modules of copper readily oxidize and it is almost impossible to remove moisture without converting metal to oxide.

Precipitation cones are employed by Kennecot for recovering copper from solution. The cones are 14 ft. in diameter by 20 ft. high and hold 15 tons of shredded scrap which is charged by overhead converyor and travelling tripper. The solution is fed from the bottom and injected through nozzles into the mass of iron. Injection of the solution has the effect not only of rapidly precipitating copper, but also of removing the metallic copper from the iron surface, thereby exposing fresh clean iron. The precipitation cone is a continuously operated unit that is self-cleaning. The copper precipitates accumulate on the sloped false bottom of the tank and are discharged inter - mittently with the use of a pneumatically operated valve on a time cycle. Consumption of scrap iron is at the rate of 2 - 2.5 lbs. per lb. of precipitate. The precipitates are de-watered in presses fitted with stainless steel plates and frames. After charging, the precipitate is washed and then blown with air. There is some oxidation in lowering moisture to 12 - 15%.

Electrowinning, a new electrolytic cell has been developed by Continental Copper and Steel Industries. The cell is less sensitive than conventional ones, to impurities in the copper solution so requires less prior cleaning of solutions. Higher current density makes possible lower capital and operating costs. Under lab and pilot plant conditions power consumption varied between 0.85 and 1.42 KWH per lb. of copper, with current efficiencies of 88 to 95 per cent. Copper solutions were reduced from 30 gms. copper per litre to 10 grams per litre compared with a typical drop of 5 gms. per litre.

E. W. Johnson,
Chief Metallurgist.

APPENDIX V

El Paso Natural Gas Company

El Paso, Texas 79999

C. L. PERKINS
SENIOR VICE PRESIDENT

December 24, 1968

Mr. A. J. Perry, Manager
Minerals Division
The Superior Oil Company
6420 East 22nd Street
P. O. Box 12487
Tucson, Arizona 85711

Dear Mr. Perry:

Following the announcement of a copper prospect on our Lakeshore property, an interest in participating in the development of this property was evidenced by many companies such as yours. This interest has continued and has become more intense as meetings have been held with interested parties and detailed data has been made available.

In order to expedite a determination as to the development of this property and consequently commence operation at the earliest possible date, we are now requesting that proposals and plans for participation be submitted in written form not later than January 15, 1969. Information on exploration, sampling, etc. now being conducted will continue to be available until such time.

We will, of course, consider any proposal submitted, however, our general desires, as previously discussed with you, are listed below.

1. A 50% interest to be carried during payout.

2. A percentage of profits during payout.

3. Minimum cash payment in lieu of a percentage of profits would commence after a mutually agreed number of years from the effective date of participation agreement.

4. A cash payment in addition to reimbursement of acquisition and exploration costs incurred to date of participation agreement.

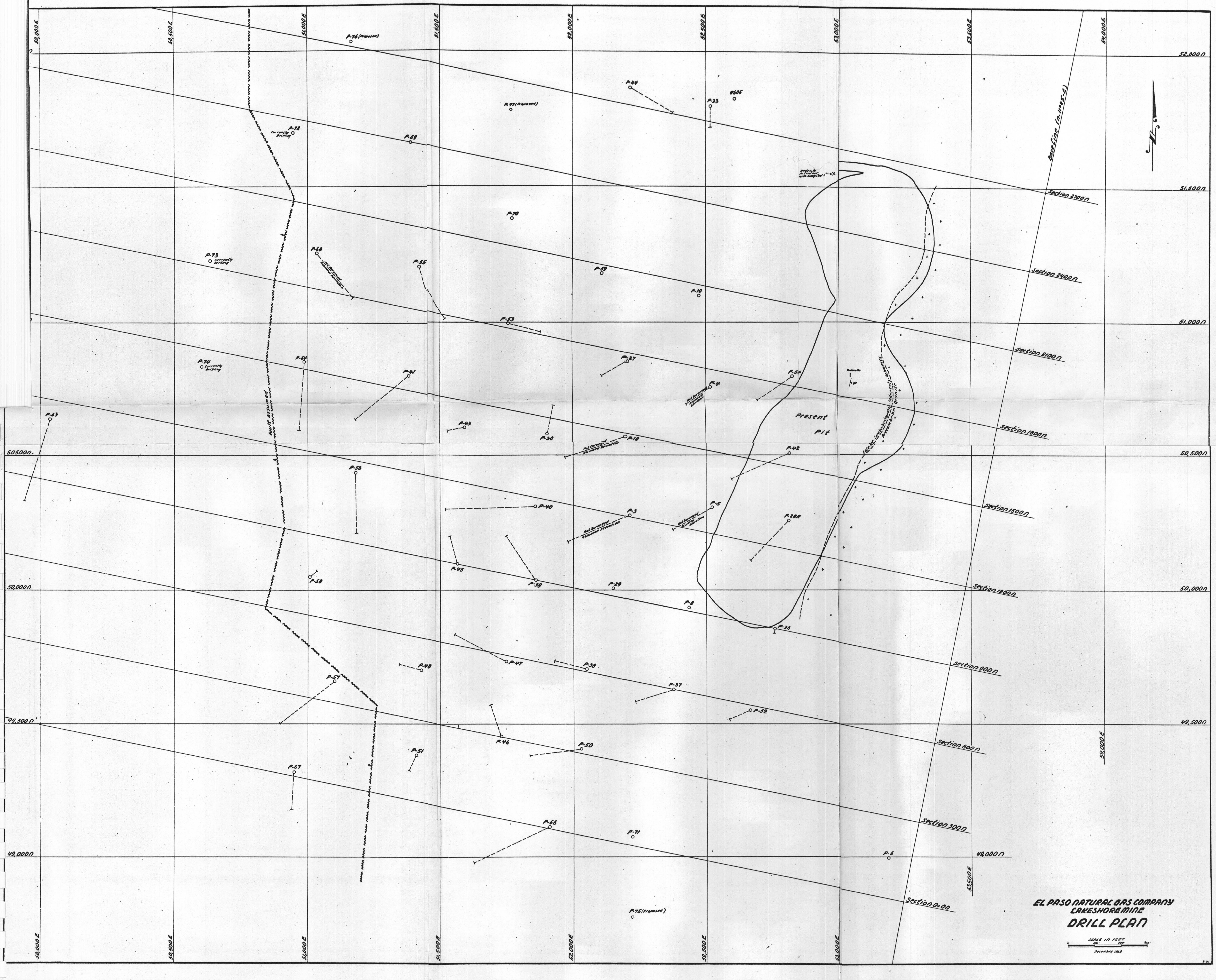
5. A call on up to 40,000,000 pounds of copper per year.

In order for us to make a proper evaluation of your proposal, we also request submittal of your plans concerning future exploration, exploration shaft and bulk sampling, mine development method, mill capacity and installation, smelting of concentrates, and refining and marketing the final product. Your proposal should include copies of your feasibility studies, geological, geophysical, and metallurgical data.

We appreciate the interest which you have shown in our Lakeshore property. After considering all proposals, we will advise you if El Paso Natural Gas Company has any further interest in pursuing your proposal. We reserve the right to reject any and all proposals.

Yours very truly,

C. Perkins


EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
DRILL PLAN
SCALE IN FEET
December 1968
Base Line (N. 11°45' E)
Present Pit
Fault assumed
Section 2700 N
Section 2400 N
Section 2100 N
Section 1800 N
Section 1500 N
Section 1200 N
Section 800 N
Section 600 N
Section 300 N
Section 0+00
52,000 N
51,500 N
51,000 N
50,500 N
50,000 N
49,500 N
49,000 N
50,000 E
50,500 E
51,000 E
51,500 E
52,000 E
52,500 E
53,000 E
53,500 E
54,000 E
P-76 (Proposed)
P-77 (Proposed)
P-75 (Proposed)
P-72 Currently Drilling
P-73 Currently Drilling
P-74 Currently Drilling
P-44
P-33
P-69
P-70
P-68
P-65
P-58
P-10
P-53
P-64
P-61
P-57
P-54
P-4
P-63
P-43
P-30
P-18
P-42
P-56
P-40
P-3
P-5
P-28A
P-45
P-39
P-29
P-8
P-36
P-47
P-48
P-38
P-37
P-52
P-57
P-46
P-50
P-51
P-67
P-66
P-71
P-6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48,000N
47,000N
Section 3900N
Section 3600N
Section 3300N
Section 3000N
Section 2700N
Section 2400N
Section 2100N
Section 1800N
Section 1500N
Section 1200N
Section 900N
Section 600N
Section 300N
Section 0+00
Section 300S
Section 600S
Section 900S
Section 1200S
100'
200'
300'
400'
500'
600'
700'
800'
900'
1000'
1500'
Pit Limit Surface
1800' Approx.
Trace of Bedrock
on Pit Slope.
Reference Line (2400N)
Base Line (N-11°45'-E)
NOTE: The Pit Design allows for a 45° Wall Slope. Later information has indicated the necessity of redesigning to a 32° Slope above the trace line shown.
NOTE:
Elevation Contours above Sea Level
EL PASO NATURAL GAS COMPANY
LAKESHORE MINE
PINAL COUNTY
ARIZONA
PRELIMINARY PIT LAYOUT
SCALE IN FEET
0 100' 200' 400' 600' 800' 1000'