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GEOLOGY OF THE HELVETIA COPPER DEPOSIT — ARIZONA

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New York, New York

For presentation at the SME Annual Meeting
Phoenix, Arizona — February 24-27, 1992

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Geology of the Helvetia Copper Deposit Arizona

The Helvetia copper deposit is a large undeveloped Laramide porphyry copper system located in the Santa Rita Mountains, Pima County, Arizona. This porphyry copper system consists of four areas of copper mineralization: Rosemont, Peach-Elgin, Broadtop Butte, and Copper World. Asarco refers to the four areas collectively as the "Helvetia Deposit". Mineralization and alteration are primarily contact pyrometasomatic, and zoning of hydrothermal alteration and sulfide mineral assemblages are similar to those observed at the Twin Buttes and Mission copper mines located approximately 32 km west of Helvetia. Asarco acquired the Helvetia Deposit in 1988 and has continued the exploration and development effort since then.

A considerable amount of excellent geological work has been completed in the area but little information has been published on developments since the mid-1950's. The geology of the deposit as developed by numerous geologists over the past 75 years and the large, bulk tonnage low grade copper deposits outlined in recent years will be briefly reviewed in this paper.

Location

The Helvetia copper deposit is located approximately 50 km southeast of Tucson, Arizona, in the northern Santa Rita Mountains (Figure 1). It lies within the Basin and Range Physiographic Province at elevations ranging from 1402 m to 1890 m.

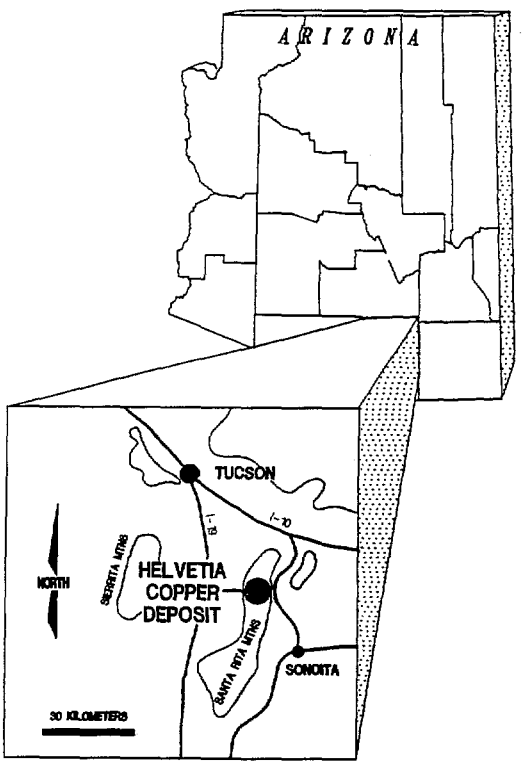
Exploration and Mining History

Copper mineralization may have been discovered in the Helvetia district prior to the Civil War, but no records are available for these early discoveries. The district has had a relatively small production of copper ore principally from underground mines. In the late 1880's copper ore from the district was treated at the Columbia Smelter, located on the west side of the Santa Rita Mountains, and the Rosemont Smelter, located on the east flank of the same range near the Rosemont Camp (Creasey, 1955). In 1903 the Helvetia Copper Company began operation and continued until 1911. Copper was produced almost continuously from 1915 to 1951. In the 1940's, some disseminated copper mineralization skarns were mined from small open pits located in the Elgin area. Total production from the Helvetia District through 1950 totalled 227 300 tons of ore containing 17 290 000 pounds of copper, 1 097 980 pounds of zinc, and 180 760 ounces of silver (Schrader, 1915; Creasey, 1955).

After 1950, activities consisted mainly of exploration and development drilling. The Lewisohn Copper Company conducted a drilling program in the Peach-Elgin area in 1955 and in 1956 outlined a possible open pit copper deposit in the Peach Hill area (Figure 3). Drilling in 1956 by American Exploration and Mining Co. in the Ingersoll breccia area, located southeast of Broadtop Butte, failed to outline an economic deposit. In the late 1950's the Helvetia Deposit was acquired by the Banner Mining Company, and a modest exploration drilling program was conducted in the area. During this Banner program, drill hole G-33 penetrated the first significant porphyry copper mineralization in the Rosemont area. This hole contained a 300 m intercept of greater than 0.90% Cu mineralization. Anaconda Mining Company acquired the property in 1963 and carried out an extensive mapping and drilling program. The vast majority of the modern information on the Helvetia Deposit was developed by Anaconda and Anamax personnel. Their efforts resulted in the delineation of the Rosemont area porphyry copper deposit, a major North American copper resource. The property was incorporated into the Anamax Mining Co. when Amax joined Anaconda in a partnership in 1973. Anamax sold the property to a real estate company in 1986 which in turn sold it to Asarco in 1988.

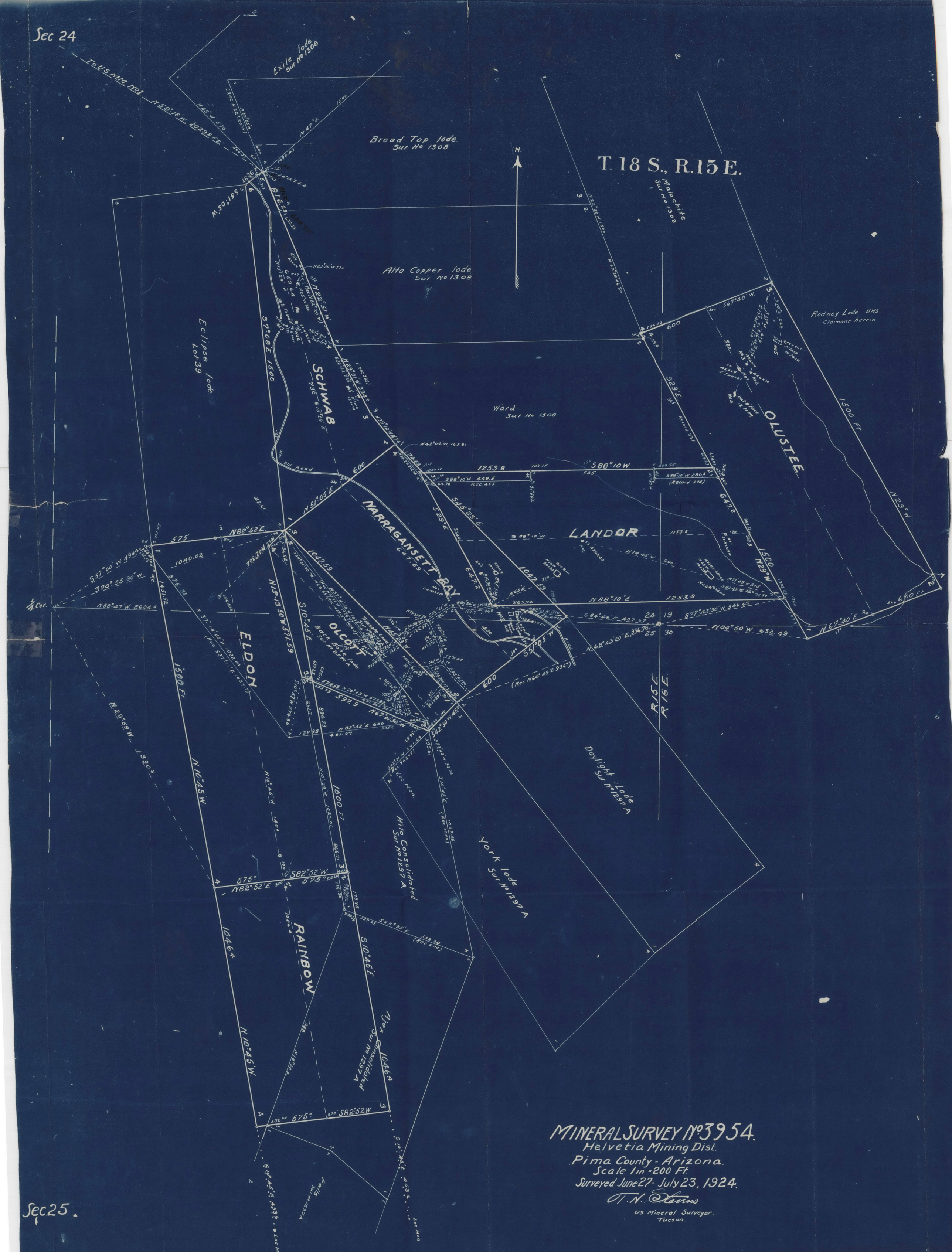
Geology

The four Helvetia copper deposits occur within a series of moderate to steeply dipping Paleozoic and Mesozoic sedimentary rocks that have been intruded by Laramide igneous rocks. Mineralization and alteration are primarily contact pyrometasomatic (Creasey, 1955), and hydrothermal alteration and zoning of sulfide



LOCATION MAP - HELVETIA COPPER DEPOSIT

FIGURE 1

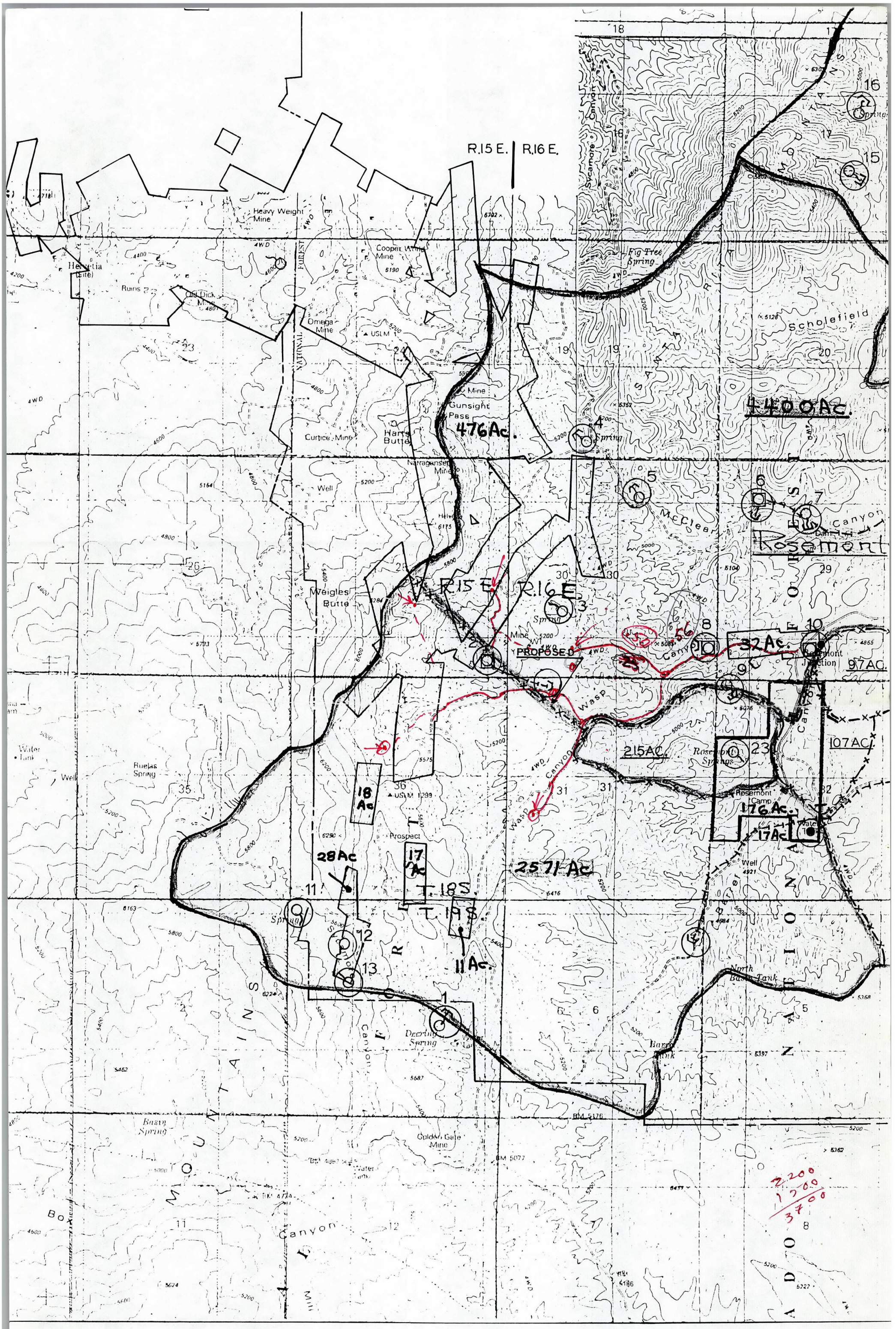

Sec 24
To U.S.M.M. No 1
N 59°18' W 20298 Ft
Exile lode
Sur No 1308
Broad Top lode
Sur No 1308
Alta Copper lode
Sur No 1308
Ward
Sur No 1308
T. 18 S., R. 15 E.
Malachite
Sur No 1308
Rodney Lode UNS
Claimant herein
Eclipse lode
Lot 39
SCHWAB
OLUSTEE
NARRAGANSETT BAY
LANDOR
ELDON
OLCOTT
back bone lode
Sur No 2350 A
Triple Lode
Sur No 2324
RAINBOW
Hilo Consolidated
Sur No 1297 A
York lode
Sur No 1297 A
Daylight Lode
Sur No 1297 A
R 15 E
R 16 E
Old Road
1/4 Cor.
Sec 25.
MINERAL SURVEY No 3954.
Helvetia Mining Dist.
Pima County - Arizona.
Scale 1 in = 200 Ft.
Surveyed June 27 - July 23, 1924.
T. N. Stearns
US Mineral Surveyor.
Tucson.

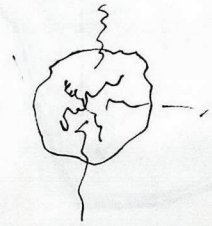
Corrected
Oct 24 1924

59 18
51 05
170 23
69·37

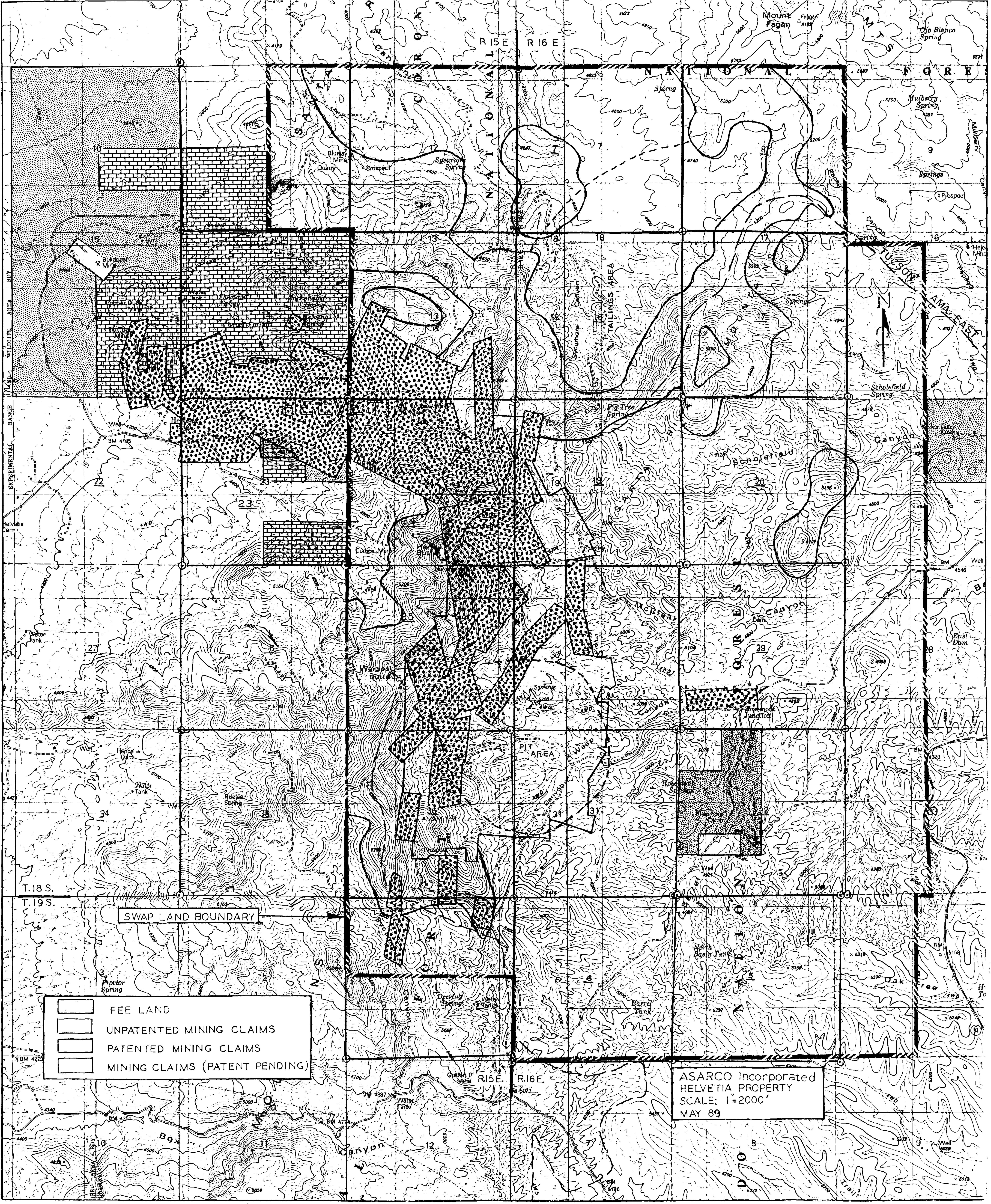
82 52
51 05
31·47

51 05
170 23
56 13
21·47

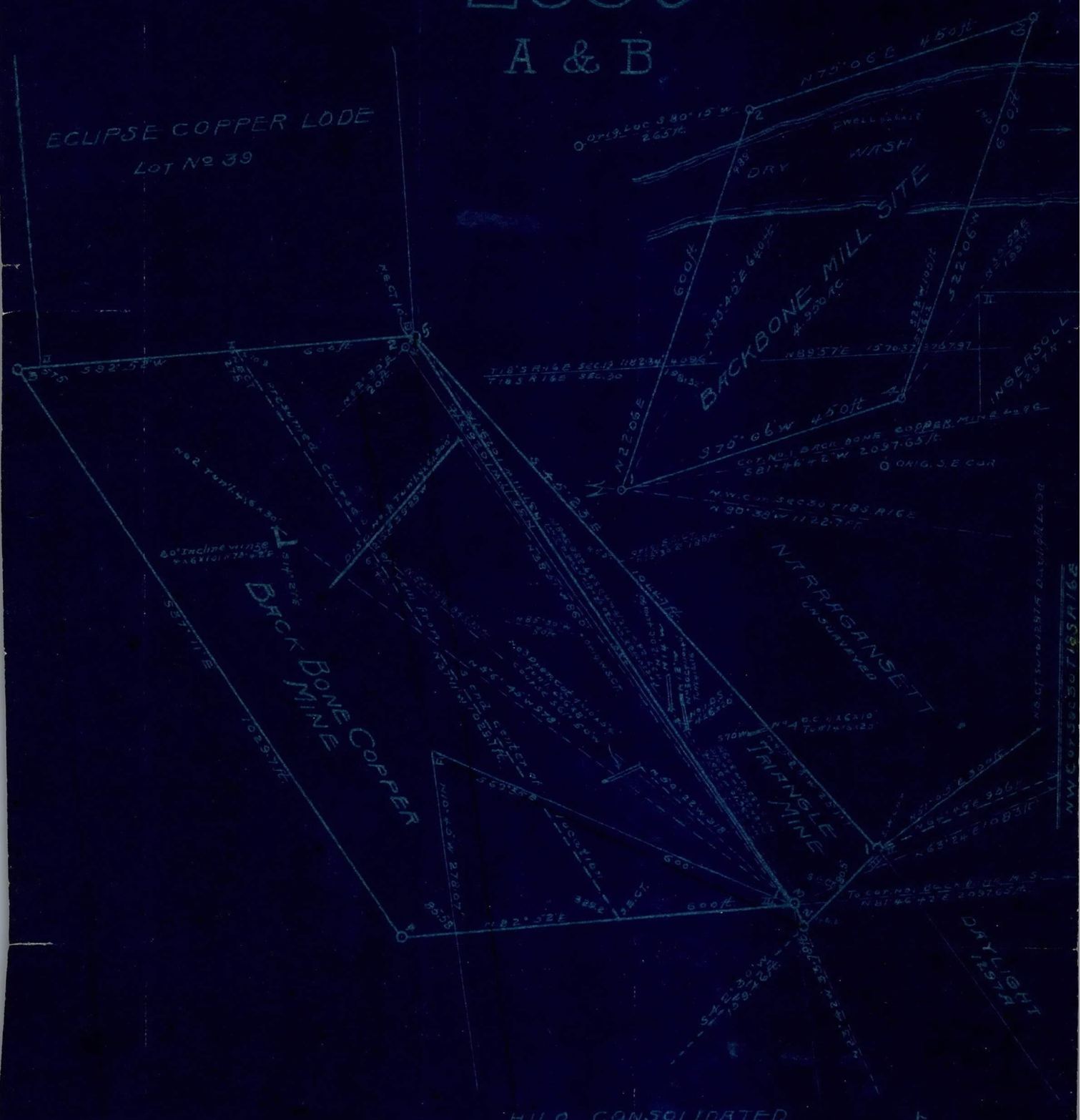

R.15 E.
R.16 E.
Heavy Weight Mine
Cooper Wing Mine
Helvetia (Site)
Ruins
Old Dick Mine
Omega Mine
USLM
Mine
Gunsight Pass
476Ac.
Curtice Mine
Harris Butte
Narragansett Mine
Well
Weigles Butte
Fig Tree Spring
Sycamore Canyon
Scholefield
440Ac.
Spring
McClear
Canyon
Rosemont
R15E
R16E
PROPOSED
32Ac.
Rosemont Junction
97AC.
Wasp
Canyon
215AC.
Rosemont Springs
107AC.
Rosemont Camp
176Ac.
17Ac.
Water
Ruelas Spring
18 Ac
Prospect
17 Ac
28Ac
2571 Ac
T.18S.
T.19S.
11Ac.
North Basin Tank
Decoring Spring
Basin Tank
Basin Spring
Golden Gate Mine
BM 5176
BM 5077
Box Canyon
Mill Canyon
2200
1700
3700



Bulldozer Mine
Copper Duke Mine
Tip Top Mine
Peach Knob
Water Tank
Shamrod Spring
Zackendorf Spring
Helvetia Spring
Chavez Spring
Prospect
Heavy Weight Mine
Copper World Mine
Helvetia
Ruins
Old Dick Mine
Omega Mine
USLM
NATIONAL FOREST BOUNDARY
EXPERIMENTAL RANGE AND WILDLIFE AREA BDY
Helvetia Cem
BM 4185
Gunsight Pass
Harts Butte
Curtice Mine
Narragansett Mine
Weigles Butte
Hanna Dam
Ruelas Spring
Proctor Spring
Basin Spring
Deering Spring
Golden Gate Mine
Sycamore Canyon
Box Canyon
Mill Canyon
Corral
Fig Tree Spring
SANTA RITA MOUNTAINS
Scholefield Canyon
Scholefield Spring
Hidden Valley Ranch
Mulberry Canyon
Papago Canyon
McCleary Canyon
Barrel Canyon
Wasp Canyon
Rosemont Junction
Rosemont Springs
Rosemont Camp
East Dam
Adobe Tank
Rest Area
Drill Holes
North Basin Tank
Barrel Tank
Oak Tree Canyon
Highway Tank
CORONADO NATIONAL FOREST
Trail
Spring
Well
Mine
4WD
GUNSIGHT 34


R15E R16E
NATIONAL FOREST
TAILINGS AREA
HELVETIA
PIT AREA
SWAP LAND BOUNDARY
T.18S.
T.19S.
FEE LAND
UNPATENTED MINING CLAIMS
PATENTED MINING CLAIMS
MINING CLAIMS (PATENT PENDING)
R15E. R16E.
ASARCO Incorporated
HELVETIA PROPERTY
SCALE: 1"=2000'
MAY 89

SURVEY
2550
A & B

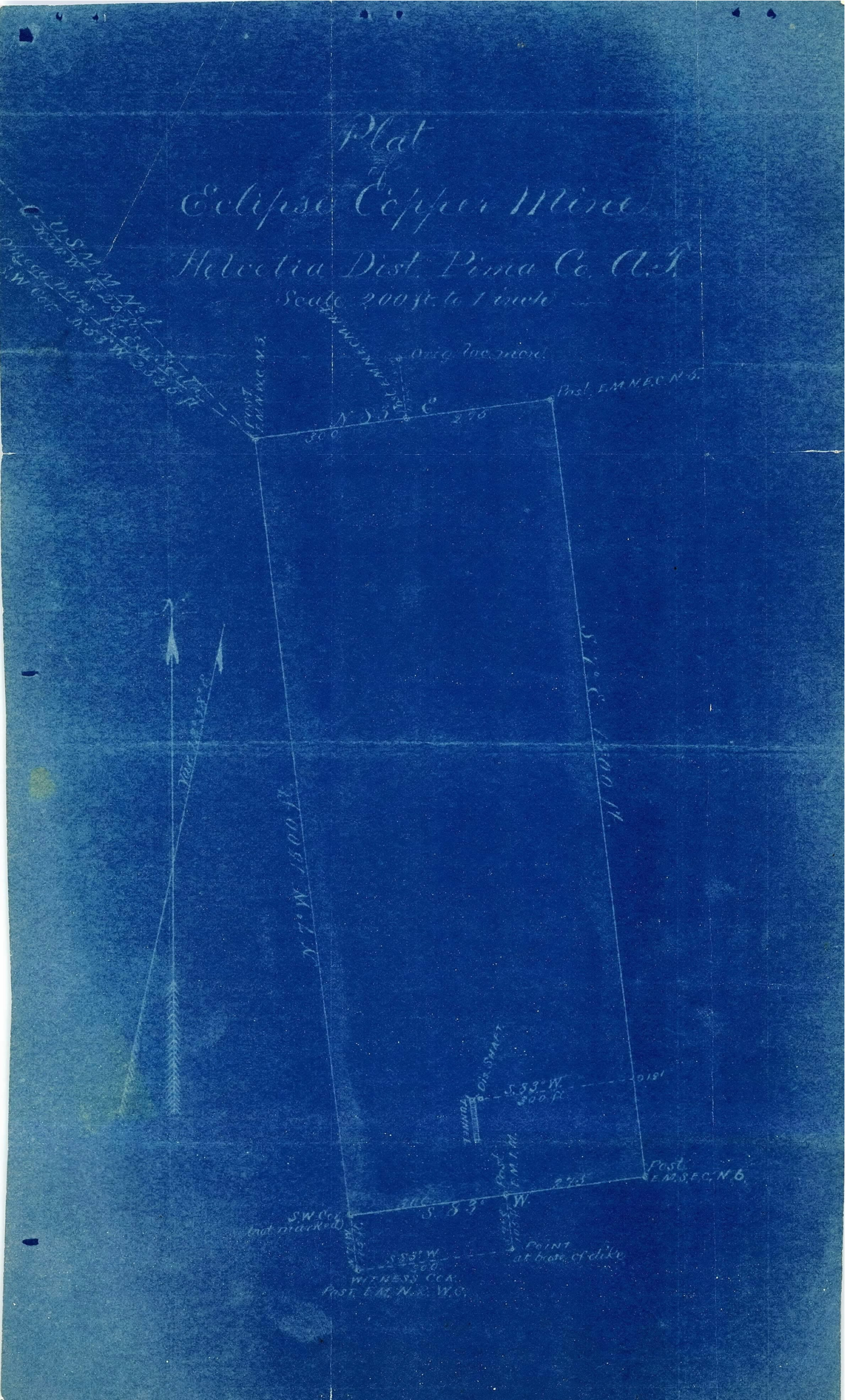


Geo. C. Clark
U.S. MINERAL SURVEYOR
AUG 22ND 1910

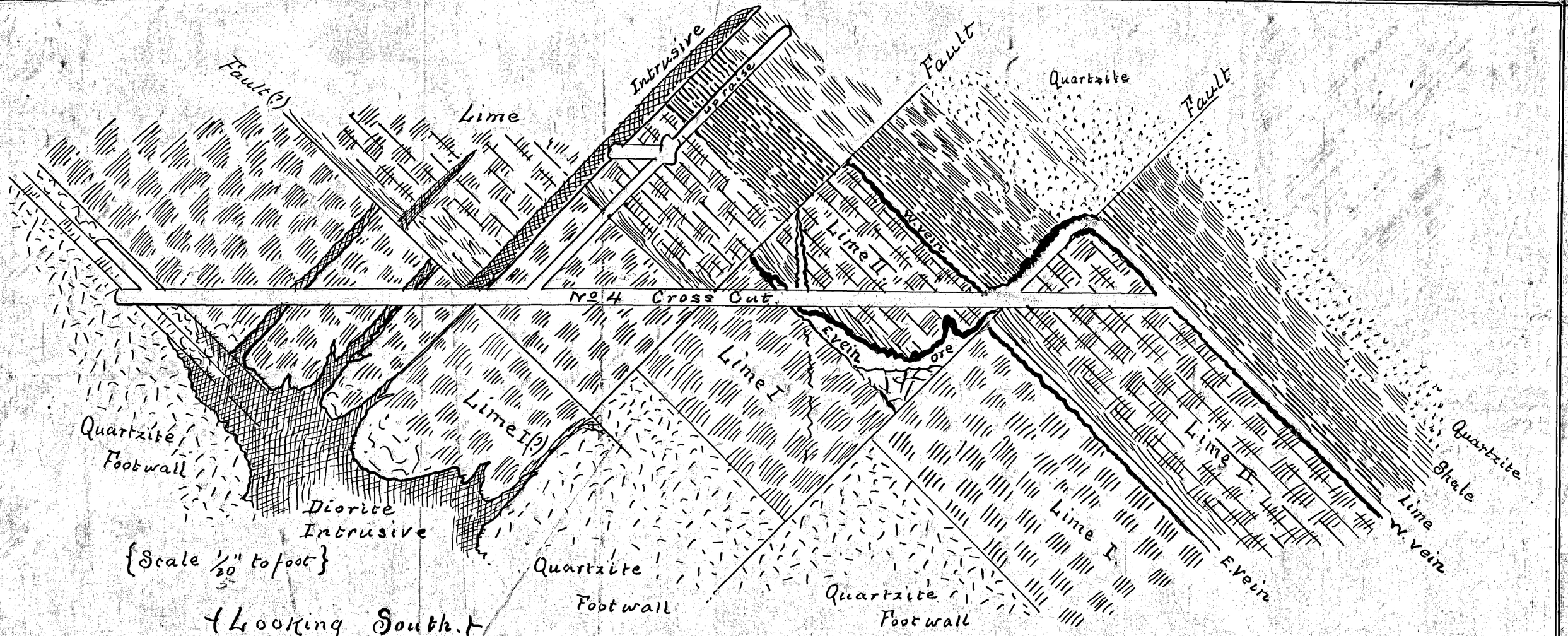
HELVETIA File

Plat
of
Eclipse Copper Mine
Helvetia Dist. Pima Co. A.T.

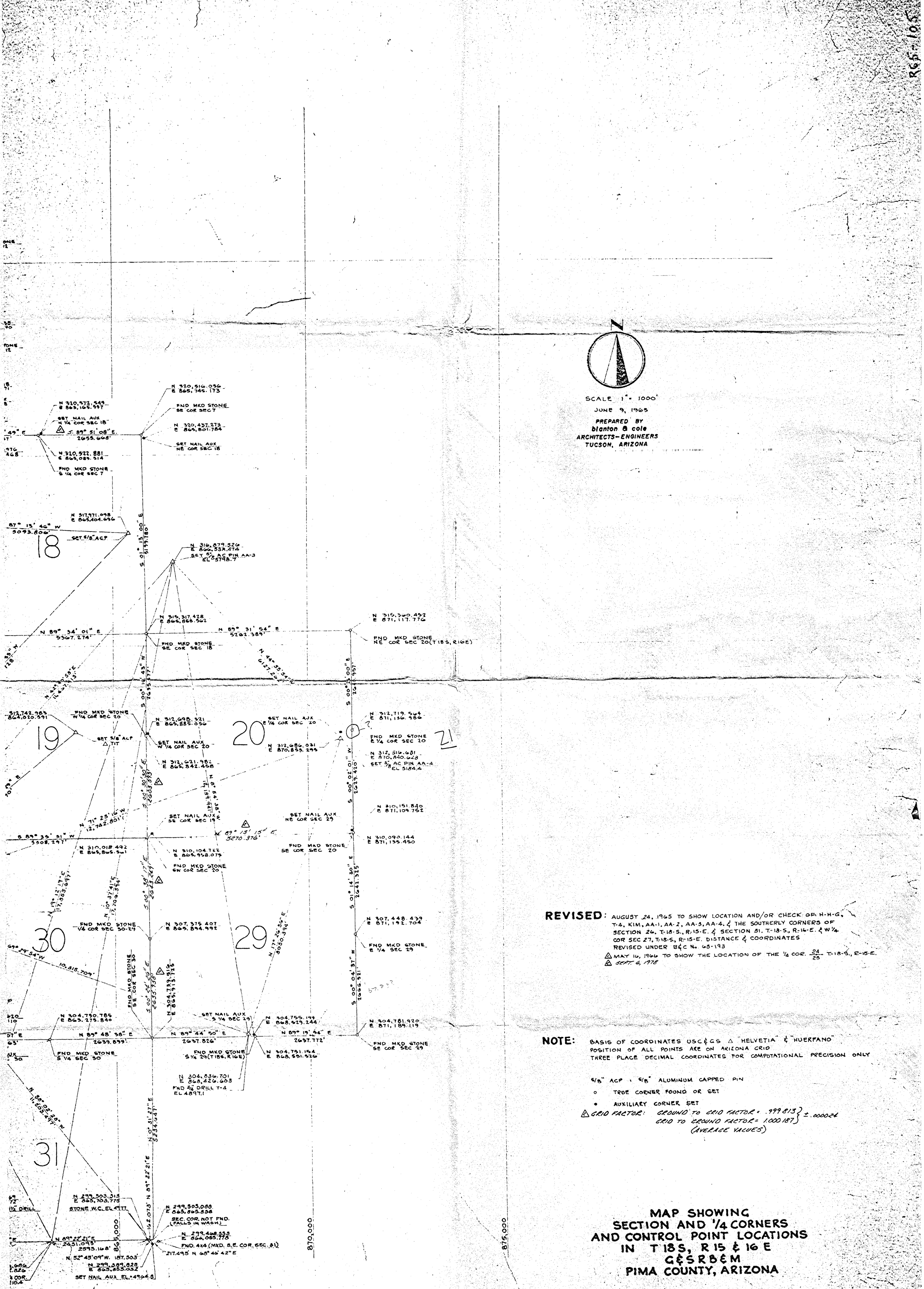
Scale 200 ft. to 1 inch



Helvetia File



Upper Graphic. No. 4.



REVISED: AUGUST 24, 1965 TO SHOW LOCATION AND/OR CHECK OF H-H-G, T-4, KIM, AA-1, AA-2, AA-3, AA-4, & THE SOUTHERLY CORNERS OF SECTION 24, T-18-S., R-15-E. & SECTION 31, T-18-S., R-16-E. & W 1/4 COR SEC 27, T-18-S., R-15-E. DISTANCE & COORDINATES REVISED UNDER B&C No. 65-193

△ MAY 16, 1966 TO SHOW THE LOCATION OF THE 1/4 COR. 24/25 T-18-S., R-15-E.

△ SEPT. 6, 1978

NOTE: BASIS OF COORDINATES USC&GS △ "HELVETIA" & "HUERFANO"
POSITION OF ALL POINTS ARE ON ARIZONA GRID
THREE PLACE DECIMAL COORDINATES FOR COMPUTATIONAL PRECISION ONLY

5/8" ACP = 5/8" ALUMINUM CAPPED PIN

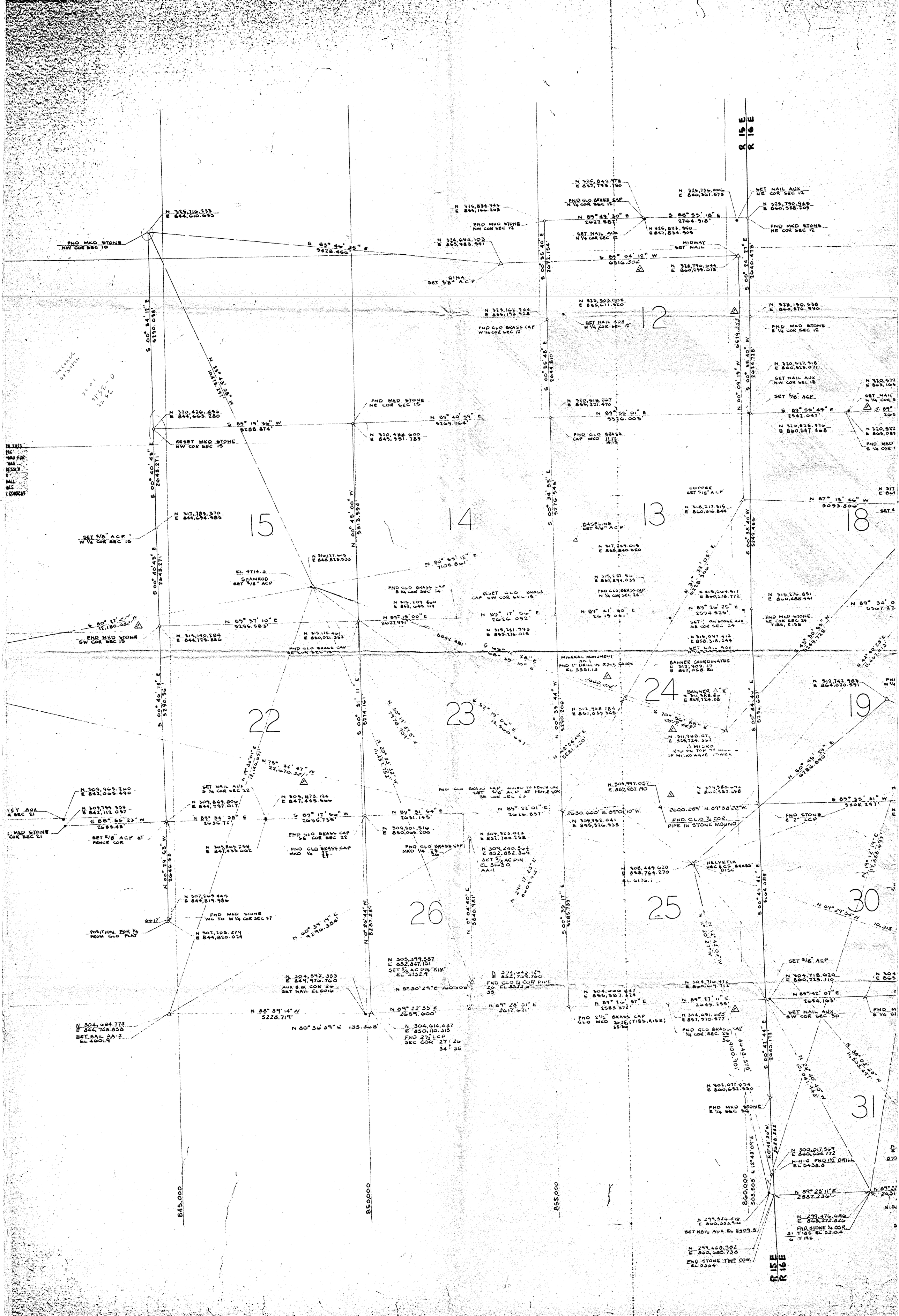
o TRUE CORNER FOUND OR SET

• AUXILIARY CORNER SET

△ GRID FACTOR: GROUND TO GRID FACTOR = .999813 } ± .000024
GRID TO GROUND FACTOR = 1.000187 }
(AVERAGE VALUES)

MAP SHOWING SECTION AND 1/4 CORNERS AND CONTROL POINT LOCATIONS IN T 18 S, R 15 & 16 E G&SRB&M PIMA COUNTY, ARIZONA

E-65-105


R 15 E
R 16 E
FND MKD STONE NW COR SEC 10
N 325,716.233 E 844,610.653
S 83° 46' 35" E 9428.466'
N 325,834.945 E 855,106.205
FND MKD STONE NW COR SEC 12
N 324,694.103 E 855,983.541
SET 5/8" A C P GINA
N 325,842.973 E 857,793.760
FND GLO BRASS CAP N 1/4 COR SEC 12
N 89° 49' 30" E 2627.587'
SET NAIL AUX N 1/4 COR SEC 12
N 325,823.950 E 857,834.905
S 88° 55' 18" E 2764.918'
N 325,736.006 E 860,361.575
SET NAIL AUX NE COR SEC 12
N 325,790.945 E 860,358.209
FND MKD STONE NE COR SEC 12
MIDWAY SET NAIL
S 89° 04' 12" W 6316.304'
N 324,796.644 E 860,299.013
S 00° 35' 40" E 2672.154'
S 00° 24' 27" E 2640.473'
N 323,303.005 E 855,611.520
N 323,162.934 E 855,193.928
FND GLO BRASS CAP W 1/4 COR SEC 12
SET NAIL AUX W 1/4 COR SEC 12
N 323,190.538 E 860,576.990
FND MKD STONE E 1/4 COR SEC 12
N 320,527.918 E 860,523.071
SET NAIL AUX NW COR SEC 18
SET 5/8" ACP
N 320,572 E 863,165
SET NAIL N 1/4 COR S
S 89° 56' 49" E 2542.047'
N 320,525.976 E 860,547.468
N 320,522 E 863,089
FND MKD S 1/4 COR
S 00° 34' 12" E 5290.098'
N 23° 43' 08" W 10,473.297'
N 320,426.456 E 844,663.280
S 89° 19' 36" W 5288.874'
RESET MKD STONE NW COR SEC 15
FND MKD STONE NE COR SEC 15
N 89° 40' 39" E 5269.764'
N 320,488.600 E 849,951.789
N 320,518.267 E 855,221.470
N 89° 55' 01" E 5326.003'
FND GLO BRASS CAP MKD 11 12 14 13
S 00° 35' 48" E 2644.810'
S 00° 34' 53" E 5276.545'
S 00° 40' 43" E 2645.271'
N 317,783.370 E 844,694.383
SET 5/8" A C P W 1/4 COR SEC 15
N 00° 45' 00" W 5318.594'
N 87° 13' 46" W 5093.806'
SET 5
COPPEK SET 5/8" A C P
N 318,217.315 E 860,316.844
BASELINE SET 5/8" A C P
N 317,249.015 E 855,840.350
N 315,251.511 E 857,894.039
FND GLO BRASS CAP N 1/4 COR SEC 24
N 315,269.917 E 860,278.772
N 315,276.851 E 860,488.441
N 89° 34' 0 5367.27
N 316,127.615 E 848,823.533
N 80° 53' 12" E 7105.861'
EL 4714.3
SHAMROD SET 5/8" A C P
FND GLO BRASS CAP S 1/4 COR SEC 14
N 315,209.660 E 862,649.119
RESET GLO BRASS CAP SW COR SEC 13
N 89° 17' 56" E 2626.092'
N 315,241.393 E 855,276.015
N 89° 41' 30" E 2619.041'
N 89° 26' 25" E 2594.525'
SET ON STONE AUX NE COR SEC 24
N 315,047.412 E 858,318.244
SET NAIL AUX N 1/4 COR SEC 24
FND MKD STONE NE COR SEC 24 T18S, R15E
S 80° 27' 51" W 12,180.682'
FND MKD STONE SW COR SEC 15
N 89° 37' 10" E 5295.583'
N 315,140.284 E 844,725.886
N 89° 15' 00" E 2627.991'
N 315,176.461 E 850,021.356
FND GLO BRASS CAP SE COR SEC 15
MINERAL MONUMENT NO. 1 FND 1" DRILL IN ROCK CAIRN EL 5351.13
BANNER COORDINATES N 312,909.27 E 867,058.80
BANNER △ "K" N 311,988.61 E 859,724.08
△ MICRO ROD ON TOP OF HILL OF MICROWAVE TOWER
N 312,742.983 E 864,020.591
N 312,958.184 E 857,059.345
S 00° 46' 13" E 5290.96'
N 19° 36' 01" E 12,146.671'
N 75° 36' 47" W 22,670.327'
N 30° 19' 37.5" W 7478.109'
N 20° 33' 32" W 11,457.152'
S 52° 19' 06" E 12,560.641'
N 50° 45' 29" E 6786.870'
N 309,965.260 E 842,065.490
1ST AUX 4 SEC 21
N 309,799.339 E 842,112.057
S 88° 55' 23" W 2686.48'
1 MKD STONE COR SEC 21
SET 5/8" A C P AT FENCE COR
SET NAIL AUX S 1/4 COR SEC 22
N 309,849.806 E 844,797.017
N 309,873.144 E 847,435.566
N 89° 34' 38" E 2636.72'
S 89° 17' 56" W 2635.735'
N 309,869.258 E 847,435.662
FND GLO BRASS CAP SE COR SEC 22
FND GLO BRASS CAP MKD 1/4 23 27
FND GLO BRASS CAP WIRED TO FENCE COR SET 5/8" A C P AT FENCE COR SE COR SEC 23
N 89° 31' 54" E 2631.145'
N 309,901.516 E 850,069.200
FND GLO BRASS CAP MKD 1/4 23 26
N 89° 22' 01" E 2626.837'
N 309,923.026 E 852,700.258
N 309,240.564 E 852,852.309
SET 5/8 A C PIN EL 5163.0 AA-1
N 309,997.057 E 867,967.190
2600.269' N. 89° 38' 22" W.
FND C.L.O. 1/4 COR. PIPE IN STONE MOUND
2630.640' S. 89° 01' 10" W.
N 309,952.041 E 855,326.935
N 309,980.693 E 860,557.298
S 89° 35' 31" W 5308.297'
FND STONE & 2" LCP
N 307,269.445 E 844,819.986
FND MKD STONE WC TO W 1/4 COR SEC 27
66.17'
POSITION FOR 1/4 FROM GLO PLAT
N 307,203.279 E 844,820.024
N 60° 39' 19" E 9296.354'
N 0° 26' 44" W 5267.289'
N 0° 04' 40" E 5840.981'
S 00° 39' 17" E 5285.789'
N 24° 17' 23" E 8609.914'
HELVETIA USC & GS BRASS DISC
N 308,449.620 E 858,764.270
EL 6176.1
N 19° 12' 19" E 11,855.697'
N 69° 29' 54" W 10,315.
N 305,399.587 E 852,847.151
SET 5/8 AC PIN "KIM" EL 5732.9
N 304,592.353 E 849,976.760
AUX S.W. COR 26 SET NAIL EL 5016
N 304,643.129 E 852,709.760
FND GLO 1/4 COR PIPE 26 35 EL 5322.0
N 5° 50' 29" E 760.408'
N 304,666.847 E 855,387.324
N 89° 26' 57" E 2583.572'
N 304,716.974 E 860,619.714
N 89° 27' 11" E 2649.255'
N 304,691.685 E 857,970.577
SET 5/8" ACP
N 304,718.620 E 860,729.110
N 89° 42' 07" E 2654.163'
SET NAIL AUX SW COR SEC 30
N 304,684.773 E 844,748.858
SET NAIL AA-2 EL 4601.9
N 89° 39' 14" W 5228.719'
N 89° 22' 35" E 2659.600'
N 89° 28' 51" E 2617.671'
N 80° 36' 39" E 135.368'
N 304,614.437 E 850,110.315
FND 2 1/2" LCP SEC COR 27 26 34 35
FND 2 1/2" BRASS CAP GLO MKD 25 26 (T18S, R15E) 35 36
FND GLO BRASS CAP 1/4 COR. SEC. 25 36
S 00° 42' 44" E 2640.172'
N 26° 10' 40" W 10,041.343'
N 35° 03' 28" W 11,505.497'
N 302,077.924 E 860,652.530
FND MKD STONE E 1/4 SEC 36
N 300,017.569 E 860,664.772
H-M-G FND 1 1/2 DRILL EL 5438.8
N 89° 28' 11" E 2587.236'
N 299,326.416 E 860,553.916
SET NAIL AUX EL 5409.5
N 299,476.686 E 863,272.826
FND STONE 1/4 COR. 31 T18S EL 5210.4 6 T19S
N 299,440.982 E 860,685.738
FND. STONE TWP COR. EL 5364
845,000
850,000
855,000
860,000
R 15 E
R 16 E
12
13
14
15
18
19
22
23
24
25
26
30
31

INTERIM APPRAISAL OF THE PEACH – ELGIN AREA
OF THE HELVETIA MINING PROPERTIES
OF LEWISOHN COPPER CORP.

By

Harrison A Schmitt

With Appended Sections

by

Howard R. Hendricks

and

Arthur M. Heyman

April 12, 1957

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Harrison Schmitt

April 12, 1957

Lewisohn Copper Company
128 North Church Street
Tucson, Arizona

Subject: Interim appraisal of the Peach-Elgin disseminated copper mineralization for open cast exploitation based on the past and current drilling results and the most recent geological information.

SUMMARY OF CONCLUSIONS

The development work on the Peach and Elgin areas at Helvetia is still not sufficiently advanced to form a basis for an accurate interim valuation. Mill concentration problems have been emphasized by the heavy oxidation disclosed, especially in the Peach area. Special research on the ore is required and for this, fresh bulk samples from underground workings will be needed. The churn drill sludges cannot be relied upon for definite conclusions. Test milling recoveries from them are likely to be too low.

The Elgin area is an excellent prospect for an additional tonnage of copper mineralization that is less oxidized than that at the Peach. Successful development of such mineralization could reduce the emphasis on the oxidized copper mineralization problem and at the same time, possibly make an ore body available for direct shipment to the operating and/or projected concentration plants in the Twin Buttes area.

The above program would not cover the Broad Top Butte area which appears to deserve testing by drilling.

The fact that the limits of the Peach area have not been defined to the east and northeast over a north-south distance of approximately 2400 ft., which includes a possible extension into the newly acquired Tip Top area and the large block of ground between the Peach and Elgin blocks, indicates the justification for a large ultimate drilling program which would start with around fifty holes. Part of this ground is about a hundred feet lower than the Peach and may be less oxidized.

The expenses for the limited program proposed for the first stage would be around $100,000. This plus the expense for the more complete program that can now be foreseen would be around $432,000. The total time required would be around fourteen months.

TEXT

At this date 27 churn drill and 17 diamond drill vertical surface holes have been completed in the Peach-Elgin area. Assay results are available up to and including hole C-49. The sampling of the churn drill holes was done according to standard procedure (Appendix 5). The diamond drill holes did not give satisfactory samples, and sone of the locations may need to be redrilled by churn drills.

Arthur Heyman made detailed geological studies of the area during the past twelve months or so. Substantial use of his work has been made when compiling the reserves, i.e., tonnage, grade, and distribution of the copper mineralization (Appendix 3). His summary of the geological setting is given in Appendix 4.

Tests of the milling or concentration characteristics of the churn drill hole sludges and diamond drill core samples have been made by Howard Hendricks. It is appreciated that sludge samples are not normally suitable for mill testing, but no other material has been available. Before underground openings for sampling could be planned, however, the position and size of the body of copper mineralization had to be determined. The substantial funds needed for such openings, furthermore, competed with the need of funds for the minimum of drilling required.

The drilling now completed indicates the high probability that the minimum tonnage believed to be required for a profitable operation, namely 20,000,000 tons of 0.8 to 0.9% copper will be blocked out or proved if drilling is continued. The minimum grade that is commercial depends, to a large degree, upon the maximum recovery of copper by milling. The uncertainty of obtaining an adequate recovery from a substantial part of the Peach mineralization, as confirmed by recent drilling, overshadows the need for immediate expansion of the drilling pattern. The Peach mineralization as now outlined appears to be around 40% oxidized. Among other things, milling recovery is decreased by the presence of chrysocolla and coated sulphides. An adequate recovery by standard flotation methods appears doubtful. The above facts, then, lead to immediate major problems: (1) the need for good samples for mill testing, and (2) the initiation of a research testing job on these samples.

Appended to this report are cross-sections through the Peach area of mineralization showing drill holes, individual assays, pertinent geology and a possible pit outline. The data are based on surveys by Robert Fordham and were compiled by Heyman, Fordham and me. Also appended is a plan map showing the locations of the cross-sections and the pertinent surface geological data.

The cross-sections transect some parts of the block of mineralization that are nearly or fully drilled out and others that are insufficiently drilled. In the first case, the largely geometrical standard terms proved (or positive), probable, and possible are used. In the second case a fourth term, geologically probable, is used. Here the geologic aspect assumes relatively great importance when judging and estimating the volume of mineralization.

CDH No. 49 was completed after the tonnage estimates were made. Its location and accompanying assays are shown on the plan map and Section 8432. A height of 250 ft. of copper mineralization includes a height of 205 ft. that assays 0.7% copper. The overburden is 185 ft. thick and waste–"ore" ratio is 230 to 205. There is 30 ft. of material in the capping that assays 0.63% copper. Possibly this material can be stockpiled and eventually milled.

The good results of this hole and of hole 33, along with fair results of hole 39, indicate the high probability that the Peach mineralization at the south end is curving eastward and pointing toward the Elgin area. CDH No. 46 was drilled on the extreme south side of the in-between area and cut 80 ft. of 0.65% copper.

The distribution of the holes suggests that the zone or belt of mineralization may be wider in this in-between area (see the surface map Appendix 1). Many more test holes are justified in this area.

The east side of the Peach mineralized area has by no means been defined. The drilling pattern is incomplete; and the diamond drill holes, especially 7, 8, 9, and 10, are of doubtful validity because the sludge was not consistent in volume or adequately recovered. Normally, the sludge assays higher than the core, especially where the core recovery is low. Holes 7 through 10, then, should be redrilled with churn drill rigs.

The data given on most of the cross-sections are largely self-explanatory. The "cut-off" grade used is around 0.4% copper depending on local conditions. A maximum backslope angle of 45 degrees is assumed for the east side where much of the area is massive limestone. On the west side, because of the known geological conditions, the backslope used can be at a very low angle with no loss of tonnage.

The more important blocks of waste in the drill holes are indicated. They have been removed from the tonnage and grade estimates and added to the waste tonnage. The tonnage reserve blocks on each cross-section were measured with a planimeter and the third dimension for obtaining the volume was derived by using the sum of one-half the distances to each of the two adjoining sections.

The grade of the copper mineralization of each cross-section was obtained by weighting the average grade of the intercept in each drill hole by the length of the intercept. The average grade of each of the main areas A (north Peach) and B (south Peach) was obtained by weighting the grade of each block of mineralization of a given category (proved, probable, etc.) By the tonnage represented.

The Elgin area, Area C, is partially developed by four drill holes, three at the apices of a 285 ft. equilateral triangle and a fourth about 1000 ft. west of the triangle. The area of outcropping favorable capping is 800,000 sq. ft., in round numbers. The average thickness of the waste and over-burden in the three holes is 25 ft., and of the "ore" 143 ft., but the good ground may extend northwestward under the mantle rock there and westward may connect with the Peach area. The apparent favorable area calculates to 1,670,000 tons of waste and 10,400,000 tons of "ore" of an average grade of 0.73% copper. This area and the possible extensions noted is, of course, inadequately drilled. The drill hole pattern needs to be filled out for the apparent favorable area and extensions sought out. In the tabulations this tonnage of copper mineralization also is called geologically probable. The grade of the block as finally outlined, because of the variation usually expected in test holes, will probably be higher or lower than the average calculated for the above holes.

It should be emphasized that the weighting of the grade of the three area, A, B, and C, is distorted by the Elgin, or area C, because the true grade of the large geologically probable tonnage of the Elgin has not been established. In other words, the combined grade of the three areas may be higher.

The critical data pertaining to the above discussion are tabulated in Appendix 3.

The value of the property will remain indefinite until exhaustive milling or concentrating tests are made on the "ore", especially the "ore" on the Peach that is so heavily oxidized, and until the drilling outlines a minimum of 20,000,000 tons or, indeed, until the maximum limits of the mineralized areas are found. The latest drilling on much or all of the east side of the Peach has yet to find the limit in that direction.

The Broad Top Butte area, in the eastern part of the area owned by the Company, is one of widespread brecciation and copper staining. This area should be kept in mind as one that has a good chance of containing another block of commercial copper mineralization. The local rock is brecciated quartzite and porphyry so standard leaching methods should be feasible on any oxidized blocks of mineralization as may be found.

Previous estimates using depletion, amortization and taxation factors have indicated that an adequate profit can be made if the following minimum basis is used.

Tonnage	20,000,000
Grade	0.9% copper
Price of copper	$0.30 per pound
Stripping ratio	1 to 1
Recovery	80%
Rate of milling	2,500 per day

The milling tests so far made by Howard Hendricks are listed in Appendix 6. The average recovery by floatation was 65.1 %. The leaching test indicate that the pulps have a high acid neutralizing power.

Conclusions.

The activity at Helvetia is still one of development of mineralized ground that cannot yet be called an ore body of ore bodies. The two chief current problems are (1), the most important, the development of a milling method that will give an adequate recovery of copper from the bulk of the Peach mine mineralization and, (2) the completion of the drilling program. The latter work, at first at least, would seem to require most immediate emphasis on the Elgin area (Area C). This is because the development of a large amount of less oxidized ground such as has so far been revealed there, would take the emphasis away from the substantially oxidized Peach "ore" and raise the overall average of anticipated mill recovery. Furthermore, the development of some millions of tons of high-recovery copper mineralization with a low overburden-to-"ore" ratio would probably make feasible an enterprise of shipping ore by truck to the Duval Sulphur and Potash Company's proposed mill at Twin Buttes or to the Mineral Hill and/or the Pima Mine Mills in the same district.

The estimated minimum reserves of proved, probable, possible and geologically probable block in the Peach area are 9,538,758 tons with a grade of 0.869 % copper (Appendix 3, Table III). Certain possibilities east of this block are not included in this total. On the Elgin ground, because of the size of the area of favorable capping and the good results of the first drill holes, there is a high geological probability of a minimum of ten million tons of high mill recovery, low waste ratio "ore" in this area. The possibilities of the area between the Elgin and the Peach are not included in this estimate.

It is proposed that, at first at least, up to 500 ft. of adit workings in the Peach area be planned and executed for the purpose of obtaining bulk samples for mill concentration testing. The two adits already started should be extended. The work proposed would be too limited to obtain a "fair assay sample" of the bulk of the mineralization. But a mineralogical sample suitable for mill testing could be obtained with roughly the average proportion of the different minerals and ores. The testing should be done on a research basis that should go beyond routine flotation treatment. Leaching methods should of course be considered. The neutralizing limestone present might swing the emphasis to ammonia leaching, although ammonia will not leach chrysocolla according to George H. Roseveare, of the State Bureau of Mines. He also stated that since chrysocolla absorbs soluble copper, a small amount of copper sulphate added to the pulp might promote the flotation of that mineral. In any case a new approach is needed.

The direct expense, for the work proposed above, would be around $15,000 for the adit driving and possibly $10,000 for the mill testing.

For the short term, only slightly less in immediate importance, is the continuation of the development of the Elgin area by drilling. Ten more holes averaging about 300 ft. deep would outline the possibilities. These holes would total around 3,000 ft. and would cost around $15 a foot for direct cost. The total direct expense, then, to carry out the suggested underground development test work and drilling would be around $70,000. This program would take about six months. For this period about $30,000 additional would be required for overhead. The total expense, then, would be around $100,000.

This above work, however, is really only a proposed immediate next step. To largely fill in the drilling pattern now indicated as needed where the limits of the "ore" have not been found and to extend it to areas now indicated as geologically favorable, would require a minimum of fifty additional holes averaging 350 ft. deep. This 17,500 ft. of hole wold cost $252,500 and take around eight months for completion. At the same time, the underground work needed for checking the sampling of he holes, proving continuity between holes and obtaining adequate samples for mill testing, could proceed. This program, which would be an extension of the 500 ft. of work proposed as a preliminary need to obtain samples for immediate mill testing work would require an additional 1000 ft. of headings, raises and winzes and the necessary operating expense of an accompanying sampling plant. Such a plant is owned by the Company and now set up at one of the adits.

If all the above indicated work is done, the total expense would be around $432,000. This breaks down as follows: First stage and/or minimum stage as described above, $100,000. Expense of additional drilling to presently indicated completion, $262,000. Expense of additional underground work, $30,000. Overhead for an additional eight months, $40,000. Total expense - $432,000.

If the completion of all this work still does not define the ultimate limits of the ore body or bodies, correspondingly more drilling will be required, but this might proceed as a plant was being built and/or mining was taking place. The tonnage developed that exceeds the immediate target of 20,000,000 tons of proved ore will accelerate the justification of still further increase at a rate greater than the increase in tonnage.

APPENDIX I

Tabulation of Results of Tonnage and Grade Calculations

TABLE I

Tonnage and Grade, Area A (North Peach)

		Copper Mineralization						Waste
		Proved		Probable		Possible		
Section Number	Block Number	Tonnage	Grade (% Cu)	Tonnage	Grade (% Cu)	Tonnage	Grade (% Cu)	Tonnage
10,380	1					91,020	0.764	231,394
	2			92,250	0.764			
	3					16,912	0.764	
10,264	4					144,590	0.970	582,403
	5			104,566	0.970			
	6					110,041	0.970	
10,161	7					153,600	0.956	561,785
	8	402,500	0.956					
	9			57,600	0.956			
	10	239,400	0.956					
10,000	11	1,015,841						1,410,177
9,351	12					212,244	0.914	442,245
	13	762,710	0.914					
9,795	14					74,983	0.841	730,559
	15	702,293	0.841					
9,602	16					75,123	0.717	479,081
	17			360,405	0.717			
9,497	18					52,473	0.627	315,550
	19			168,770	0.627			
	20					62,621	0.627	
9,380	21			206,427	0.759			308,501
	22					74,029	0.759	
9,236	23					155,952	0.591	846,494
	24	447,998	0.891					
9,123	25					132,699	O.882	1,080,481
	26	508,295	0.882					
TOTAL		**4,076,837**	**0.926**	**990,018**	**0.758**	**1,359,287**	**0.858**	**6,988,770**
Combined Total:			**6,424,142 Tons**		**0.883 % Copper**			**6,988,770**

Note: 1. Eleven cubic feet per ton of copper mineralization.
2. Twelve cubic feet per ton of waste.
3. Waste tonnage determined by using same ratio of ore to waste as found in average of All Sections, except 9602, 9497, and 9380.

TABLE II
Tonnage and Grade, Area B (South Peach)

Section Number	Geologically Probable Copper Mineralization Block Number	Tonnage	Grade (% Copper)	Waste Tonnage
9,011	27	587,693	0.780	
9,846	28	608,124	0.902	
8,720	29	458,073	1.013	
8,618	30	281,002	0.751	
8,482	31	398,045	0.819	
8,366	32	504,067	0.772	
8,133	33	277,612	0.670	
TOTAL		**3,114,616**	**0.829**	**3,426,078**

TABLE III
Tonnage and Grade, Area C (Elgin)

Favorable (Sq. Feet)	Geologically Probable Copper Mineralization Average Drill Hole Intercept (Feet)	Tonnage	Average Grade (% Copper)	Average Drill Hole Intercept (Feet)	Waste Tonnage
800,000	143	10,400,000	0.732	25	1,670,000

TABLE IV
Combined Tonnage and Grade: Areas A and B (Total Peach)

Area	Copper Mineralization Tonnage	Grade (% Copper)	Waste Tonnage
A	5,424,142	0.888	6,988,770
B	3,114,616	0.829	3,426,078
TOTAL	**9,538,758**	**0.869**	**10,414,848**

TABLE V
Combined Tonnage and Grade; Areas A, B, and C (Total Peach and Elgin)

Area	Copper Mineralization Tonnage	Grade (% Copper)	Waste Tonnage
A	6,424,142	0.888	6,988,770
B	3,114,616	0.829	3,426,078
C	10,400,000	0.732	1,670,000
TOTAL	**19,938,758**	**0.798**	**12,084,848**

1. Area insufficiently drilled to establish certainty of tonnage and grade.
2. Eleven cubic feet per ton of copper mineralization.
3. Twelve cubic feet per ton of waste.
4. Waste tonnage determined by using same ratio of ore to waste as found in Area A.

APPENDIX 4

Geological Setting

Arthur M. Heyman

The copper mineralization of the Peach-Elgin area is a low-grade, disseminated, pyrometasomatic deposit in silicated limestone rocks. The metamorphosed sedimentaries are thrust over basement granite on a curving fault plane which dips moderately east under the Peach hill and is level under the Elgin area. A wedge of quartzite is thrust between the granite and the lime silicates in the western part of the Peach hill (Cross-sections 1 - 18), and the hill is capped by a weakly silicated limestone bed containing disseminated copper mineralization. A Tertiary (?) quartz monzonite porphyry weakly mineralized with disseminated pyrite and chalcopyrite intrudes and is faulted into the lime silicates of the Elgin area. The meta-sediments dip and strike similarly to the main thrust fault and maintain constant attitudes. Several high angle faults cut the beds, but little displacement is evidence. Maximum and average thickness of the lime silicate beds are 700 and 350 feet, respectively. The granite and quartzite form the footwall of strong copper mineralization.

The silicate beds are of two general types: aluminum-rich, red-brown dominantly garnet rocks; and aluminum-poor, dark green diopside- (or salite-) bearing rocks. They represent alteration of shaly limestone and relatively pure limestone beds, respectively.

Emplacement of copper mineralization is controlled by a combination of structural features and favorable beds. Intersection of the thrust planes with high angle faults are the gross structural controls. Extensive brecciation prepared the ground for copper-bearing solutions. Aluminum-poor beds are the most favorable host rock, but "ore-grade" copper mineralization occurs to a lesser degree in all other rocks except the granite and the porphyry. The brittle nature of the garnet rock and the quartzite allowed more complete oxidation of the copper mineral than occurred in the diopside-bearing rocks.

Dominant ore minerals are chalcopyrite, bornite, covellite, and chrysocolla. Copper pitch malachite, cuprite, and native copper occur in smaller quantities. Pyrite and magnetite are associated withe the copper sulfides. Silver averages 0.3 ounces per ton. Gold and molybdenum values are low and have not yet been accurately determined.

Oxidation reaches a dept of 100 feet in the Peach area, less than 50 feet over half of the Elgin area, and is nearly negligible over the rest of the Elgin. Oxidation results in a irony capping which is prominent over most of the area of "ore-grade" mineralization except where the mineralization is overlain by the Peach lime capping or thick overburden. Due to the reactive nature of the gangue, secondary enrichment is limited, being significant only in the strongly broken north Peach area.

APPENDIX 5

Drilling and Sampling Procedure

Arthur M Heyman

Drilling

Churn drill holes are collared and drilled through surface talus with a 14 in. Bit. Collar pipe is set, usually less than 10 ft. and drilling is continued with the 14 in. bit. When mineralization of probable ore-grade is reached, 10 in. casing is run into the hole at the discretion of the sampler, a representative of the Company. The hole is then drilled with a 10 in. bit and is reamed and cased every 40 ft. so that raw hole below the casing never exceeds the length of the tools. The contractor may reduce the hole to 8 in. but is required, within practical limits not to reduce farther. A few holes were bottomed with 7 in. tools. All drill runs are 5 ft. in length. When the hole is bottomed, all casing is pulled except 5 to 8 ft. of collar pipe which is left in the hole and capped.

Dart-valved bailers are used to remove the sludge from the hole after each drill run. The dart is cut off so that its tip is 3 in. below the bailer when the bailer valve is open. The hole is bailed until the sludge is removed as completely as possible. This usually requires 4 to 6 runs with the bailer.

Drilling water is introduced by lowering the water to the bottom of the hole in the bailer to avoid splashing the sides of the hole. Excess lubricating oil and thread dope are removed from the drilling tools and casing before they are lowered into the hole. All equipment is kept clean and free of excess grease, and no diesel oil is used in the hole because of its strong sulphide flotation properties. When the tools are withdrawn, the hole is covered to prevent steel fragments from welding or other extraneous matter from falling in.

Salting in the hole is thought to have been kept to a minimum for the following reasons:

1. Casing is always kept below the top of the tools when drilling.
2. The sharp assay cut-offs indicate clean sampling.
3. The grade of the mineralization varies systematically with the rock type and degree of breaking.

Sampling

The bailer discharges the sludge into a dump box from which it is carried by a launder to a four-tier cascade Jones splitter. The steel launder reduces flow surges to a minimum. The dump box and launder are set up so that there is no loss of sludge by splash or leakage and so that the sludge flows at an optimum rate. The splitter is set level and stable. The 1/16 split from the cascade splitter is caught in a square steel tub and, if necessary, split again using a Jones splitter until a sample containing one to two gallons of solid material is obtained. The number

of splits depends on the size of the hole, 1/32 and 1/64 being the usual samples taken. All sampling equipment is thoroughly cleaned after each drill run; the wash water and sludge are run through the same splitting procedure and are added to the sample. The sample is finally poured into a special two-gallon sample bucket with a water-tight top and is appropriately tagged.

The rejects from the final split are washed for a character sample, then sieved and panned for a concentrate sample. The sampler on each shift fills out a report describing the activities of the shift and the number and depth of samples taken. The driller's report, sampler's report, character and concentrate samples, and sludge samples are collected once a day and brought to the Company's field office. The two reports for each shift are compared, discrepancies are resolved, and the result is recorded in a running log of the hole. Portions of the character and concentrate samples are glued onto "sludge boards" on which depth and assay of the sample are recorded and are used for demonstration purposes. The sludge sample is taken to the drying room where preparation for assaying is started.

Since the size of a representative sample is determined by the coarsest particle size of the whole sample and by proportion by weight of the size fractions, sieve analyses were made of samples from each rock type that carries ore-grade mineralization. It was determined that less than one per cent of the cuttings are plus 1/4 mesh and about 50% of the material is minus 100 mesh. Thus, following McKinstry, the minimum sample taken has a dry weight of 10 lbs. Average sample weight is about 15 lbs.

In the process of making sieve analyses, the Jones splitter was checked and found to give a representative sample.

Drying and Preparing Pulps.

The two gallon sample is poured onto a steam table, no water being decanted, and dried at 115°C. This temperature is maintained at 4500 ft. elevation by adjusting the table's pressure release valve to 10 lbs. Per square inch. 115°C is used for maximum speed drying without altering the nature of the sulfide minerals.

The dried sample is scraped from the steam table and taken to the pulp preparation room. The steam table is thoroughly washed between samples. Because the sludge is generally fine, it has not been found necessary to crush it; larger fragments are broken manually. The sample is split by Jones splitter to one gallon which is stored in a labeled, water-repellent paper container for metallurgical testing. One-half pint of the gallon is split out for assaying. This is run through the pulverizer until no grit can be felt in the pulp obtained. The powder is rolled on a 2 ft. square rubber cloth, then run through a small Jones splitter. Three 10 gm.samples are taken from the final split, put into pulp sacks and labeled. One pulp is sent for immediate assay; the other two are stored for later check assaying.

Assaying

Most of the samples were assayed by Dickerson Laboratories, El Paso, for total copper using the short iodide method. Non-sulfide assays are made on composites of the probable ore zone in each drill hole. Total copper and non-sulfide copper assays were made for individual samples in three drill holes. Systematic checks on the assaying and sample preparation are made by sending a second split of a given pulp to Dickenson and by sending pulps assayed by Dickenson to Hawley and Hawley of Douglas, Arizona, who use the standard electrolytic method. The average variation between original assays and check assay is 0.2 % copper. Two drill water samples collected on different days contained 246 ppm. Copper. Gold, silver and molybdenum assays are run on composite samples of the estimated ore zone in each drill hole. Composites of all drilling were used for two spectrographic analyses.

APPENDIX 6

Summary of Laboratory Milling Tests and Conclusions

Howard R. Hendricks

General

A summary of laboratory milling tests on drill samples from the Peach-Elgin copper deposit is herewith submitted.

A total of 103 laboratory milling tests have been made on 35 of a total of 46 holes drilled.

The principal known copper minerals are as follows:

Chalcopyrite	Covellite
Chrysocolla	Carbonates
Bornite	Malaconite

Summary of Laboratory Milling Tests

Laboratory tests were first conducted by straight flotation of the sulfide copper minerals followed by sulphidizing to promote flotation of the oxide, carbonate and silicated of copper.

In a number of the tests in which the occurrence of chrysocolla predominated, recoveries of the copper minerals was relatively poor. It then became apparent that some other method was necessary in order to recover more of the copper minerals and to accomplish this, a leach-float method was tried. This gave improved results in the recovery of the copper minerals, but indicated high acid consumption.

The tests indicated that additional metallurgical research was necessary to develop a satisfactory process for maximum economic recovery of the copper minerals.

The material available for laboratory testing was chiefly composed of churn drill sludges. These sludges are normally not considered suitable material for conclusive metallurgical testing and result in low recoveries of wanted minerals.

It is necessary to have freshly broken ore from the various ore zones on which to conduct testing as follows:

(1) preliminary laboratory testing followed by
(2) full scale pilot plant testing to substantiate results obtained in the preliminary laboratory testing.

Table VI. Summary of Laboratory Milling Tests

Hole No.	Assay Total % Cu	Assay Non-sulfide % Cu	Interval Ft.	Per Cent Copper Recovery Flot. Only	Leach-Float Leach	Leach-Float Float	Leach-Float Total	H2SO4 Used lbs/ton
6	0.89	0.85	217	33.1	51.7	14.6	66.3	40
7	0.73	0.34	135	*	65.3	10.0	75.3	120
8	0.82	0.17	25	*	16.2	72.1	88.3	100
9	0.53	0.19	57	*	40.3	40.4	80.7	100
13	0.36	0.19	275	*	24.4	29.8	54.2	120
15	0.82	0.27	105	79.4	5.3	83.5	88.8	240
16	1.02	0.21	172	81.7	*	*	*	*
17	0.56	0.12	140	83.3	0.8	82.2	83.0	100
19	0.58	0.22	205	83.9	44.4	38.6	83.0	120
20	0.90	0.71	59	23.6	61.7	15.3	77.0	104
21	1.00	0.38	160	73.3	6.9	80.5	87.4	40
22	0.95	0.54	64	81.7	19.6	38.3	57.9	69
23	1.38	1.28	75	48.3	*	*	*	*
24	1.50	0.07	107	89.2	*	*	*	*
25	0.95	0.54	200	53.4	36.2	37.5	73.7	220
26	0.96	0.66	135	69.5	15.6	59.8	75.4	60
27 B	1.11	0.59	145	72.2	*	*	*	*
27A&B	0.95	*	240	*	25.3	38.6	63.8	40
28	0.92	0.67	175	70.3	*	*	*	*
30	0.84	0.60	45	66.7	32.8	32.6	65.4	140
31	0.77	0.31	105	68.6	33.5	61.8	95.3	140
32	0.81	0.05	75	83.3	5.5	78.1	83.6	80
33	0.84	0.48	70	57.5	53.0	35.3	88.3	90
34	0.68	0.51	115	48.9	49.2	26.5	75.7	100
35	0.48	0.22	80	35.8	20.9	45.1	66.0	90
36	0.68	0.60	110	35.6	46.5	18.9	65.4	50
37	0.85	0.07	125	81.0	*	*	*	*
38	0.75	0.41	170	*	20.0	52.8	72.8	150
39 A	0.56	0.41	45	*	23.4	16.0	39.4	130
39 B	0.72	0.05	35	*	2.3	71.1	73.4	90
40	0.64	0.05	70	79.8	5.6	79.8	85.4	110
41 A	0.90	0.82	45	*	26.9	13.1	40.0	130
41 B	1.82	0.29	25	74.8	16.7	75.9	93.6	80
42	0.94	0.05	35	*	2.0	84.0	86.0	70
43	0.88	0.36	95	*	22.1	46.6	68.7	110
44 A	0.54	0.39	70	*	21.3	20.7	42.0	150

(Continued)

Table VI. Summary of Laboratory Milling Tests

Hole No.	Assay Total % Cu	Assay Non-sulfide % Cu	Interval Ft.	Per Cent Copper Recovery Flot. Only	Leach-Float Leach	Leach-Float Float	Leach-Float Total	H2SO4 Used lbs/ton
44 B	1.33	0.39	80	*	11.2	71.1	82.3	80
45 A	0.56	0.48	70	*	50.3	15.8	66.1	180
45 B	0.68	0.17	85	*	18.3	70.9	87.2	120
46 A	0.60	*	30	*	43.3	16.8	80.1	120
46 B	0.81	0.51	50	*	9.5	64.6	74.1	80
WEIGHTED AVERAGE				**66.1**	**28.3**	**44.6**	**72.9**	**102**

A - Dominantly oxide zone.
B - Dominantly sulfide zone.
* - Test or Assay not made.