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A FEASIBILITY STUDY

OF THE

EXPLORATION, DEVELOPMENT & OPERATION

OF THE

CALTA MINES, LTD. PROPERTY

AT

CONTACT, NEVADA

By

C. R. Hagen
L. K. Requa

April, 1970

INTRODUCTION

The mining claims of the Calta Mines, Ltd., are located in the Salmon River Mining District, in northern Elko County, Nevada, which is a mining district that was first discovered in 1870.

For an exact description of the Calta Mines property reference should be made to the company maps and files.

Small scale copper mining has been carried on in a sporadic manner since 1906, but mostly at times of high copper prices. Such ores were mostly shipped to the Garfield, Utah smelter of the American Smelting and Refining Company, but this outlet was closed in 1959 when this smelter was sold to the Kennecott Copper Corporation. Since that time the district has been largely inactive, except the exploration by Calta Mines and others starting in 1967.

The Contact copper ores occur in an extensive area that is largely associated with the periphery of a large granodiorite batholith. These ores are found in a series of fissure veins, some of great length. The discovery of a disseminated sulfide deposit has been a hope that has so far not been realized, but may still be found with additional exploration. This report deals with the ore in the fissure veins.

This district is well located for favorable operating conditions and is served by the Rogerson branch of the Union Pacific Railroad and U. S. Highway 93. A high tension power line of the Idaho Power Company passes within a mile of the property, and good motel and restaurant facilities are to be found at Jackpot, which is fifteen miles north of Contact, but there are only poor accommodations at Contact.

The present feasibility study is directed at the following matters:

1. Evaluating the geological and geophysical information and how they influence future exploration for additional ore.

2. Estimating tonnages and copper content of the ores developed by the Calta drilling program.

3. Estimating, on a preliminary basis, the cost of developing and equipping the mine.

4. An investigation of the physical nature of the ore to determine the portion that is non-sulfide and metallurgical tests to arrive at the best mill flow sheet for the recovery of the optimum metal content.

5. Estimating, on a preliminary basis, the cost of buying and erecting an adequate mill at the property.

6. Estimating the cost of mining and milling and the profit arrived therefrom.

7. Surveying the market for the best sales outlet for concentrates and precipitates and making a contract with the most favorable buyer.

8. Reviewing the future trend of the supply and demand for copper and the future trend of copper prices.

The above matters are covered in the following sections of this report and summarized at its conclusion.

GEOLOGICAL FACTORS INVOLVED IN FURTHER EXPLORATION

The ore reserves of this property, as calculated in this report, are 2,148,696 tons with an average grade of 2.190% copper and 0.601 ounces silver per ton that are contained in several large fissure veins. There are reasons to believe that significant additions to this tonnage will result with more drilling.

Previous production from the Contact district has been estimated unofficially at as much as 300,000 tons of ore. The grade has been around 5% copper. One partial estimate for the years 1913-1949 was 34,404 tons with an average grade of 4.85% copper, 0.2 ounces gold and 1.7 ounces silver per ton. At present metal prices the gross value of this ore would be $1,870,000.

In 1928- a series of 129 shipments from the Marshall (Delano) Mine averaged 5.3% copper.

This old production is the best indication of where the better ores occur, and it should be noted that such ores were mined from veins that are near the contact of the granodiorite with the surrounding sedimentary rocks and mostly in the north contact zone that occurs in a zone that starts at old Contact and runs for about four miles to the west. The presently calculated ore reserve lies entirely within this zone.

Also in this zone lies the Palo Alto Mine, which had an early production of 300 tons of 32% copper ore before 1910 and 1,400 tons of 5% copper ore between 1910 and 1928.

Blue Bird ores mined in early days contained from 17% to 35% copper, 20 ounces silver, and $3.50 gold per ton. Bryan ores were stated to average 5% copper, 8 ounces silver and $20 in gold per ton.

The Brooklyn Mine, also on the contact, is said to have had spectacular sulfide ore, of which the only visible proof is a dump said to average 15% copper.

The above mentioned mines contained some of the best ore mined in the district and are located on, or near, the contact. This high-grade ore was largely mined from erratic ore shoots in the sediments, which could open up into more consistent orebodies in the underlying granodiorite. This zone has not been explored except in the old Delano workings where the limestone, slate and quartzite overlie much of the good ore that was mined in this vein in the granodiorite. The extension of this orebody and the North Vein to the east, under the sediments, are two of the best exploration objectives on the property.

Such an exploration project would carry the drilling onto the Palo Alto, Bryan and Blue Bird claims where the previously mentioned high-grade ores occurred in the sediments and where the objective should be the extensions of these veins into the underlying granodiorite.

In 1903 J. T. Bailey stated in an article in Engineering & Mining Journal that there was a vein only $1\frac{1}{2}$ feet wide in the slate on the Delano No. 2 claim which, when followed onto the Palo Alto claim and into the granodiorite enlarged by replacement to a maximum of seven feet.

The exploration, the geophysics, the geological mapping and the drilling that has been done on the Calta property in the past three years have produced a considerable ore reserve, as calculated herein. This work has also furnished excellent information that warrants several conclusions in regards to future exploration, as follows:

1. The Central Granodiorite Batholith: The western extension of this batholith that is contained on the Calta property has been closely explored by a grid of induced polarization lines that was run with the objective of discovering a disseminated sulfide copper deposit.

   No encouraging anomalies were found in the central batholith even though this batholith is laced by a swarm of syenite dikes, many of which have substantial showings of copper carbonates. Therefore, unless some new and encouraging evidence is presented, this section of the batholith can be eliminated from further consideration.

   There was a partial I. P. anomaly developed in the limestone north of the contact zone which will be discussed later.

2. Favorable Ore Zone: Recent exploration and older mining operations definitely indicate that the better and more consistent orebodies are located along the contact zone between the granodiorite and the limestone, slate, quartzite series and that this zone extends from the site of old Contact for about four miles in length to the west.

   There is much ground yet to be explored along this contact, but the most obvious objectives would be to drill for the eastern extensions of the Delano ore shoot and the North Vein ore shoot in Palo Alto, Bryan and Blue Bird ground.

3. Zoning of Ore Shoots: There is a definite vertical zoning of ore shoots in the limestone, slate, quartzite series, where the veins tend to be narrow, but rich, and in the lower granodiorite where the

veins are much wider, either from replacement or supergene enrichment, or both, with widths and values of ore being more consistent.

Except in the Delano Vein the veins in the granodiorite have not been adequately explored under the capping of sedimentary rocks.

4. Supergene Enrichment and Oxidation: These processes play an important role in the contact zone on the Calta property even though it is impossible to define a definite supergene, or secondary sulfide zone. It is quite probable that a rich secondary, or supergene, sulfide zone was formed which was later reoxidized due to a shifting of the water table.

There is a zoning of better grades and widths of ore that is shown by the isograms of the ore shoots that accompany this report.

Oxidation went deeper in the veins that are further to the south of the contact. In the Delano Vein oxidation went to 600 feet below water level, but oxidation in the Brooklyn was shallower with sulfides being encountered at 200 feet in depth due to the fact it was exactly on the contact.

The proximity to the contact seemed to hinder oxidation, and this may be due to a chemical or physical impedance of oxidation, but while the oxidation process is not well understood, its results are and can be reasonably well mapped.

Most of the ore cut in the recent drill holes averaged about 67% oxide, or non-sulfide ore. A few of the holes cut only sulfide, but no true supergene sulfide zone has been definitely established.

5. Rake of Ore Shoots: This is a very important factor to establish as the success of future drilling may depend on it. In the Delano Vein a definite east rake can be seen on the isogram pattern on the longitudinal section of that vein. This rake is approximately minus

20° to 30° to the east. This rake may be controlled by the overlying sedimentary rocks or other factors, but it is a definite feature and indicates that drilling for more ore should be laterally to the east into Palo Alto, Bryan and Blue Bird ground rather than trying to explore the Delano Vein to greater depths with vertical holes.

The eastern extension of the North Vein ore shoot does not have as definite a rake established, but it is about to go under the sedimentary rocks which would be a real reason for a flat, easterly rake. This vein too should be explored to the east on Palo Alto and Blue Bird ground.

6. More Geological Mapping: An excellent geological map was prepared of the Calta property, but at that time the Palo Alto, Bryan, Blue Bird and other adjacent claims were not part of the Calta holdings and were consequently not mapped. It is quite essential to have this area mapped if more drilling is contemplated.

   The time required for such a mapping job is estimated at one month and the cost at $4,000.

The discussion of geological factors as related to future ore discovery will be limited to the four-mile zone already described, although many good prospects exist to the west of this zone. Also the contact deposits on the Allan No. 2 claim will not be considered for the present although they present certain features that warrant close study.

Specific Drilling Objectives

Drilling objectives that can be considered are--

Delano Vein:

This ore shoot has not been delimited on its eastern edge where the granodiorite is under the sediments. Several drill holes on Palo Alto ground are warranted to explore for this ore shoot on its flat rake. If such drilling is successful, drilling should be extended eastward onto the Bryan or Blue Bird ground.

North Vein:

The eastern extension of this ore shoot too is logical. It is not cut off by Drill Holes 15A and 15B. An error in assaying, recently discovered, shows that there is $2\frac{1}{2}$ feet of 6.98% copper ore in Hole 15A instead of 1.7% copper as first reported.

No drilling has been done on this vein where it goes under the sediments to the east, and there should be good possibilities for ore if a flat rake is allowed for. Two drill holes are warranted as suggested.

Brooklyn:

At least two drill holes should be drilled to explore this vein at depths of 300 feet. No drilling has been done on this ore shoot which is reputed to contain the best sulfide ore in the district.

Bellevue:

There has been only one drill hole drilled into this ore shoot below the previously known one. This hole was very encouraging and showed 4 feet of 4.4% copper ore. At least two more holes are warranted to better define the pattern of this ore shoot.

To explore these objectives the following drilling is suggested:

Eastern Extension of the Delano Ore	
2 Holes each 700 ft. deep	1,400 ft.
Eastern Extension of the North Vein Ore	
3 Holes each 600 ft. deep	1,800 ft.
Brooklyn Vein	
2 Holes each 400 ft. deep	800 ft.
Bellevue Vein	
2 Holes each 500 ft. deep	1,000 ft.
Total Suggested Drilling	5,000 ft.

At an estimated cost of $11 per ft., this program would cost $55,000.

The above drilling does not include any deep drilling that might be warranted after further induced polarization studies have been made.

ANALYSIS OF ALL INDUCED POLARIZATION SURVEYS AT CONTACT

The induced polarization surveys at Contact dated from May, 1967, when Heinrichs Geoexploration Co., ran a series of I. P. lines over the area. This work was done for Mr. Paul Kayser who required only verbal reports on results. However, Heinrichs did submit a rather rudimentary written report to the Calta Mines staff on December 11, 1967, and this has been available for the purposes of this report.

Of the fourteen lines that Heinrichs ran at that time, three were on the east side of the river on the Stewart claims and had no bearing on the Calta property. Eight I. P. lines were across the central portion of the batholith on what is called the Blue Jay area, and three lines, No's. 12, 12A and 13, were on the Banner (Calta Mines) property. Of all these lines only these last three showed anomalous results. The other lines were all negative and for this reason it has been concluded that this central part of the batholith contained no disseminated sulfide deposits.

A drill hole, BK-1, was drilled where lines 12 and 12A cross and where an anomaly was indicated, and some fissure ore was found, but no disseminated sulfides.

This early I. P. work indicated that, in the Contact area, this technique could not consistently outline fissure vein ore, most of which was oxidized, but it did give the first indication of the weak anomaly in the limestone at the end of line 12A. This weak anomaly is yet to be fully outlined and defined, but with the later I. P. results, it assumes some importance.

Another I. P. survey was run between July and September, 1967, by the Heinrichs organization for the Circuit Foil Corporation, who held claims adjoining Calta ground. Now these Circuit Foil claims are encorporated in the Calta holdings. The report that Heinrichs made at that time to Circuit Foil is available and was reviewed for this analysis.

At that time Heinrichs was the first to substantially indicate a weak anomalous zone on the north end of the property, but did not fully delimit the anomaly as it appeared to be running off the property. This north area was later taken up by three tiers of Liz claims for Calta.

Heinrichs lines 3, 4, 5 and 6 show this anomaly in part but did not define the north limit. Their lines 8 and 9 also show weak anomalies. From these results Heinrichs recommended further I. P. detailing (which was not done). They estimated that the west end of this anomalous zone was 500 feet deep and the east end 1000 feet deep.

Between August and October, 1969, the Mineral Surveys, Inc., ran an I. P. grid over the Banner ground and adjacent claims. In their report, dated November 7, 1969, they said that background count was determined to be about 4.0 milliseconds and that the only place where this was significantly exceeded was on the north ends of their lines 0 + 00, 8 E and 8 W where the maximum reading was 9.5 milliseconds.

Mr. Parker Gay, Jr., the local manager of Mineral Surveys estimates that the depth of their I. P. probing did not exceed 900 feet and also that it would be possible to increase the depth of their I. P. probing to as much as 1,500 feet. He thought that such additional work might be warranted.

Conclusions

It is fairly well established that the central portion of the Contact batholith holds little promise for disseminated copper ores and that the ore potential of this district is closely associated with the periphery of the Contact batholith and not with its central part despite the copper showings there.

All forms of ore deposits may occur on this peripheral zone of the batholith, including disseminated copper ores, but any ore deposits developed will probably require underground mining.

As the only consistent I. P. anomaly found was in the limestone, any further I. P. work should be confined to that area and should be so set up that depths of 1,500 feet will be probed.

Of this situation Heinrichs says:

"A halo of mineralization is a possible cause of the long band of induced polarization anomalism near

the north edge of the claim group and roughly paralleling the igneous contact inferred from the aerial photograph. Since the anomaly was not completely crossed, it is impossible to say how wide this anomalous zone is or how deep the polarizable material may be. The data on hand indicate that it is less deep on the west, possibly less than 500 feet deep, but appears to be at least 1,000 feet deep on the east side and very likely entirely north of the claim group."

Heinrichs goes on to mention that they tested one suspicious limestone for graphite content, but with negative results. They do caution that further studies in this regard should be made as graphite can cause definite I. P. anomalies.

Heinrichs concludes with the statement,

"Should no positive explanation of the anomaly be obtained by surface examination, we recommend at least four more spreads of 1,000 foot dipole I. P."

The Mineral Surveys report, while recording this weak anomaly, has little to say about this feature. It should be noted that the Heinrich work on the west end and the Mineral Surveys work on the east end all seem to combine to form a continuous weak anomalous zone, which has been indicated on the attached map as "Suggested sulfide zone." The extent, nor the depth of this zone, have not been completely defined, and it would seem that such additional exploration work is worthwhile.

Such an anomalous zone, if found to persist with further I. P. work, could mean any of the four following situations:

1. A disseminated sulfide deposit in porphyry under the limestone.

2. A contact metamorphic sulfide deposit between granodiorite and limestone.

3. A sulfide replacement deposit in limestone.

4. A pyrite or graphite deposit which would be of no value.

The Calta Mines geological maps do not extend as far north as the "Suggested sulfide zone" and further reconnaissance geological mapping seems indicated.

Recommendations

1. No additional claims need be staked at this time as the anomalous zone appears well covered.

2. A reconnaissance geological map should be made of the anomalous area with particular attention being paid to any occurrence of graphitic limestone.

3. If no graphitic limestone is detected, then additional I. P. lines should be run which will probe to depths of at least 1,500 feet. A few reconnaissance I. P. lines should first be run across the east and west ends of the anomalous, or "Suggested sulfide zone" and, if these lines are encouraging, a detailed grid of deep I. P. lines should be accomplished.

4. If Calta Mines is interested in furthering this program, it should be broken down into several phases and costs of all phases estimated.

5. The ultimate of such a program would be drilling, which should be initiated with several pioneer holes.

Costs

The time required for a reconnaissance geological map of the suggested anomalous area is one month and the cost is estimated at --

The time required for an induced polarization survey of the suggested anomalous area is three weeks and the cost at $6,000.

If this further I. P. work indicates that an attractive anomaly exists, then pioneer drilling will be warranted on the following basis:

Two 600 foot holes	1,200 feet
Two 1,000 foot holes	2,000 feet
	3,200 feet

At an estimated cost of $11 per foot the total cost would be $35,200.

If this pioneer drilling is successful, more drilling will be needed, but this cannot be estimated on present information.

ESTIMATE OF ORE RESERVES

CALTA MINE

CONTACT, NEVADA

This estimate of the ore reserves is made for the Calta Mines, Ltd., at the request of Mr. Kenneth Chattin. The data used were taken almost entirely from the very excellent compilation of data obtained from exploration results at the property of the Calta Mines, Ltd., at Contact, Nevada. This compilation was made by the Cordilleran Engineering Ltd., 1418 - 355 Burrard Street, Vancouver 1, B. C., under the direction of J. W. Stollery, a professional engineer of British Columbia, Canada.

In addition to the above report by Mr. Stollery, reports of several other engineers have been closely examined; nearly all are credible and of authentic conditions existing at the time of their report.

In addition to the information and data of the Stollery compilation of facts of the drilling the reports of the following engineers have been read and studied. Although these reports did not enter into the calculation of the ore reserves, they were extremely helpful to give a background of the conditions that prevailed at the time of their respective reports. Therefore I wish to acknowledge the following reports: M. F. Hickey, April, 1906; W. E. Dunkle, April, 1923; Arthur H. Lawry, January, 1930; S. V. Kemper, January, 1930; F. C. Schrader, Bulletins 497 and 847, 1930 and 1935; S. A. Taylor, Utah Construction Co., November, 1966; Frank Lewis, I. P. work by Heinrichs for Circuit Foil, August, 1967; W. J. Franklin and L. K. Requa, March, 1968; F. L. Croteau, June, 1968, July, 1969; W. F. Bondar, Geo Chemical, August, 1968; and Talis Kalnin, Trenches, December, 1968.

Mining investors are coming more and more to the engineer for sound advice because such advice can be given within certain limits. It is to the engineer's interest to give his best advice and to protect the investor so that the industry, which concerns his own life's work, may be of honorable repute and that capital may be forthcoming to provide the necessary work to place the property in a profitable position.

Therefore to the investor, valuation is the knowledge of the sum of the following factors:

1. Determination of the average metal content of the ore;
2. Determination of the quantity of the ore;
3. Prospective value;
4. Recoverable percentage of gross values;
5. Price of metals;
6. Cost of production;
7. Amortization of capital and interest;
8. Valuation without ore indicated in report; and
9. General conduct in the examination of old reports.

Sometimes the engineer can make even a short visit to the mine for data purposes--more often he cannot. In the former case he may resolve for himself an approximation of factors bearing upon value, excepting the quality of the ore.

Reports of other engineers are to be studied for their value, which are accepted in the ratio of the estimator's confidence in the men who presented the reports. This, then becomes a case of judgment of his own experience, of men, reports and things, and for this, no rule applies.

The ore deposits of copper may be considered to have four horizons, explained as follows:

1. The zone immediately below the surface outcrop where the main feature of the zone is the oxidation and leaching of the soluble minerals.
2. Under the first zone is a lower zone, still in the zone of oxidation, where the main feature is the deposition of the metals as native and the oxides and carbonates of copper.

3. Below zone two is the upper horizon of the sulfide zone where the special feature is the enrichment of secondary deposition and sulfides. The best values usually are in this zone.

4. The lowest horizon is where the deposition is in the primary sulfide state and undisturbed.

The above zones are seldom sharply defined but may be recognized in a general manner by the knowledge of which mineral is where and by the increase and decrease of metal values. No attempt has been made to define the zones in this report; but it was noticed that the maps and sections of the old reports show that a high percentage of the stope area of the Bellevue, Dunkle and Delano workings indicate a continuation of vein deposition laterally and in depth. The values shown, as sampled in the lower levels of the respective mines, show values in excess of the values obtained by diamond drilling as a rule. One instance is the penetration of the North vein by drill hole N-10, which is merely in agreement with the cited values of the ore on the lower level of the Bellevue Mine. In cases where values of width of the vein and copper values were cited as in the old reports, such information, when compatible with Mr. Stollery's factual report, was an influence but not averaged with the data of the Stollery report.

The copper ore deposits of the Contact area are of three types.

1. The veins found in the seyenite dikes, whose dip and strike are confined within the dike, are of least importance as of this date.

2. The contact metamorphic veins--the Brooklyn Mine and other mines along the limestone-granodiorite contact are examples.

3. Fissure veins--this report consists of metals in veins, namely, the North vein, the Delano vein, the Delano Split vein, and the four South veins. The South veins did not show sufficient mineral to be included in this report, although shows promise of possible ore deposition.

Rough estimates of the cost of mining, milling and amortization indicated that ores having a vein width of five feet, or expanded to five feet with an assay value of 2.00% copper, or near its equivalent, should be the target of the estimate, which is considered as conservatively adequate at the present time. If necessary, a lesser amount of higher-grade ore is possible to estimate.

Examination of the old reports and the later drilling program proved that the best "new" ore will be found below the presently deepest workings of the Bellevue, Dunkle, and Delano mines. It is suggested that below the deepest workings is to be found the zone of enrichment.

From long-held definitions of the classification of ores, the most commonly held definitions are as follows:

1. PROVED or POSITIVE ORES

   Ores where there is practically no risk of failure of extraction or continuity. Usually ore blocked out on four sides--this is sometimes called developed or blocked-out ore.

2. PROBABLE ORES

   Ores where there is some risk, yet a warrantable justification for the assumption of continuity--ore on two sides as a usual requisite.

3. POSSIBLE ORES

   Ore where the whole or part is below the range of knowledge and/or without proof of existence.

For the classification of this estimate, under the strict terms above, this ore has been developed by diamond drill holes. Therefore its existence and copper content at that small area of penetration spot on the vein confirm the existence of the vein at the depth or elevation being the same as that vein of the outcrop and the mine workings, and that it is but the extension of knowledge previously accumulated. The ore of this estimate cannot be called Positive ore and cannot be called Possible ore. It can probably be best named PROBABLE MINABLE ORE for want of a better name.

At best an ore estimate is based solely on the information gained by drilling, however accurate, old reports, secondhand production data, tons and assays. Such an estimate, of course, can only be approximate or indicative made from the best information available.

The specific gravity of the ore in place was determined to be 2.676, weighing 167 lbs. per cu. ft., or 12 cu. ft. per a 2000 pound ton. This was determined by using the average specific gravity found in several handbooks and in Herbert C. Hoover's Principles of Mining, as follows:

	Specific Gravity	Lbs. /Cu. Ft.	Cu. Ft. /Ton	
Granite *	2.6122	163 lbs.	12.30	
Diorite *	2.7564	172 lbs.	11.73	
Granodiorite	2.6761	167 lbs.	12.00	Average of above (ore)
Schrader	3.1251	195 lbs.	10.25	
Croteau	3.2051	200 lbs.	10.00	

The selection of 12 cu. ft. per ton was made to be more on the conservative side of estimating.

Then the amount of ore was calculated by the prismoidal formula ordinarily used--

$\frac{TWLD}{C}$ = tonnage of a prismoid, where

TW = true width of ore in feet (width x sine of intersect angle between vein dip and hole dip)

L = horizontal length of block

D = dip length of ore run (Difference of elevation divided by sine of dip.)

C = number of cubic feet per ton

$V_x \frac{TWLD}{C}$ = total metal content.

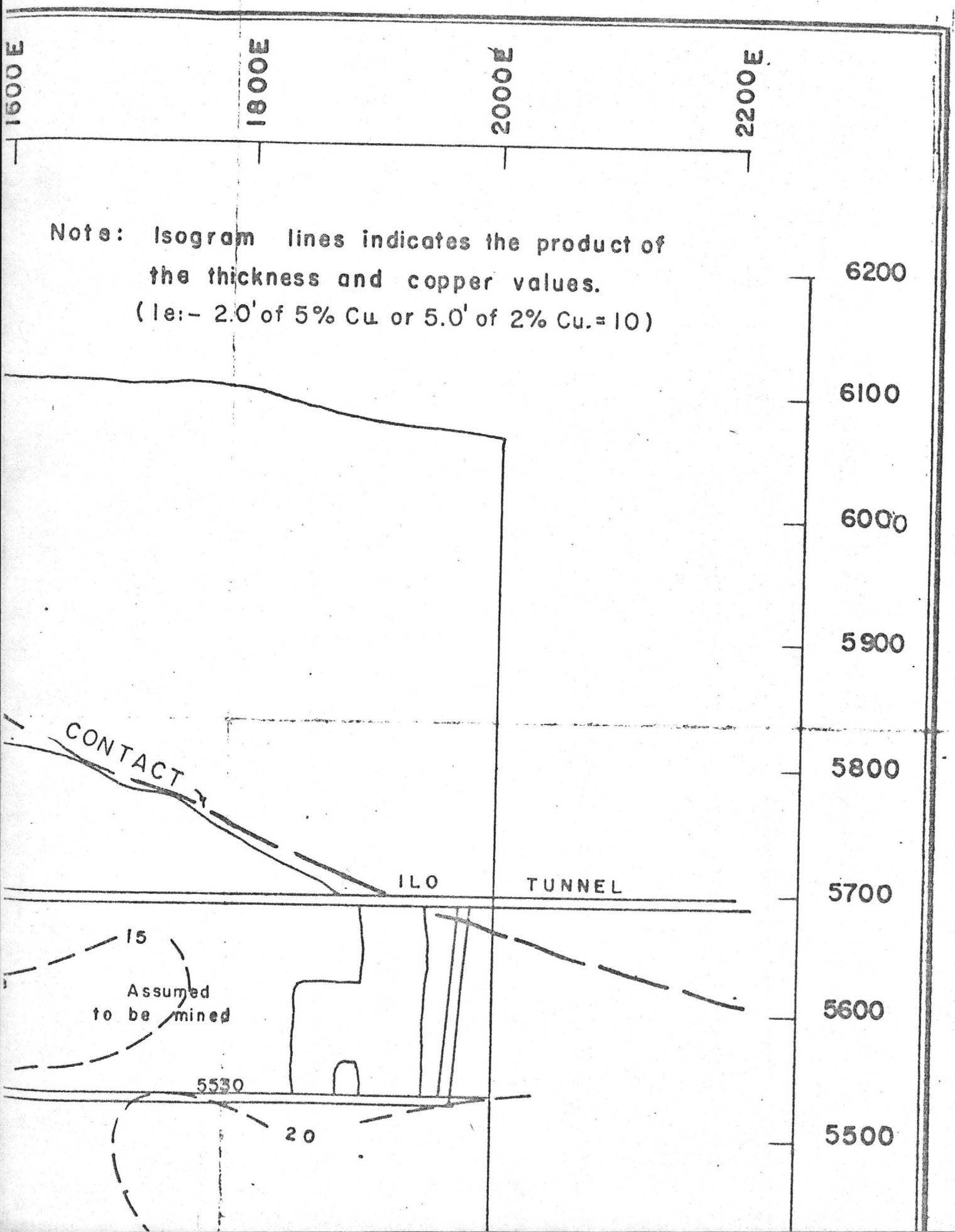
* Average cubic feet per ton of granite and diorite.

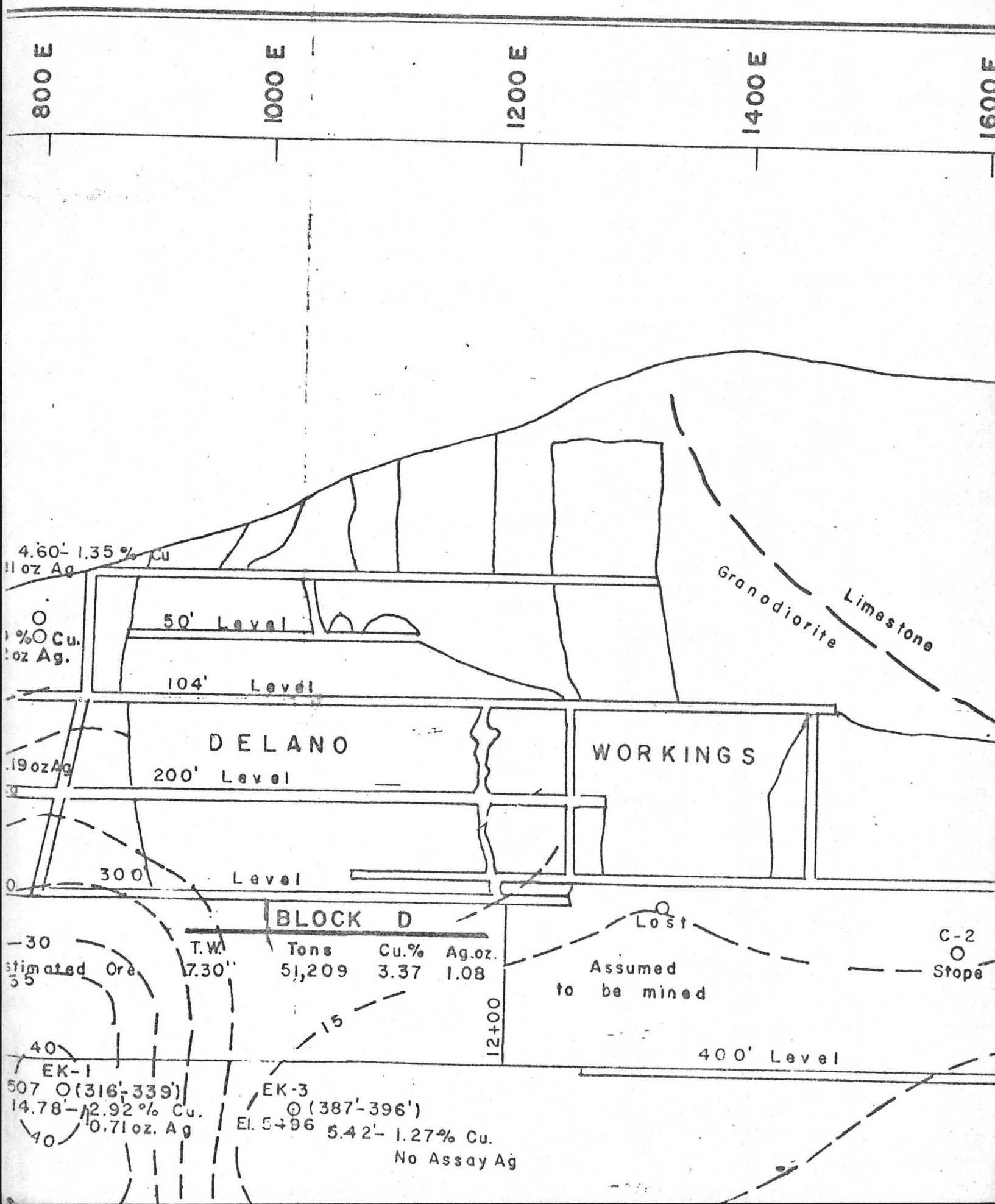
The following tables are a compilation of data required to determine tonnages and ore values:

All true thicknesses averaged arithmetrically.

All assays are weighted when averaged.

$$\text{Tons Ore} = \frac{(\text{True Width}) \times (\text{Horizontal Dist.}) \times (\text{Dip Length or Fissure Run})}{12 \text{ (cu. ft. per ton)}}$$


1600E
1800E
2000E
2200E
Note: Isogram lines indicates the product of the thickness and copper values.
(Ie:- 2.0' of 5% Cu. or 5.0' of 2% Cu. = 10)
6200
6100
6000
5900
5800
5700
5600
5500
CONTACT
ILO TUNNEL
15
Assumed to be mined
5530
20


800 E
1000 E
1200 E
1400 E
1600 E
4.60'- 1.35 % Cu
11 oz Ag
50' Level
104' Level
Granodiorite
Limestone
DELANO
WORKINGS
200' Level
300 Level
BLOCK D
T.W.
Tons
Cu.%
Ag.oz.
7.30'
51,209
3.37
1.08
Lost
Assumed
to be mined
C-2
Stope
12+00
400' Level
EK-1
14.78'- 2.92 % Cu.
0.71 oz. Ag
EK-3
(387'-396')
El. 5+96
5.42'- 1.27% Cu.
No Assay Ag

E00W 200E 400E 600E

BLOCK B

Estimated Ore:

T.W.	Tons	Cu.%	Ag.oz
2.31	34,251	0.79	0.15

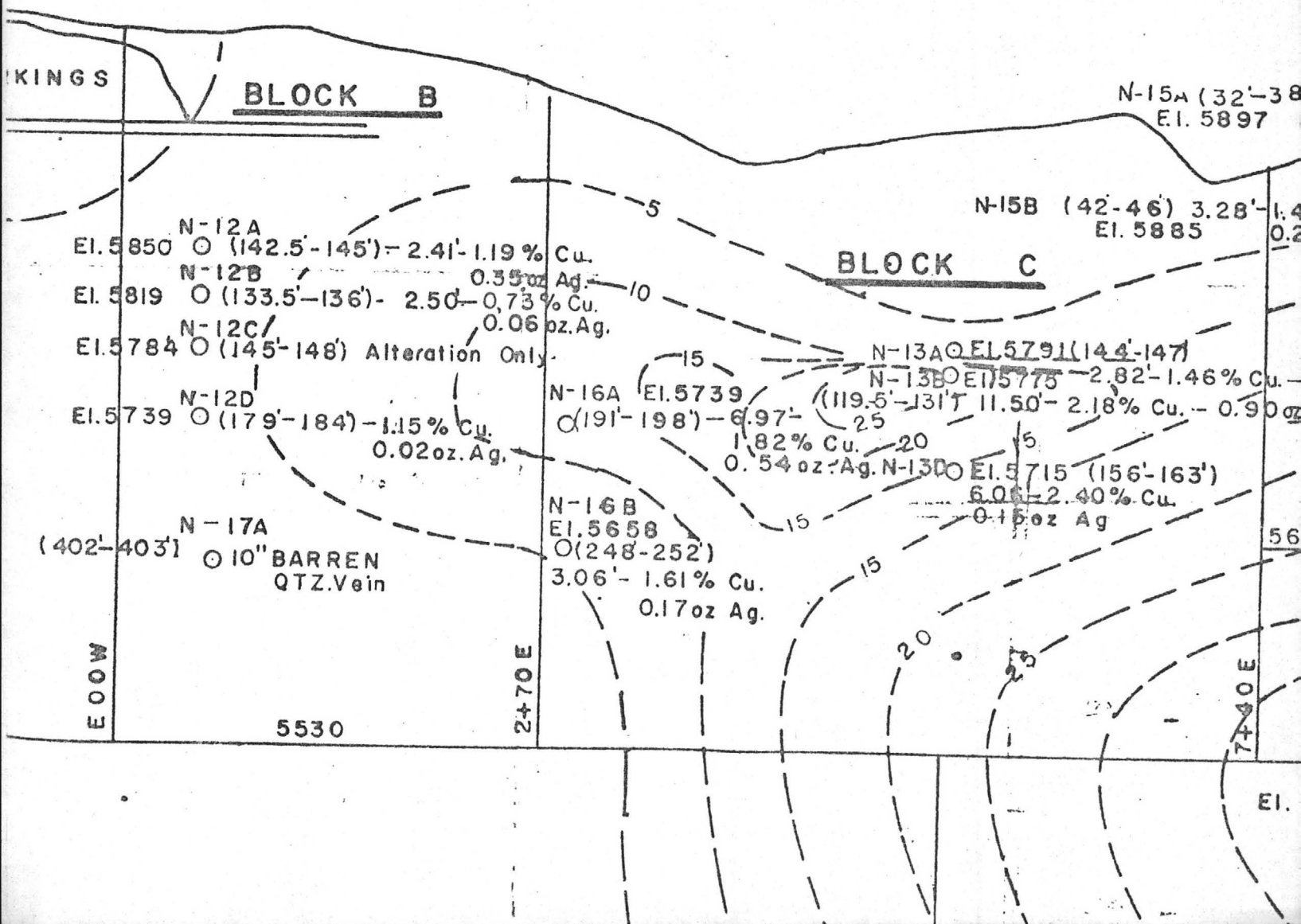
Expand to 5.0':

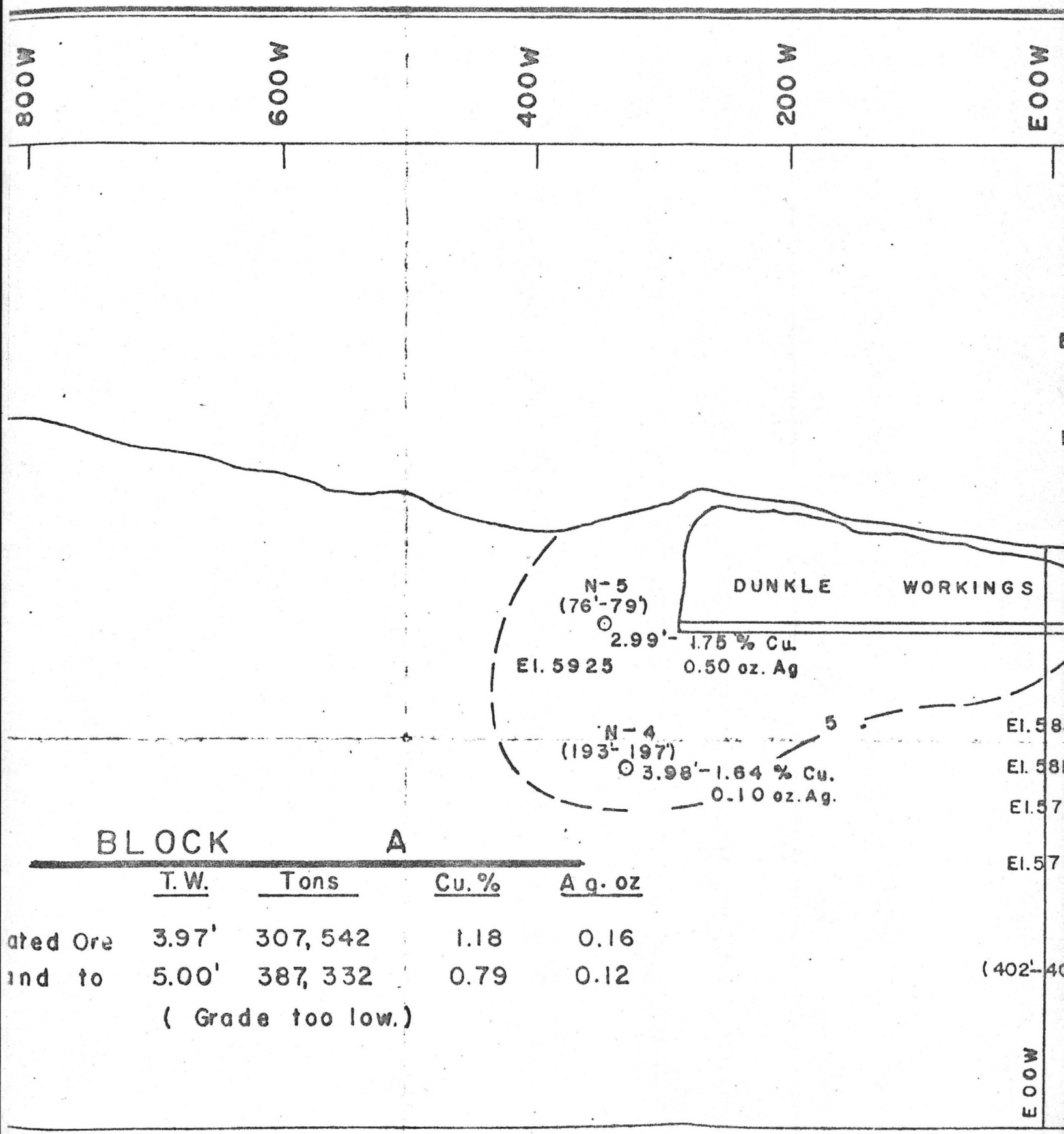
— 74,136 0.36 0.07

(Grade too low.)

BLOCK C

	T.W.	Tons	Cu.%	Ag.oz.
Estimated Ore	5.70'	147,345	2.31	0.81

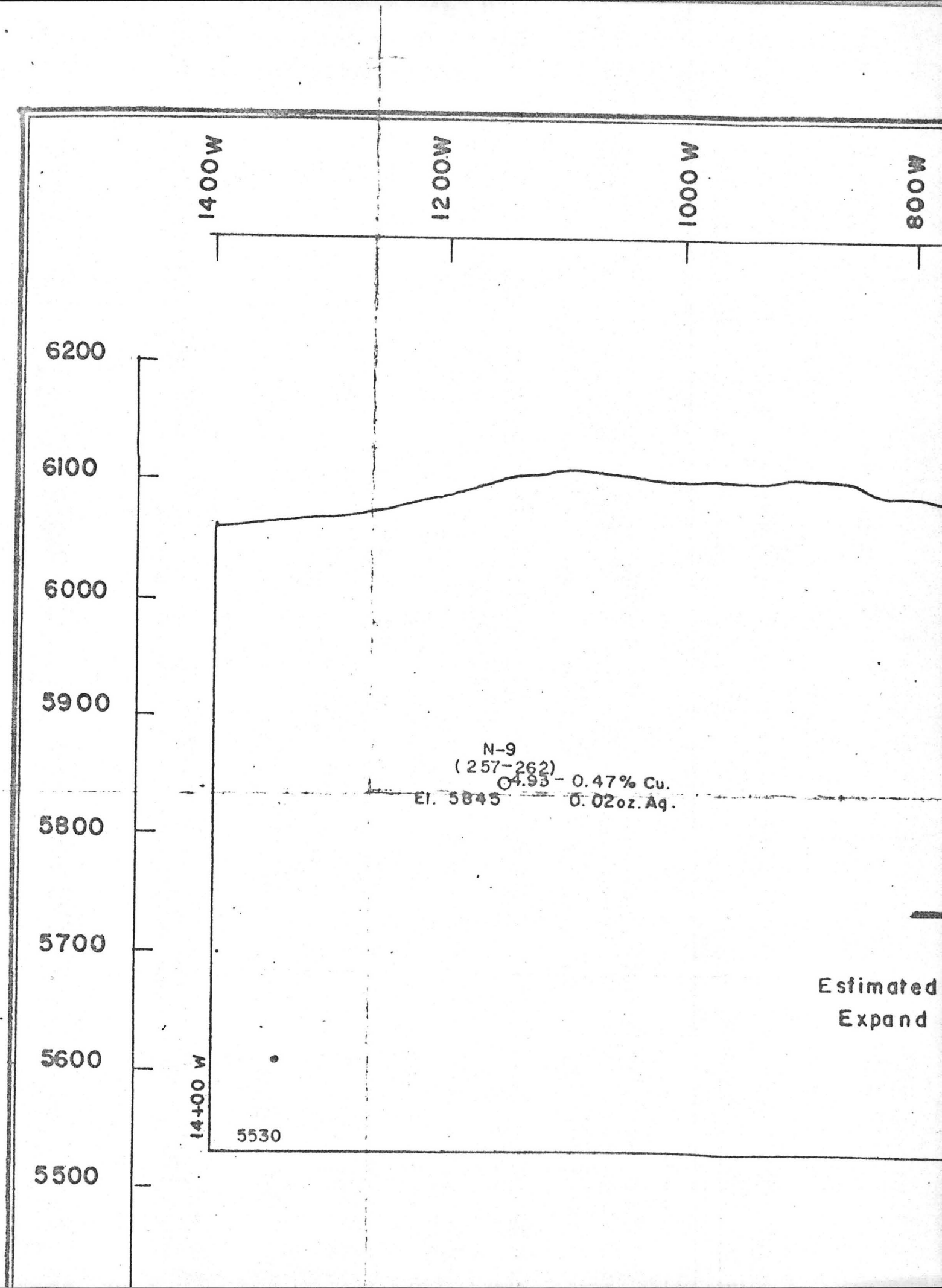


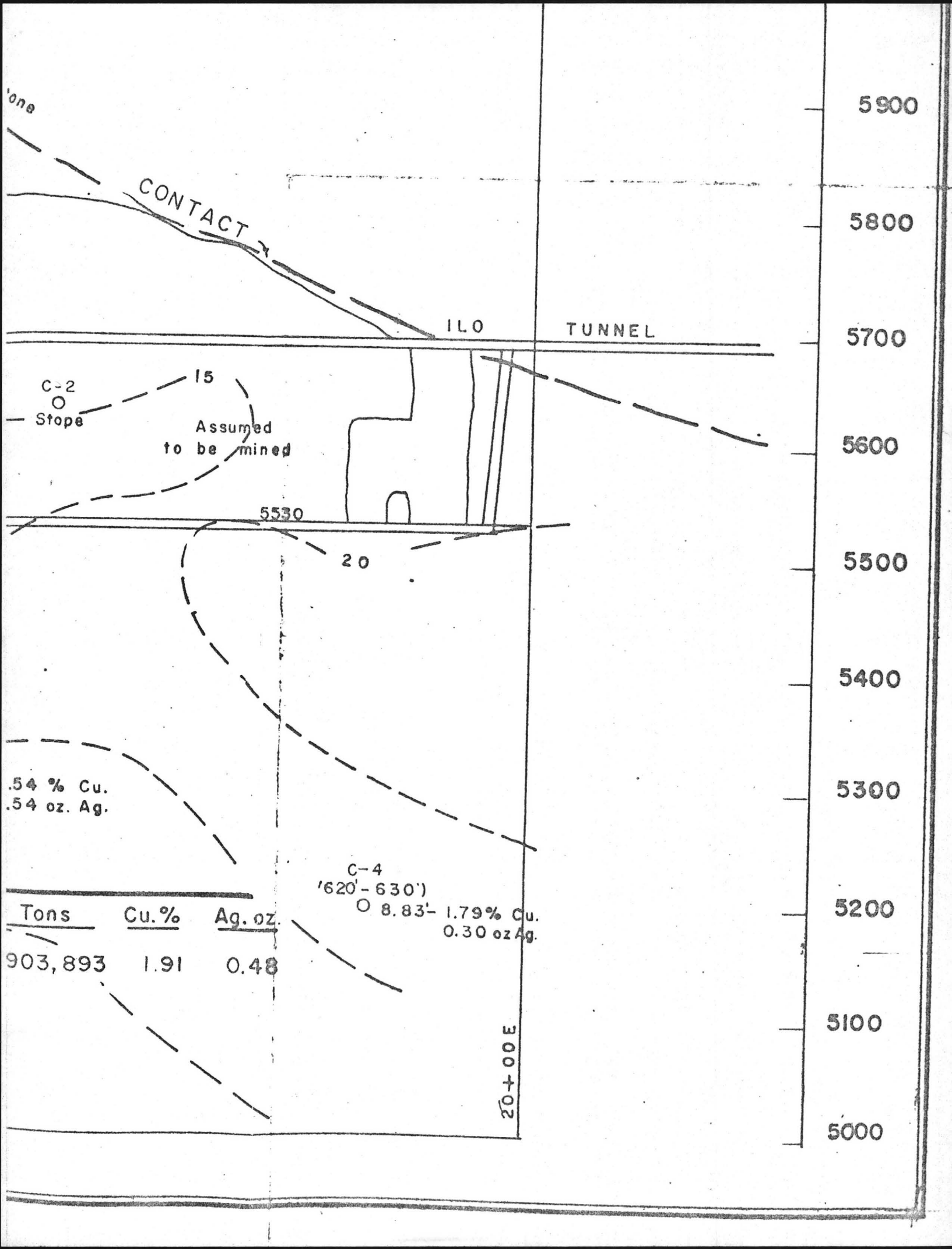


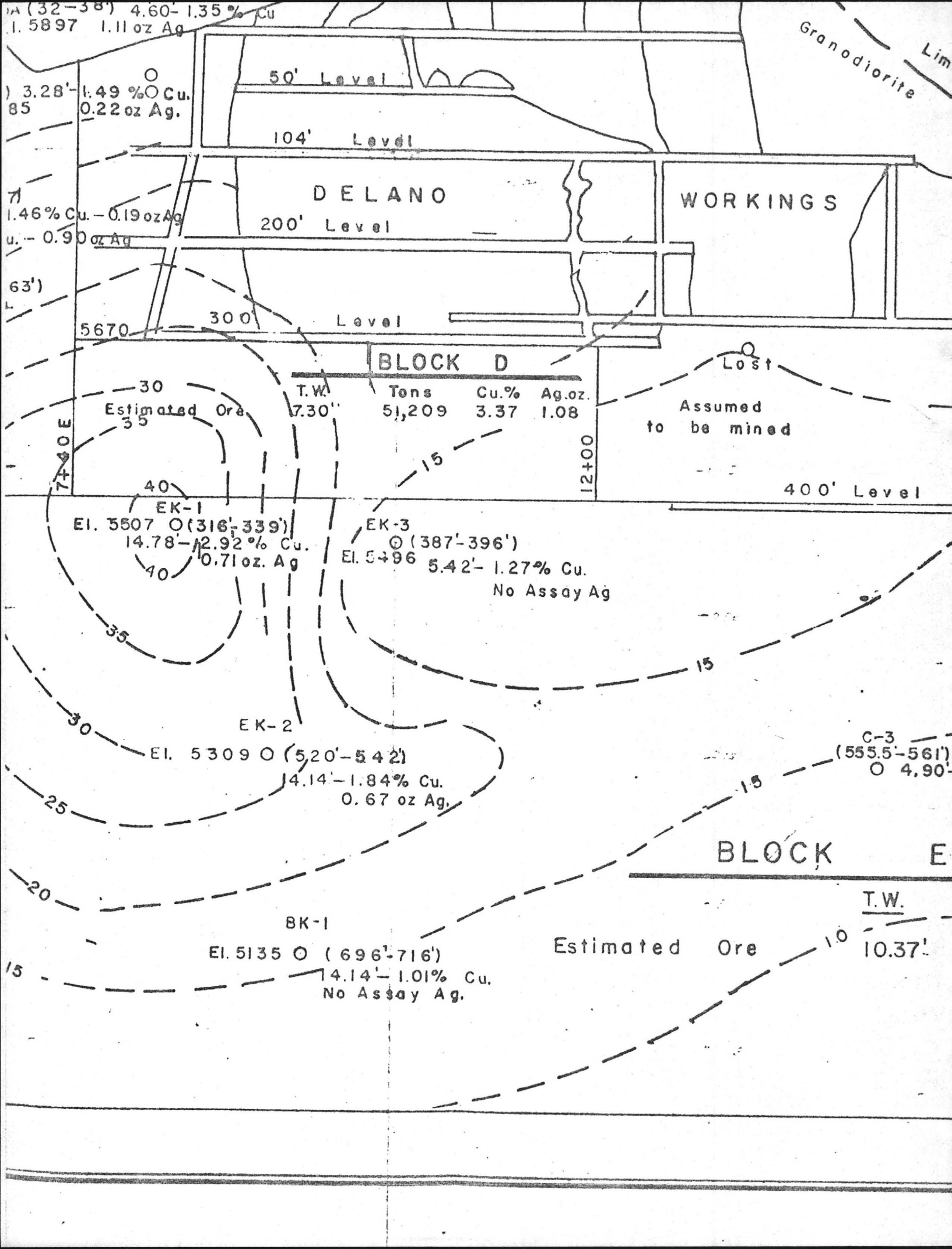
BLOCK A

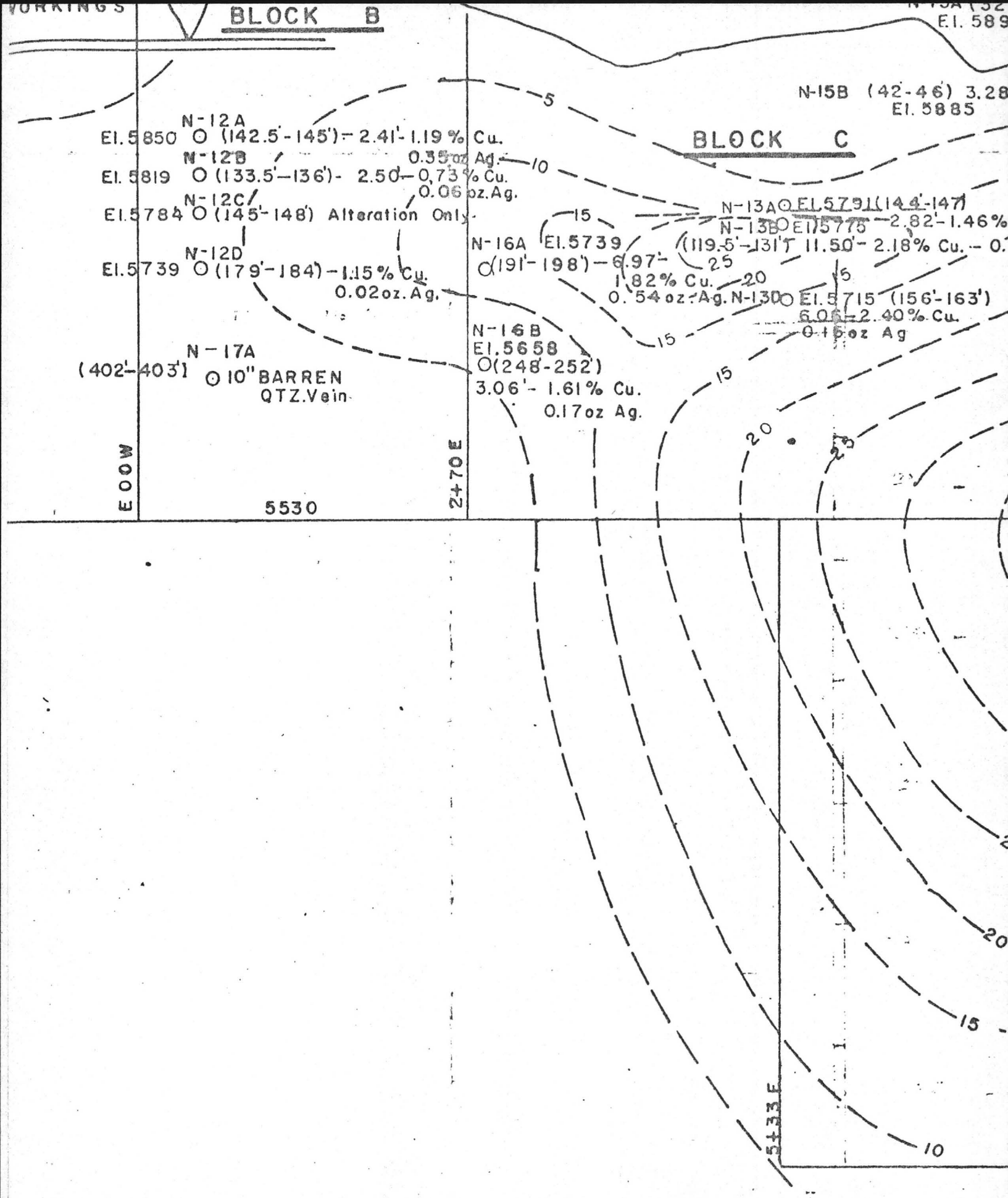
	T.W.	Tons	Cu.%	Ag. oz
ated Ore	3.97'	307,542	1.18	0.16
and to	5.00'	387,332	0.79	0.12
	(Grade too low.)			

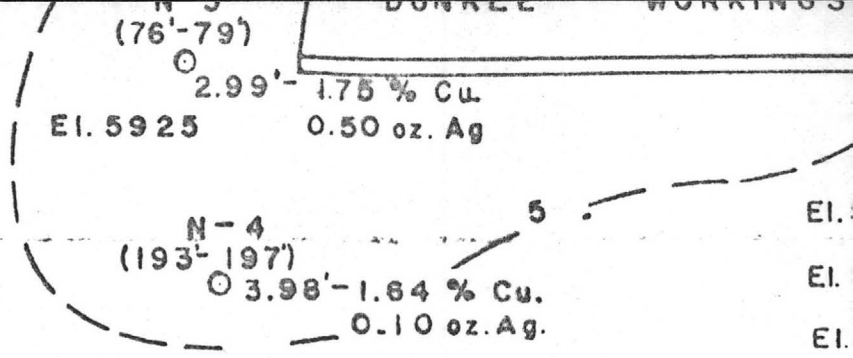
LONGITUDINAL SECTION


1400W
1200W
1000W
800W
6200
6100
6000
5900
5800
5700
5600
5500
N-9
(257-262)
4.93 - 0.47% Cu.
El. 5845
0.02oz. Ag.
Estimated
Expand
14+00 W
5530


'one
CONTACT
ILO
TUNNEL
5900
5800
5700
5600
5500
5400
5300
5200
5100
5000
C-2
Stope
15
Assumed
to be mined
5530
20
.54 % Cu.
.54 oz. Ag.
C-4
'620'-630')
8.83'- 1.79% Cu.
0.30 oz Ag.
Tons
Cu.%
Ag. oz.
903,893
1.91
0.48
300+02E


(32-38') 4.60-1.35% Cu
1.5897 1.11 oz Ag
Granodiorite
Lim
50' Level
3.28'-1.49% Cu.
85 0.22 oz Ag.
104' Level
DELANO
WORKINGS
1.46% Cu.-0.19 oz Ag
u.-0.90 oz Ag
200' Level
63')
300 Level
5670
BLOCK D
Lost
T.W.
Tons
Cu.%
Ag.oz.
30
Estimated Ore
7.30'
51,209
3.37
1.08
Assumed
to be mined
35
7+40E
12+00
15
40
EK-1
El. 5507 (316'-339')
14.78'-2.92% Cu.
0.71 oz. Ag
EK-3
(387'-396')
El. 5496
5.42'-1.27% Cu.
No Assay Ag
400' Level
35
15
30
EK-2
El. 5309 (520'-542')
14.14'-1.84% Cu.
0.67 oz Ag.
C-3
(555.5'-561')
4.90'
25
15
BLOCK E
20
T.W.
BK-1
El. 5135 (696'-716')
14.14'-1.01% Cu.
No Assay Ag.
Estimated Ore
10
10.37'
15


WORKINGS
BLOCK B
BLOCK C
N-15B (42'-46') 3.28
El. 5885
N-12A
El. 5850 (142.5'-145') - 2.41' - 1.19% Cu.
0.35 oz Ag.
N-12B
El. 5819 (133.5'-136') - 2.50' - 0.73% Cu.
0.06 oz. Ag.
N-12C
El. 5784 (145'-148') Alteration Only.
N-12D
El. 5739 (179'-184') - 1.15% Cu.
0.02 oz. Ag.
N-13A El. 5791 (144'-147')
N-13B El. 5775 - 2.82' - 1.46%
(119.5'-131') 11.50' - 2.18% Cu. - 0.
N-16A El. 5739
(191'-198') - 6.97' -
1.82% Cu.
0.54 oz. Ag.
N-13C El. 5715 (156'-163')
6.06' - 2.40% Cu.
0.16 oz Ag
N-16B
El. 5658
(248'-252')
3.06' - 1.61% Cu.
0.17 oz Ag.
N-17A
(402'-403')
10" BARREN
QTZ. Vein
E 00 W
5530
2+70 E
5+33 E
5
10
15
20
25



BLOCK A

	T.W.	Tons	Cu. %	Ag. oz
mated Ore	3.97'	307,542	1.18	0.16
pand to	5.00'	387,332	0.79	0.12

(Grade too low.)

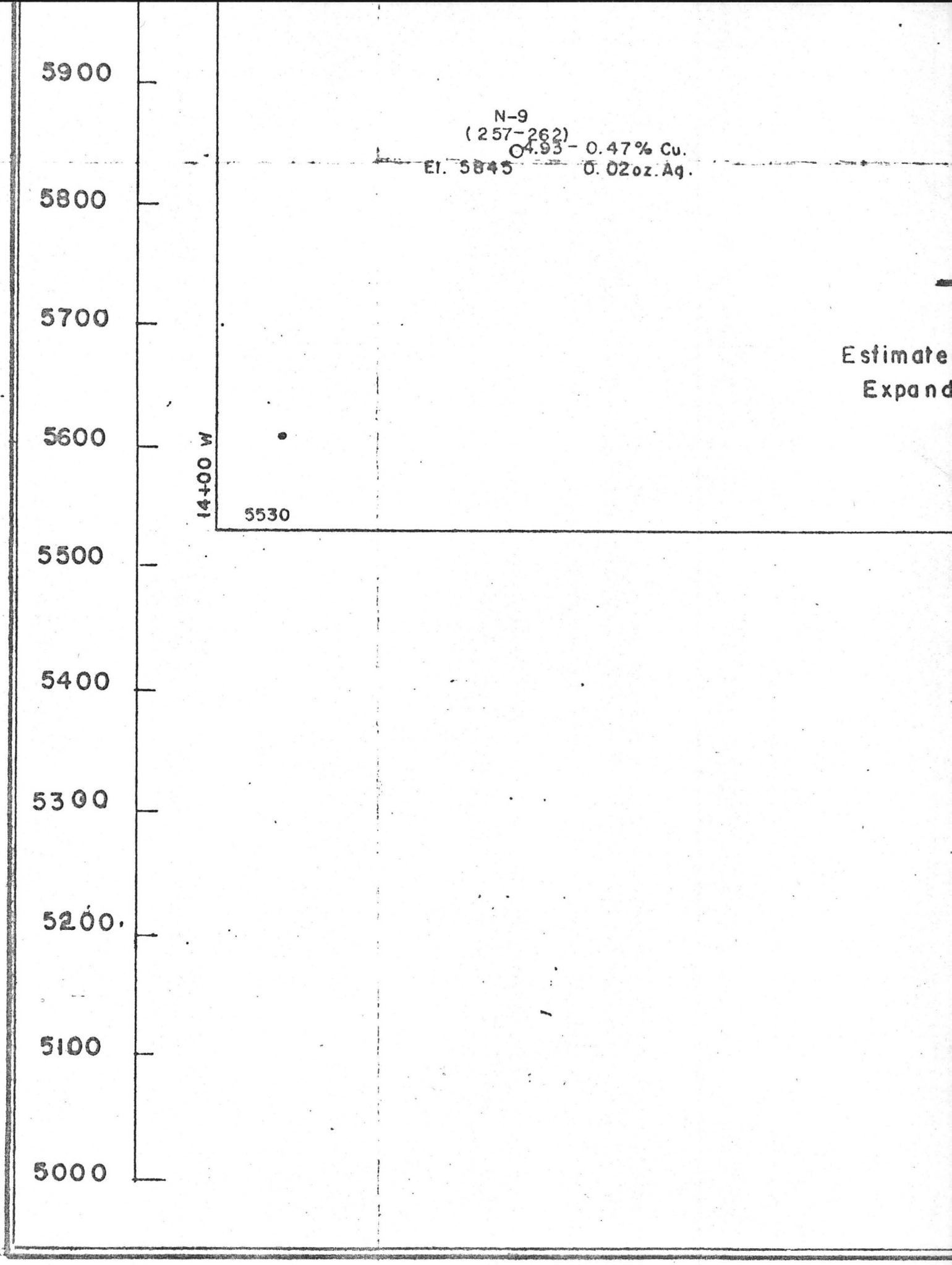
LONGITUDINAL SECTION

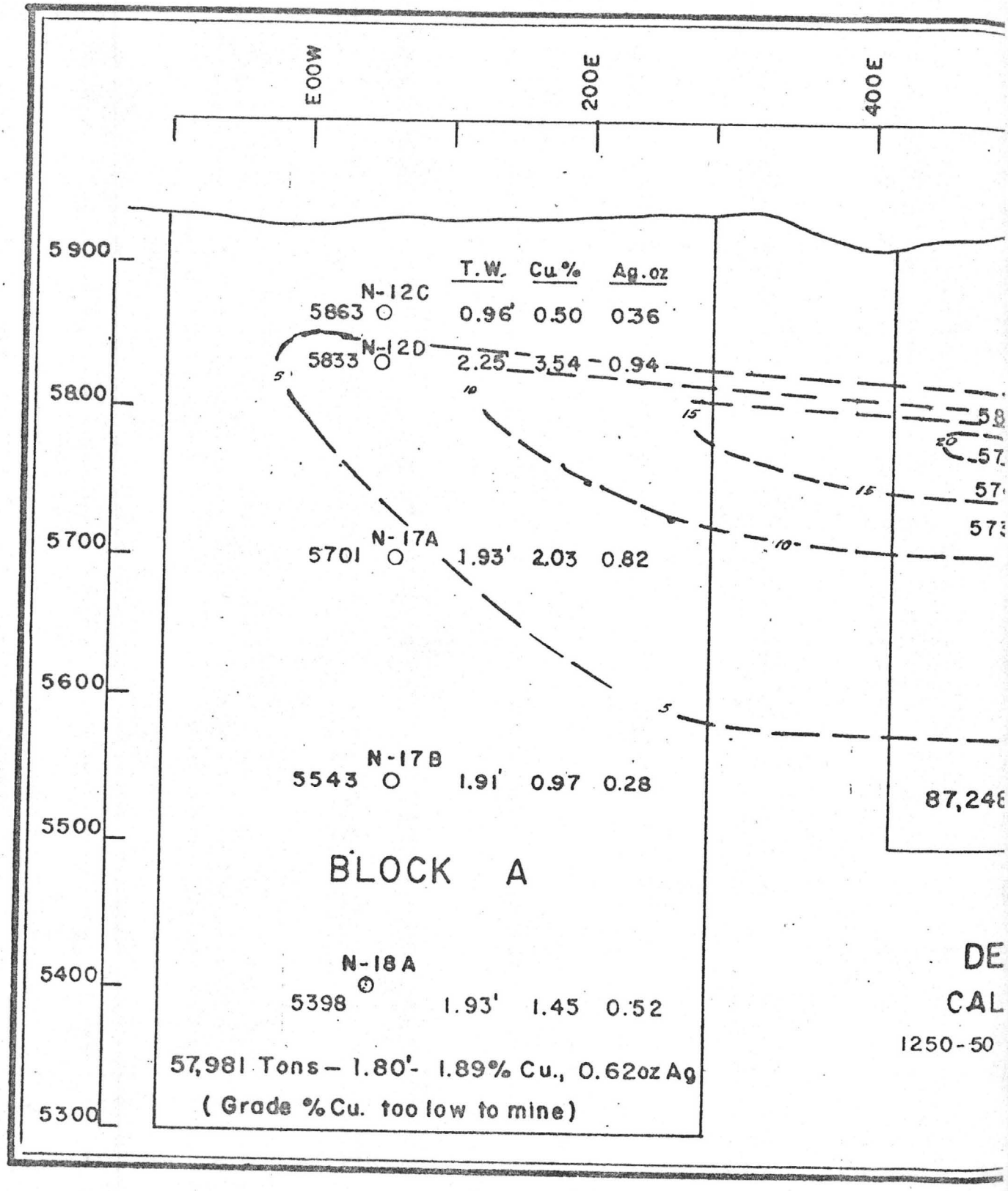
DELANO VEIN

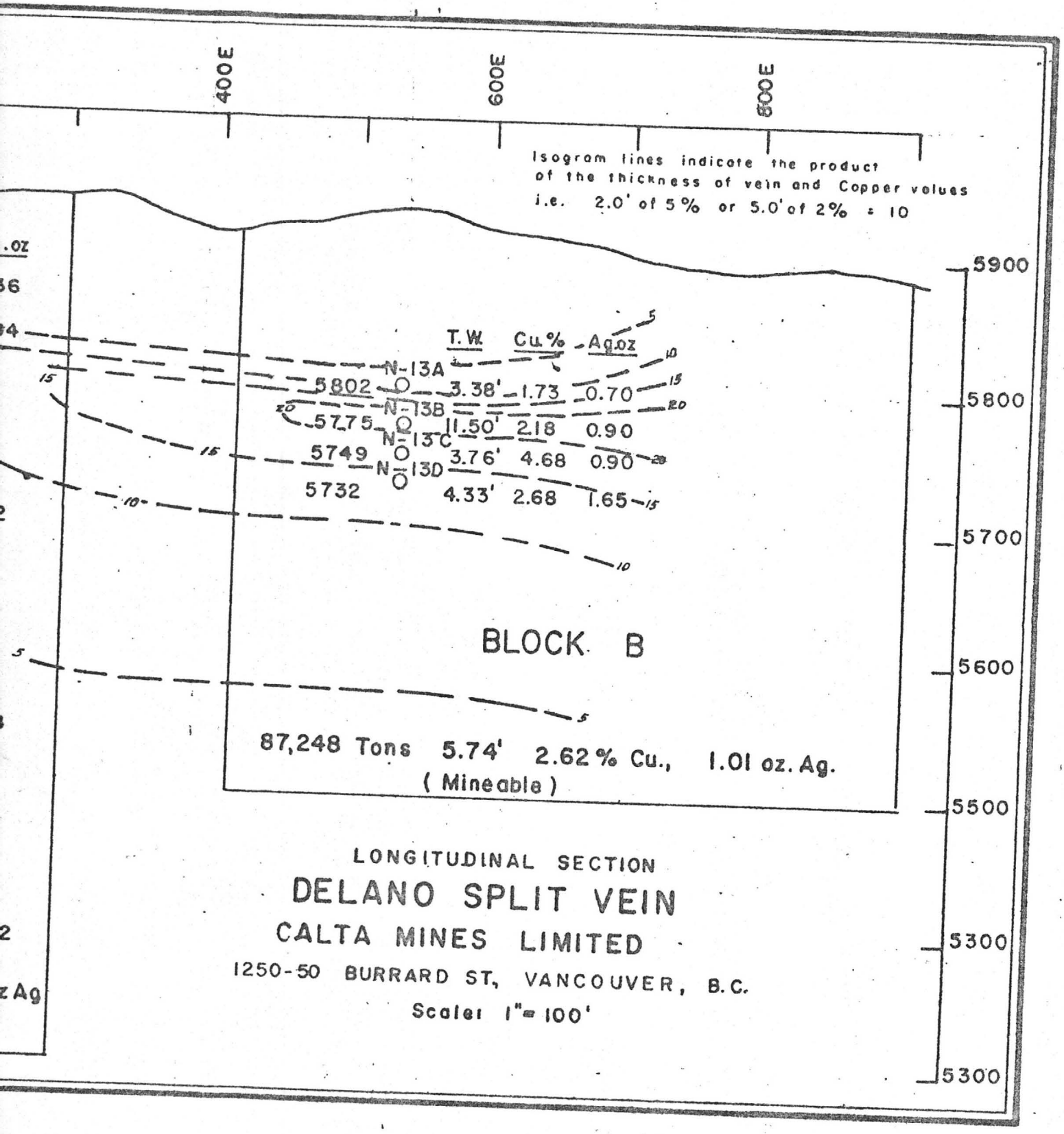
CALTA MINES LIMITED

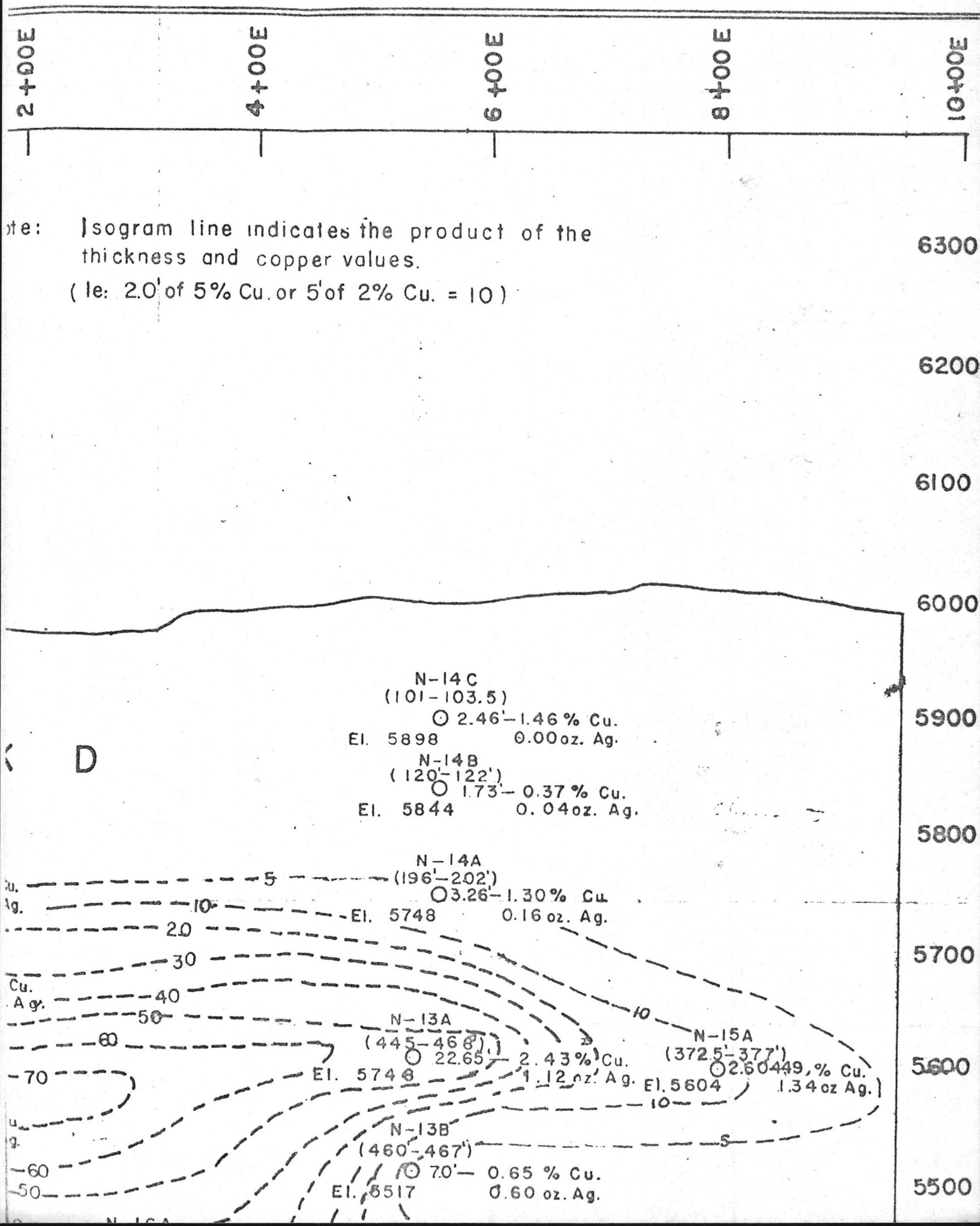
1250-50 BURRARD ST. VANCOUVER, 1, B.C.

SCALE 1" = 100'


5900
5800
5700
5600
5500
5400
5300
5200
5100
5000
N-9
(257-262)
4.93 - 0.47% Cu.
El. 5845
0.02oz. Ag.
Estimate
Expand
14+00 W
5530


E00W
200E
400E
5900
5800
5700
5600
5500
5400
5300
T.W.
Cu.%
Ag.oz
N-12C
5863
0.96'
0.50
0.36
N-12D
5833
2.25
3.54
0.94
N-17A
5701
1.93'
2.03
0.82
N-17B
5543
1.91'
0.97
0.28
BLOCK A
N-18A
5398
1.93'
1.45
0.52
57,981 Tons – 1.80'- 1.89% Cu., 0.62oz Ag
(Grade %Cu. too low to mine)
87,248
DE
CAL
1250-50


400E
600E
800E
Isogram lines indicate the product of the thickness of vein and Copper values i.e. 2.0' of 5% or 5.0' of 2% = 10
5900
5800
5700
5600
5500
5300
5300
T.W. Cu.% Ag.oz
N-13A
5802 O 3.38' 1.73 0.70
N-13B
5775 O 11.50' 2.18 0.90
N-13C
5749 O 3.76' 4.68 0.90
N-13D
5732 O 4.33' 2.68 1.65
BLOCK B
87,248 Tons 5.74' 2.62% Cu., 1.01 oz. Ag.
(Mineable)
LONGITUDINAL SECTION
DELANO SPLIT VEIN
CALTA MINES LIMITED
1250-50 BURRARD ST, VANCOUVER, B.C.
Scale: 1" = 100'


2+00E
4+00E
6+00E
8+00E
10+00E
te: Isogram line indicates the product of the thickness and copper values.
(Ie: 2.0' of 5% Cu. or 5' of 2% Cu. = 10)
6300
6200
6100
6000
5900
5800
5700
5600
5500
D
N-14C
(101-103.5)
2.46'-1.46% Cu.
El. 5898
0.00oz. Ag.
N-14B
(120'-122')
1.73'-0.37% Cu.
El. 5844
0.04oz. Ag.
N-14A
(196'-202')
3.26'-1.30% Cu.
El. 5748
0.16 oz. Ag.
N-13A
(445-468')
22.65'-2.43% Cu.
El. 5748
1.12 oz. Ag.
N-15A
(372.5'-377')
2.60449,% Cu.
El. 5604
1.34 oz Ag.
N-13B
(460'-467')
7.0'-0.65% Cu.
El. 5517
0.60 oz. Ag.
5
10
20
30
40
50
60
70
10
10
5
60
50

2+00 W

E 00 W

2+00E

4+00E

Note: Isogram line indicates the produc
thickness and copper values.
(Ie: 2.0' of 5% Cu. or 5' of 2% Cu. = 10

BLOCK D

N-14C
(101-103.5
2.46
El. 5898

N-14B
(120'-122'
1.73
El. 5844

N-12A
(328.5-330.5)
1.97'- 1.72 % Cu.
El. 5758 0.20 oz. Ag.

N-14A
(196'-202'
3.26
El. 5748

4 % Cu.
2 oz. Ag.

9 Cu. 0.15Ag

Cu. 0.08Ag

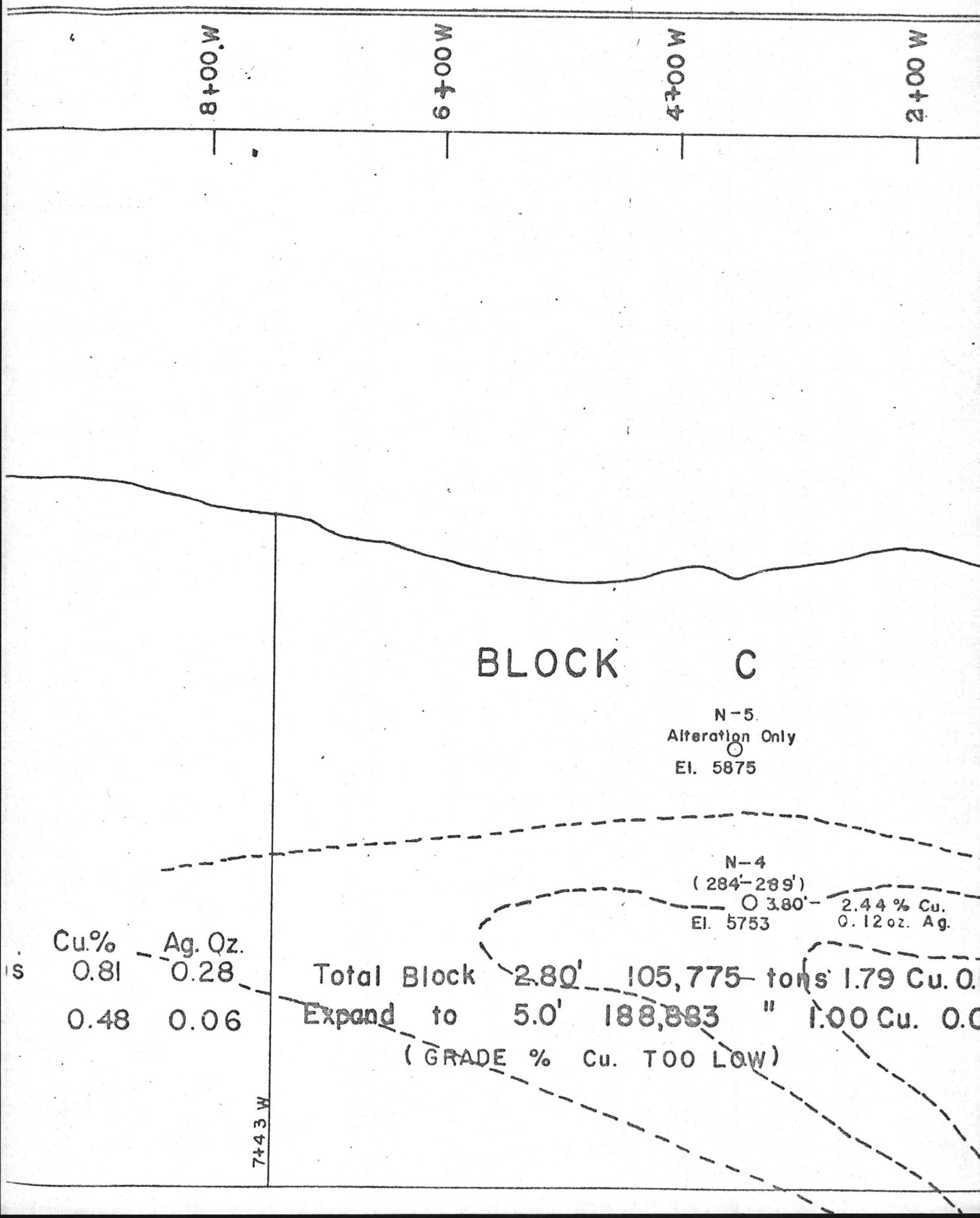
N-12B
(320-332')
11.82'- 2.06 % Cu.
El. 5669 0.92 oz. Ag.

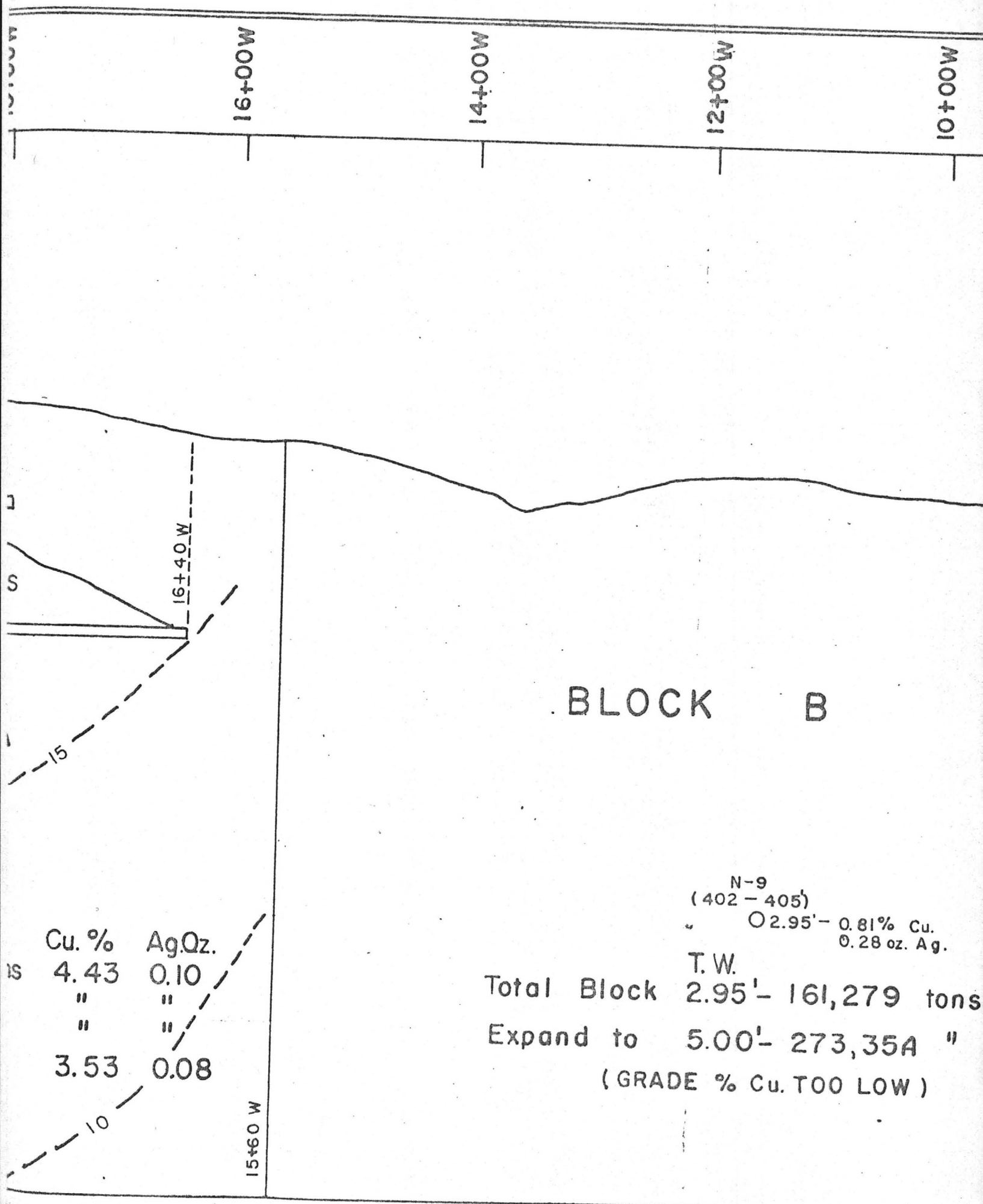
N-13A
(445-468')
22.65'
El. 5748

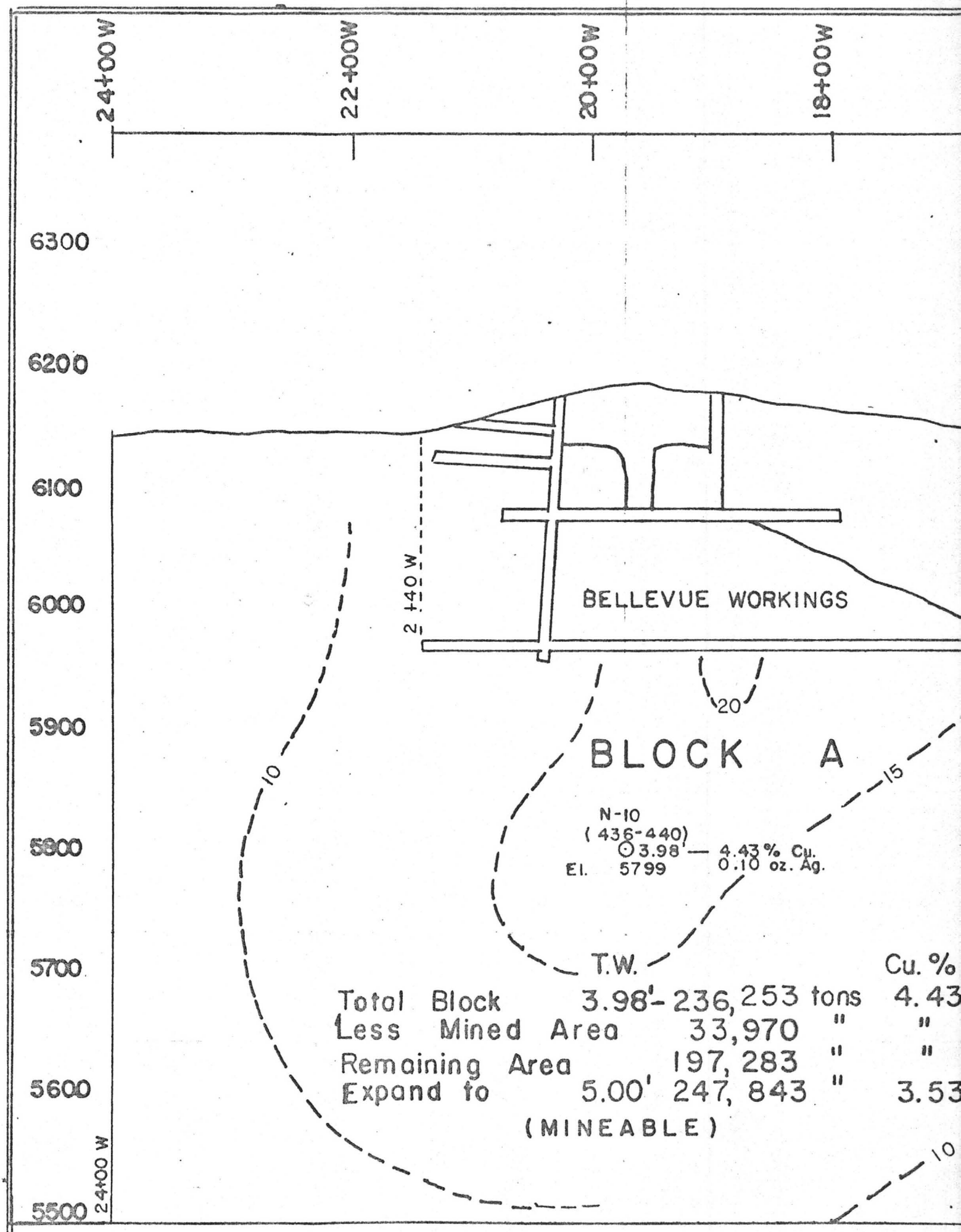
N-12C
(392'-396')
3.28'- 2.30% Cu.
El. 5551 0.15 oz Ag.

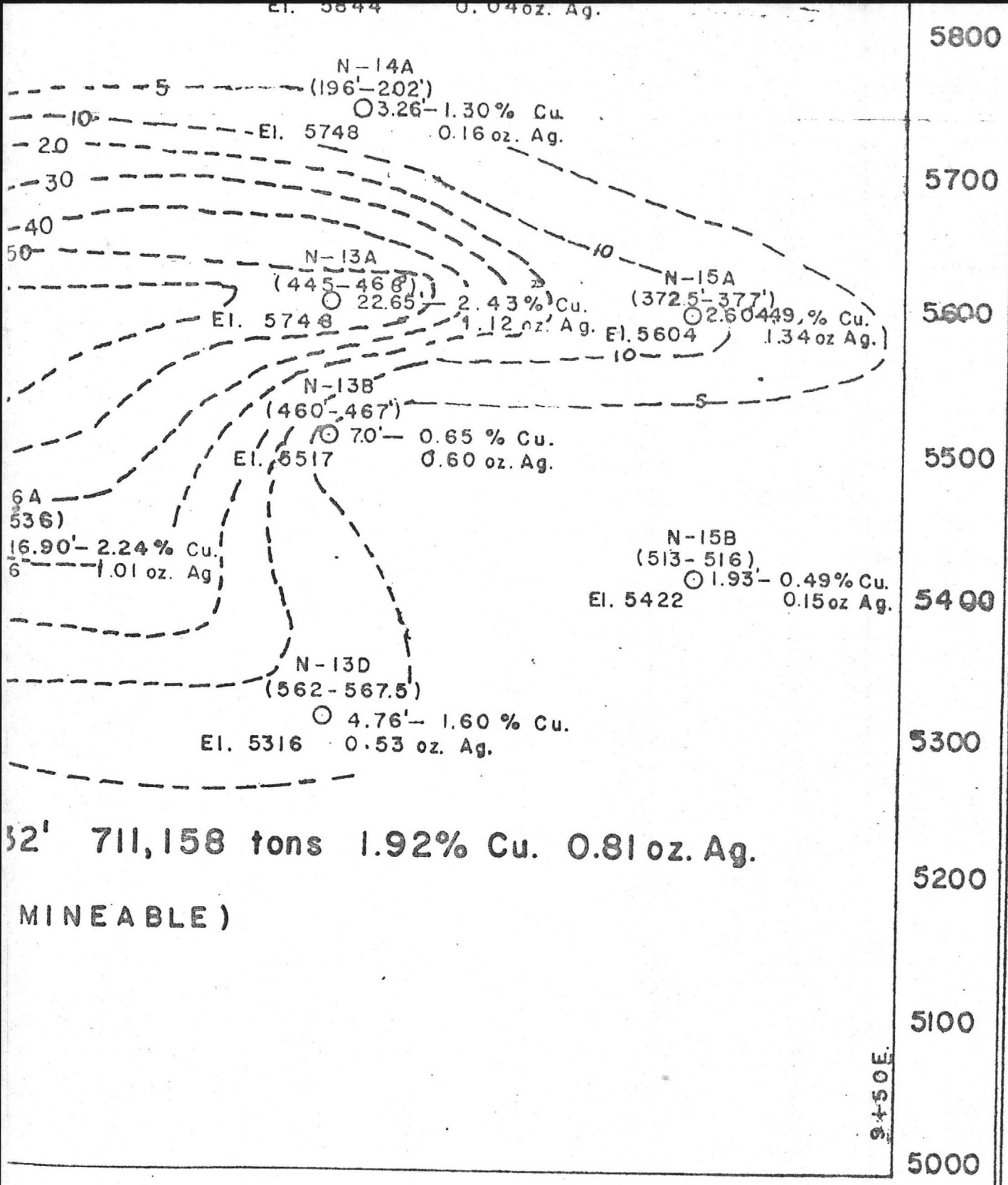
N-13B
(460'-467')
7.0'-
El. 5517

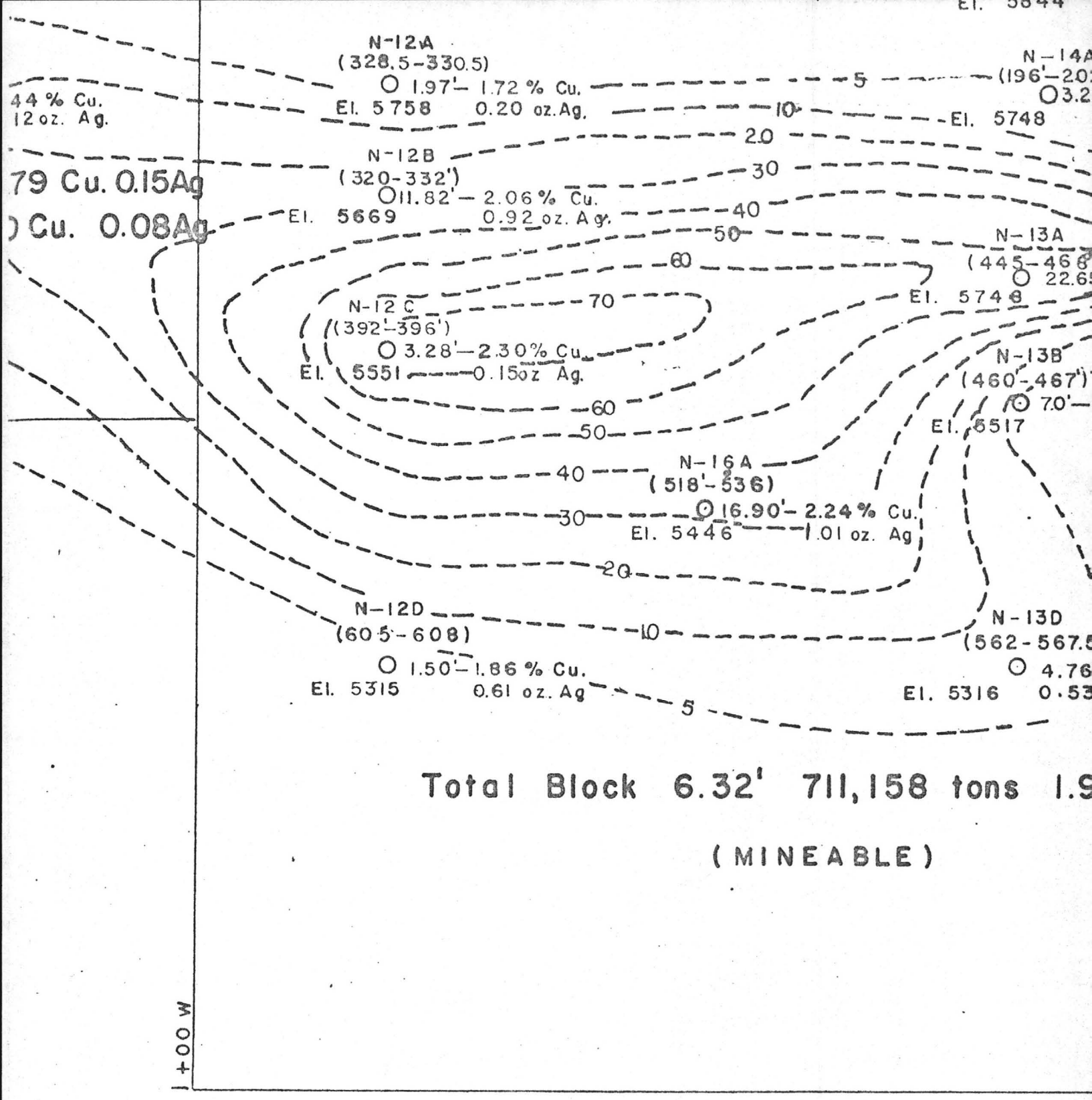
5
10
20
30
40
50
60
70
80
60
50

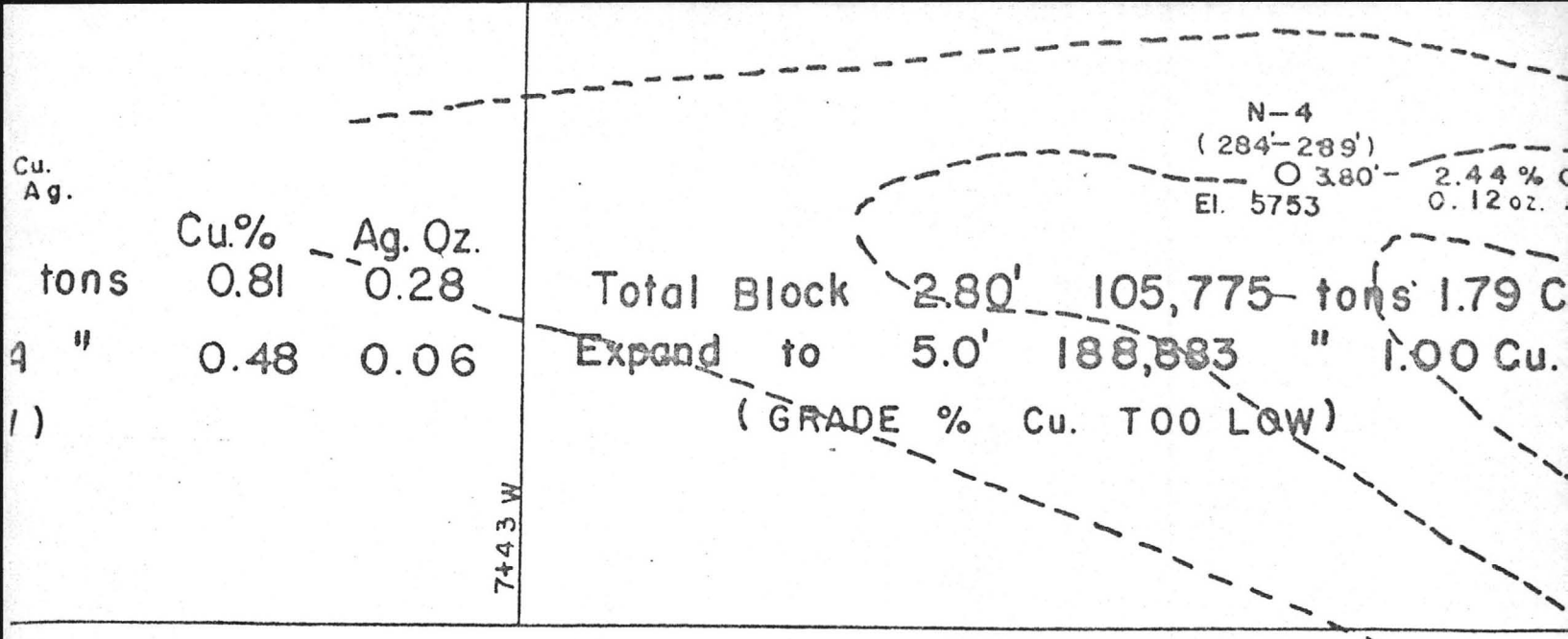

8+00 W
6+00 W
4+00 W
2+00 W
BLOCK C
N-5
Alteration Only
El. 5875
N-4
(284'-289')
3.80'
El. 5753
2.44 % Cu.
0.12 oz. Ag.
Cu.%
Ag. Oz.
0.81
0.28
0.48
0.06
Total Block 2.80' 105,775 tons 1.79 Cu. 0.
Expand to 5.0' 188,883 " 1.00 Cu. 0.0
(GRADE % Cu. TOO LOW)
7+43 W


16+00W
14+00W
12+00W
10+00W
16+40 W
15
10
15+60 W
BLOCK B
N-9
(402 – 405')
O 2.95' – 0.81% Cu.
0.28 oz. Ag.
T.W.
Total Block 2.95' – 161,279 tons
Expand to 5.00' – 273,35A "
(GRADE % Cu. TOO LOW)
Cu. % Ag.Oz.
4.43 0.10
" "
" "
3.53 0.08


24+00W
22+00W
20+00W
18+00W
6300
6200
6100
6000
5900
5800
5700
5600
5500
24+00 W
2+40 W
BELLEVUE WORKINGS
20
10
15
10
BLOCK A
N-10
(436-440)
3.98' — 4.43% Cu.
0.10 oz. Ag.
El. 5799
T.W.
Cu. %
Total Block 3.98'- 236,253 tons 4.43
Less Mined Area 33,970 " "
Remaining Area 197,283 " "
Expand to 5.00' 247,843 " 3.53
(MINEABLE)


El. 5844 0.04 oz. Ag.
5800
N-14A
(196'-202')
3.26'- 1.30% Cu.
0.16 oz. Ag.
El. 5748
5
10
20
30
40
50
5700
N-13A
(445-468')
22.65'- 2.43% Cu.
1.12 oz. Ag.
El. 5748
N-15A
(372.5'-377')
2.60 449,% Cu.
1.34 oz Ag.
El. 5604
10
5600
N-13B
(460'-467')
7.0'- 0.65 % Cu.
0.60 oz. Ag.
El. 5517
5500
6A
536)
16.90'- 2.24% Cu.
1.01 oz. Ag
N-15B
(513-516)
1.93'- 0.49% Cu.
0.15 oz Ag.
El. 5422
5400
N-13D
(562-567.5')
4.76'- 1.60 % Cu.
0.53 oz. Ag.
El. 5316
5300
32' 711,158 tons 1.92% Cu. 0.81 oz. Ag.
5200
MINEABLE)
5100
9+50E.
5000


El. 5844
N-12A
(328.5-330.5)
1.97'- 1.72 % Cu.
El. 5758 0.20 oz. Ag.
N-14A
(196'-20
3.2
El. 5748
44 % Cu.
12 oz. Ag.
N-12B
(320-332')
11.82'- 2.06 % Cu.
El. 5669 0.92 oz. Ag.
79 Cu. 0.15Ag
) Cu. 0.08Ag
N-13A
(445-468
22.6
El. 5748
N-12 C
(392'-396')
3.28'-2.30% Cu.
El. 5551 0.15oz Ag.
N-13B
(460'-467')
7.0'
El. 5517
N-16A
(518'-536)
16.90'- 2.24 % Cu.
El. 5446 1.01 oz. Ag
N-12D
(605-608)
1.50'-1.86 % Cu.
El. 5315 0.61 oz. Ag
N-13D
(562-567.5
4.76
El. 5316 0.53
5
10
20
30
40
50
60
70
Total Block 6.32' 711,158 tons 1.9
(MINEABLE)
1+00 W



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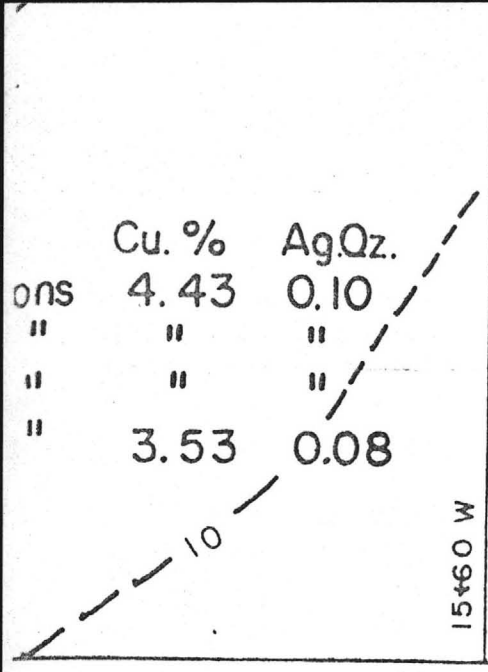
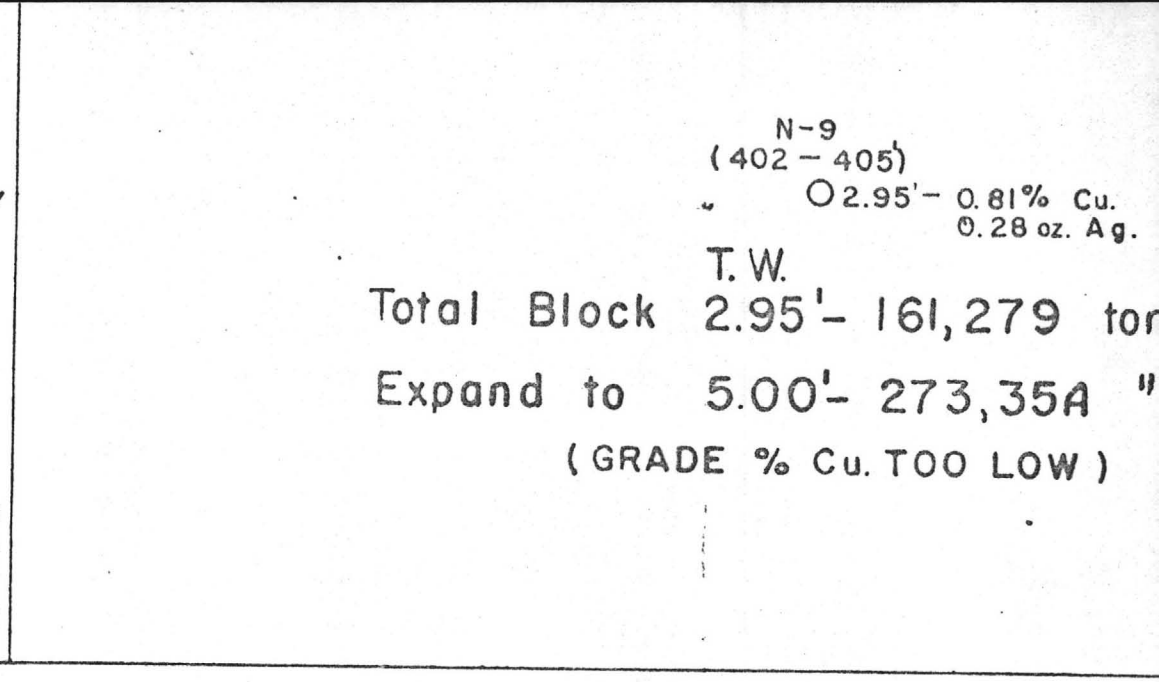
IN

MITED

UVER, I, B. C.





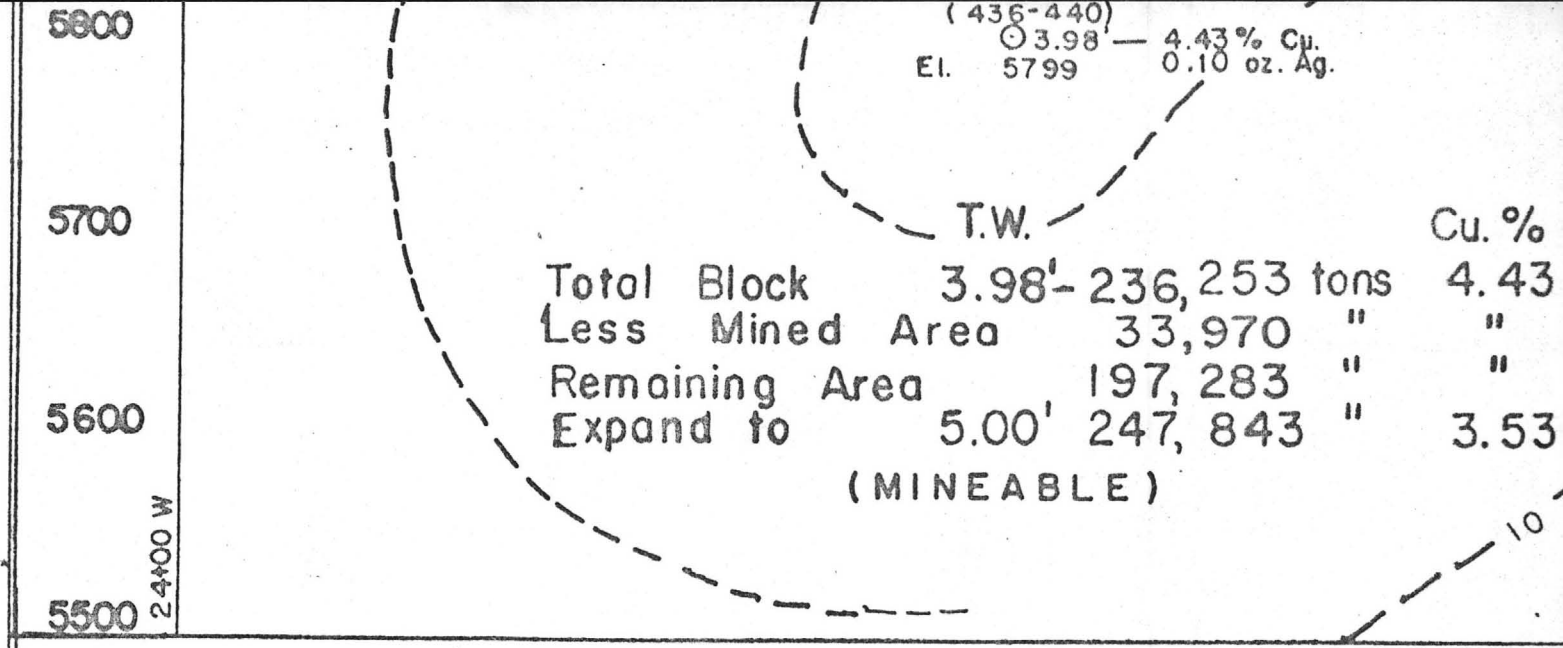
LONGITUDINAL SECTION

NORTH VEIN

CALTA MINES LIMIT

1250 – 50 BURRARD ST. VANCOUVER

SCALE 1" = 100'


5800
5700
5600
5500
24+00 W
(436-440)
3.98' — 4.43% Cu.
0.10 oz. Ag.
El. 5799
T.W.
Cu. %
Total Block 3.98'- 236,253 tons 4.43
Less Mined Area 33,970 " "
Remaining Area 197,283 " "
Expand to 5.00' 247,843 " 3.53
(MINEABLE)
10

NORTH VEIN

Hole No.	Coordinates East	Coordinates South	Elev.	Dip Hole	Dip Fiss.	Hole Depth Pierce Point	Vein Apprnt. Width	Elev. Pierce Point	Angle Intersect Hole-Vein	True Width	Vert. Dist. or Diff. Elev.	Fissure Run	Distance East-West	Tons Ore @ 12 ft. per ton	Cu %	Ag oz.
Block "A" From 24+00 W to 15+60 W = 840' E-W																
From Elev. 6150 to Elev. 5500 = 650' Diff. = 848' Run																
N-10	19+70W	5+10S	6110	45°	50°	436-440'	4.0'	5799	85°	3.98'	650'	848'	840'	236,253	4.43	0.10
Less Bellevue "Mine" Area										3.98'	470'	235'	500'	38,970	4.43	0.10
Remaining Tons														197,283	4.43	0.10
Expand Remaining Tons to 5.00' to Mining Width														247,843	3.53	0.08
Tons = $\frac{3.98 \times 848 \times 840}{12} - \frac{3.98 \times 235 \times 500}{12}$ = 197,283 tons @															4.43	0.10
Tons expanded to 5.0 ft. width																
= $\frac{197,283}{0.796}$ = 247,843 tons @															3.53	0.08
Block "B" From 15+60 W to 7+43 W = 817'																
From Elev. 6115 to Elev. 5500 = 615' = Diff. Elev. = 803' Run																
N-9	11+50W	3+70S	6030	45°	50°	402-405'	3.0'	5744	70°	2.95'	615'	803'	817'	161,279	0.81	0.28
(Grade of Ore Too Low to Mine) Expand 5.0'														273,354	0.48	0.16
Tons = $\frac{2.95 \times 803 \times 817}{12}$ = 161,279 tons @															0.81	0.28
Tons expanded to 5.0 ft. width																
= $\frac{161,279}{0.590}$ = 273,354 tons @															0.48	0.16

NORTH VEIN
(Con't.)

Hole No.	Coordinates East	Coordinates South	Elev.	Dip Hole	Dip Fiss.	Hole Depth Pierce Point	Vein Apprnt. Width	Elev. Pierce Point	Angle Intersect Hole-Vein	True Width	Vert. Dist. or Diff. Elev.	Fissure Run	Distance East-West	Tons Ore @ 12 ft. per ton	Cu %	Ag oz.
						Block "C" From 7+43 W to 100 W = 643' E-W										
						From Elev. 6040 to Elev. 5500 = 540' Diff. = 705' Run										
N-4	3+35W	2+10S	5957	45°	50°	284-289'	5.0'	5753							2.44	0.12
N-5	3+50W	0+10S	5981	45°	50°	149-150.5'	1.5'	5875							0.00	0.00
						Outcrop (Please Average)										
							4.0'								2.10	0.60
							3.50'			2.80'	540'	705'	643'	105,775	1.79	0.15
							Expand to Mining Width			5.00'				188,883	1.00	0.08
						(Grade of Ore Too Low to Mine)										
						Tons = $\frac{2.80 \times 705 \times 643}{12}$ = 105,775 tons @									1.79	0.15
						= $\frac{105,775}{0.560}$ = 188,883 tons @									1.00	0.08
						Block "D"					(Outcrop El.)					
N12A	0+50E	2+10S	5923	30°	50°			5758	100°	1.97'	N-12 5995				1.72	0.20
N12B	0+50E	2+10S	5923	50°	50°			5669	80°	11.82'	N-13 5975				2.06	0.92
N12C	0+50E	2+10S		70°	55°			5551	55°	3.28'	N-14 6005				2.30	0.15
N12D	0+50E	2+10S		90°	60°			5315	30°	1.50'	N-15 5980				1.86	0.61
N16A	2+90E	3+90S	5910	60°	50°			5446	70°	16.90'	N-16 5970				2.24	1.01
N13A	5+33E	3+60S	5875	35°	35°			5607	110°	22.65'	Ave. 5985				2.43	1.12
											Less Bottom					
N13B	5+33	3+60S	5875	50°	35°			5517	90°	7.00'	5000				0.65	0.60
N13D	5+33	3+60S	5875	80°	40°			5316	60°	4.76'					1.60	0.53

NORTH VEIN
(Con't.)

Block "D" (con't.)

Hole No.	Coordinates East	Coordinates South	Elev.	Dip Hole	Dip Fiss.	Hole Depth Pierce Point	Vein Apprnt. Width	Elev. Pierce Point	Angle Intersect Hole-Vein	True Width	(Outcrop) Elev.) Diff. Elev.	Fissure Run	Distance East-West	Tons Ore @ 12 ft. per ton	Cu %	Ag oz.
N14A	5+58E	1+00N	5950	90°	57°			5748	33°	3.26'	(Diff. Elev.		from 1+00W)		1.30	0.16
N14B	5+58E	1+00N	5950	60°	57°			5844	60°	1.73'	(985'	1286'	to 9+50E)		0.37	0.04
N14C	5+58E	1+00N	5950	30°	40°			5898	110°	2.46'			1050'		1.46	0.00
N15A	8+00E	1+97S	5930	60°	60°			5604	40°	2.89'					1.60	1.34
N15B	8+00E	1+97S	5930	80°	60°			5422	40°	1.93'					0.49	0.15
							Weighted Average			6.32'	985'	1286'	1050'	711,158	1.92	0.81

$$\text{Tons} = \frac{6.32 \times 1286 \times 1050}{12} = 711,158 \text{ tons @}$$

1.92 0.81

TOTAL - NORTH VEIN

			Cu %	Ag oz.					
Block A	3.98'	Expanded to Width 5.0'			5.0'		247,842	3.52	0.08
Block B	2.95'	Expanded to Width 5.0'	0.48	0.16	Not minable grade				
Block C	2.80'	Expanded to Width 5.0'	1.00	0.08	Not minable grade				
Block D		Not Expanded			6.32'		711,158	1.92	0.81
Total Minable Ore - Blocks A and D						Weighted Values	959,000	2.33	0.62

DELANO VEIN

Hole No.	Coordinates East	Coordinates South	Elev.	Dip Hole	Dip Fiss.	Hole Depth Pierce Point	Vein Apprnt. Width	Elev. Pierce Point	Angle Intersect Hole-Vein	True Width	Vert. Dist. or Diff. Elev.	Fissure Run	Distance East-West	Tons Ore @ 12 ft. per ton	Cu %	Ag oz.
					Block "A"	From 14+00 W to E 00 W				= 1400' E-W						
						From Elev. 6060 to Elev. 5530				= 530'						
N-9	11+50W	3+70S	6030	45°	56°	257-262'	5.0'	5845	80°	4.93'					0.47	0.02
N-4	3+35W	2+10S	5957	45°	50°	193-197'	4.0'	5818	85°	3.98'					1.64	0.10
N-5	3+50W	0+10S	5981	45°	54°	76-79'	3.0'	5925	80°	2.99'					1.75	0.50
	Average Block "A"				53°					3.97'		664'	1400'	307,542	1.18	0.16
							Expand to Minable Width			5.00'				387,332	0.79	0.12
						Tons = $\frac{3.97 \times 664 \times 1400}{12}$ = 307,542 tons @									1.18	0.16
						Expand to 5.0 ft. = $\frac{307,542}{.794}$ = 387,332 tons @									0.79	0.12
														Too low grade to mine		
					Block "B"	From E 00W to 2+70 E				= 270' E-W						
						From Elev. 5975 to Elev. 5530				= 445'						
N12A	0+50E	2+10S	5923	30°	45°	142.5-145'	2.5'	5850	105°	2.41'					1.19	0.35
N12B	0+50E	2+10S	5923	50°	45°	133.5-136'	2.5'	5819	90°	2.50'					0.73	0.06
N12C	0+50E	2+10S	5923	70°	40°	145-148'	3.0'	5784	70°	2.82'					Alt.	Alt.
N12D	0+50E	2+10S	5923	90°	40°	179-184'	5.0'	5739	50°	3.83'					1.15	0.20
N17A	0+61E	5+23S	5998	60°	45°	402-403'	10"	5649	45°	0.00'					10" Qtz.	0.00
	Average Block "B"				42°30'					2.31'		659'	270'	34,251	0.79	0.15
							Expand to Minable Width			5.00'				74,136	0.36	0.07

DELANO VEIN
(Con't.)

Block "B" (con't.)

Hole No.	Coordinates East	Coordinates South	Elev.	Dip Hole	Dip Fiss.	Hole Depth Pierce Point	Vein Apprnt. Width	Elev. Pierce Point	Angle Intersect Hole-Vein	True Width	Vert. Dist. or Diff. Elev.	Fissure Run	Distance East-West	Tons Ore @ 12 ft. per ton	Cu %	Ag oz.
						Tons = $\frac{2.31 \times 659 \times 270}{12}$ = 34,251 tons @									0.79	0.15
						Expand to 5.0 ft. = $\frac{34,251}{.462}$ = 74,136 tons @									0.36	0.07
												Too low grade to mine				
						Block "C" From 2+70 E to 7+40 E				= 470'						
						From Elev. 5925 to Elev. 5530				= 395'						
N16A	2+90E	3+90S	5910	60°	40°	191-198'	7.0'	5739	85°	6.97'					1.82	0.54
N16B	2+90E	3+90S	5910	90°	40°	248-252'	4.0'	5658	50°	3.06'					1.61	0.17
N13A	5+33E	3+60S	5875	35°	35°	144-147'	3.0'	5791	70°	2.82'					1.46	0.19
N13B	5+33E	3+60S	5875	50°	35°	119.5-131'	11.5'	5775	90°	11.50'					2.18	0.90
N13C	5+33E	3+60S	5875	70°	35°	130-134'	4.0'	5750	70°	3.76'					4.68	0.90
N13D	5+33E	3+60S	5875	80°	35°	156-163'	7.0'	5715	60°	6.06'					2.40	0.15
						Block "C" Weighted Averages				5.70'		660'	470'	147,345	2.31	0.81
						Tons = $\frac{5.70 \times 660 \times 470}{12}$ = 147,345 tons @									2.31	0.81
												Minable				

DELANO VEIN
(Con't.)

Hole No.	Coordinates East	Coordinates South	Elev.	Dip Hole	Dip Fiss.	Hole Depth Pierce Point	Vein Apprnt. Width	Elev. Pierce Point	Angle Intersect Hole-Vein	True Width	Vert. Dist. or Diff. Elev.	Fissure Run	Distance East-West	Tons Ore @ 12 ft. per ton	Cu %	Ag oz.
												Cu Wgt.	Ag Wgt.			
Block "D" From 7+40E to 12+00E = 460' (Assay values from adjacent holes and From Elev. 5670 to Elev. 5530 = 140' 300 Level--from A. H. Lawry)																
EK-1	8+25E	5+35S	5846	90°	50°	316-339'	23.0'	5507	40°	14.78'		43.157'	10.49'		2.92	0.71
EK-3	10+20E	5+40S	5883	90°	50°	387-396'	9.0'	5496	37°	5.42'		6.883'	4.15'		1.27	No assay
From A. H. Lawry - Section Map																
300 Level-Ave. 90' Drift--(15 samples) +100-190'								5670		3.14'		22.985'	9.10'		7.32	2.90
300 Level-Ave. 115' Drift--(7 samples) +285-400'								5670		5.85'		25.33'	7.89'		4.33	1.35
										29.19'		98.355'	31.63		(3.37)	1.08
Block "D" Weighted Averages										7.30'	140'	183'	460'	51,209	3.37	1.08
Block "E" From 5+33E to 20+00E = 1467' From Elev. 5530 to Elev. 5000 = 530'																
EK-1	8+25E	5+35S	5846	90°	50°	316-339'	23.0'	5507	40°	14.78'					2.92	0.71
EK-2	9+10E	7+15S	5829	90°	50°	520-542'	22.0'	5309	40°	14.14'					1.84	0.67
BK-1	9+45E	8+70S	5831	90°	45°	696-716'	20.0'	5135	45°	14.14'					1.01	No assay
EK-3	10+20E	5+40S	5883	90°	50°	387-396'	9.0'	5496	37°	5.42'					1.27	No assay
C-3	14+55E	9+65S	5883	70°	46°	555.5-561'	5.5'	5306	63°	4.90'					2.54	1.54
C-4	18+15E	12+00S	5798	70°	48°	620-630'	10.0'	5207	62°	8.83'					1.79	0.30
Block "E" Weighted Averages										10.37'		713'	1467'	903,893	1.91	0.48
															1.91	0.48

$$\frac{140}{.766} = 183' \text{ Fiss. Run}$$

$$\text{Tons} = \frac{7.30 \times 183 \times 460}{12} = 51,209 \text{ tons}$$

$$\text{Tons} = \frac{10.37 \times 713 \times 1467}{12} = 903,893 \text{ tons @}$$

TOTAL DELANO VEIN

	Width	Cu %	Ag oz.	True Width	Tons Ore @ 12 ft. per ton	Cu %	Ag oz.
Block A	3.97'	1.18	0.16	Not minable grade			
Block B	2.31'	0.79	0.15	Not minable grade			
Block C				5.70'	147,345	2.31	0.81
Block D				7.30'	51,209	3.37	1.08
Block E				10.37'	903,893	1.91	0.48
Total Minable Ore - Blocks C, D, and E				Weighted Values	1,102,447	2.030	0.551

DELANO SPLIT VEIN

Hole No.	Coordinates East	Coordinates South	Elev.	Dip Hole	Dip Fiss.	Hole Depth Pierce Point	Vein Apprnt. Width	Elev. Pierce Point	Angle Intersect Hole-Vein	True Width	Vert. Dist. or Diff. Elev.	Fissure Run	Distance East-West	Tons Ore @ 12 ft. per ton	Cu %	Ag oz.
Block "A" From 1+00 W to 2+85 E = 385' E-W																
From Elev. 6100 to Elev. 5300 = 710' Diff. Elev. = 1004' Run																
N12C	0+50E	2+10S	5923	70°	50°	62.5-64'	1.5'	5863	40°	0.96'					0.50	0.36
N12D	0+50E	2+10S	5923	90°	50°			5833	40°	2.25'					3.54	0.94
N17B	0+61E	5+23S	5998	60°	46°			5701	75°	1.93'					2.03	0.82
N17A	0+61E	5+23S	5998	90°	44°			5543	50°	1.91'					0.97	0.28
N18A	0+45E	10+67S	6012	60°	35°			5398	85°	1.93'	710'	1004'	385'		1.45	0.52
Weighted Average Block "A"					45°					1.80'	710'	1004'	385'	57,981	1.89	0.62
Expand to Minable Width										5.00'	710'	1004'	385'	161,058	0.68	0.22

$$\text{Tons} = \frac{1.80 \times 1004 \times 385}{12} = 57{,}981 \text{ tons @} \quad 1.89 \quad 0.62$$

$$\text{Expand to 5.0' Width} = \frac{57{,}981}{.360} = 161{,}058 \text{ tons @} \quad 0.68 \quad 0.22$$

Too low grade to mine

DELANO SPLIT VEIN
(Con't.)

Hole No.	Coordinates East	Coordinates South	Elev.	Dip Hole	Dip Fiss.	Hole Depth Pierce Point	Vein Apprnt. Width	Elev. Pierce Point	Angle Intersect Hole-Vein	True Width	Vert. Dist. or Diff. Elev.	Fissure Run	Distance East-West	Tons Ore @ 12 ft. per ton	Cu %	Ag oz.
Block "B" From 4+10 E to 7+10E = 300' E-W																
From Elev. 5915 to Elev. 5500 = 415' Diff. Elev. or 608' Run																
N13A	5+33E	3+60S	5875	35°	43°	124.5-128'	3.5'	5802	105°	3.38'					1.73	0.70
N13B	5+33E	3+60S	5875	50°	43°	119.5-134'	11.5'	5775	90°	11.50'					2.18	0.90
N13C	5+33E	3+60S	5875	70°	43°	130-134'	4.0'	5749	70°	3.76'					4.63	0.90
N13D	5+33E	3+60S	5875	80°	43°	140-145'	5.0'	5732	60°	4.33'					2.68	1.65
Weighted Average Block "B"										5.74'		608'	300'	87,248	2.62	1.01

$$\text{Tons} = \frac{5.74 \times 608 \times 300}{12} = 87,248 \text{ tons @}$$

2.62 1.01

Minable

RECAPITULATION OF TOTAL MINE

	Width ft.	Weighted Values Cu	Weighted Values Ag	Tons Ore @ 12 ft. per ton	Cu %	Ag oz.
North Vein - Block "A"	5.0	872,407	19,827	247,843	3.52	0.08
North Vein - Block "D"	6.32	1,365,423	576,038	711,158	1.92	0.81
Delano Vein - Block "C"	5.70	340,367	119,349	147,345	2.31	0.81
Delano Vein - Block "D"	7.30	172,574	55,305	51,209	3.37	1.08
Delano Vein - Block "E"	10.37	1,726,435	433,868	903,893	1.91	0.48
Delano Split Vein - Block "B"	5.74	228,589	88,120	87,248	2.62	1.01
TOTAL TONS & GRADE	7.83	(4,705,795)	(1,292,507)	2,148,696	2.190	0.601

RECORD OF DRILL HOLE MINERALIZATION
CALTA MINES, LTD.
CONTACT, NEVADA

SOUTH VEINS

	% Cu	Oz. Ag	PP Elev.
N-1	0.68		5294
N-2	2.00	1.84	5301
N-2	1.32	0.36	5118
N-3	0.86	0.38	5179
N-3	0.40	0.06	5099
N-6	0.51	0.05	5436
N-6	0.91	No	5261
N-11	0.64	No	5538
N-8	0.74	0.04	5613
N-8	0.47	No	5520
N-8	0.43	No	5378
N-8	0.45	No	5337
N-18A	1.04	0.06	5951
N-18B	0.21	0.14	5906
N-19A	1.12	0.12	5788
N-19A	0.99	0.09	5650

DELANO VEIN

Hole	T. W.	% Cu	Oz. Ag	PP Elev.
N-9	4.93'	0.47	0.02	5845
N-5	2.99'	1.75	0.50	5925
N-4	3.98'	1.64	0.10	5818
N12A	2.41'	1.19	0.35	5850
N12B	2.50'	0.73	0.06	5819
N12C	3.00'	Altered		5784
N12D	3.83'	1.15	0.02	5739
N13A	2.82'	1.46	0.19	5791
N13B	11.50'	2.18	0.90	5775
N13D	3.76'	4.68	0.90	5750
N14A	6.06'	2.40	0.15	5715
N15A	4.60'	1.35	1.11	5897
N15B	3.28'	1.49	0.22	5885
N16A	6.97'	1.82	0.54	5739
N16B	3.06'	1.61	0.17	5658
EK-1	14.78'	2.92	0.71	5507
EK-2	14.14'	1.84	0.67	5307
EK-3	5.42'	1.27	No	5496
BK-1	14.14'	1.01	No	5135
C-3	4.90'	2.54	1.54	5306
C-4	8.83'	1.79	0.30	5207

NORTH VEIN

Hole	T. W.	% Cu	Oz. Ag	Elev. PP
N-10	3.98'	4.43	0.10	5799
N-9	2.95'	0.81	0.28	5744
N-5	Alteration Only			5875
N-4	3.80'	2.44	0.12	5753
N-12A	1.97'	1.72	0.20	5758
N-12B	11.82'	2.06	0.92	5669
N-12C	3.28'	2.30	0.15	5551
N-12D	1.50'	1.86	0.61	5315
N-13A	22.65'	2.43	1.12	5748
N-13B	7.00'	0.65	0.60	5517
N-13D	4.76'	1.60	0.53	5316
N-14A	3.26'	1.30	0.16	5748
N-14B	1.73'	0.37	0.04	5844
N-14C	2.46'	1.46	0.00	5898
N-15A	2.60'	4.49	1.34	5604
N-15B	1.93'	0.49	0.15	5422
N-16A	16.90'	2.24	1.01	5446

DELANO SPLIT VEIN

	T. W.	% Cu	Oz. Ag.	PP Elev.
N-12C	0.96'	0.50	0.36	5863
N-12D	2.25'	3.54	0.94	5833
N-13A	3.38'	1.73	0.70	5802
N-13B	11.50'	2.18	0.90	5775
N-13C	3.76'	4.68	0.90	5749*
N-13D	4.33'	2.68	1.65	5732
N-17A	1.93'	2.03	0.82	5701
N-17B	1.91	0.97	0.28	5543

* N-13C Hit a stope below assay.

MINABLE ORES ESTIMATED

		Width	Tons	% Cu	Oz. Ag
Block A	NORTH VEIN	5.0	247,843	3.52	0.08
Block D	NORTH VEIN	6.32	711,158	1.92	0.81
Block C	DELANO VEIN	5.70	147,345	2.31	0.81
Block D	DELANO VEIN	7.30	51,209	3.37	1.08
Block E	DELANO VEIN	10.37	903,893	1.91	0.48
Block B	DELANO SPLIT	5.74	87,248	2.62	1.01
		7.43			
TOTAL ORE MINABLE (Weighted)			2,148,696	2.191	0.601

MINING COSTS

The following mining costs are estimated on a daily production of 500 tons with the mining to be done by shrinkage stoping, or a modification thereof, in the Delano, Delano Split, North Vein and Bellevue orebodies, entry to which is to be gained by repairing the Ilo tunnel and sinking a 700-foot 2½-compartment shaft, from which levels will be run as required. Details of the various charges against mining are itemized in the following estimates.

Inasmuch as the old mines at Contact are mostly inaccessible, it would be the better part of judgment to open a representative part of the Delano Mine and actually mine a reasonable quantity of ore to establish, to a more certain degree, the physical characteristics of this ore deposit. With such information in hand a more exact mining method can be estimated.

SUMMARY OF MINING COSTS
CALTA MINES, LTD.
Contact, Nevada
April 11, 1970

CAPITAL COSTS

Surface Plant	$ 133,000
Equipment	189,000
Development	620,610
Total Capital Costs	$ 942,610

ESTIMATE

Based upon 2,148,696 tons mined at a rate of 175,000 tons/year; 15,000 ton/month; 500 tons/day. 12 years time.

INTEREST

942,610 @ 9.5% per year	$ 89,548
12 years x $89,548	1,074,576
Plus principal	942,610
Total (P + I)	$2,017,186

$$\frac{\$2,017,186}{144 \text{ months}} = \$14,008 \text{ per month}$$

$$\frac{\$14,008}{15,000 \text{ tons per month}} = \$0.93388 \text{ per ton}$$

SUMMARY OF COSTS

	Per Month	Per Ton
Principal & Interest	$ 14,008	$ 0.934
Explosives	28,394	1.893
Labor Salaried	13,020	0.870
Labor Underground	57,436	3.829
Supplies	21,010	1.400
Maintenance	7,020	0.468
Freight & Incidentals	10,000	0.666
TOTAL MINING COSTS	$150,888	$10.060

PRELIMINARY SURFACE PLANT COSTS
CALTA MINES, LTD.
Contact, Nevada
March 28, 1970

1.	Office & Warehouse, complete with desks, chairs, calculator, adding machine, duplicator, supplies	$ 18,000
2.	Assay Office, also complete, with furnace, burners, beakers, tubing, crucibles, acids, everything for assaying, Copper, Gold, Silver, and other Mineral	$ 17,000
3.	Trailer Homes, 5, 2-bedroom, living room, bathroom, storage, connections to facilities, for supervisory personnel	$ 18,000
4.	Water Tanks, foundations, pipe lines to mill, portal of mine and camp buildings, etc.	$ 8,000
5.	Fuel Storage, tanks, pumps, gasoline & oils, grease	$ 5,000
6.	Sub Station, transformers, drops lines to facilities	$ 19,000
7.	Sanitation & Sewers, connections to camp, mill, offices	$ 10,000
8.	Change House (Dry) - for mine and mill combined	$ 12,000
9.	Roads & Grading & Garages	$ 10,000
10.	Yard Equipment, auto, truck, fork lift, grader, Michigan loader #977	$ 16,000
	TOTAL SURFACE PLANT	$133,000

Note: No allowance is made herein for a bunkhouse and mess hall for the employees. More than likely this expense may be avoided as there are some facilities at Jackpot which could be expanded by the motel and restaurant owners now there to accommodate employees. It is fifteen miles from the mine to Jackpot on Highway No. 93, a good paved road.

A PRELIMINARY ESTIMATE
OF EQUIPMENT COSTS FOR A
500 TON PER DAY MINING AT THE
CALTA MINES, LTD.
Contact, Nevada
March 31, 1970

EQUIPMENT ITEMS		Approx. Cost
Compressor - 1200 cu. ft., IR-350 HP.	Used	$ 30,000
Mucking Machines - Eimco 21-B @ $2500		5,000
Hoist - Nordberg, Double Drum, 700' Rope Speed, 175 HP	Used	25,000
Mancha Trammer - Extra Battery, Panels, Generator		6,000
Mine Cars - 40 cu. ft. - 12 @ $300 each		3,600
Jackleg & Jackhammers - 10 @ $1500 each		15,000
Tuggers - 10 @ $2615 each		26,150
Skips - 2 2-ton, guides & dump & sheaves		1,600
Pump - at shaft - 200 gpm, 700' lift, submersible		6,000
Drill Steel & Carbide Bits 2, 4, 6, 8-ft. changes		3,000
Slushers - 5 @ $2000 each	Used	10,000
Dump Pockets - 5 small and one large @ $1200 & $2500		8,500
Miner's Lamps - Charging Panel - 30 Lamps		1,500
Air Receivers - 2 @ $400		800
Blower - Ventilation		2,450
Track - 3000' - 24" gauge, ties, rail, bolts, splice		7,200
Pipe - 3000' - 1", 2", 3", fittings and valves		9,500
Small Tools - shovels, axes, wrenches, etc.		1,000
Freight & Trucking - S. L. C. to Contact " @ $11.60/cwt.		22,350
Total		$164,000
Contingency 15%		25,000
TOTAL EQUIPMENT		$189,000

PRELIMINARY DEVELOPMENT COSTS
CALTA MINES, LTD.
Contact, Nevada
March 26, 1970

The scope of work prior to mining ore is counted herein as development costs. Upon completion of the development work, there would be 22 faces of ore, tunnel sections, from that point each 100 ft. of drifting on ore would provide a place to raise in ore to create stopes.

Following are the several waste development work contemplated.

1.	Clean up the Ilo Tunnel of rock fall and small caves to a point 1550 ft. from the portal @ $20.00 per foot	$ 31,000
2.	"Shoo Fly" around the caved portion of the Delano vein from portal + 1550' for 850 ft. @ $65.00 per ft.	$ 55,250
3.	Drift westerly on Delano Vein to 5+80E, a distance of 520 ft. @ $65.00 per ft.	$ 33,800
4.	Cross cut from Delano Vein to North Vein, 342 ft. @ $65.00 per ft.	$ 22,230
5.	Cross cut southerly to Shaft Station and 150 ft. beyond. 200 ft. to Station and 150 ft. tail track = 350 ft. @ $65.00 per ft.	$ 22,750
6.	Sink a 7' x 22' shaft a distance, vertical of 700 ft. Shaft timbered, complete, bearing sets, guides, hanging rods @ $400 per ft.	$280,000
7.	Raise shaft to sheave, together with rope raise at station hoist to sheave. Sheave and dump tracks installed, total raising 160 ft. @ $65.00 and dump arrangement, main level loading pocket @ $1800	$ 12,200
8.	Ore & Waste loading pockets at each of 5 Levels 5 pockets & chutes @ $800 each (80-ton pockets)	$ 4,000
9.	Cross cut from shaft to North Vein and to Delano Vein as follows: 125' Level, 444'; 250' Level, 460'; 375' Level, 492'; 500' Level, 516'; 625' Level, 540'. Total cross-cut footage = 2452' @ $65.00	$159,380
TOTAL DEVELOPMENT COST ABOVE ESTIMATED		$620,610

A PRELIMINARY ESTIMATE
OF MINING COSTS AT
CALTA MINES, LTD.
Contact, Nevada
April 4, 1970

LABOR

Salaried Personnel		Per Month	Per Ton
Main Office		$ 3,000	$0.20
Manager		2,000	0.13
Asst. Manager		1,500	0.10
General Foreman		1,000	0.07
Chief Accountant		750	0.05
Accountant' Help	(2)	900	0.06
Chief Engineer		800	0.05
Engineering Help	(2)	900	0.06
Assayer		700	0.05
Sampler		450	0.03
	Total	$12,000	$0.80
Nevada Workman's Compensation	0.20%		
Unemployment	3.50%		
F.I.C.A.	4.80%		
Total	8.50%	1,020	0.07
Total Salaried & Clerical		$13,020	$0.87

Underground Labor	Per Day	Per Month	Per Ton
25 Contract Stope Miners	$37.50	$28,125	$1.875
4 Trammers	27.00	3,240	
2 Hoistmen	30.00	1,800	
2 Pocketmen	24.00	1,440	
4 Timermen	24.00	2,880	
2 Machine Muckers	27.00	1,620	
2 Mechanics	27.00	1,620	
2 Electricians	27.00	1,620	
2 Electrical Helpers	23.00	1,380	
4 Maintenance	23.00	2,760	
2 Pumpmen	25.00	1,500	
2 Yard & Track Maint.	23.00	1,380	
53 Men		$49,365	$3.291
Nevada Workman's Compensation	8.05%		
Unemployment	3.50%		
F.I.C.A.	4.80%		
Total	16.35%	8,071	0.538
Total Underground		$57,436	$3.829

Salaried	$13,020	$0.870
Underground	57,436	3.829
TOTAL LABOR	$70,456	$4.699

PRELIMINARY ESTIMATE OF
OPERATING COSTS
CALTA MINES, LTD.
Contact, Nevada
April 8, 1970

Items		Per Month Supplies	Per Month Maintenance	Per Ton
Explosives -	Powder	$24,075		$1.605
	Caps	3,050		.203
	Fuse	1,269		.085
Total Explosives		$28,394		$1.893
Electric Power		$10,800	$ 400	$0.747
Compressor		400	400	0.053
Drills		200	1,500	0.113
Drill Steel & Bits		1,500	1,500	0.200
Air Lines & Hose		125	125	0.016
Small Tools		150	150	0.020
Ventilation		200	200	0.027
Chutes & Manways		1,260	125	0.092
Timber - Tunnels & Raises		225	120	0.023
Mancha Trammer		400	150	0.037
Mine Cars - Haulage		400	600	0.067
Hoist, Pockets, Shaft		200	100	0.020
Pumps & Drainage		200	100	0.020
Autos & Truck		240	50	0.019
Surface Loader		150	50	0.013
Fork Lift		50	50	0.006
Pipes - Air & Water		250	50	0.020
Welding		370	50	0.028
Table Saw - Surface		200	50	0.016
Office		200	50	0.016
Engineering		250	10	0.017
Assaying		250	25	0.018
Mine Track		250	250	0.033
		$18,270	$6,104	
Contingencies 15%		2,740	916	
	Total	$21,010	$7,020	
	Per Ton Basis	$1.400 + $0.468 =		$1.868
Freight & Haulage		$10,000		$0.666

OUTLOOK FOR THE PRICE OF COPPER
AND
SALES OUTLET FOR CALTA COPPER

The outlook for the price of copper can be summarized by the following recent statement of George B. Munroe, President of Phelps Dodge Corporation, which is the number two U. S. copper producer:

> "There is yet no evidence of a lessening of the world wide demand for copper in spite of the world price which seems high by historical standards."

A more detailed statement of this price situation is contained in the March issue of the Nevada Mining Association Newsletter, as follows:

> "COPPER:- The domestic producer price for copper remains at 56¢ per pound. However, prices on the exchanges have been appreciably higher and many consumers felt the heavy demand for copper contracts was sparked by rumors, started in London, that The Anaconda Company, which had ordered "force majeure" on 40% of copper shipments out of Chile during March, would extend or broaden the curtailment next month.
>
> "A study group, subcommittee of the Cabinet Committee on Economic Policy, has been reviewing the copper market and their principal concern is the two-tier pricing system which places most domestic primary copper production at the low end of the price scale and all other copper at the high end. The current price range is from 56¢ to 79¢. During the first round of talks, the study group discovered, as has long been known in the copper market, that there are vast differences in how consumers fare in the acquisition of low-priced copper. Some users, large and small, are able to get virtually all the copper they need at the producer price; others, again both large and small, can get little or none at the producer price. The key to the differences in treatment, the study group feels, is in the proportion of copper needs a buyer was getting from major producers in 1963, the last year of a copper surplus. After free-market prices began to rise,

producers rewarded customers who had stuck with them during the surplus by selling them low-priced U. S. produced copper. Available supply is still being allocated pretty much on this basis. The study group learned that this situation 'severely restricts market entry by new copper customers who do not have access to producer-priced copper, and it discriminates against many firms.' Although no specific solution has been offered by the study group, indications are that they would favor a move to a single world price, even though this would mean a sharp increase in the cost of copper for many U. S. fabricators. Such a move would allow consumers to compete for supply on a first-come-first-served basis and would be more in keeping with the modus operandi of a free market."

On April 1st the Phelps Dodge Corporation advanced the price of copper to 60¢ per pound. The rest of the industry seems to be following suit and, as of April 10th, the quotation for domestic copper was 60¢-60¼¢ per pound.

Inquiries regarding the sale of Calta concentrates and precipitates met with negative answers from the usual buyers of domestic copper concentrates and precipitates. The American Smelting and Refining Company might take some of the Calta product, but well in the future. They did quote us a schedule, which they said was for calculations only and such schedules are attached. The freight rate from Contact to El Paso is presently quoted at $27 per ton but might be lowered by application. A copy of the A. S. & R. schedules is attached.

No other smelter was interested in quoting a schedule. They all seem to be filled to capacity.

A much better arrangement was found possible with Space Metals, Inc., of Salt Lake City who have signed a contract, subject to approval by Calta Mines, Ltd., under which Space Metals agrees to buy 80,000,000 pounds of Calta copper in concentrate form FOB Contact, to start one month after completion of the Calta mill. The purchase price is to be the E. & M. J. price on the day of delivery. A copy of this contract is attached.

This arrangement with Space Metals, if approved by Calta, will be a most unusual and favorable development which practically assures the success of the Calta projects. So far as is known to the writers of this report, no more favorable purchase agreement, either domestic or foreign, could be made for the sale of this copper.

METALLURGICAL INVESTIGATIONS

It was known that the Calta Mines ores were, as far as developed, partly sulfide and partly oxide. To determine the degree of oxidation 95 samples from the ore intervals in the various Calta drill holes were sent to Union Assay Office and assayed for non-sulfide copper content. The average of these 95 assays was 67% non-sulfide and 33% sulfide copper.

From the above samples a 40-pound lot was combined for testing and sent to The Galigher Company of Salt Lake City for metallurgical testing.

By flotation and acid leaching an overall extraction of 95.8% of the contained copper was obtained.

The complete report by The Galigher Company follows:

THE GALIGHER COMPANY
Salt Lake City, Utah

April 14, 1970

Mr. L. K. Requa
Felt Building
Salt Lake City, Utah

Laboratory Report on the Testing
of your Sample of Copper Ore
Our Lot No. 1919

Dear Mr. Requa:

In accordance with the arrangement made with you, we are pleased to summarize the results obtained in the preliminary testing of your copper ore. The sample was received in our laboratory on March 11, 1970, and was designated as our Lot No. 1919.

Sample Preparation & Analysis:

The sample that was tested consisted of a mixture of assay reject samples obtained from the Union Assay Office of this city.

The sample was prepared under the guidance of Mr. Clare Hagen and consisted of the various samples shown on the tabulation of samples contained in the appendix.

About 40 pounds of sample was prepared by stage crushing through 20 mesh using a laboratory roll crusher in closed circuit with a vibrating screen. The sample was then mixed by riffling and a head sample was removed by splitting. The Union Assay Office reported the following assay:

	% Cu	% Oxide Cu	% S
Head Sample Lot No. 1919	2.37	1.726	0.23

Based on the above analysis, it is seen that 72.8% of the copper contained in the sample is acid soluble and 27.2% is indicated as being present as sulfide copper.

Investigation Objective:

The objective of this investigation was that of conducting preliminary tests on the submitted sample to determine the copper recovery which could be expected as well as probable copper concentrate grade.

Summary of Test Results:

Six tests were conducted in this investigation, the complete details of which are contained in the attached test data sheets. The most satisfactory results were obtained in test No. 5, the testing procedure and results obtained being summarized as follows:

1. 1,000 grams of minus 20 mesh ore was acid leached for 24 hours at 50% solids. The pH during leaching was held at 1.6, the total addition of H_2SO_4 amounting to 42.3 grams.

2. The leached material was filtered and washed, the pregnant solution being analyzed for copper content.

3. The leach tailing was ground in a laboratory ball mill for ten minutes with lime and collector (Reagent 350 - potassium amyl xanthate). This grinding was performed at 50% solids, the resulting screen analysis being minus 48 mesh.

4. The ball mill discharge was diluted to 30% solids and a copper rougher concentrate was floated for five minutes using additional collector (R-350) and frother (a 3:1 mixture of methyl isobutyl carbinol and frother 65) the pH being 9.6.

5. The copper rougher concentrate obtained in (4) was subjected to one stage of cleaning for five minutes.

Lot No. 1919 - Test No. 5:

Product	Weight %	% Cu	% Distribution
Leach Extraction		17.001*	73.5
Leach Tail (Calc.)	95.9	0.638	26.5
Head (Calc.)	100.0	2.31	100.0
Cu Cleaner Concentrates	1.4	36.82	22.3
Cu Cleaner Tail	4.0	0.510	0.9
Cu Rougher Tail	90.5	0.085	3.3
Leach Tail (Calc.)	95.9	0.638	26.5

* Grams of Cu per liter of pregnant solution

The results obtained in Test No. 5 allow for the following observations:

1. It is seen that 73.5% of the copper was recovered by leaching. The weight loss encountered during leaching was 4.1%. The grade of copper precipitate which would be obtained from this leach solution would be expected to be in the range of 70% to 90% copper depending upon the precipitation method employed. The indicated acid consumption was 2.3 pounds of H_2SO_4 per pound of copper recovered in leaching.

2. An additional 22.3% of the copper contained in the ore was recovered by grinding the leach tailing, after washing, to 48 mesh and by subjecting same to flotation. In this test a concentrate grade of 36.82% Cu was obtained. However, based on other test results (see Tests Nos. 2, 3 and 4) it is believed

that a flotation concentrate grade of better than 40% Cu can be maintained in plant practice.

A line flow sheet depicting a proposed method for handling ores as represented by the submitted sample is contained in the appendix. It must be realized that this is based on preliminary testing and that "optimumization" of various unit operations will depend upon the results obtained in further detailed testing.

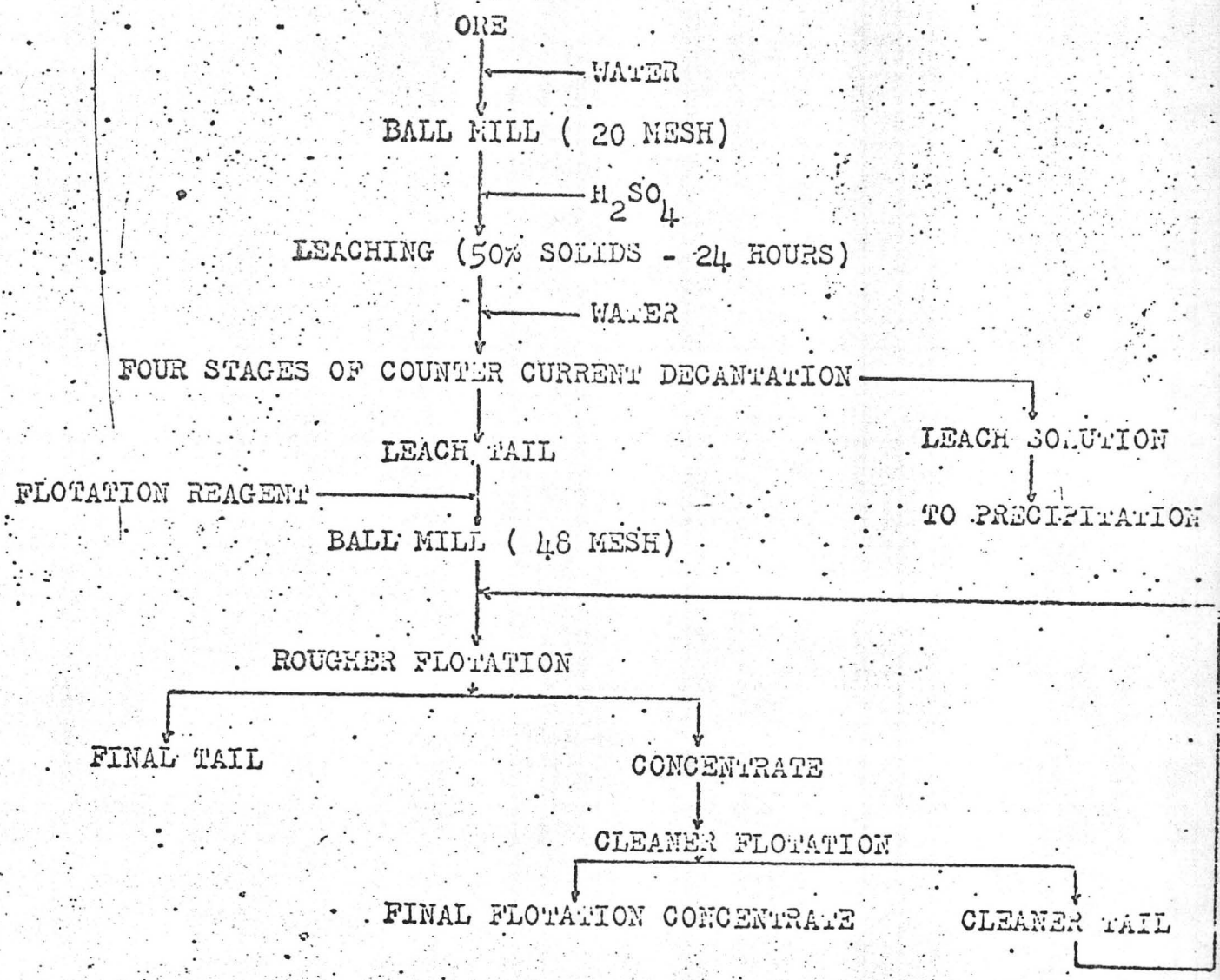
We've appreciated the opportunity to conduct these tests for you and, if we can be of further service to you, please contact us.

Respectfully submitted,

THE GALIGHER COMPANY

/s/ R. O. Huch

PROPOSED FLOWSHEET



LOT NO. 1919 COMPOSITE

		% Cu	Weight Grams
N 17 A	341-343	2.032	598
N 16 B	248-250	1.219	450
N 16 B	250-252	2.006	328
N 12 A	63113	1.720	678
	63153	2.226	270
	63156	1.087	321
	63157	1.437	892
	63158	1.012	763
	63178	6.167	333
	63179	1.568	54
	63180	1.138	517
	63188	1.340	798
	63186	1.391	146
	63458	1.386	373
N 13 B	119-125	4.161	1103
N 13 C	130-134	4.680	404
N 4	285-292	1.84	404
N 5	76-79	1.745	566
N 12 A	142-5-145	1.189	616
N 12 B	320-332	2.06	2643
N 12 C	393-396	2.297	1077
N 13 A	445-468	2.434	6031
		2.20	19,365

MILL CONSTRUCTION COSTS AND MILLING CHARGES

Only preliminary costs estimates were made for the cost of mill construction at Contact. The Galigher Company figured an overall cost of $2,000,000 for a new 500 ton per day combination flotation and leach plant.

There will be an opportunity to buy the 500 ton per day flotation mill of Federal Resources at Hailey, Idaho, which is closing down on May 1st. It is possible by using this mill that an adequate 500 ton per day plant could be constructed at Contact for a possible cost of $1,250,000. The operating charges and capital charges for such a plant are figured on the following page.

If it should be decided to build the new plant, at a cost of the estimated $2,000,000, the operating charges would be 76½¢ per ton more than those of the cheaper mill.

SUMMARY OF MILLING COSTS
CALTA MINES, LTD.
Contact, Nevada

CAPITAL COST

Reported purchase price of mill @ 500 tons per day capacity	$1,250,000

INTEREST

Pay 9.5% interest for 12-year period:

0.095 x $1,250,000 = $118,750 per year

$118,750 x 12 years =	$1,425,000
Plus Principal	1,250,000
Total cost to be retired in 12 years	$2,675,000

$$\frac{\$2,675,000}{12 \text{ years}} = \$222,916.00 \text{ per year}$$

500 tons per day = 175,000 tons per year

$$\frac{\$222,916.00}{175,000} = \$1.273 \text{ per ton}$$

MILLING COSTS

	Per Ton
Purchase Payments (as above)	$1.273
Reagents: Acid, Iron, Flot, Floc.	3.500
Labor	1.700
Power	0.800
Total	$7.273

SCOPE OF THE REPORT AND SOURCE OF DATA

This is a preliminary report and should be followed by detailed investigations into the various costs of operating and equipping the Calta Mine, before actual development or construction is started.

As this is essentially an engineering report, the matter of legal title to the claims constituting the Calta property was not investigated, although it is believed that such title is valid.

The source material of this report was as follows:

Geology, Geophysics & Exploration - L. K. Requa, who has been familiar with this district since 1930.

USGS Bulletins No's. 497 and 847a by F. C. Schrader.

Maps, reports and various data on recent exploration. Security Mining Co., and K. Chattin.

Costs of Mining and Mine Development - C. R. Hagen.

Assaying - Union Assay Office.

Metallurgy and Milling Costs - The Galigher Company.

A Compilation of Exploration Results from Property of Calta Mines, Ltd., by Cordilleran Engineering Ltd., which contained the maps and drill hole data on which the ore estimate was made.

MINING TAXES

In addition to the federal income tax there are certain other tax regulations that should be mentioned.

The federal income tax regulations permit a depletion allowance of 15 per cent of the gross income, which cannot exceed 50 per cent of the net taxable income as figured without any depletion allowance.

Elko County levies a net proceeds tax. Net proceeds is defined as the gross value of the ore, less all expenses. On this net proceeds a tax of 3.2677 per cent is payable to the county assessor.

SUMMARY AND CONCLUSIONS

It has been estimated in this report that there are 2,148,696 tons of minable ore with an average grade of 2.190% copper and 0.601 ounces of silver per ton in the Delano, Delano Split and the North Vein ore shoots. Minable ore is considered a combination of proven and probable ore. Possible ore is not estimated in this figure, but there are good reasons to believe that these ore reserves can be significantly increased by additional drilling.

A few simple geological rules appear important --

1. The periphery of the batholith and not its interior portion is the favorable location for ore deposits, especially the northwest contact.

2. The pattern of ore oxidation is unusual and extends well below water level. This water level has undoubtedly changed in geological times and there appears to be a reoxidation of a previously enriched supergene sulfide zone.

   This reoxidized sulfide - oxide zone is the better zone for ore values and ore widths, and it takes

flatly to the east. It probably lies mostly in the granodiorite and under the sediments. These features should control further drilling. In other words lateral and not deeper drilling is indicated.

3. The ores at Contact, at least for the present, will be mostly confined to the fissures, but the chances for more massive deposits exist in the partially outlined induced polarization anomaly which is indicated along the northern edge of the claims.

As the known ore is a mixture of sulfides and oxides, it was necessary to determine the extent of the oxidation. To determine this matter a series of 95 samples was assayed for non-sulfide copper. From these results the non-sulfide copper content was determined to be 67 per cent.

For this reason it was necessary to devise a mill to handle both the sulfide and the oxide copper ores. Such a flow sheet was devised by The Galigher Company of Salt Lake City and is shown in the text.

The metallurgical test work by The Galigher Company can be summarized as follows. A representative 40-pound sample of core hole rejects from the Calta drill holes was used. This sample assayed 2.37% total copper and 1.726% non-sulfide copper. Several tests of flotation and leaching were made. The best test showed 73.5% of the copper recovered by leaching and 22.3% by flotation, which is a combined recovery of 95.8% of the contained copper. The grade of the copper precipitate should run from 70% to 95% depending on the method of precipitation and the grade of the flotation concentrate should average better than 40% copper.

The sale of the copper products is assured by a contract obtained from the Space Metals, Inc., of Salt Lake City for the purchase of 80,000,000 pounds of copper FOB Contact at the current E. & M. J. copper price. This is an unusual and very favorable development. Space Metals will treat the Calta copper products chemically to obtain powdered copper and copper chemicals which will mean a great savings in freight and smelter charges. The usual buyers of copper concentrates and precipitates are not in the domestic market as their capacities are loaded.

The domestic market is presently very favorable and copper is quoted at 60¢ to 60¼¢ per pound. Foreign prices are higher at 70¢ to 79¢ per pound. The outlook for future copper prices will be influenced by the trend of the domestic economy, which at the present time shows signs of leveling off. As the Calta mill will not be operating for at least a year, the present price of copper may not apply when operations begin. However the present outlook for future copper prices is favorable.

SUMMARY OF CAPITAL REQUIRED

Exploration:		
Geological Mapping:		
Palo Alto-Blue Bird Area	$4,000.00	
I. P. Anomaly Area	4,000.00	
Additional I. P. Survey	$6,000.00	
Drilling:		
Main Ore Shoots	$55,000.00	
I. P. Anomaly	35,200.00	
Total Exploration		$ 104,200.00
Mine Development and Equipment:		
Surface Plant	$133,000.00	
Mine Equipment	189,000.00	
Mine Development	620,610.00	
		942,610.00
Cost of Mill:		
500 tons per day combination flotation and leach mill using secondhand mill now at Hailey with leach circuit added		1,250,000.00
TOTAL CAPITAL REQUIRED		$2,296,810.00

To this figure must be added interest at the rate of 9.5% for 12 years, the estimated time for mining and milling the calculated ore reserve. On the above basis this interest would amount to $2,618,363.40 for the 12-year period. This could be reduced by several methods of financing that are not within the scope of this report.

SUMMARY OF MINING, MILLING AND OVERHEAD COSTS

	Per Ton
Mining	$10.06*
Milling	6.00
Mill Amortization	1.27
Overhead	1.00
Total	$18.33

* Includes development cost

SUMMARY OF OPERATING PROFIT

Average ore assays 2.19% copper = 43.80 pounds Cu per ton

43.8 x 95.8% = 41.96 pounds Cu recovered per ton

41.96 x 60¢ = $25.18 per ton; present value of Calta ore.

	Per Ton
Present value of ore	$25.18
Estimated operating costs	18.33
Estimated operating profit	$ 6.86 (on present price copper)

The Calta operation, as herein estimated, should run on a break even basis with copper at 44¢ per pound. Any price above 44¢ should result in an operating profit. However, if the present price of copper, at 60¢ per pound, prevails, as well as present costs, it would mean an operating profit of $14,718,567 on the estimated ore reserves, over a twelve-year period, at the rate of 500 tons per day mined and milled.

RECOMMENDATIONS

The above estimated operation for the Calta Mines, Ltd., property appears feasible and, if approved, the exploration work, as suggested, should be done. The mining and milling procedures should be detailed as to materials and costs. Further metallurgical testing should be done to determine the more refined mill flow sheet.

Respectfully submitted,

C. R. Hagen

C. R. Hagen
Nevada Professional Engineer No. 363

L. K. Requa

L. K. Requa
Nevada Professional Engineer No. 1904

Salt Lake City, Utah
April 15, 1970

L. K. REQUA
SCHEDULE OF PROFESSIONAL EXPERIENCES

1919	B. A. in Geology, University of California.
1919-25	Oil Geologist, Sinclair Consolidated Oil Corp., and Standard Oil Company of California.
1926-38	Geologist and mine operator, Manager, Lewiston gold dredge. Exploration for lode and placer gold deposits for Herbert Hoover and Mark Requa. Manager, West Dip gold mine, Mercur, Utah.
	Consulting Engineer, U. S. Bureau of Mines.
1938-43	President, Idaho Almaden Mines, Weiser, Idaho, for Hoover family.
	Lieutenant Commander USNR, World War II
1944	Vice President of Combined Metals Reduction Co., during reorganization for Milbank-Hoover interests.
	Chief of Party in Ethiopia for Herbert Hoover, Jr., and Freeport Sulphur Co.
1945-51	Vice President and General Manager, Compania Minera de Guatemala in Guatemala. A silver, lead, zinc property of Hoover-Milbank interests.
1951-60	Director of Exploration for Combined Metals Reduction Co., at Salt Lake City, specializing in silver, lead, zinc and manganese properties.
	Consulting geologist for Jeremiah Milbank, American Zinc, Lead and Smelting Co., The Cyprus Mines Corporation in various western states.
	General exploration for W. R. Grace & Co., and The Bunker Hill Co., a joint venture in Central and South America, 1959-60.
1961-66	Consulting geologist for The Bunker Hill Co., in various western states.

1967 Consulting geologist for Paul Kayser, Houston, Texas, on various geophysical projects in Nevada.

1968 Consulting geologist for The Bunker Hill Company on various projects in western states and Alaska.

1969 Consulting geologist on various projects in Nevada.

1970 Consulting geologist Security Resources, Inc., of Boise.

Legion of Honor member AIME (1919).

Registered Professional Engineer in both Utah and Nevada.

CLARENCE R. HAGEN

SCHEDULE OF PROFESSIONAL EXPERIENCES

1920-1925	University of Utah - Mining Engineering, Major in Geology, B. S. - Summer work - Mucker, Miner, Millwork, Surveys.
1925-1926	Utah State Road Commission - Concrete Laboratory, Surveys and Design.
1926-1928	Utah Copper Co., Bingham, Engineering Dept., Geological Dept., Design of Precipitation Plant at Copperton. In charge alternately of Churn Drilling, Shovel Cuts, Sampling, Precipitation, Leaser Boss, Supt. Underground.
1928-1928	Consolidated Cortez Silver Mine, Nevada, Supt. of Mine and Mill.
1928-1936	Combined Metals Reduction Co., Pioche, Nevada, Mine Engineer in charge of Surveys, Geology, Design, and Production. U. S. Mineral Surveyor since 1936.
1936-1939	Lincoln County Power District, Pioche, Nevada, Chief Engineer, Design, Surveys, Purchasing, Construction of Hoover Dam - Pioche 138 KV line and all branch lines.
1939-1947	Combined Metals Reduction Co., Salt Lake City, Utah, Examination of new prospects, Ore Purchasing, Estimates, Geology, Exploration, Exploitation, Available to all mines of the Company.
1947-1948	Examination of Mine and Transportation Facilities of Guatemala Mines for Milbank and Hoover, New York City, N. Y.
1948-1951	Assistant Manager of Mines at Guatemala for Milbank and Hoover, under L. K. Requa, Manager. Speak Spanish.
1950-1952	Examination of mines, domestic and foreign, for Combined Metals Reduction Co., Milbank and Hoover.
1952-1970	Professional Engineering & Geology, Claim Surveys, Legal, Appraisals, Mine Engineering.

Registered Professional Engineer in both Utah and Nevada.

Mr. Larry Requa
Calta Mines, Ltd.
Felt Bldg.
Salt Lake City, Utah 84111

SCHEDULE FOR CALCULATION PURPOSES ON COPPER PRECIPITATES

Delivery: F.o.b. American Smelting and Refining Company, El Paso, Texas.

Payment Copper: Deduct from the copper assay 30 pounds and pay for remaining copper at the daily net domestic refinery quotation for electrolytic wire bars averaged for the calendar week including date of delivery, less a deduction of 5¢ per pound of copper accounted for.

Base Charge: $28.00 per net dry ton of 2,000 pounds, plus 15¢ per ton for each unit of copper in excess of 25%, fractions in proportion.

(Based on average hourly labor cost of $4.209)

No payment would be made for metals other than copper, and purchases would be based upon the product having no more than 18% moisture. Moisture over 18% would be charged for at the rate of 10¢ per unit.

Shipments should be prepared and delivered in solid-bottom open gondola railroad cars.

AMERICAN SMELTING AND REFINING COMPANY

By Reed Welch Manager
Southwestern Ore Purchasing Office

Tucson, Arizona
March 31, 1970

American Smelting and Refining Company

EL PASO SMELTING WORKS
EL PASO, TEXAS

PURCHASE SCHEDULE

Date Effective March 31, 1970

Mine

Location Near Contact, Nevada

Shipper CALTA MINES, LTD.

Address c/o Larry Requa, Felt Bldg., Salt Lake City, Utah 84111

Character Ore COPPER CONCENTRATES

R. R. Station

The following purchase terms are subject to the General Clauses shown on the back of this sheet, and are subject to change on 30 days notice. Unless shipments are begun within 30 days this quotation is automatically cancelled.

DELIVERY: F. O. B. unloading bins American Smelting and Refining Company, El Paso, Texas. The rates quoted are based upon shipment in gondola equipment. Extra unloading charges of $1.00 per dry ton will be assessed for products received in box cars.

PAYMENTS

GOLD: If .03 of an ounce per dry ton or over pay for 92.57 percent of the gold content at the daily London Final Gold Quotation, as published in Metals Week, averaged for the calendar week including date of delivery of product. All purchases of gold will be subject to United States Governmental regulations pertaining to transactions in gold.

SILVER: Pay for 95% (minimum deduction of ½ ounce) at the average Handy and Harman New York Silver Quotation adjusted to the basis applicable to silver contained in unrefined silver-bearing material, for the calendar week including date of delivery of last car of each lot at plant of Buyer, less a deduction of ~~2¾~~ 3¢ per ounce.

COPPER: Deduct from the wet copper assay 20 pounds and pay for 100 percent of the remaining copper at the daily net domestic refinery quotation for electrolytic wire bars as published in Metals Week averaged for the calendar week including date of delivery of last car of each lot at the plant of Buyer, less a deduction of 5 cents per pound of copper accounted for. Nothing paid for copper if less than one-half percent by wet assay.

No payment will be made for any metal or content except as above specified.

DEDUCTIONS

BASE CHARGE: ~~$ ---------- per net dry ton of 2,000 pounds; provided the sum of payments for gold, silver, and copper does not exceed $ ------- per ton. Add to the base charge ton per cent of the excess over $ ------- to a maximum charge of $ ------- per dry ton.~~

$28.00 per net dry ton of 2,000 pounds, plus 15¢ per ton for each unit of copper in excess of 25%, fractions in proportion.

(Based on average hourly labor cost of $4.209)

ZINC: Allow five units free; charge for the excess at thirty cents per unit, fractions in proportion.

ARSENIC: Allow one unit free; charge for the excess at $0.50 per unit, fractions in proportion.

ANTIMONY: Allow one unit free, charge for the excess at $1.50 per unit, fractions in proportion.

BISMUTH: One-tenth of one percent (0.1%) by wet assay allowed free, excess charged at 50¢ per pound, fractions in proportion.

TAXES: See Clause 1 on reverse side of this schedule.

FREIGHT: All railroad freight and delivery charges for account of shipper. Deduct from settlement freight and other advances made by Buyer.

TONNAGE: Limited to tons per month except by special arrangement.

AMERICAN SMELTING AND REFINING COMPANY

By Reed F. Welch
REED F. WELCH

(over)

HILL PRINTING CO.—EL PASO

PURCHASE ORDER

SPACE METALS INC.

2040 FORTUNE ROAD • TELEPHONE 801 486-8406

SALT LAKE CITY, UTAH 84104

PURCHASE ORDER

No. 1702

SHOW THIS NUMBER ON ALL B/L'S, PACKAGES, INVOICES, AND CORRESPONDENCE.

INVOICE in duplicate to us at above address unless otherwise indicated below.

Date April 7, 1970

Terms Net 30 days after delivery

F.O.B. Contact, Nevada

Ship Via

Date Required One month after completion of concentration plant

Ship To: Space Metals Inc
2040 Fortune Rd
Salt Lake City, Utah 84104

Calta MinesLtd.
P.O. Box 1644
Boise, Idaho 83701

Confirming

Do Not Duplicate

Attention Of:

Item	Quantity	Description	Price
	80,000,000 lb or total perduction of mine @ Contact, Nevada	Contained Cu	E. & M.J. quoted price on day of delivery

This purchase order approved by G.A. Armstrong

Date_____, 1970

G.A. Armstrong
Pres. Calta Mines Ltd.

Griff E. Williams

SPACE METALS INC.

METALLURGICAL RESEARCH DEPT.

CABLE "GALGAL"

THE GALIGHER COMPANY

P. O. BOX 200 — 345-355 WEST 6TH SOUTH STREET
SALT LAKE CITY, UTAH 84110 — U.S.A.

TELEPHONE 359-0731

OUR LOT NO. 1919
DATE 3/16/70
BY JX

TEST NO. 1 NAME CALTA

Leach Test at 20 mesh, 4 hr. Leach

PRODUCT	Weight	Percent Weight	ASSAY										DISTRIBUTION		
				Cu		Ox Cu						Cu			
Extraction				13.933	gpl							61.7			
Leach Tail		96.0		0.904		0.226						38.3			
Head (Calc)		100.0		2.26								100.0			
# H_2SO_4/#Cu =	2.14														

OPERATION	Start													GRIND PRODU
TIME	9:20	9:14	9:20	9:25	9:35	9:40	9:55	10:20	11:05	11:55				
REAGENTS — LBS. PER TON														Leach
													MESH	%
													+10	
H_2SO_4	27.2	5.9	3.7	5.9	2.9	2.9	6.6	3.7	5.9	0.7			+14	
													+20	
													+28	10.4
Extraction	3.7 cc = 2.9 gpl H_2SO_4												+35	20.4
													+48	15.4
													+65	11.7
													+100	8.9
													+150	7.8
MACHINE													+200	3.7
R.P.M.													+325	5.2
pH	1.6												−325	16.5
% SOLIDS	50													
TEMPERATURE														

REMARKS:

METALLURGICAL RESEARCH DEPT.

CABLE "GALCHER"

THE GALIGHER COMPANY

P. O. BOX 209 — 565-535 WEST 5TH SOUTH STREET
SALT LAKE CITY, UTAH 84110 — U.S.A.

TELEPHONE 355-9701

OUR LOT NO. 1919
DATE 3/16/70
BY JK

TEST NO. 2 NAME CALTA

PRODUCT	Weight	Percent Weight	ASSAY Cu	Ox. Cu						DISTRIBUTION Cu			
Cl Conc		1.74	48.36	0.47						35.5			
Cl Tail		1.30											
Ro Tail		97.16											
Head (Assay)		100.00	2.37										

OPERATION	Cl	Ro	Ro	Ro									
TIME	10	4	2	3									
REAGENTS — LBS. PER TON													
Lime	5.0												
1-250	0.07		0.02										
Ro 65 - MBC		0.16											
MACHINE		1650		500									
R.P.M.		1600		1600									
pH	11.5												
% SOLIDS	50												
TEMPERATURE													

MESH	%
+10	
+14	
+20	
+28	
+35	
+48	1.6
+65	8.6
+100	16.0
+150	19.5
+200	10.0
+325	14.0
-325	29.7

REMARKS: Light froth at beginning of Ro.

METALLURGICAL RESEARCH DEPT.

CABLE "CALSAL"

THE GALIGHER COMPANY

P. O. BOX 209 — 545-555 WEST 5TH SOUTH STREET
SALT LAKE CITY, UTAH 84110 — U.S.A.

TELEPHONE 359-8731

OUR LOT NO. 1919
DATE 3/19/70
BY JK

TEST NO. 3 NAME CALSA

Same as Test 2 except float at 20 mesh.

PRODUCT	Weight	Percent Weight	ASSAY								DISTRIBUTION			
				Cu								Cu		
Cl Conc		1.12		44.64								22.3		
Crib Ro & Cl Tail		98.88		1.77								77.7		
Head		100.0		2.25								100.0		

OPERATION	Ro	Ro	Cl										GRIND PRODUCT
TIME	3	2	3										
REAGENTS — LBS. PER TON													See Test
												MESH	%
												+10	
												+14	
Lime	3.5											+20	
K-350	0.02	0.02										+28	
Pro 65 - MIBC	0.16											+35	
												+48	
												+65	
												+100	
												+150	
MACHINE	1000		500									+200	
R.P.M.	1000		1000									+325	
pH	11.4											-325	
% SOLIDS													
TEMPERATURE													

SAMPLER

METALLURGICAL RESEARCH DEPT.

CABLE "GALGAL"

THE GALIGHER COMPANY

P. O. BOX 200 — 145-185 WEST 8th SOUTH STREET
SALT LAKE CITY, UTAH 84110 — U.S.A.

TELEPHONE 359-3731

OUR LOT NO. 1919
DATE 3/20/70
BY JX

TEST NO. 3 - A NAME CALTA

500 gms of combined Re and Cl Tails leached for 4 hrs.

PRODUCT	Weight	Percent Weight	ASSAY Cu		DISTRIBUTION Cu	Cu %
Extraction			15.305	gpl	58.4	75.2
Leach Tail		97.8	0.450		19.3	24.8
Whole Tail		100.0	1.77		77.7	100.0
@ H_2SO_4/# Cu		2.51				

*Based on Float Tail

OPERATION			
TIME	9:25	10:50	1:25
REAGENTS — LBS. PER TON			
H_2SO_4	65.5	9.6	
Extraction (cc)	5.3 @	4.2 gpl	H_2SO_4
MACHINE			
R.P.M.			
pH	1.0		1.4
% SOLIDS			
TEMPERATURE			

REMARKS:

MESH	Gain Product %
+10	
+14	
+20	
+28	
+35	
+48	
+65	
+100	
+150	
+200	
+325	
-325	

METALLURGICAL RESEARCH DEPT.

CABLE "GALGAL"

THE GALIGHER COMPANY

P. O. BOX 200 — 845-865 WEST 6TH SOUTH STREET
SALT LAKE CITY, UTAH 84110 — U.S.A.

TELEPHONE 359-8731

OUR LOT NO. 1919
DATE 3/30/70
BY JX

TEST NO. 4 NAME CALTA

Leach at 20 mesh, grind and float leach Tail.

PRODUCT	Weight	Percent Weight	ASSAY								DISTRIBUTION			
				Cu							Cu*	Cu		
Cl Conc		1.2		49.24							61.9	24.7		
Cl Tail		2.1		0.926							2.0	0.8		
Re Tail		93.1		0.372							36.1	14.4		
Leach Tail		96.4		0.991							100.0	39.9		
Extraction				14.392	gpl							60.1		
Leach Tail		98.4		0.991								39.9		
Head		100.0		2.39								100.0		
											*Based on Leach Tail			

OPERATION	Start			@	Leach Tail RM	Re	Re	Cl					Grind Product
TIME	9:05	11:30	1:10	4 hr.	10	2	2	3					
REAGENTS—LBS. PER TON													Leach
												MESH	%
Lime					5.0							+10	
R-350					0.02		0.02					+14	
Pro 33-31NC						0.08	0.04					+20	
												+28	10.4
H_2SO_4	65.5	19.6	8.1									+35	20.4
												+48	15.4
												+65	11.7
												+100	8.9
Extraction	7.7 cc ~ 6.1 gpl H_2SO_4											+150	7.8
MACHINE						1000		500				+200	3.7
R.P.M.						1000		1000				+325	5.2
pH	1.3			1.5		9.2						−325	16.5
% SOLIDS	50												
TEMPERATURE													

REMARKS:

METALLURGICAL RESEARCH DEPT.

CABLE "GALEAL"

THE GALIGHER COMPANY

P. O. BOX 209 — 645-665 WEST 9th SOUTH STREET
SALT LAKE CITY, UTAH 84110 — U.S.A.

TELEPHONE 359-8731

OUR LOT NO. 1919
DATE 4/6/70
BY JX

TEST NO. 5 NAME CALTA

-20 mesh, 24 hr. leach, followed by -48 mesh grind and flotation.

PRODUCT	Weight	Percent Weight	ASSAY								DISTRIBUTION			
				Cu							Cu*	Cu		
Cl Conc		1.8		36.82							84.1	22.3		
Cl Tail		4.0		0.510							3.3	0.9		
Ro Tail		90.5		0.035							12.6	3.3		
Leach Tail		95.9		0.033							100.0	26.5		
Reconstitution				17.001	mpl							73.5		
Leach Tail		95.9		0.033								26.5		
Head		100.0		2.31								100.0		
												*Based on Leach Tail		

OPERATION	Start			PM	AM	off	RM	Ro	Ro	Cl			GRIND PRODUCT
TIME	9:25	10:15	1:35	4:45	8:10	9:25	10	2	3	5			
REAGENTS — LBS. PER TON													Leach
												MESH	%
Lime												+10	
H_2SO_4	65.5	4.4	32.9		1.8		5.0					+14	
R-350												+20	
Pro 65 - MIBC							0.02		0.05			+28	Same
								0.08	0.04			+35	Test
												+48	
												+65	
												+100	
Dilution (cc)	5.2 ml	4.1 ml	H_2SO_4									+150	
MACHINE								1000		500		+200	
R.P.M.								1000		1000		+325	
pH	1.1	1.6	1.6	1.452	1.6	1.6		9.6				—325	
% SOLIDS	50												
TEMPERATURE													

REMARKS:

METALLURGICAL RESEARCH DEPT.

CABLE "GALSAL"

THE GALIGHER COMPANY

P. O. BOX 200 — 545-585 WEST 6TH SOUTH STREET
SALT LAKE CITY, UTAH 84110 — U.S.A.

TELEPHONE 359-8731

OUR LOT NO. 1919
DATE 4/7/70
BY JY

TEST NO. 6 NAME CALTA

-48' mesh grind, 4 hr. leach followed by flotation.

PRODUCT	Weight	Percent Weight	ASSAY								DISTRIBUTION			
				Cu							Cu			
Cl Conc		1.5		38.02							23.5			
Cl Tail		1.8		0.774							0.6			
Ro Tail		92.1		0.198							7.5			
Leach Tail		95.4		0.803							31.6			
Extraction				16.549	gpl						68.4			
Leach Tail		95.4		0.803							31.6			
Head		100.0		2.42							100.0			

OPERATION	BM	Start					eff	Ro	Ro	Cl			Grind PRODU
TIME	10	10:10	10:25	10:40	12:00	1:00	2:10	2	3	4			Leach
REAGENTS — LBS. PER TON													& Wash
												MESH	%
Lime								5.0				+10	
H_2SO_4		65.5	3.7	5.1	2.2	0.7						+14	
R-350								0.02	0.05			+20	
Fro 65-MIBC								0.08	0.04			+28	
												+35	
												+48	1.6
												+65	
												+100	
Titration (cc)	4.1 =	3.2gpl	H_2SO_4									+150	
MACHINE								1000				+200	
R.P.M.								1000				+325	
pH		1.1					1.6	9.6				-325	29.7
% SOLIDS	50												
TEMPERATURE													

REMARKS: