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Inspiration, Arizona.
July 30, 1930.

Mr. R. H. Sales,
Anaconda Copper Mining Company,
Butte, Montana.

Dear Mr. Sales:

I will enclose this letter with the maps you requested of the Inspiration - Miami ore area, and will outline in it the information we have been able to gather relative to a solution of the Miami fault problem as it concerns the eastward extension of the Miami orebody.

On the plan maps enclosed I have cross-hatched in two directions the high grade portion of the Miami orebody which averaged around 2% in copper, and had individual assays as high as 5%. This portion of the orebody had a length along the fault of 1200 - 1400 feet, and the remainder of the 2500 feet of ore against the footwall side of the Miami fault averaged from 0.8% - 1.0%. The ore is covered by capping averaging about 200 feet in thickness, and, below the high grade portion of the orebody, ore averaging 0.7% - 1.0% Cu. has been developed down to the Miami 1000 level. This level is

920 feet below the collar of the Miami #5 Shaft, so they have originally had a thickness of around 700 feet of ore, the upper 300 feet, immediately below the capping, being the high grade portion.

I have attached to this letter a 1000 scale composite sketch of the Miami 1000, and the 1800 and 2600 levels as they were remembered by A. V. Taylor. The 1800 and 2600 are approximately 800 and 1600 feet lower respectively than the 1000 level. The geology of the 1000 level is sketched roughly as we have it.

Soon after my return here from Butte we spent a day along the fault south of the cave in an endeavor to discover an area of fresh granite that might correspond to the granite encountered on the Miami 1000 in the hanging wall of the fault. No granite which did not contain more or less prominent quartz seams could be found, although a small area of fresh coarse granite with quartz seams was found in a position about 2000 feet south of that on the 1000, measuring parallel to the fault.

The porphyry which we had mapped between strands of the fault is possibly not offset as much as we supposed. Further mapping within the caved area showed a mass of porphyry of indefinite structure in the footwall of the fault, just to the south of that mapped between strands of the fault. If the two are the same the throw on the footwall strand is not over 200 feet, if

that much, depending on the attitude of the porphyry dike. On the plan maps I have projected the dike mapped by Ransome through the porphyry we mapped in the footwall of the fault, but it is still possible that the dike Ransome mapped is located further south in the cave and cannot now be mapped. In that case, the porphyry we mapped in the footwall does not appear at the surface between strands of the fault.

We have traced the Miami fault south of the town of Miami for a short distance, finally losing it in conglomerate. The fault can now easily be traced for a distance of about three and one-half miles, and is undoubtedly a strong and persistent structure. The dips at the surface have varied from 35° - 60° to the east, and from the surface to the 1000 level the dip was about 55° E. Striations on the fault plane as observed at the surface have a direction of N 55° E - N 65° E, and, as can be seen on the sketch of the 2600 level, the work done by the Miami Copper Company was directed to prospect a portion of the hanging wall ground which would correspond to a N 60°E movement of the high grade portion of the orebody.

Perry was convinced after his last talk with Mr. Graybeal of Miami Copper that the ore they encountered east of the fault was at the elevation of the Miami 2600, approximately 2520 feet below the collar of the #5 Shaft.

Assuming that this is true, I have made a few calculations to show the amount of movement necessary on the Miami fault to throw the high grade portion of the orebody into ground owned by the Inspiration Company. In these calculations the top of the orebody has been assumed to lie about 200 feet below the collar of the #5 Shaft, and the measurements are from the south boundary of the ore at the fault, on the footwall side, to the projected south boundary on the hanging wall side.

Direction of Movement.	Horizontal Component of Movement.	Vertical Component of Movement.	Approximate Dip of Fault to Satisfy Foregoing Conditions.	Amount of Fault Movement.
N 40°E	4300 Ft.	2320 Ft.	62°E	4885 Ft.
N 50°E	5000 "	2320 "	45°E	5510 "
N 60°E	6100 "	2320 "	35°E	6525 "

Assuming the movement to be N 60°E, not all of the ore would lie within Inspiration ground even with a displacement of 6525 feet because of the northward jog in the property line near the smelter.

I have lately again gone through the annual reports of the Miami Copper Company since 1921, and it appears to me that it is more likely that the ore found by the Miami Company east of the fault was above the 2600 and below the 1800 level. The #5 Shaft was driven to the 1000 level in 1921; in 1922 there is a charge against

"Shaft #5 Development". On April 18, 1923, a special announcement of the ore on the hanging wall side of the fault was made, described as "the top of a new sulphide orebody". In 1923, 2020 feet of diamond drilling is noted in the General Manager's report; this is the only mention of diamond drilling in any of the reports, although diamond drilling to outline ore reserves was in use at that time, I believe. The 1924 report states that some good ore had been found, but that no ore worthy of commercial exploitation was discovered. In 1925, the report simply states that work was discontinued without adding anything to the ore reserves. The total charges against Shaft #5 Development from 1922-1925 aggregated $430,000.

Mr. A. V. Taylor was employed by Miami Copper in 1924 as an engineer, and he said on his visit to Perry in Cananea that he had surveyed the workings on the 1800 and 2600 levels. His recollection was that there was about 1000 feet of drift driven N 20° E from the #5 Shaft on the 1800 level; that a winze was driven to the 2600 level from a point near the north end of the 1800 drift; and that another 1000 feet of drift was driven N 40° E from this winze on the 2600 level. Diamond drilling was then done from the 2600 drift near its north end. Taylor believed the work was finally stopped because the State Mining Inspector would not permit them to continue so far from a main shaft.

A comparison of the time of striking the ore and the amount of work done makes it appear that the ore was not first encountered on the 2600 level. It seems more likely that diamond drilling from the 1800 level first disclosed the ore in the early part of 1923; and the winze to the 2600 and subsequent drifting N 40°E was for the purpose of getting under the ore and delimiting it with up-holes. No reason was ever given to the stockholders for the failure to find the ore so enthusiastically reported in 1923, and the greatest secrecy has been maintained about the work since its abandonment.

Of course any information regarding the elevation of the ore on the hanging wall side of the fault is decidedly important, for as the projected vertical component of the fault movement is made less, the greater must be the horizontal component to have the ore appear in Inspiration ground; and at the same time the less attention can be paid to the striations noted at the surface.

With the foregoing in mind I have only placed the projection of the orebody upon one of the maps, the one that I have used as a work sheet. If this position conforms to your ideas, I know that it will be a simple matter to have one of the draftsmen place it on the remainder of the prints. I am also enclosing the tracing, in case you may need a greater number of prints than I have been able to send.

I expect to have the field work that was outlined when you were here in June finished by about August 15th, and at present we are planning to be in Cananea by September 1st. I have written Perry and asked him to stop at Inspiration on his trip from Long Beach to Cananea; at that time he can decide if there is any more field work which should be done here at this time.

I sincerely hope that the maps will be satisfactory.

Yours very truly,

OCT 15 1956

THE ANACONDA COMPANY

25 Broadway

New York 4, N. Y.

OFFICE OF THE
CHIEF GEOLOGIST

October 11, 1956

Mr. Roland B. Mulchay
Asst. Chief Geologist
The Anaconda Company
818 Kearns Building
Salt Lake City, Utah

Dear Mul:

Enclosed are prints of pictures taken in the Inspiration Pit when we visited there last May.

I have the original Kodachrome slides if you want additional prints.

Best regards.

Sincerely yours,

V. D. Perry

V. D. Perry

VDP:bc
encl.

MINING GEOLOGY DIVISION - ARIZONA SECTION A.I.M.E.

SPRING MEETING: APRIL 30, 1966

INSPIRATION OPEN PIT TOUR

STOP 1

Southeast side of Thornton Pit -- approximate elevation 3700. Your attention is directed to the northeast corner of the pit where the Pinto Fault is indicated in the pit wall by the color change. This is a reflection of both moisture and mineralization. Next your attention will be directed to the benches in the lower portion of the pit and also in the vicinity of the shovel to the northwest. You will note the oxidized copper ore in the old draw blocks which had been mined by block caving but was not recovered. Your attention will now be directed to the lower working benches in the southwestern portion of the Thornton Pit. This area shows as a white to light grey and is an area of unmined or virgin ore which is principally sulfide -- some Cu_2S, $CuFeS_2$, FeS_2 and quartz veins containing MoS_2. The grade approximates 1.00% copper in this area.

STOP 2

The Bulldog fault zone is represented by the structure which can be noted in the 50 foot bench to the north of the buses. This structure, which you will recall from the morning speeches, is a low angle normal fault, striking approximately N 35° E. At this location there is a footwall of sulfide mineralization which varies in grade from 0.60% to 2.00%+. The hanging wall is leached capping. The rock type is sericitized Pinal Schist with a few, small, porphyry dikes cutting at random strikes and dips. These are weakly mineralized.

STOP 3

On this dump you will find the surface leached capping from the Thornton West stripping. We interpret the seal brown to shades of maroon as indicative of prior copper mineralization. The blacks, bright reds and yellows are indicative of iron. The boxworks present are primarily of pyrite as the original ore tenor was very low in copper. It was necessary for the process of supergene enrichment to establish the present grade of mineable ore from the Inspiration open pit. Relief limonite is probably derived from secondary chalcocite.

STOP 4

Live Oak Pit. The 3450 bench displays ore in block cave draw blocks. The overlying formations to the north are dacite and porphyry. To the northwest you may note the scarps caused by the underground caving operation. The high hill to the northeast is known as Red Hill and production is expected from the area in the future.

ROLAND B. MULCHAY

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GROUND MOVEMENT AND SUBSIDENCE FROM BLOCK CAVING AT MIAMI MINE

James Bishop Fletcher
Chief Mine Engineer

Miami Copper Company
Miami, Arizona

This paper is to be presented at the Annual Meeting of the American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., San Francisco, February 15 to 19, 1959.

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GROUND MOVEMENT AND SUBSIDENCE FROM BLOCK CAVING AT MIAMI MINE

By James Bishop Fletcher, Chief Mine Engineer, Miami Copper Company,
Box 100, Miami, Arizona

59AU27

HISTORY: Miami Mine started mining 1910. The orebody was divided as follows:

High Grade 24,400,000 tons 1910 to 1925	Mined by top slicing, sub level caving, etc. Shown on subsidence map (Fig. 1) solid lines (brown shading).
Low Grade 102,000,000 tons 1926 to 1954	Mined by block caving. Shown on subsidence map (Fig. 1) 1. Solid lines stopes #1 to #603, 720 level haulage, (red shading). 2. Dotted lines stopes A to G and 101 to 150, 1000 level haulage (purple shading).
Mixed Ore 9,800,000 tons 1936 to 1943	Mined by block caving, shown on subsidence map (Fig. 1). Solid lines stopes #201 to 263, 720 level haulage, (green shading).
Low Grade #2 23,000,000 tons (expected) 1955 to	Mined by block caving. Shown on subsidence map (Fig. 1). Dotted lines stopes #301 to #320, 1000 level haulage (Blue shading).

GEOLOGY: Ore minerals largely in pre-Cambrian Pinal schist intruded by tertiary Schultz granite porphyry and covered to some extent by Quarternary Gila conglomerate. Structures highly faulted and shattered. Miami fault on east cuts off ore. Pinto fault on southwest caused reoxidation of enriched sulphides producing mixed ore.

MINERALOGY: Mainly chalcocite, with chalcopyrite, bornite, covelite, malachite, Azurite, chrysocola, cuprite, native copper, and molybdenite. Ore minerals occur in seams, veinlets and disseminated particles.

ROCK CLASSIFICATION: For simplicity the orebody can be divided into three classes as follows:

1. Strong - requires no timbering.
2. Medium - requires timbering.
3. Weak - requires close timbering and repairs.

Definition:

The term "pillar stope" as used in this paper refers to a stope which is bounded on one or more sides by subsided capping from a previously mined stope or stopes. The term "pillar" refers to the partitions between stopes. No attempt was made to mine the 7-½' to 15' pillars. Some of the 30' to 50' pillars were mined after the adjacent stopes were completed.

EXCERPTS FROM PREVIOUS PAPERS WRITTEN ON SUBSIDENCE

"SUBSIDENCE FROM BLOCK CAVING AT MIAMI MINE, MIAMI, ARIZONA" - BY F.W. MACLENNAN, Feb. 1929.

Mr. Maclennan[1] states that in starting block caving the problems were:

"1. How the ground would cave above the stopes.

2. Whether in the first stopes caving would encroach on the pillar stopes as laid out.

3. Whether in subsequent mining of the pillar stopes, waste filling would be drawn from the previously caved adjacent stopes."

Fig. 2 shows the area mined and limit of the escarpment and extreme cracking on the surface as of Dec., 1928. Mr. Maclennan states there is very little caving outside the boundary of the stopes up to the top of the ore. He states that special care was taken to prevent caving outside the stope boundaries by driving boundary caving drifts. Fig. 3 shows the method of mining at this date.

Fig. 4, E 250 section, shows the only instance of movement found in the ore adjacent to stopping operations. Crack E records the movement of a few inches of the drifts on the various levels.

Fig. 5, is a reproduction of the sections from Mr. Maclennan's paper with pertinent data added.

Mr. Maclennan concludes his paper by stating the maximum subsidence to date is 79.4% of the ore drawn as shown on the E 250 section, but that the average to Dec., 1928 is 2/3 of the volume of ore removed.

Mr. Maclennan[2] in his paper on Miami Copper Method of mining low grade orebody (Feb. 1930) answers two of the questions brought out in his paper of 1929:

"Up to date 13 stopes have been completely drawn. Nine of these were original stopes surrounded on all four sides by solid ground and four were pillar

stopes, two of which were adjoined by broken waste on two sides and one end, and two adjoined waste on one side and one end."

Table #1 gives the results of mining of the first 13 block caving stopes:

Table #1 - Tonnage and Grade Extraction

Partitions Not Included in Expectancy

	Expectancy		Mined	
		Copper		Copper
	Tons	Per Cent.	Tons	Per Cent.
Total of 13 completed stopes	11,038,070	1.0260	12,710,378	0.9124
Best original stope (2)	998,016	1.0388	1,210,424	1.0091
Best pillar stope (11)	319,560	1.0640	387,827	0.9348
Poorest original stope (7)	1,071,535	0.8701	1,053,153	0.7786
Poorest pillar stope (9)	1,098,313	1.1067	1,025,032	0.8995

	Percentage Extraction		
	Tonnage	Grade	Copper
	Per Cent	Per Cent.	Per Cent.
Total of 13 completed stopes	115.15	88.93	102.40
Best original stope (2)	121.28	97.14	117.81
Best pillar stope (11)	121.36	87.86	106.63
Poorest original stope (7)	98.28	89.48	87.94
Poorest pillar stope (9)	93.33	81.28	75.86

To continue quoting Mr. Maclennan, "The extraction of more than 100 per cent. of the estimated copper content of the ore may be due to drawing some of the partition ore, and to copper not included in the estimate, coming from overlying capping or gob.

The best pillar stope, 11, was bordered by waste on one end and one side, and the extraction from it compares favorably with the best original stope, the tonnage extraction being almost identical while the grade extraction is 10 per cent. lower. The poorest pillar stope, No. 9, is the poorest of the lot, but it was the first pillar stope mined. It was 150 x 300 ft. in plan, and so much weight developed during its extraction that results were far from good. The experience gained here resulted in reducing the size of the stopes to one-half their original size. No. 11, the best pillar stope, was a one-half size stope.

In drawing the pillar stopes, there is no serious trouble experienced from waste drawing in from the adjoining previously mined original stopes. It has been the experience in the past that the drawing spreads very little and tends to pipe up vertically once the ore has been broken. This characteristic necessitates the close spacing of draw points, but at the same time protects the operation from waste rock drawing in laterally. The waste capping which has settled down into the original stopes becomes well packed and consolidated, due to the pressure exerted by a vertical column of 300 or 400 ft. of this material, and this waste filling constitutes substantial side support in mining the pillar stopes. Before the first pillar stope, No. 9, was mined, crosscuts were driven into the adjoining fill of the original stopes 1 and 2 on the 510-ft. level, which is 80 ft. above the sills of these stopes, and this material at this point appeared to be as solid as the original ore along the sides of pillar stope 9, through which approach drifts were driven."

THE NEXT DESCRIPTION OF SUBSIDENCE AT MIAMI IS GIVEN IN A MEMORANDUM, 12-19-39, BY MR. R.W. HUGHES[3] TO MR. GEORGE O. RICE, CHAIRMAN OF THE COMMITTEE ON GROUND MOVEMENT AND SUBSIDENCE, A.I.M.E. Pertinent data is quoted from this memorandum:

"Since the presentation by Mr. F.W. Maclennan of his paper, "Subsidence From Block Caving" at the New York meeting in February 1929, there have been mined an additional 34,000,000 tons of ore." Fig. 6 shows the area mined and limit of the escarpment and extreme cracking on the surface as of July 31, 1929.

"As mining was extended it became increasingly difficult to obtain accurate data on subsidence. This was due to the fact that none of the pit area was safe for traveling, and parts of it were seen only from remote points, due to the extension of the cracked boundary.

In Mr. Maclennan's paper of 1929, he stated three problems which were unanswered at that time. During the past ten years these problems have been somewhat simplified. They were stated at that time as follows:

1. "How the ground would cave above the stopes."

Our experience has been that any difficulty in caving has been in starting the first fifteen to twenty feet overlying the undercut level. From this point to surface there has been no instance of hang.

2. "Whether in the first stopes caving would encroach on the pillar stopes as laid out."

This refers to the practice of mining 150 x 300 ft. or 150 x 150 ft. blocks, by a "checkerboard" scheme, the pillar stopes to be mined later. To what extent the original stopes have encroached is still debatable. Certainly there is some encroachment as is shown by:

A. Consistent overdrawals from the outside chutes of original stopes indicating a horizontal drift of ore from pillar stopes.

B. Pillar stopes found to be structurally weaker than adjoining original stopes indicating stresses caused from caving the original stope.

C. Ore as drawn from the pillar stope is found to break much finer than the adjoining original stope, which would also indicate stress from caving original stope.

These three facts are considered proof that original stopes encroach on pillar stopes, the amount probably depending on the presence of slips, faults and other structural weaknesses.

3. "Whether in subsequent mining of the pillar stopes, waste filling would be drawn from previously caved adjacent stopes."

This has been answered very definitely. The mining of pillar stopes is considerably complicated by the waste filling from the original stopes. The amount of such depends on the number of stope sides exposed to waste, increasing rapidly with increased exposure. Present practice is to mine these pillar stopes by leaving approximately 30 ft. of peripherical pillar to shut out adjoining fill.

In comparing the subsidence angles of 1929 and 1939, it is found that angles have in most instances flattened considerably. For instance the #250 section went from 50 degrees to 46 degrees, the E 0 section from 52 degrees to 44 degrees while the W 250 section dropped from 73 degrees to 42 degrees. Sections at right angles to these show similar flattening. This is probably due to the increased periphery of the pit. During the earlier stages of caving, with a smaller diameter of pit, there undoubtedly was a horizontal arching effect tending to resist slippage of the pit walls. As the pit grew larger, this arching effect was weaker with resultant greater tendency to slip. This would hold true only in a roughly circular pit, such as is found at Miami.

In the section (Fig. 7) showing "Marker Block Travel Adjacent to Pinto Fault" the effect of a major slip is shown in relation to ore extraction. It is routine practice at Miami to plant 12" x 12" timber blocks in the boundary caving drifts. These are placed at 25 foot intervals along the drifts and are marked with tags showing the elevation and coordinates of each block. All ore is screened thru ten-inch grizzlies so that the marking blocks are caught on the grizzlies and the caving progress of the stope is charted.

The Pinto Fault is a major feature of the Miami orebody, limiting it on the west. Ore occurs in the hanging wall schist, the foot wall granite being waste. Dip is roughly 45 degrees with many local deviations. Fault gouge varies from a few inches to ten feet.

As shown on the section, stope #202, as a matter of expediency, was mined prior to the higher situated stope #207. With the intent of causing a movement along the fault, no boundary caving drifts were driven along the stope boundary common to stopes #202 and #207.

That movement was effected is proven by the recovery of two marking blocks planted in stope #207 while drawing stope #202, their angle of travel being 50 degrees and 54 degrees. The additional ore drawn from without the vertical limits of the stope cannot be measured, but the final results of mining #202 stope resulted in 163% tonnage extraction, while final results in #207 stope were 55%."

EXTRACTS FROM A PAPER PRESENTED AT THE AMERICAN MINING CONGRESS IN SEPT. 1954 BY MR. J. W. STILL:[4]

"The original mining scheme on this low grade was covered by F.W. Maclennan in the A.I.M.E. publication #314-A-34 - "Miami Copper Company Methods of Mining Low Grade Orebody" issued with "Mining and Metallurgy", March 1930.

Briefly this scheme proposed the mining of about 550-600 vertical feet of ore in two lifts (720 and 1000 level haulages) in individual stope units and on a full gravity basis. There have been few changes in the basic scheme and only the following were of a major nature.

(1) Pillars were enlarged from 15 ft. on the 720 lift to pillars either 30 or 50 ft. on the 1000 lift. This change offered effective protection against dilution from adjoining mined-out areas. Mining experience has shown that an unknown, but substantial, pillar tonnage is recovered thru side movement of pillar ore into the adjoining producing stopes.

(2) Square set control-draw points at the top of the grizzly raises were abandoned, the spacing of the draw points changed from 12.5 ft. center to center to 16.66 by 18.75 ft. and drawing is done from the bottom of the grizzly raise (now called the control raise). This simplified drawing and maintenance with no adverse change in copper extraction.

(3) During the war years, due to the shortage of men, boundary slice level work was also discontinued, with little or no change in the copper extraction picture apparent."

Continuing to quote Mr. Still: "There was a mining loss" (at that time) "of about 19 percent of the copper content. Of this loss, the larger part, or about 81 percent, was left in pillars. This represents a sizeable tonnage of copper, totaling 163,800 tons of copper metal. Leaching of the caved area will undoubtedly recover an appreciable portion of this; for since leaching started in 1942 (on only a limited portion of the area eventually available) some 28,340 tons of copper have been recovered.

SUBSIDENCE AND GROUND MOVEMENT TO DATE (1958) AT MIAMI COPPER COMPANY

Fig. 1 shows the area mined and limit of the escarpment and extreme cracking on the surface as of Jan. 1, 1958.

The following table is given to bring the block caving picture up to date:

Table #2 - 720 level sulfide orebody

(7-½ to 15 ft. pillars and boundary caving)

	Tons in 1000's				Extraction Percentages		
	Expectancy		Drawn		Tons	Grade	Copper
Stopes	40,577 @	.878	40,522 @	.763	99.87	86.90	86.79
Pillars	7,130 @	.859	-	-	-	-	-
TOTAL	47,707 @	.875	40,522 @	.763	84.93	87.20	74.06
720 level mixed orebody							
(No pillars boundary caving drift driven except on Pinto fault side)							
Stopes	7,358 @	1.788	9,791 @	1.460	133.06	81.66	108.58
1000 level stopes							
(30 to 50 ft. pillars with some boundary caving drifts)							
Stopes	53,331 @	.749	59,023 @	.690	110.67	92.12	101.96
Pillars	13,866 @	.766	1,888 @	.704	13.61	91.91	12.51
TOTAL	67,197 @	.753	60,911 @	.691	90.65	91.77	83.18
1000 level pillars mined							
Pillars	4,290 @	.788	1,888 @	.704	44.01	89.34	39.31

The above table shows that the best extraction was from the mixed orebody where no pillars were left between stopes, however this orebody was in hard to medium-hard rock.

The next best extraction is shown in the 1000 level sulfide stopes where 30 to 50 ft. pillars were left. However, the following table from Mr. Still's paper gives a further breakdown of these stopes as to type of rock:

Table #3

Stope Mtce. Tons/MS	No. of Stopes	% of Total Tons Drawn	% Ton Extr.	% Grade Extr.	% Cu. Extr.
300 Plus - Little Mtce.	12	21.5%	120.06	93.23	111.93
300-100 - Av. Mtce.	44	72.9%	111.94	94.27	105.52
Under 100 - Heavy Mtce.	9	5.6%	85.33	85.62	73.06
Totals & Av.	65	100.0%	111.65	93.39	104.27

The poorest extraction is shown in the 720 level stopes where 7-½ to 15 ft. pillars were left. The 720 level stopes were in medium to weak rock. The following table from Mr. Still's paper gives a breakdown of these stopes:

Table #4

720 Lift Stopes	Sides Exposed	% of Total Tonnage Drawn	% Ton Extr.	% Grade Extr.	% Cu Extr.
16	0	33.3%	115.48	92.56	106.89
17	1	24.4%	101.52	87.72	89.05
13	2	15.2%	97.47	84.55	82.41
15	3	16.3%	82.28	78.54	64.62
8	4	10.8%	90.84	79.54	72.65
69	43% Av. Perimeter Exposure	100.0%	99.83	86.79	86.64

1000 Lift Stopes	Sides Exposed	% of Total Tonnage Drawn	% Ton Extr.	% Grade Extr.	% Cu Extr.
37	0	65.5%	116.67	94.43	110.17
12	1	14.5%	105.02	96.00	100.82
14	2	17.0%	103.15	90.24	93.08
2	3	3.0%	96.02	86.98	83.52
65	13.4% Av. Perimeter Exposure	100.0%	.65	93.39	104.27

GROUND MOVEMENT OBSERVED UNDERGROUND AT MIAMI

(a) With boundary cutoff: The early stopes all had boundary cutoff drifts which served as observation points and in no case did the stopes cave outside a vertical line. In only one case was there cracking outside the stope boundary as shown on the E 200 section.

A drift was driven through a completely drawn stope 80 ft. above the undercut which showed no caving beyond the vertical boundaries of the stope.

(b) Without boundary cutoff: All of the later stopes on the 1000 level were mined without boundary cutoff. In every case where it has been possible to observe these stopes they caved inside the vertical stope line. In 132 stope a drift was driven through the caved material and an angle of 83 degrees was found inside the stope boundary (Fig. 8). Fig. 9 shows the same condition to hold true in 310 stope.

(c) Caving next to broken ground (pillar stopes):

1. Stopes next to waste tend to draw from outside the undercut limits. Table #4 shows how the copper and grade extraction varies with the number of sides of a stope exposed to waste.

2. The mixed orebody was mined without any pillars between the stopes. This orebody gave a better grade and copper extraction than any of the others. The mixed ore was in strong rock.

3. Pillar stopes are structurally weaker than the original stopes. Where pillar stopes were mined in weak ground they were heavier stopes than the original stopes and required a great deal of repair which resulted in a poor draw. Table #3 shows how the grade and copper extraction varies with the maintenance required in a stope.

4. Individual stopes (1000 level haulage) bordering waste have been pulled without dilution from the sides being a problem. These stopes have been in firm ground with little repairs where the draw could be well controlled.

(d) Caving next to Pinto fault: The mixed orebody was bounded on one side by the Pinto fault and no boundary cutoff drifts were driven on that side. Fig. 7, showing marker block travel, definitely proves that there was drawing outside the stope adjacent to this fault.

(e) Rock surrounding a caving stope: The rock surrounding a caving block is under stress. This is very evident in fringe drifts and extraction drifts under the block. The distance from the caving stope that this stress is noticeable depends upon the strength of the rock. The drift shown in Fig. 9 which is in firm rock showed no timber failure 20 ft. back from the caving ground. The timber in an elevator penthouse in weak ground 100 ft. from a stope and 75 ft. above the undercut level, failed when the stope was mined. After a stope has been mined the waste fill consolidates and the stress in the adjacent rock is relieved. Workings can be driven alongside and into this material.

SURFACE SUBSIDENCE

(a) Angle of draw or ground movement outside stope boundaries: The attached sections (Figs. 10 & 11) show this better than any description. The cave angle from the undercut level to the outside crack or movement averages 45 degrees in both the schist and conglomerate; however, it must be pointed out that this crack does not extend from the surface to the undercut. The best evidence to show that the ground does not move along a 45 degree shear plane is shown in the open pit at Inspiration Consolidated Copper Co. where they are mining adjacent to their block caved stopes.

The shovels are able to dig the subsided material without blasting. At the boundary, or a small distance inside the boundary, unbroken rock is encountered and has to be blasted. This vertical cutoff, is in some cases a slight overhang, between subsided capping and unbroken rock can be seen by the differences in color of the material.

The mining boundaries to the northwest of the Miami orebody are an assay cutoff, so if the ground were broken at some angle outside of the stoping limits a considerable amount of copper could be recovered by leaching. In 1956 an attempt was made to leach this material. Solution was introduced into the cracks and along the escarpment north of 17, 50, 32 and 48 stopes (#2 Coord.) The solution worked its way under the surface and entered the caved material within the above named stopes high up in the column and leached the material from the northern portions of these stopes. In stope No. 48, which had been leached prior to 1956, the solutions from the outside cracks entered this stope with no pickup of copper.

(b) Movement on the Pinto fault: There was definite movement on this fault underground; however, the dip of the fault was so close to the average draw angle that it gave the same picture on the surface as the conventional subsidence.

(c) Amount of subsidence: Mr. Maclennan in his paper states that the maximum subsidence on #250 section was 79.4% of the ore removed, and that the average up to 1929 was 66.6%. This figure of 66.6% in general holds true up to date; however, as shown in the N 2600 section (Fig. 12) mining 86 ft. of ore at a depth of 870 ft. gave a subsidence of 34 ft. or a subsidence of 40% of the ore removed. The other extreme was 256 stope which was mined at a depth of 160 ft. from the surface and pulled to daylight giving a 100% subsidence.

(d) Increase in volume of subsided capping: It is evident from observation underground that at depth the broken material packs and approaches the density of the original material. Owing to the difficulty of obtaining an accurate survey of the caving ground this figure is very hard to obtain; however, careful measurements tend to show that the swell varies with the depth. The rock at Miami averages 12.5 cu. ft. per ton in place and 20 cu. ft. per ton broken, or a ration of $\frac{20}{12.5} = 1.6$ Assuming that the broken rock at the surface is 20 cu. ft. per ton and that at depth the rock approaches the original density, and plotting the observed data at Miami we get an average curve as shown in Fig. 13. This is an average curve and individual stopes vary from it, which tends to show that the swell also varies with the type of rock. Inspiration Consolidated Copper Co. is open pit mining adjacent to their block caved stopes and they have found an average

figure of 16 cu. ft. per ton or a ration of $\frac{16}{12.5} = 1.28$ where they have dug in subsided material.

(e) Time element in the subsidence: The rate of progress of the cave from the undercut level to the surface depends to a great extent on the strength of the rock. It also depends to some extent on the size of the stope and the rate of draw which in turn are both determined by the strength of the rock. Mr. Maclennan states in his paper that No. 12 stope, which was in very weak rock, caved the surface 650 ft. above the undercut in 104 days. More recent observation shows that undercutting from 600 to 700 ft. below the surface in weak to medium rock with a normal pull of 9 inches per day will break the surface in 100 to 150 days after the stope is undercut.

137 stope (Fig. 12), 870 ft. below the surface, did not cave to the surface until more than one year after the undercut, and not until the adjacent stope began to draw. The exact date of the break-through was not observed but it was more than one year and less than two years after the undercut. It is apparent in this stope that the swell of the broken material is approaching the amount removed by mining. If the curve (Fig. 13) is accurate 61 ft. of mining would just balance the swell and there would have been no subsidence; 870 ft. - 61 ft. = 809 ft.; 809 ft. x 1.075 = 870 ft.

(f) Description of subsidence: A typical report as turned in by the survey party is as follows:

312 Stope - Subsidence Survey:

Number 312 stope is located between the N 0 - S 300 and E 108.0 - E 258.0 (No. 5 Coordinate). The undercut is on the 870 level. The block encompassed by the boundaries is 300 x 150 x 600 ft. or 27,000,000 cut. ft. or 2,160,000 tons. The ore column above the undercut level extends upwards to mean distance of approximately 250 ft. The actual calculated tonnage of this block is 895,330 tons of ore.

Undercutting of 312 stope was completed January 8, 1955. Undercutting started at the northern end of the stope and finished at the southern end. A slow draw followed the undercut. As a result, upon completion of the project, 11.0% of the ore and 4.56% of the total head had been pulled.

On March 11, 1955 two holes appeared on the surface above the stope (Fig. 14). These holes were of about the same size (20 ft. in diameter). Accompanying these two break-throughs were two noticeable ground fractures. These fractures occurred 340 ft. and 410 ft. from the center of the stope and completely encircled it, although it seemed apparent that the southernmost fractures showed a greater displacement than at any other sector. Movement of 1 ft. and 2 ft. respectively was indicated by the fracture scarps. Underground activity charts on this same date showed that 24.06% of the ore and 9.97% of the total head had been drawn.

A check on April 14th showed that the two holes in the surface above the stope had enlarged into a more oval shape with dimensions 25 ft. x 40 ft. An entirely new 30 ft. x 40 ft. hole appeared in the southeast section of the stope. Fractures between this hole and the northernmost hole of the other two indicates that one large

is in the making. Measurements of the two fracture scarps showed a movement (March 11 to April 14) of 5 ft. and 7 ft.; 340 ft. and 410 ft. scarps respectively. In addition ten small fractures had appeared between the 340 ft. scarp and the center of the stope. The downward displacement of this area cannot be measured because of the danger of getting too close to the caving area, but actual total displacement seems very slight. The ore drawn up to this time amounted to 33.4% of the stope tonnage and 13.85% of the total block.

(g) Size of the mined block to cause surface subsidence: The experience at Miami has been any block that can be successfully block caved will cause surface subsidence. The minimum size of block that can be block caved is dependent upon the strength of the rock. In medium to weak rock blocks as small as 37-½ ft. x 75 ft. have been caved. The only block that has not successfully caved to date is 125E - 126 pillar which is in hard rock. The dimensions are 37-½ ft. x 120 ft. Some of the mixed ore stopes which were in hard rock with dimensions of 150 ft. x 200 ft. had to be blasted with powder pockets above the undercut to start caving.

MECHANICS OF SUBSIDENCE

The great number of tension cracks in the subsidence area and the absence of any observed shear planes (except along the Pinto fault) suggest that the tensile and compressive strength are the governing factors in subsidence and block caving at the Miami mine. Immediately after undercutting a block the roof begins to act as a beam and fails in tension on the under side and forms an arch which continues to fail as the span of the arch is too great for the strength of the rock. (The word arch is commonly used in describing block caving but it could be more accurately described as a dome.)

When the arch reaches the capping the crown breaks into capping before the haunches, which is the major source of dilution in block caving at Miami. This dilution is more serious if two or more arches are formed in the stope.

"Piping" is the local term for a small break-through to the surface, which is very descriptive of what occurs. Piping can and does occur after the entire stope has broken the surface. This is caused by small arches occurring in the broken material and working up to the surface. The best preventative of piping is a uniform draw. Piping can also be caused by too wide a spacing of draw point.

As the arch approaches the surface there is a small sag in the surface which causes tension cracks to open up at the outside of the stope limits. At this point the collapse of the surface is very rapid and has rarely been observed.

There is considerable evidence that the ground surrounding a caving block is under stress, and from the observation of the timber failures it is evident that there is a small movement of the solid ground toward the caving stope. This small movement underground causes the hale of tension cracks to form at the surface. These tension cracks become blocks which fail by tipping toward the cave, or in some cases they tilt away from the cave, or individual pie-shaped wedges will drop into a crack. Mr. John W. Vanderwilt[5] describes the same type of surface failure at Climax, which he calls rock-slump and rock-slide, or a combination of the two.

He also observed tension cracks at depth alongside the caved block. No cracks have been observed underground at Miami except in the one case mentioned in Mr. Maclennan's paper. The absence of any tension cracks at Miami underground is probably due to the much weaker rock. Mr. Vanderwilt[5] mentions the possibility of residual strains in the rocks being responsible for the tension cracks but he states that the tensile strength of Climax rock is so small that the relative sudden release of pressure in one direction, as subsidence over a caved block reaches surface, may be sufficient without the aid of residual compressive stresses to produce the tension cracks. The rock at Miami in general is much weaker than the Climax rock.

The rock failure observed outside the caved block is a common failure in all materials having cohesion where a steep slope, or in this case where a vertical break, is encountered. There are times during caving when the entire side of a stope from the undercut to the surface is subject to stresses. The subsided capping and the broken ore do support the walls; however, the outside cracks and the escarpment can be expressed as an angle from the undercut proving that they vary with the depth of mining rather than with the depth of the unsupported sides of the block. The best explanation based on observation is that the walls adjust themselves by a bending movement at depth. On approaching the surface, the bending has been transformed to tension which results in the observed surface movement.

Fig. 15 shows stress condition within a caving block.

SURFACE FEATURES OF SUBSIDENCE

The appearance of the subsided area is shown better in the accompanying slides than in any description:

Slide #20 - Tension crack, formed in concrete block, caused by subsidence. From this photograph it can be seen that the ground failure is a tilting or bending movement toward the cave, opening tension cracks.

Slide #21 - Shows a wedge dropped down in conglomerate.

Slide #22 - Tension cracks in the schist, showing how these cracks diminish at depth. Notice that the right-hand block has failed by bending away from the cave. This photograph shows how individual pie-shaped wedges are formed and drop down.

Slide #23 - Shows the subsidence in strong rock.

From the N 400 section (Fig. 10) it is evident that any permanent surface structures should be located well outside a 45 degree line from the deepest mining. This angle of 45 degrees holds true only where large tonnages are mined. Where small tonnages or individual stopes are mined this angle is much steeper.

Figs. 1A, 2, and 6 show the horizontal arching of the caved area.

MODEL EXPERIMENT

In an endeavor to duplicate the action that takes place in block caving at Miami a model experiment was performed. The model consisted of two glass plates 1-½" apart, supported in a wooden base and wooden ends. Using a scale of 1" to 100' four of the stopes drawn on the glass are 150 ft. x 150 ft., and one is 187 ft. x 150 ft. No pillar is left between the left-hand stopes. The material used to represent the rock is crushed ore and crushed capping. The results of this experiment are shown in the accompanying slides. This experiment visually verifies the conclusions drawn as to ground movement and subsidence in block caving at Miami.

Slide #1 - Before drawing - 300 ft. ore and 300 ft. capping.
Slide #2 - First stope drawing - arch formed.
Slide #3 - Arch working its way up the column of ore - broken ore in stope.
Slide #4 - Crown of arch breaking into capping.
Slide #5 - Haunches of arch have broken capping.
Slide #6 - Arch approaching surface - tension cracks forming. Angle to outside crack = 70 degrees. Arch forming in broken ore.
Slide #7 - Tension cracks forming blocks which are tipping toward stope. Dilution approaching draw points.
Slide #8 - Capping has reached draw points. Tension crack at 55 degrees.
Slide #9 - Second stope pulled - tension cracks forming.
Slide #10 - Surface block tipping into stope.
Slide #11 - Subsidence over second stope.
Slide #12 - Third stope - arch breaking into capping.
Slide #13 - Third stope - surface subsidence, inclusion of capping in ore.
Slide #14 - Third stope completed.
Slide #15 - Fourth stope - partially drawn.
Slide #16 - Fourth stope - completely drawn.
Slide #17 - Fifth stope breaks surface. Note pillar tipping toward stope.
Slide #18 - Fifth stope ½ drawn, with no dilution from #3 stope.
Slide #19 - Fifth stope completely drawn.

The experimental results are undoubtedly not an exact picture of block caving; however, the main features are shown very clearly.

In actual mining the outside draw points in the #1 stope would have continued to draw after the center of the stope was pulling capping. The slides tend to show more ore remaining than would be the actual case.

In order to induce caving, the model had to be continually tapped with a hammer, which caused compaction of the material representing the rock. This accounts for some of the sag in the surface and the sag in the line between ore and waste.

SUMMARY AND CONCLUSIONS

1. From the observed data at Miami it can be stated that the ground caves vertically or inside the undercut limits. Although the Miami orebody was highly faulted the only place this affected subsidence was at the Pinto fault which is a major fault with from 6" to 12" of very slick gouge.

2. The vertical tension cracks surrounding a caved block do not represent shear planes extending from the surface to the mining level.

3. Ground movement does occur outside the caved area and the distance that this movement occurs varies with the strength of the rock.

4. A line drawn from the mining level to the escarpment does not represent a zone of broken ground.

5. Dilution from capping is an inherent characteristic of block caving. There is abundant evidence in the drawing records of Miami Copper Company to prove that an irregular draw, whether caused by repair problems, packed chutes, hung transfer raises, or any other type of interruption of draw, causes a poor extraction. It is evident that intelligent draw supervision is the key to the control of dilution.

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Trans A.I.M.E. (1929) 161

2. F.W. Maclennan: Miami Copper Company Method of Mining Low-grade Ore Body
Trans A.I.M.E. (1930)

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(In files at Miami Copper Company)

4. J.W. Still: Block Caving at Miami
Trans. Mining Congress Journal (April 1955) 89

5. J.W. Vanderwilt: Ground Movement Adjacent to a Caving Block in the Climax Molybdenum Mine
Trans. A.I.M.E. (1949) 60

TITLES OF ILLUSTRATIONS

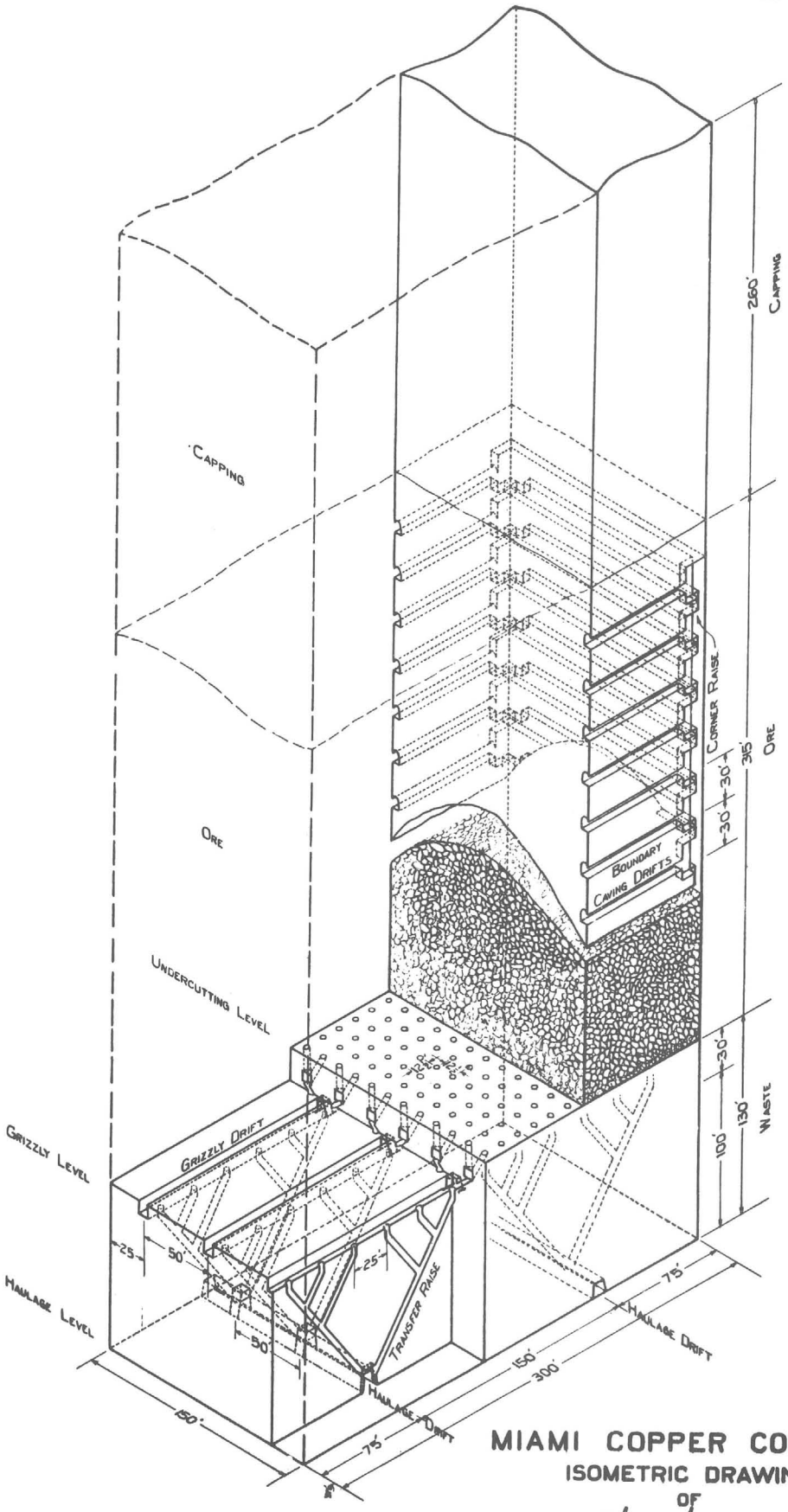
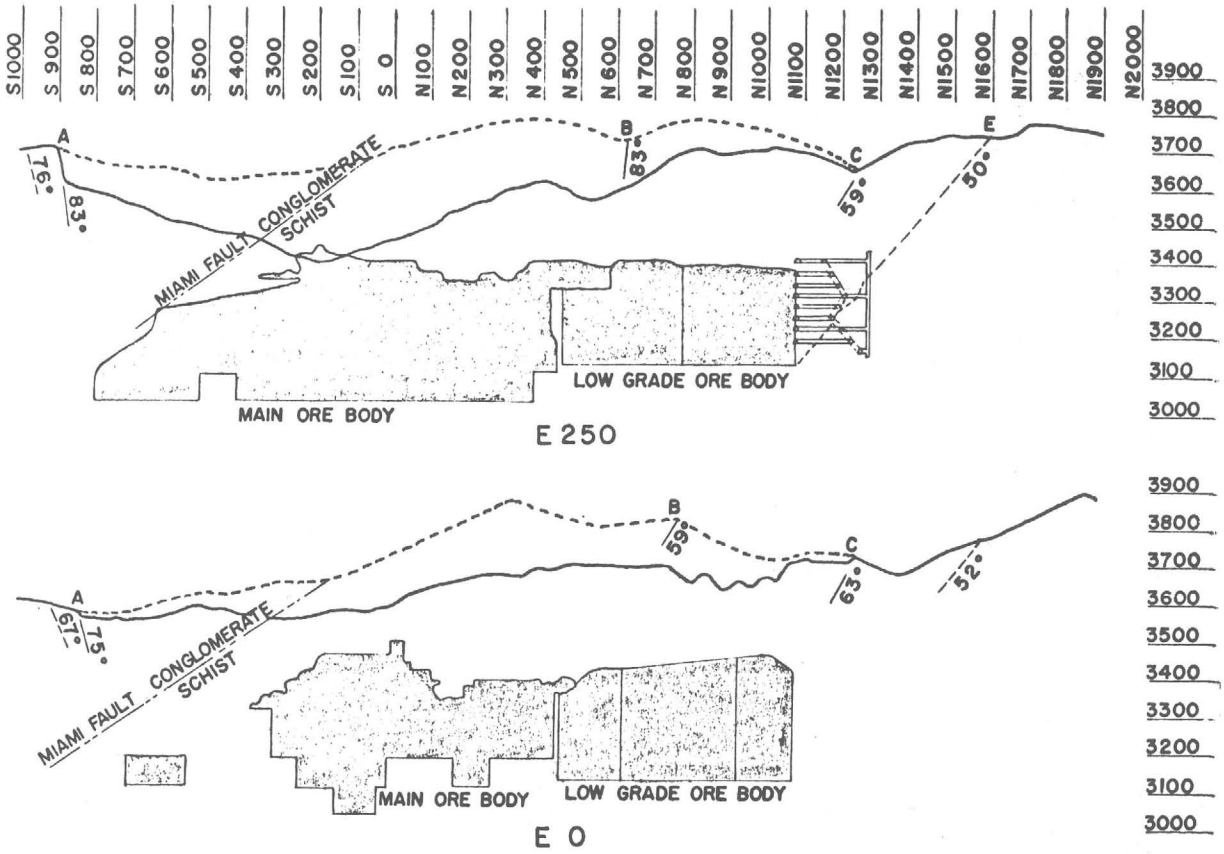


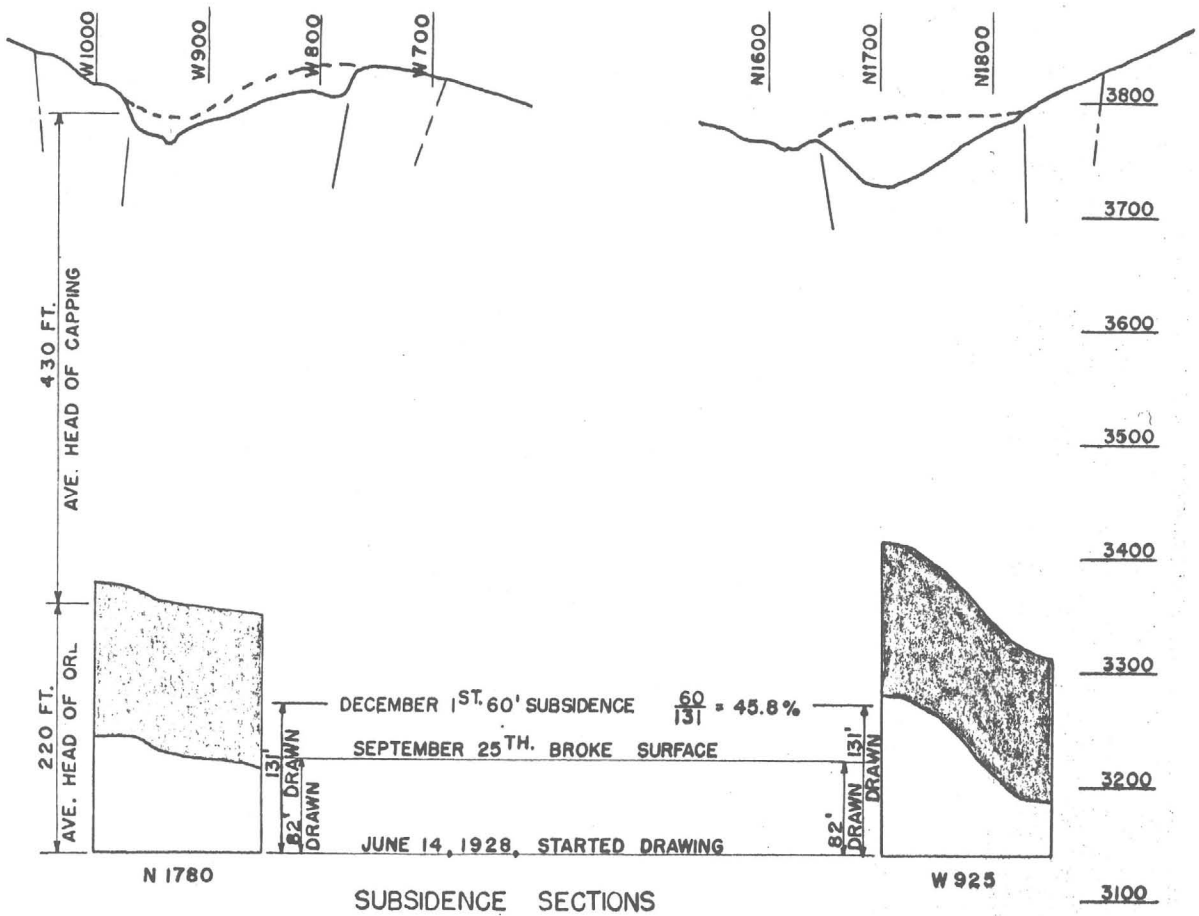
FIG. 3

MIAMI COPPER COMPANY
ISOMETRIC DRAWING
OF
150' x 300' STOPE
MIAMI, ARIZONA.
OCT. 1, 1929.



SUBSIDENCE SECTIONS
DECEMBER 1, 1928 1" = 300'

FIG. 4



SUBSIDENCE SECTIONS
DECEMBER 1, 1928 1" = 100'

FIG. 5

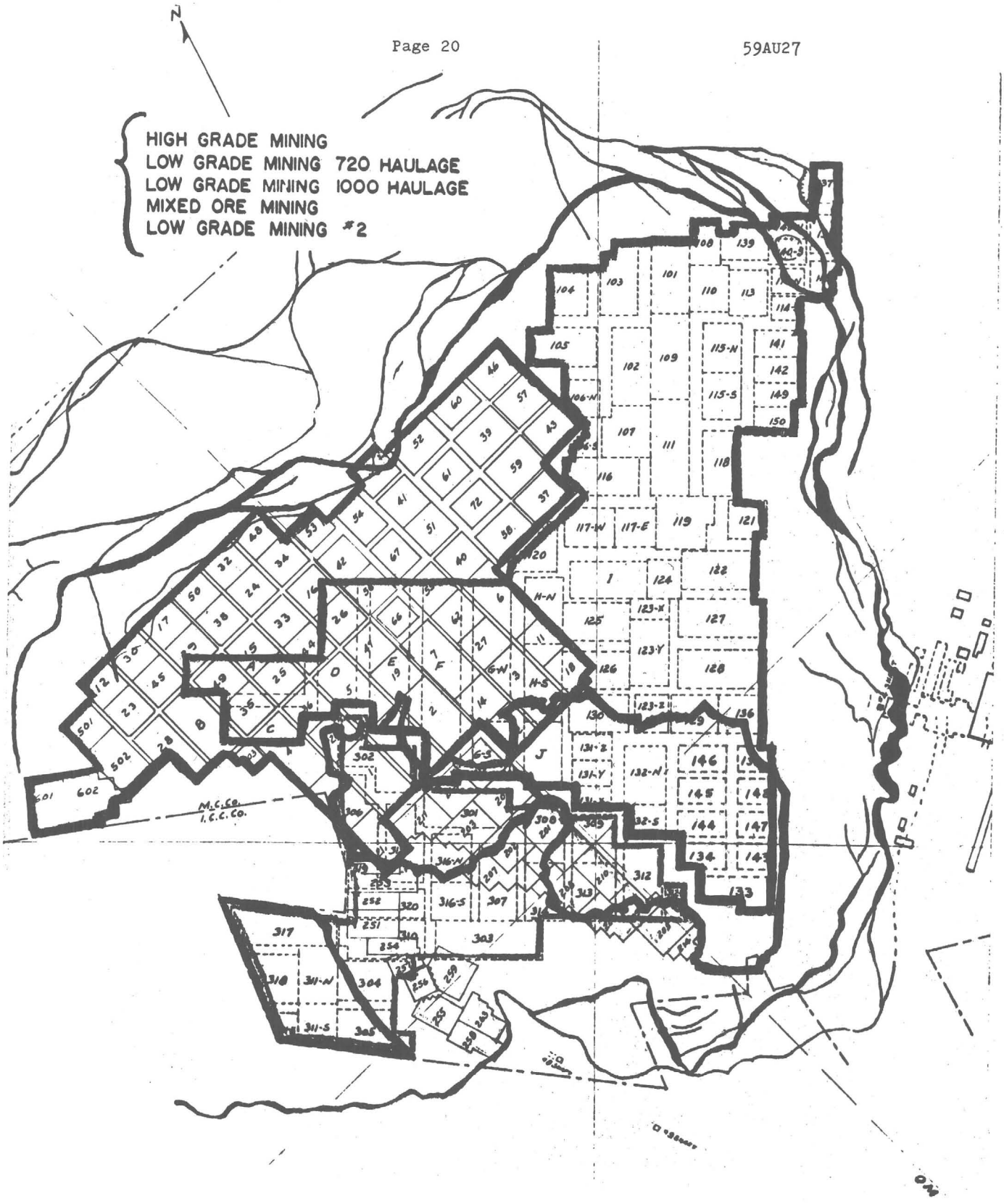

N
HIGH GRADE MINING
LOW GRADE MINING 720 HAULAGE
LOW GRADE MINING 1000 HAULAGE
MIXED ORE MINING
LOW GRADE MINING #2
M.C.Co.
I.C.C.Co.

FIG. 1

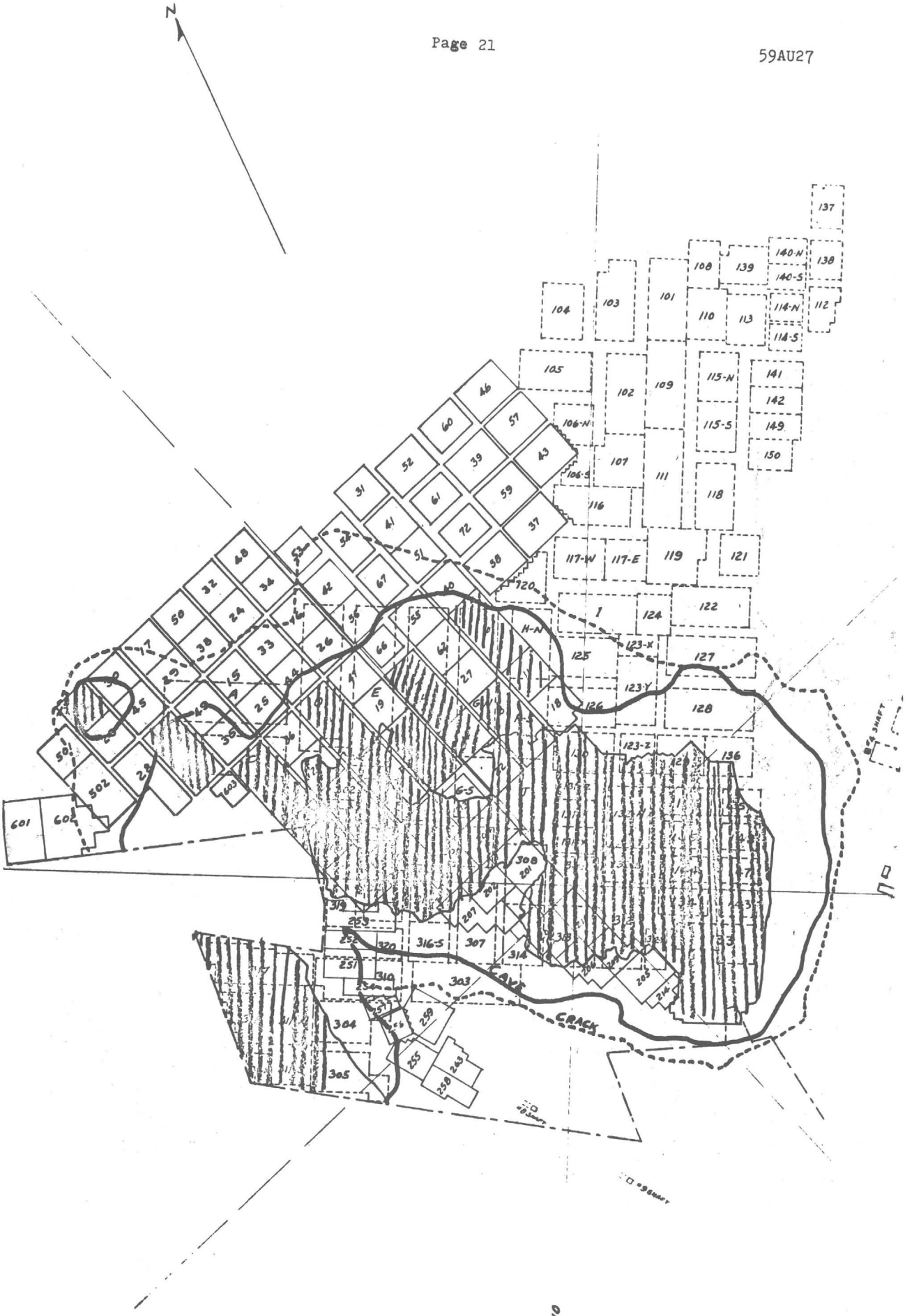
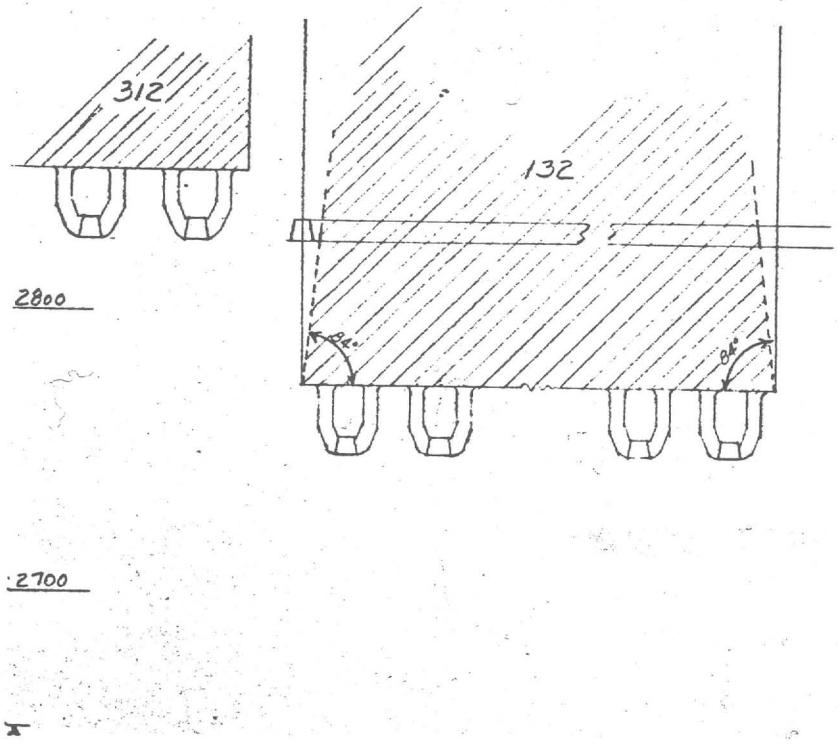


FIG. 2

FIG. 6

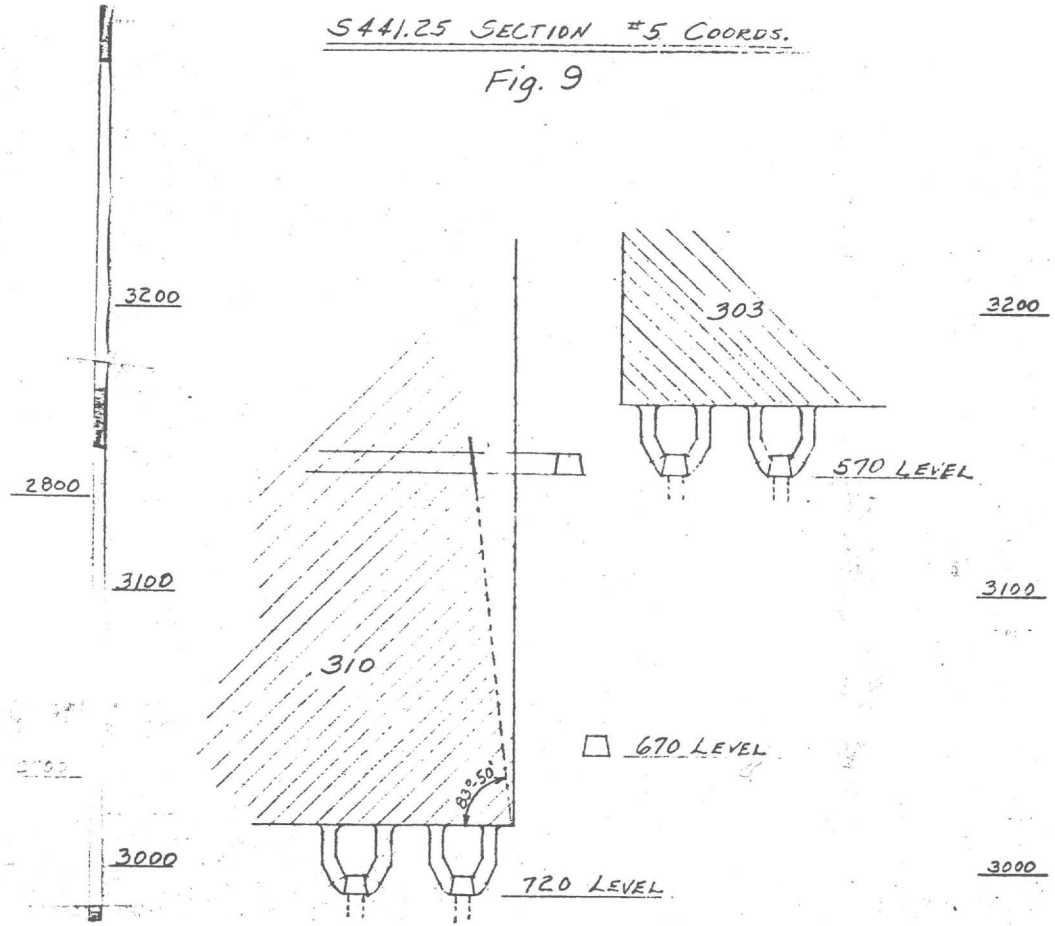
E 190 SECTION #5 COORDS.

Fig. 8



S 441.25 SECTION #5 COORDS.

Fig. 9



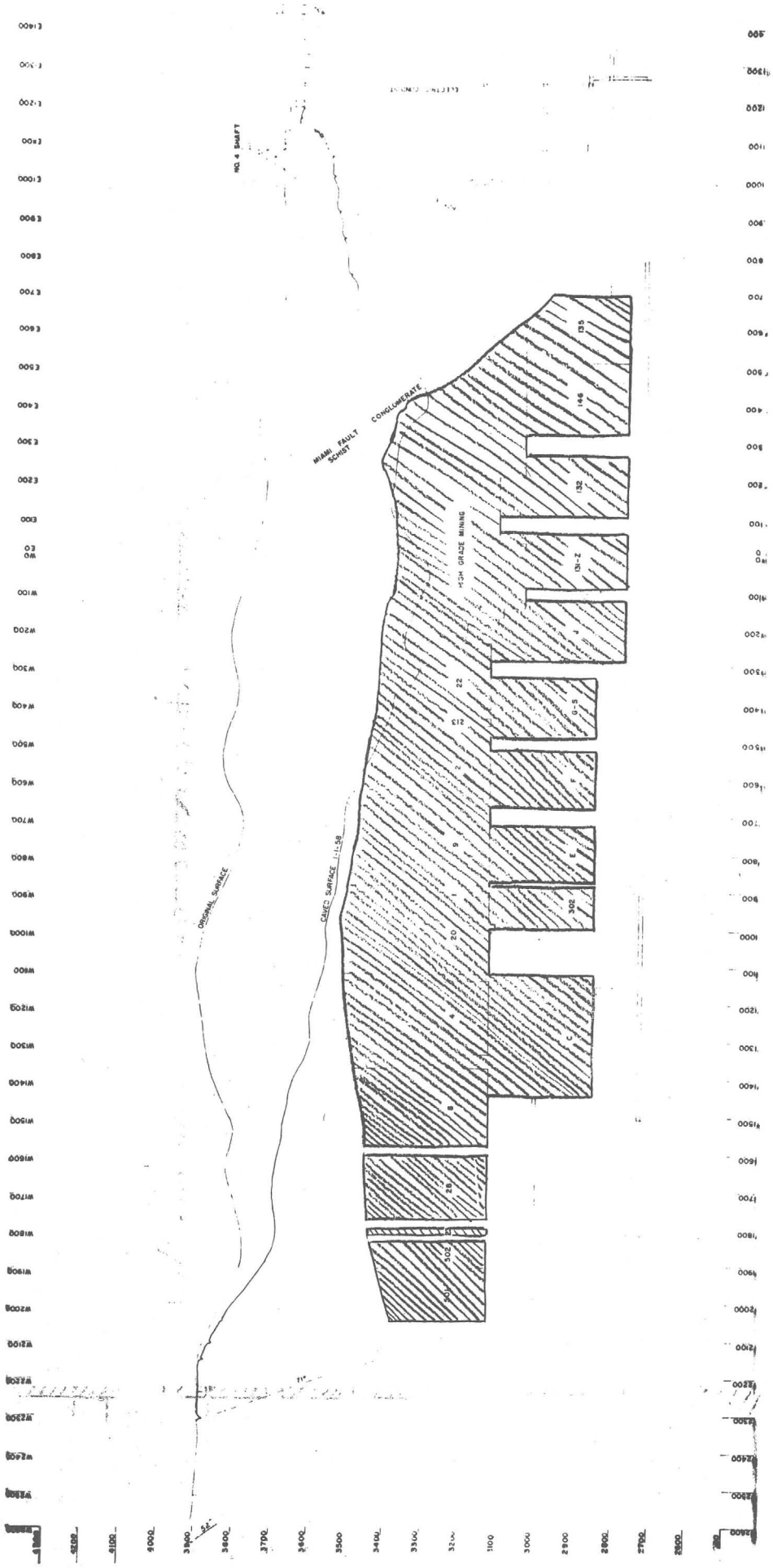
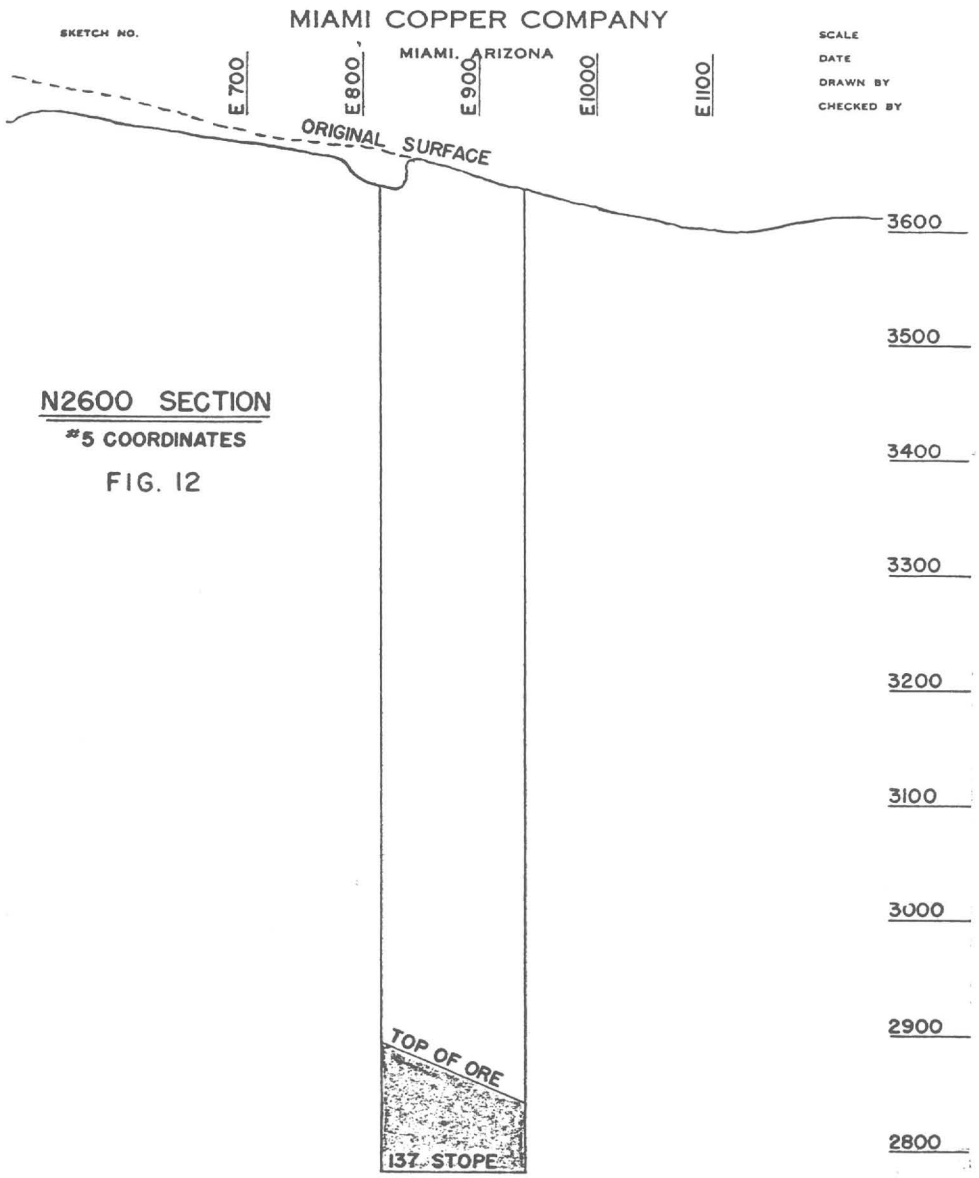

NO. 4 SHAFT
MIAMI FAULT
SCHIST
CONGLOMERATE
HIGH GRADE MINING
ORIGINAL SURFACE
CAVED SURFACE 1-1-58
135
146
132
131-Z
G-S
F
E
302
C
B
28
502
300
22
213
9
20
4
3500
3400
3300
3200
3100
3000
2900
2800
2700
2600

FIG. 10



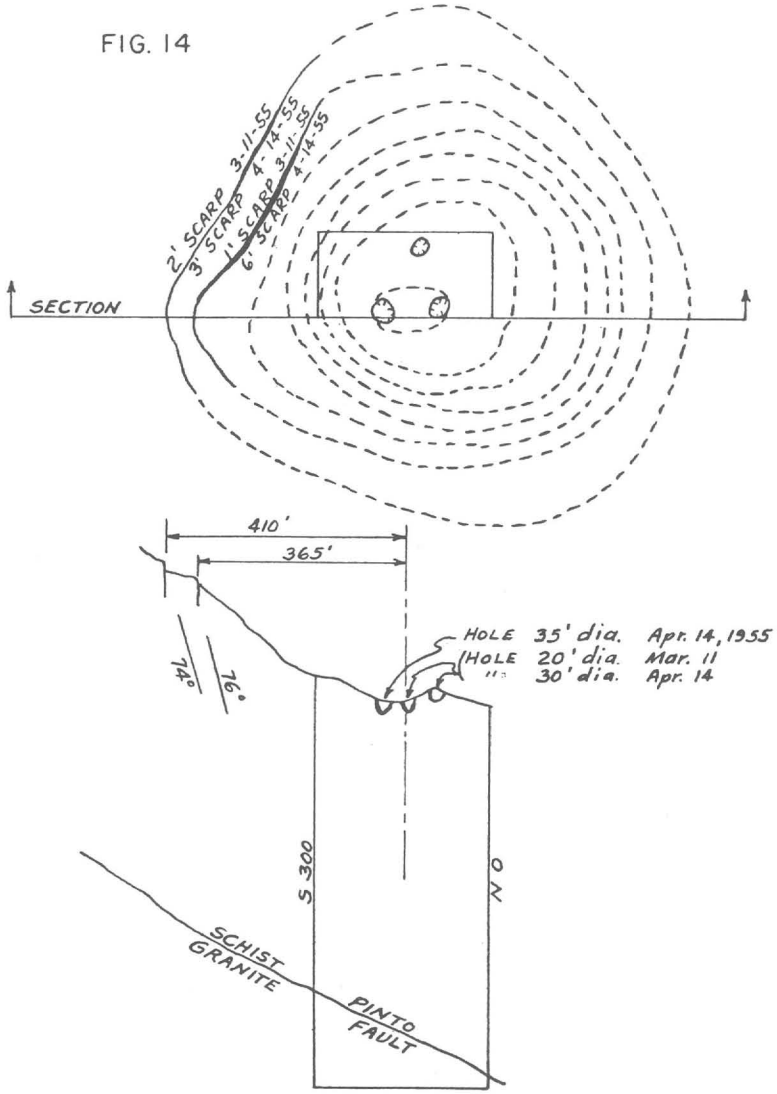
N2600 SECTION

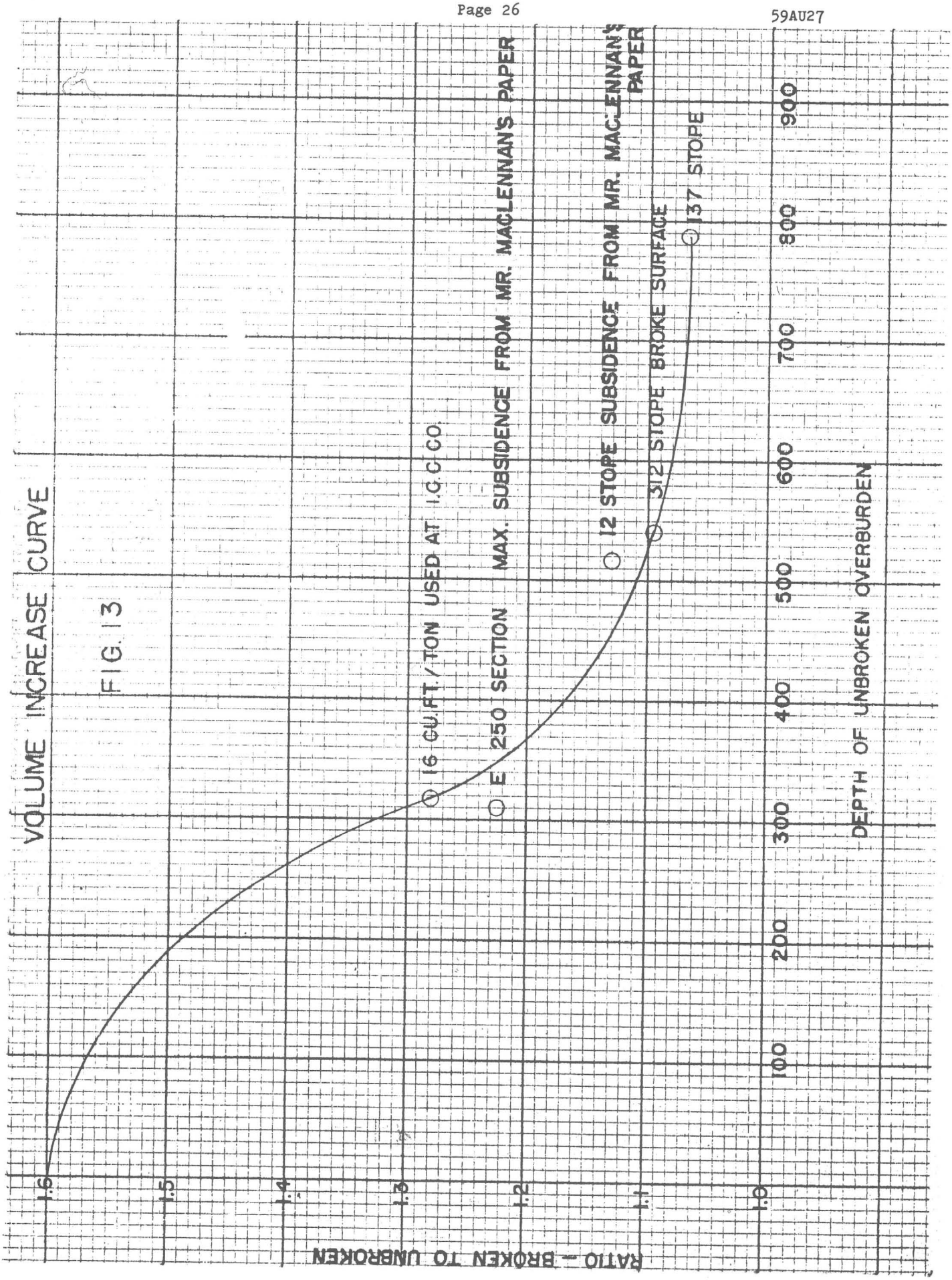
#5 COORDINATES

FIG. 12

EARLY. SUBSIDENCE 312 STOPE

FIG. 14




VOLUME INCREASE CURVE
FIG. 13
16 CU.FT./TON USED AT I.G.C.CO.
E 250 SECTION MAX. SUBSIDENCE FROM MR. MACLENNAN'S PAPER
12 STOPE SUBSIDENCE FROM MR. MACLENNAN'S PAPER
312 STOPE BROKE SURFACE
137 STOPE
1.6
1.5
1.4
1.3
1.2
1.1
1.0
100
200
300
400
500
600
700
800
900
DEPTH OF UNBROKEN OVERBURDEN
RATIO - BROKEN TO UNBROKEN

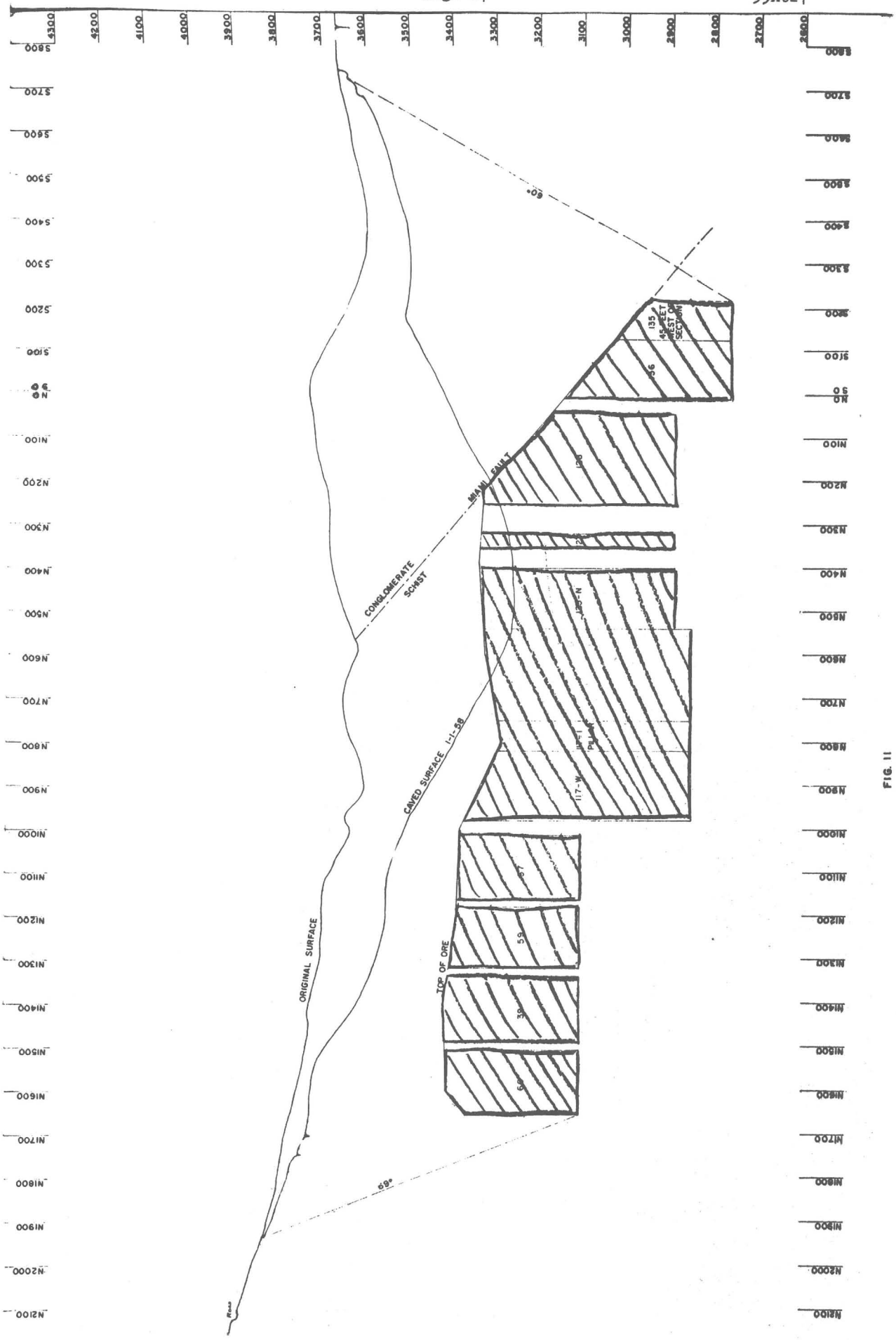
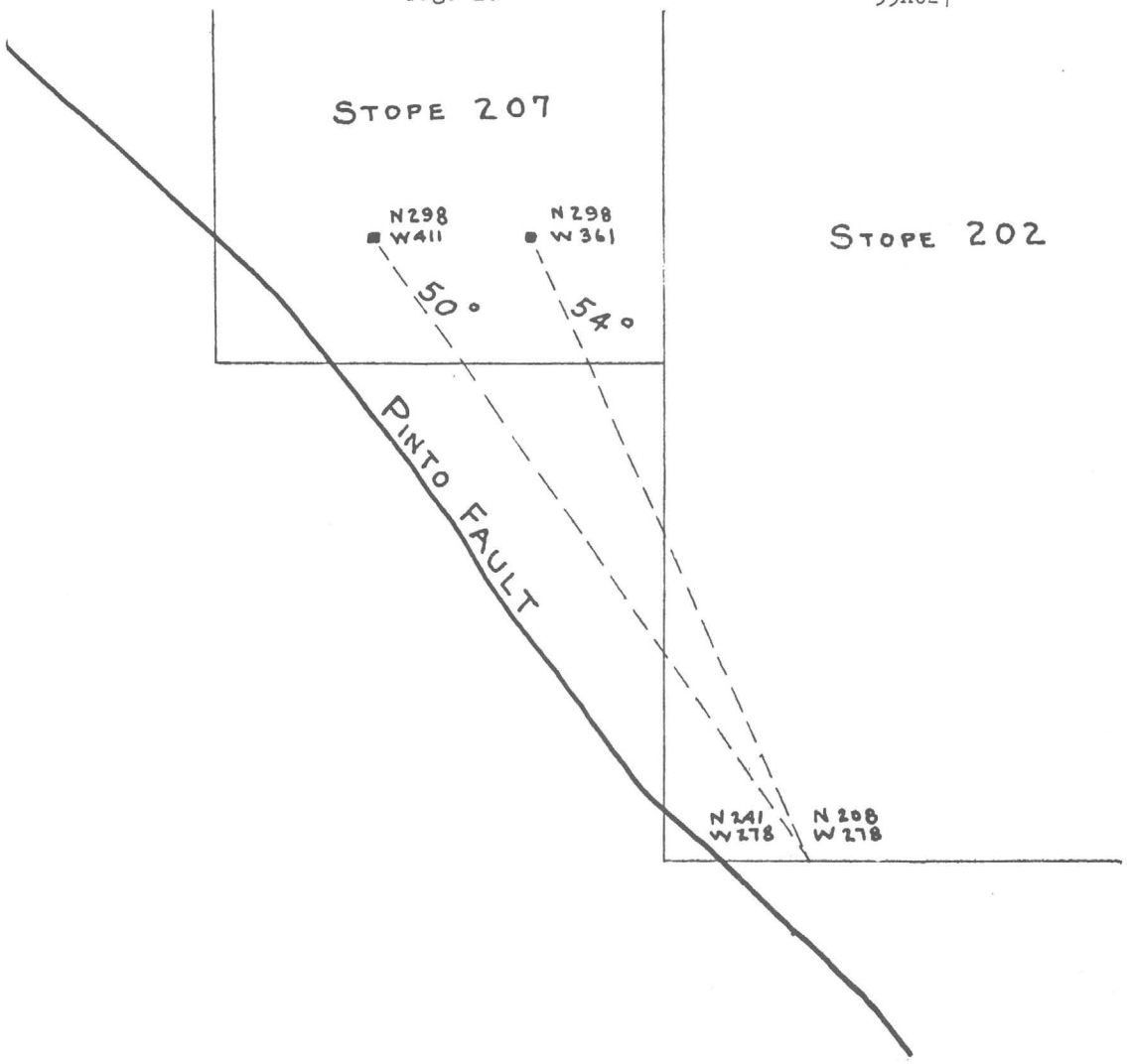
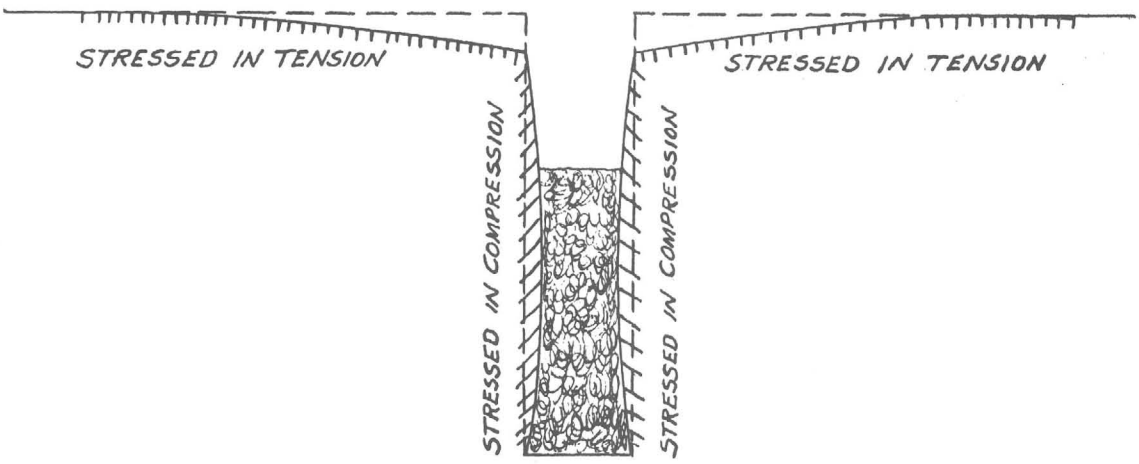

4300
4200
4100
4000
3900
3800
3700
3600
3500
3400
3300
3200
3100
3000
2900
2800
2700
2600
60°
135
45 FEET
SECTION
56
MAIN FAULT
CONGLOMERATE
SCHIST
CAVED SURFACE 1-1-58
ORIGINAL SURFACE
TOP OF ORE
59
39
60°

FIG. 11


STOPE 207
STOPE 202
N298
W411
N298
W361
50°
54°
PINTO FAULT
N241
W278
N208
W278


STRESSED IN TENSION
STRESSED IN TENSION
STRESSED IN COMPRESSION
STRESSED IN COMPRESSION

ROLAND B. MULCHAY

CALCULATION OF PIT LIMITS AND ORE RESERVES, COPPER CITIES MINING COMPANY

BY J. H. Gray and W. W. Simmons

INTRODUCTION

Copper Cities Mining Company operates an open pit mine in a disseminated copper deposit which is located in the Globe-Miami district, Gila County, Arizona.

Some attention was given the deposit early in the history of the district, but the first major exploration was begun in 1917 by the Louis d'Or Mining and Milling Company. This work, consisting of a shaft 360 feet deep and 12 drill holes, totaling nearly 9000 feet, was completed in 1922. Further exploratory drilling was done in 1929 and 1930. These efforts showed the presence of disseminated copper minerals, but the grade was too low to be considered ore at that time.

Miami Copper Company purchased the property in 1940 and organized a wholly-owned subsidiary, Copper Cities Mining Company, to operate the property. This paper is chiefly concerned with exploration by this company.

Systematic exploration by churn drilling was started in 1943 and completed in 1948 with the blocking out of 33,800,000 tons of low grade ore amenable to open-pit operations. Stripping was begun in November, 1950, and mill production was started in August, 1954.

ACKNOWLEDGEMENTS

The writers are indebted to many people for help of various kinds. We are particularly indebted to Dr. N. P. Peterson and Mr. E. N. Pennebaker. Their excellent work, Dr. Peterson's published paper

and Mr. Pennebaker's private reports on the Copper Cities ore body are the basis for much of the geological thought in the present work.

GENERAL GEOLOGY

Figure 1 is a generalized geologic map of the Copper Cities pit and the immediate area around it. The Lost Gulch quartz monzonite is the predominant rock in the pit. The other important rock in relationship to the ore is granite porphyry. In general, the quartz monzonite is the better grade material. A small amount of diabase is included within the pit limits.

STRUCTURE

The most important structures in relationship to the ore body are the Coronado and Drummond fault zones. The Coronado which lies near the west side of the pit, strikes north and dips steeply west. The Drummond, near the eastern pit limit, strikes northwest and dips about 60° northeast. Along the northern edge of the pit is the Sleeping Beauty fault which strikes northeast; its dip is unknown. The area bounded by these faults has been raised relative to the adjacent blocks. The ore body itself is intricately dissected by many minor fractures with no dominant pattern.

MINERALOGY

The principal hypogene minerals in the deposit are quartz, pyrite, chalcopyrite and molybdenite. Chalcocite is the predominant supergene sulfide mineral, and malachite, azurite and turquoise are the principal acid-soluble copper minerals in the ore body.

CHURN DRILLING

The base pattern for churn drilling was a 250 foot grid. From previous experience in the district, this was thought to be sufficiently close for accuracy of grade and tonnage calculations, but as a measure of insurance, some holes were drilled at intermediate points. Drilling at intermediate points was also used to more precisely define the pit limits on some sections. The base grid was oriented to make the sections at right angles to the supposed elongation of the ore body. As finally developed by the drilling, the right angle relationship did not hold exactly, but no serious error was introduced by this fact.

The churn drill holes were sampled at 5 foot intervals using the conventional Jones splitter. Each 5-foot sample was assayed for total copper and oxidized copper. Composite samples of each 50 feet were assayed for gold, silver and molybdenum. All samples were logged for rock type and other geological features.

Preceding and concurrent with the drilling, a geological map of the surface was made as a guide for the exploration. At the completion of the drilling, a map was prepared showing surface geology, topography and drill hole collars.

COMPILATION OF DATA FROM EXPLORATION PROGRAM

Using the plan map as a base, cross sections and longitudinal sections were made showing the assay data of drill holes. By inspection, it was obvious that correlation between drill holes on assay data alone did not show a coherent nor probably true relationship. By plotting rock types and other

geology on the sections, more reasonable ore outlines could be drawn.

On the basis of previous district experience, it was decided that a 45 foot bench height would give the best mining operations. By inspection and cut and try, the elevation of the bottom level of the pit was set at 3600 feet. Factors which influenced this selection were maximum ore production, working room, pit drainage, haulage and others, but to some extent the final figure was arbitrary and depended on the judgment of the planners. Using this base, the upper bench elevations were drawn on the sections and the average grade of the holes through each bench was plotted along the hole. These sections were the work sheets for the determination of the ultimate pit limits.

DETERMINATION OF ULTIMATE PIT LIMITS

The exploration indicated an ore body of small tonnage and low grade. It was improbable that mining would disclose enough additional ore, or that market conditions would change sufficiently, to justify enlarging the initially set pit limits. For these reasons, it was of prime importance that the pit as initially planned be the best economically for the life of the operation. This meant that the pit limits must be extended to the theoretical slope lines which would just meet a set of conditions to give an acceptable minimum profit.

From the exploration, it was known that the waste to be stripped was a relatively uniform cover. It was also evident that the copper mineralization was largely gradational which meant that the pit limits would be assay boundaries rather than some other geologic feature. For these reasons, it was believed that the pit limits could

be set by consideration of a line rather than by calculating a three-dimensional tonnage figure. This concept materially reduced the required calculations.

To locate the theoretical slope lines certain assumptions were necessary. These were grouped as the cost to concentrate one ton of ore, the costs per pound of copper, and the per cent copper extraction.

The cost to concentrate one ton of ore includes:

a. Cost per ton mined.
b. Cost per ton milled.
c. Miscellaneous costs per ton.

The costs per pound of copper include:

a. Smelting cost per pound of copper.
b. Miscellaneous costs per pound copper.
c. Minimum acceptable profit per pound of copper.

The gross value per pound of copper can be defined as the market value minus the costs per pound of copper.

The mine grade which will just satisfy the assumed costs per ton of ore and per pound of copper can be calculated by the following formulae.

A. $\dfrac{\text{Total cost to concentrate 1 ton ore}}{\text{Gross value/lb. Cu}}$ = Net lbs. Cu/ton ore required to give minimum acceptable profit.

B. $\dfrac{\text{Net lbs. Cu/ton ore}}{\text{\% Extraction}}$ = Gross lbs. Cu/ton ore required to give minimum acceptable profit.

C. $\dfrac{\text{Gross lbs. Cu/ton ore}}{2000}$ = % Cu required in mill heads to give minimum acceptable profit.

To illustrate the use of the above formulae, certain figures, not necessarily those used at Copper Cities, are shown as follows:

Cost to concentrate 1 ton of ore: $0.94

Cost/lb. Cu: $0.0575 = Cost/lb. Cu to put 1 lb. Cu in concentrate into a market product.

Market price/lb. Cu	$0.18000
Minus cost/lb. Cu	0.05751

$0.12249 = Gross value/lb. Cu at mimimum acceptable profit.

$$\frac{\$0.94 \text{ (cost to concentrate 1 ton)}}{0.12249 \text{ (Gross value/lb. Cu)}} = 7.674 = \text{Net lb. Cu per ton of ore required to give minimum acceptable profit.}$$

$$\frac{7.674 \text{ (Net lbs. Cu/ton ore}}{.87511 \text{ (\% extraction)}} = 8.769 = \text{Gross lbs. Cu/ton ore required to give minimum acceptable grade.}$$

$$\frac{8.769 \text{ (Gross lbs. Cu/ton)}}{2000} = .438\% \text{ Mine grade necessary to satisfy above assumptions.}$$

The mine grade of ore necessary to meet the above conditions plus the mining of 1 ton of waste is calculated by the same basic formula, but must include the additional cost of mining 1 ton of material, i. e.,

$$\frac{\dfrac{\$0.94 + \$0.20}{.12249}}{\dfrac{.87511}{2000}} = 10.635 \text{ lbs. Cu ton} = 0.532\% \text{ mine grade.}$$

The grade necessary to give the minimum acceptable profit with a different tonnage of waste removal is calculated similarly.

A table of waste to ore ratios for which a corresponding mine grade of ore will meet the assumed conditions including profit is presented in Table 1.

With the aid of this table, the determination of theoretical slope lines which fix the ultimate pit limit can more easily be made.

At Copper Cities, it was decided that a 45° backslope could be safely maintained. The initial step in locating a backslope line on any section was to arbitrarily draw a 45° line on the section.

This initial line was, of course, located near one extremity of the ore body and as near the correct position as judgment based on quick visual inspection permitted.

In the investigation of the slope lines, it was assumed that ore grade in any prospect drill hole could be projected to the mid point between 2 adjoining holes. This assumption was subject to modification by geological conditions. The investigation consisted of several steps. First the total length of the backslope line from the bottom of the lowest ore bench to point where the line intersected the surface was measured. Next, the integral lengths of lines along the slope through each ore bench was measured, and each length multiplied by its related grade of ore. The summation of these "Grade Lengths" divided by the summation of the integral length of slope line through the ore benches is the average grade along this section of the backslope line. The total length of backslope line minus the length in ore is the measured length of slope line in waste.

By reference to Table 1, a waste to ore ratio corresponding to the average grade as computed can be found. By multiplying this waste figure by the length of ore line, a theoretical length of waste line is obtained. If this theoretical length of waste line is greater than the length of line actually measured, the ultimate slope line lies in the direction away from the center of the pit. Conversely, if the theoretical length of waste line is less than the measured waste line, the ore along this slope will not pay for the waste and the ultimate pit limit lies in a direction toward the pit center. By sufficient trial and error, a slope line can be located along which the measured waste will just equal the theoretical waste that can be carried by the ore under the assumed conditions.

Figure 2 is a section of Copper Cities. On line A, the total length of the backslope line from pit bottom to intersection with surface is 545 feet. The summation of the integral length of line through each ore bench multiplied by its related grade of ore divided by the total length of line in ore is .674%, which is the average grade of that portion of the slope line in ore. The length of the slope line in waste is 290.40 feet. By reference to Table 1, the waste ore ratio corresponding to .674% is 2.5. By multiplying the waste figure by 254.60, the length of line in ore, we see that the slope line can be moved outward. By similar calculations on Line B, the waste-ore ratio was found to be 1:1, and the waste figure multiplied by the ore length exceeded the measured waste length and the slope line must be moved inward. On Line C, the theoretical waste line was 323.33 feet and the measured waste was 329.05 feet. It was felt that this was as close as the accuracy of the original data and assumptions permitted, and this line was used. The detailed calculations of the lines are given below as illustrative of the method.

CALCULATIONS: INVESTIGATION OF SLOPE LINES

LINE A

Measured line			Slope Line Thru Ore		Grade
Designation	Length Feet	Bench	Length	Grade	Length Units
Ore	255.0	3645	45.40	.55	25.02
Waste	290.40	3645	18.15	.47	8.53
	545.40	3690	63.65	.70	44.56
		3735	63.65	.58	36.92
		3825	63.65	.89	56.65
			254.60	.674	171.68

From Table 1: .674% ore grade will carry 2.5:1 waste to ore
∴ 254.6 (ft. ore) x 2.5 = 636.50 ft. of waste which 254.6 ft. of .674% ore will carry. Since this is greater than the 290.4 ft. of waste actually measured along this slope (A), line B was tried.

LINE B

Measured line Designation	Length Feet	Bench	Slope Line Thru Ore Length	Grade	Grade Length Units
Ore	88.65	3645	63.65	.47	29.92
Waste	456.35	3690	25.00	.70	17.50
	545.00		88.65	.535%	47.42

From Table 1: .535% ore grade will carry 1:1 waste to ore ∴ 88.65 ft. x 1.0 = 88.65 ft. of waste which 88.65 ft. of .535% ore will carry. Since this is less than the 456.35 ft. of waste measured along this slope line B, line C was tried.

LINE C

Measured line Designation	Length Feet	Bench	Slope Line Thru Ore Length	Grade	Grade Length Units
Ore	230.95	3645	63.65	.47	29.92
Waste	329.05	3690	63.65	.70	44.56
	560.00	3735	63.65	.58	36.92
		3825	40.00	* .50	20.00
			230.95	.569	131.40

* Assume .5% grade from midpoint to fault.

From Table 1: .569% ore grade will carry 1.4:1 waste to ore. ∴ 230.95 ft. of .569% ore will carry 323.33 ft. of waste. The slope line thru waste actually measures 329.05 ft. As this is as close as the accuracy of data, this line was used as theoretical slope line.

The back slope lines on all sections were calculated by a similar process.

The portion of the line near the Drummond fault illustrates a modification of the basic method based on geological conditions. Ore was figured to the fault rather than to the midpoint between holes.

After completion of the sections, the data was transferred to plan maps. The initial step was to pick a key level near the midpoint of the ore column. A composite plan map was constructed by mechanical development from the key level up slope to surface, and down slope to the pit bottom. Individual bench maps were then made by transfer of theoretical bench outlines from the composite map and plotting of prospect holes with average grade and geology as derived from the sections.

Polygonal areas of grade influence modified by the geology were constructed around each hole. By planimeter, the area of influence of each hole was determined. From this data, the average grade and the tonnage of ore and waste on each bench was computed. The total tonnage of ore and waste in the pit was obtained by addition of individual bench tonnages and average mine grade by the bench average grade weighted by bench tonnage.

Based on these calculations, the Copper Cities pit contains 33,800,000 tons of ore. The waste to be removed at the start of mining totaled 34,700,000 tons which is a 1.03 to 1 waste to ore ratio.

TABLE 1

ORE-WASTE RATIO AND ITS REQUIRED MINE GRADE

Ratio Waste to Ore	Mine Grade Ore	Ratio Waste to Ore	Mine Grade Ore	Ratio Waste to Ore	Mine Grade Ore
0.0:1	.438%	1.7:1	.597%	3.4:1	.756%
0.1:1	.447	1.8:1	.606	3.5:1	.765
0.2:1	.457	1.9:1	.616	3.6:1	.774
0.3:1	.466	2.0:1	.625	3.7:1	.784
0.4:1	.475	2.1:1	.634	3.8:1	.793
0.5:1	.485	2.2:1	.644	3.9:1	.803
0.6:1	.494	2.3:1	.653	4.0:1	.812
0.7:1	.504	2.4:1	.662	4.1:1	.821
0.8:1	.513	2.5:1	.672	4.2:1	.831
0.9:1	.522	2.6:1	.681	4.3:1	.840
1.0:1	.532	2.7:1	.690	4.4:1	.849
1.1:1	.541	2.8:1	.699	4.5:1	.858
1.2:1	.551	2.9:1	.709	4.6:1	.868
1.3:1	.560	3.0:1	.718	4.7:1	.877
1.4:1	.569	3.1:1	.727	4.8:1	.886
1.5:1	.578	3.2:1	.737	4.9:1	.896
1.6:1	.588	3.3:1	.746	5.0:1	.905

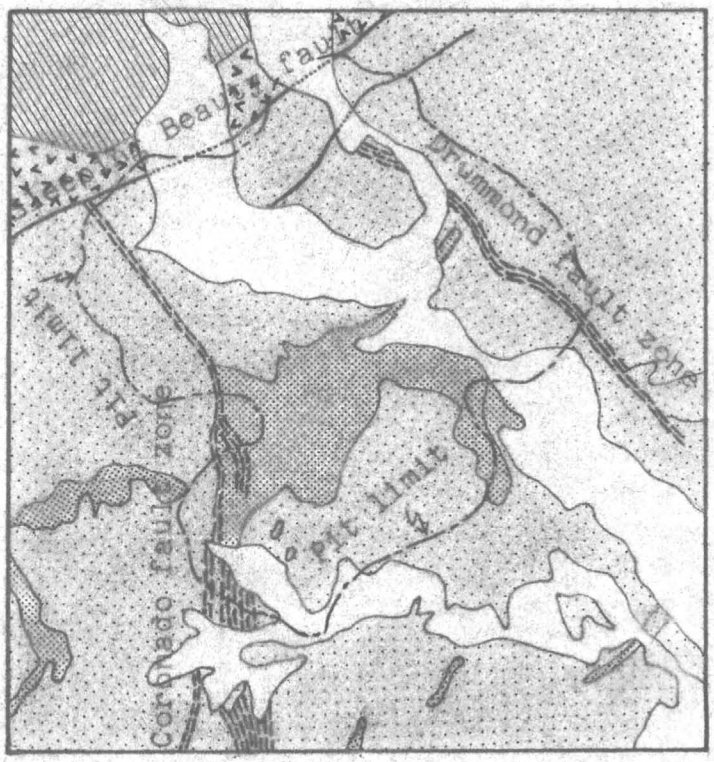


Fig. 1 Geologic map of the copper-bearing area
(After: N. P. Peterson)

EXPLANATION

Alluvium

Paleozoic limestones
and Apache group

Granite porphyry

Diabase

Lost Gulch
quartz monzonite

Fault

Contact

Scale
1 inch = 1000 feet

N

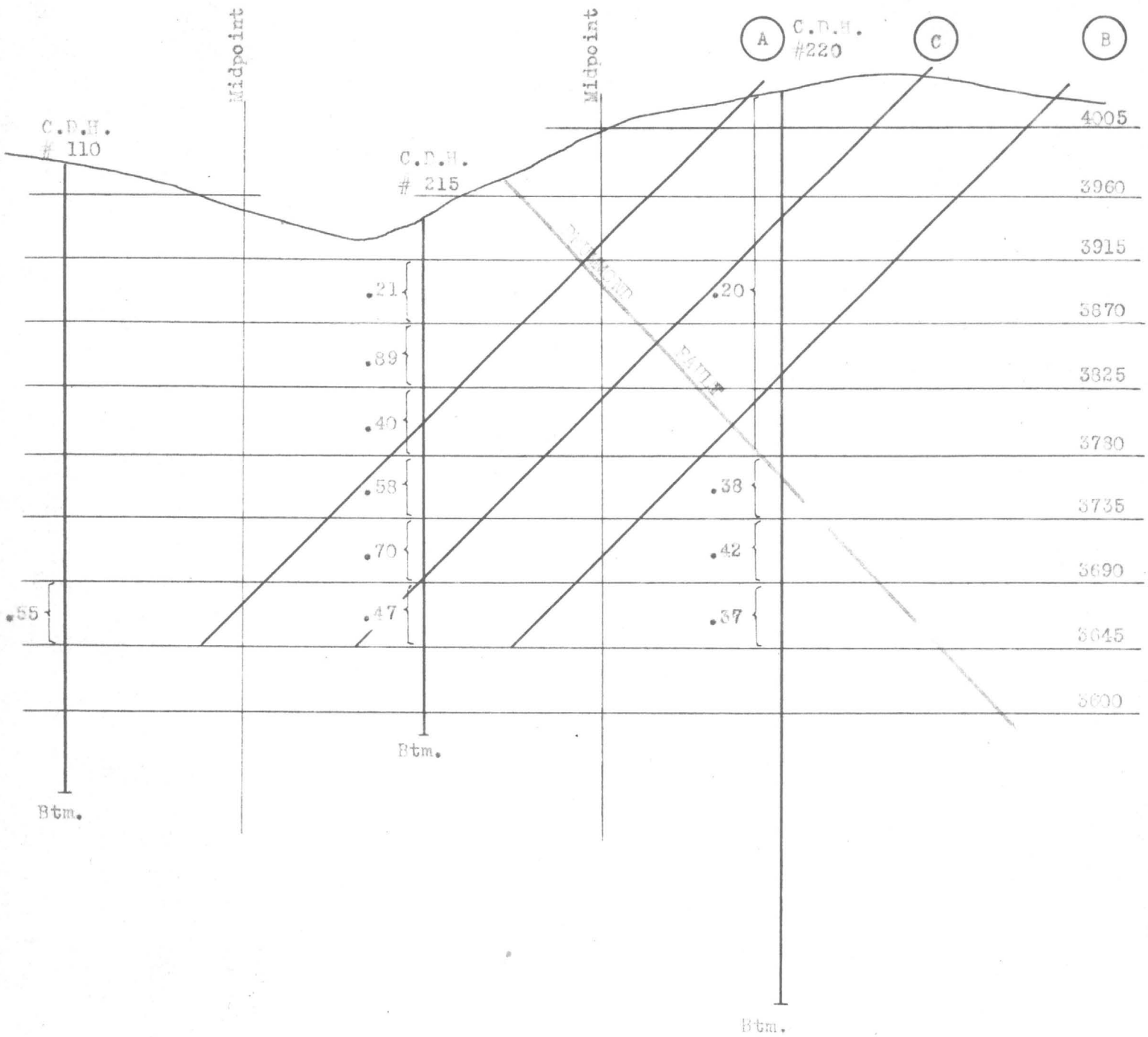


Fig 2. Portion of section N 3500 illustrating investigation of slope line.

The Use of a Caved Block as an Ore Pass and Its Application to Open-Pit Mining

Dual Process Metallurgy Stretches Inspiration Ore Reserves

Reprinted from Mining World

September 1957

respect Inspiration is moving ahead and during the last few years notable progress has been made.

Enlarge Leaching Vats

In 1948 it was realized that in order to maintain copper production with lower grade of ore, more leaching capacity was necessary. Room for horizontal extension of the leaching vats was not available. Therefore, it was decided to increase their height and the walls were raised 18 in. giving a capacity increase of about 10 percent. This job presented an interesting problem, as the walls had to be raised without interfering with normal operation.

The presence of slimes always complicates vat leaching. Percolation of solutions and wash waters through the bedded ore is interfered with and extraction suffers. This had long been recognized at Inspiration and suitable facilities had been provided to deslime the Leaching Plant feed. In recent years, the proportion of primary slimes in ore being mined from certain sections of the orebody was on the increase. This, together with a 10 percent increase in plant capacity, called for revision in fine crushing and classifying practice to meet the situation.

For the successful leaching of sulphide ores, contact time, strength, and temperature of the solvent are all important. To gain more contact time, a revision of solution flow made it possible to adopt a "continuous" wash in place of the previous "batch" wash system. By this means it has been possible to gain 24 hr additional contact time for leaching. Further research was successful in developing improved control over the ferric sulphate content of the leaching solvent. Taken together, these several forward steps have vastly improved the outlook for successful leaching of the increased sulphide content of present-day ores.

Miscellaneous

In bringing Inspiration operations up to date, other forward steps have been taken. At Inspiration we have 17 miles of standard gauge railroad. This railroad has been completely dieselized and five diesel electric locomotives are in service.

Railroad operations, as well as open pit operations, are controlled by two-way short-wave radio communication.

The main coarse crushing plant has been completely modernized. Four vibrating rod deck grizzlies and two seven-ft standard cone crushers have replaced the original plant which contained twelve individual crushing units.

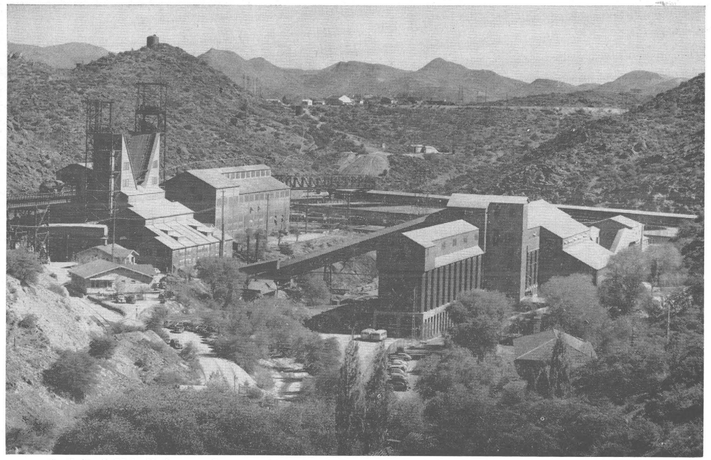
All in all, with the assurance of longer life brought about by conversion to open pit mining and the improvements in metallurgical practice, Inspiration has gone ahead to meet the new situations constantly arising. Every effort has been made to improve control, flexibility, and efficiency of operations.

In this era of change, a "new look" is necessary. At Inspiration the "new look" involves a broader approach, by management and staff, in considering the ever-changing problems of the day. Only by recognizing the constant need for improvement will it be possible to meet the challenge presented by these changing times!

The "New Look" at Inspiration

P. D. I. HONEYMAN
Vice President and General Manager
INSPIRATION CONSOLIDATED COPPER COMPANY

Reprinted from
MINING CONGRESS JOURNAL
September, 1954



Inspiration's Main Shaft area. Coarse crushing plant is between twin shafts. Leaching tanks are in middle ground

The "New Look" At Inspiration

Broad Approach Lets Management and Staff Foresee Problems and Solve Them as They Arise

By P. D. I. HONEYMAN
Vice-President & General Manager
Inspiration Consolidated Copper Co.

IN the beginning there was the Inspiration Copper Co. with its Joe Bush, and the Live Oak Development Co. Consolidation of these properties, with the later acquisition of the Keystone together with various other additions here and there, resulted in the birth in 1912 of the Inspiration Consolidated Copper Co. This mine has, during its life, ranked as one of the great porphyry copper producers. The forward looking engineers of those early days realized that bold, new, low-cost methods of mining would be necessary to make Inspiration a success. As a result, it might be said that Inspiration became the mother of "block caving" as a method of mining. Copper was to be recovered by concentration and smelting and a 20,000-ton mill was installed. However, the concentrator operation was continued only until 1927. Then, because of the ever-increasing oxide content of the ore, milling was superseded by leaching. The leaching process adopted was developed after ten years of research and is known as the ferric-sulphate leach. In this process both sulfide and oxide minerals are simultaneously dissolved and the copper is produced as an electrolytic cathode. The 13,000-ton leaching plant built at that time has continued to serve the needs of Inspiration up to the present.

From these beginnings through 1945, there were mined from the Inspiration orebody some 109,000,000 tons of ore, from which were produced 2,000,000,000 lb of copper. But about 1946, it began to be apparent that perhaps the end might be in sight. Underground mining was becoming more difficult and production costs were soaring at an alarming rate. Further, the question of control of both grade and the ratio of sulphide-oxide content of the ore, so necessary to the efficient operation of the ferric-sulphate leaching process by which the mixed ore was treated and its copper content recovered, was also tending to get out of hand.

Study Open Pit Methods

Facing these problems, urged on by a vital desire to perpetuate the life of the property, the company initiated a study to determine whether matters could be improved by the adoption of open pit methods of mining. It soon became evident that, by the use of modern electric shovels and truck haulage, not only would an open pit operation be feasible, but also such a method would be economic and would go a long way toward solving the vital problems of ore control. Furthermore, minable ore reserves would be increased, as fringe ore, too low in grade or otherwise unavailable to underground mining, could be included in the scope of pit operations.

From this point the initial development of the Live Oak open pit proceeded. Much of the remaining ore in this area actually came through to surface. The stripping ratio was low and some 20,000,000 tons of good ore would be made available. Plans were soon completed for the development of this pit on this basis. The necessary equipment was purchased and soon the job of building haulage roads, most of which are on a seven percent grade, and developing the ore for mining was well under way. A new primary crusher plant, with ore storage bins and railroad connections with existing lines, for delivery of pit ore to the treatment plants, was constructed. By April of 1948 the first open pit ore was being mined and a new chapter in the life of Inspiration had begun.

Open Thornton Pit

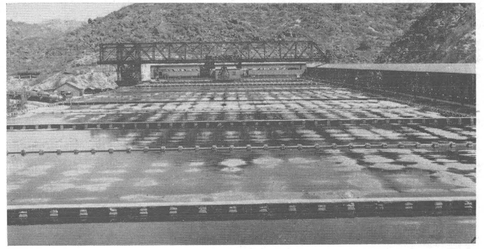
From that time forward both progress and expansion of pit operations was rapid. Eagerly casting an eye around the staff soon decided there were other areas in the remainder of the orebody which could be brought in as open pit operations. As it generally does, one thought led to another and by an eventual bold decision which virtually involved the moving of a mountain, the "Thornton Pit" came into existence. This pit, named after the recent president of Inspiration, W. D. Thornton, is a sizable project. In the area the ore lies deep and to uncover it, some 60,000,000 tons of waste will have to be moved. Fortunately, some of this stripping contains a few pounds of oxidized copper per ton. Waste is carefully segregated and that having an appreciable copper content is being stored in suitable disposal areas in such a way that it can be subsequently "leached in place" and much of the copper content recovered.

The great problem, however, in the development of the Thornton Pit was that of ore transportation. It appeared economically unsound to attempt to haul this ore by truck, up out of the deep pit, and around a mountain to the crusher plant. In fact, the entire success of the Thornton Pit proposal depended upon a solution to this problem.

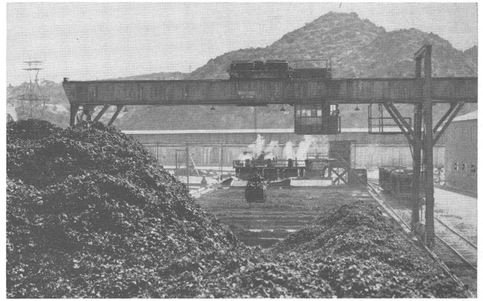
Haul Ore Under Mountain

As has been described in other published papers, this problem was solved in rather a unique way. We were fortunate at Inspiration! The original main haulage level of the Inspiration mine extended out in the direction of the Thornton Pit. By a short extension of the level it was possible to get out under the pit area. Back into the picture came "block caving." A small caving block in area 65 by 100 ft was developed, and caved through to surface from which the waste had previously been stripped. Now we had the desired ore transfer system! Ore from the pit is trucked or moved by bulldozers and carryalls and dumped into the transfer block. From the transfer block the ore is pulled on the grizzley level as in regular caving operations, discharged to ore trains on the 600 haulage level, and trammed to the main shaft, where it is hoisted and dumped into the main crusher plant bins. We simply found it cheaper and better to go under the mountain, rather than to go over it!

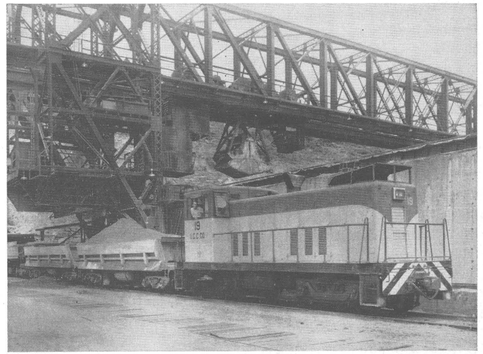
The system has worked perfectly and is capable of moving tonnage at a high rate and a very low cost. To date we have passed some 3,300,000 tons of ore from the pit through this transfer block. Operations have been simple and maintenance has been nominal. Ultimately other transfer blocks will be installed in the Thornton Pit as the developing situation demands. In the west end of the Live Oak pit a similar transfer block has recently been put into service. It, too, is working out well and ore from this area is moved underground to the Live Oak main shaft, from which point it is hoisted and hauled overland on the railroad to the main shaft crusher plant. This arrangement in the Live Oak pit eliminates a truck haul of 5000 ft up a seven percent grade. Much of the remaining ore in the upper benches of the Live Oak pit will still be transported by truck haulage to the pit primary crusher, but ore from the lower benches will be handled through the transfer block.



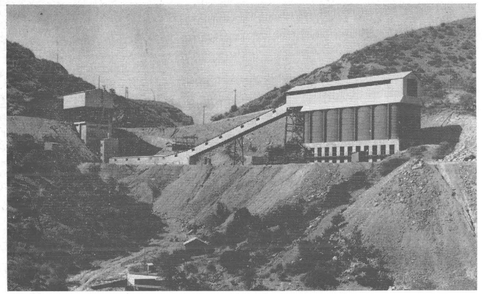
Additional capacity was obtained by building walls of leaching vats 18 in. higher



Cement copper is precipitated when pregnant solution is run over iron in launders



Inspiration has 17 miles of standard gauge railroad. Diesel electric locomotives and open pit operations are controlled by two-way radio



Primary crusher and ore bins at open pit

Leaching in Place

It was long known at Inspiration that much of the broken capping overlying the old caved areas of the mine had an appreciable copper content, a good deal of which was in oxidized form. By early 1950 plans had been completed for the leaching-in-place of these broken capping areas. Water supply was a problem, but this was solved by the decision to use that stored in a lake behind the leaching plant tailings dam. Actually, this lake water is on the acid side, having a pH of 2.5 and contains some ferric iron. It is pumped to a reservoir from which it flows to the leaching areas by gravity. Acid is added at the distributing area so that the leaching solvent contains about 4.5 grams per liter of acid.

The solution is well distributed over the surface and percolates through the broken capping and is recovered on the 850 level of the mine. Concrete-lined ditches transport the pregnant solution to the pump station at the main shaft, where it is pumped to iron launders on surface in which the copper is precipitated out as cement copper. Iron launder "off" solution is returned as creek flow to the lake. This installation has been most satisfactory and to date has recovered approximately 40,000,000 lb of copper.

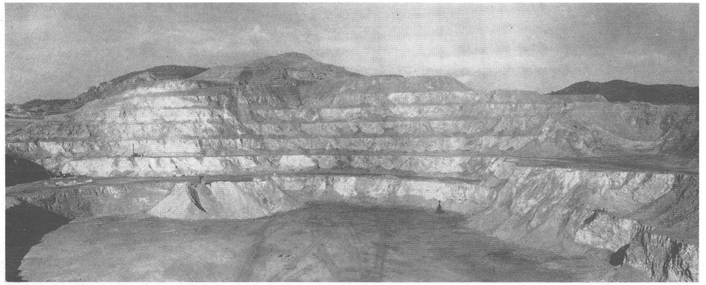
As previously mentioned, waste dumps containing a few pounds of copper to the ton will subsequently be similarly treated.

Metallurgical Problems

Metallurgy at Inspiration is unique in that substantially all copper is recovered by leaching. The ore is a mixture of sulphide (as chalcocite cu_2S) and the usual run of oxide minerals, chrysocolla predominating. The leaching solvents are a mixture of sulphuric acid and ferric sulphate.

Early metallurgists, under the able leadership of the late Dr. L. D. Ricketts, planned the leaching operation at Inspiration for its treatment of a mixed sulphide-oxide ore containing about 1.2 percent copper, of which about 50 percent was in the sulphide form and 50 percent was present as oxide. Today Inspiration faces the new problems as mining progresses to lower horizons. The grade of ore is lower and the ratio is steadily swinging more and more to the sulphide side. This complicates the metallurgical problems and the trend will continue for the remaining life of the mine.

The metallurgical effort of the times is toward increasing and improving the extraction of copper from every ton of ore sent to treatment. In this



Bringing the Thornton Pit into existence meant moving a mountain

SAMPLING DRILL CUTTINGS AT INSPIRATION —

A FIRST STEP IN QUALITY CONTROL

By James H. Lundy Jr.

"Drill holes make it possible to investigate blocks of ground that by any other means would be accessible only at much greater expense, if at all. In some investigations drill holes are intended merely to secure geologic information — the position of a contact, the attitude of a formation, or the sequence in a stratigraphic column. In others they are designed to determine the presence of veins or other guides to ore. In still others, drill holes are used to provide all the information that is required for an estimate of tonnage and grade."

Hugh E. McKinstry [1]

The last sentence of Professor McKinstry's opening paragraph on Drilling contains the scope of the role drilling has played at INSPIRATION. Indeed, no single phase of scientific endeavor has born a greater responsibility in supporting the industrial complex at Inspiration than has the compilation and utilization of the data derived from drilling.

In order to properly evaluate this basic part of the mining program today, it is necessary to quickly review the History of Inspiration.

Located in East-Central Arizona in Gila County, Inspiration is one of the oldest of the typical Porphyry Copper deposits of the Southwest. Systematic prospecting by churn drilling was started in 1910, with much of this early drilling data still being profitably studied today. The mine was a pioneer in block caving, with this system being used exclusively until 1948. At this time the open pit was started, and the two systems were continued until 1954 when the underground mining was phased-out in favor of the open pit exclusively.

[1] McKinstry, Hugh E., Mining Geology - New York - Prentice-Hall, Inc. 1948

From its inception until the fall of 1926, all the ores from Inspiration's several divisions were treated by flotation. In fact, this mill was the first of its kind ever to treat high tonnages of low grade ore by flotation. However, the high percentage of silicates and carbonates encountered in our orebody were not recoverable by this process, thus bringing about the introduction of the leaching system now used. The present Leaching Plant was then put into operation using a ferric sulfate leach. This process uses ferric sulfate-sulphuric acid leaching to recover Chalcocite, Chrysocolla, Malachite and Azurite, and perhaps some other lessor know copper minerals in minute quantities. All soluble copper ore at Inspiration is referred to as "oxide copper".

Inspiration is considered to be a typical Porphry-Copper deposit. These are generally agreed to have a leached Gossan Cap, a low grade "oxide" zone, a highly productive sulphide secondary-enrichment zone, and finally they grade out to a low grade primary-sulfide zone at depth. Since the primary sulfides at the bottom of the ore body could not be taken into solution with the same techniques employed in the existing leaching plant, and since a considerable amount of "oxide" ore remained, Inspiration rebuilt its flotation mill in 1957 and has employed a dual metallurgical process since that time.

Current production is supplied nearly equally from the west Live Oak Pit, and the east Thornton Pit by six electric shovels — 4, 5 and 6 yard sizes equipped with 5, 6 and 8 yard buckets — and a fleet of some 34, 40-ton trucks. Both these divisions are mined in 25 or 50 ft. benches, which are drilled for blasting by truck mounted 9 in. rotary drills. The cuttings from these blast holes furnish the immediate data for quality control of daily run-of-the-mine ore.

Necessity for close control of daily production is based on the fol-

lowing described operation. All of the ore bedded in the leaching tanks is subject to dual-process beneficiation and must be a mixed "oxide-sulfide" product of such grade and proportions to insure that both plants may treat it economically. Approximately 7% of all production is removed from the main flow of ore as slimes. This is done by running the ore through a classifier plant. The slimes are piped to a flotation unit at the mill; following this they are leached in acid bath thickeners. The copper from this last process is recovered in an iron launder as cement copper. The ore which is "practically-all-sulfides" may by-pass the leaching plant and go directly to the flotation plant in the mill. This tonnage is a variable dictated by local plant conditions and has not been a standard part of the flow sheet until recently.

Without going farther into the field of beneficiation it is apparent that the mine must evaluate all blocks of ground into the following classifications: 1. Sulfide ore which may be treated directly by flotation. 2. A mixture of oxide and sulfide ore which may be treated by dual-process. 3. Readily soluble ore known as oxide which is used as a leacher feed sweetner. 4. Sub-marginal ore which may be economically heap-leached on selected leaching dumps, and 5. Barren rock which must be removed to waste dumps in a stripping operation (if it is inside the pit slopes).

Classification into the five mentioned catagories is not dependent on copper assay alone. In some few areas the Molybdenite content must be evaluated in terms of equivalent units of copper and credited to the mill-feed product. Also the solubility of sulfides must be determined as percent "oxide" so that dual-process feed may be blended to meet the needs of both Leaching Plant and Mill.

It is now apparent that multiple process metallurgy does not allow the mine to send "anything above cut-off" to the crusher at any time, but rather demands a dynamic quality and grade control from shovel to final beneficiation plant feed.

This responsibility has been met by a program of careful sampling and evaluation of drill hole cuttings as a first step in mine planning and production quality control. The program as carried on now varies from past practice only in that blast holes were formerly drilled by churn drill, but now, as has been mentioned, are drilled by rotary drills using compressed air as the drilling fluid to cool and lubricate the bits and bring all cuttings to the surface.

Even though some Diamond Drilling was done in the mine in earlier days, and a little surface drilling may be carried on by this method today, the burden of exploration and development has been carried by Churn Drills, and only Churn Drilling is considered in this paper.

EARLY PRACTICE

The first systematic drilling was started in 1910, with holes being put down on the corners of 200 ft. squares. This pattern with fill-in holes in anomalous areas has been continued till the present. The results of this drilling revealed a very homogenous ore body. Logical contouring of ore grade lines between holes plotted on either plans or sections has given remarkably reliable pre-mining evaluation. This fact has been an important factor in establishing drilling proceedure. Many other mines use the equilateral hole spacing system, and then evaluate benches by single value polygons. Inspiration has been very successful in using the method first outlined, perhaps due to the fact that the ore body is very consistent. We do not quarrel with the polygonal

approach; we simply prefer the square pattern.

The original drill holes were logged on individual section sheets which included rock types, formation penetrated, and classification as capping, oxide, mixed and sulfide. Unfortunately for us today, only sulfide samples were assayed, that being the only ore which could be treated in those days. However, great care was exercised in taking samples on 5 ft. runs. Splitting was carefully supervised, equipment kept clean to avoid salting and composite samples taken to be checked by several assayers. After evaluation by drilling was nearly completed in the initial phase, underground samples were taken to check the validity of projected values. Subsequent mining verified the reliability of this evaluation.

RECENT PRACTICE

With the coming of the open pit mining in the late '40's, an additional drilling program was initiated. The original pattern was expanded, with fill in holes located to expose indicated anomalies. Much care was exercised in drilling previously caved ground, as it became necessary to re-mine large areas which were previously undercut or had subsided due to nearby mining.

This program of continued drilling to develop additional low grade tonnage has been carried on somewhat intermittently since the pit began, with the previously described sampling program followed. However, now assays are run for both oxide and sulfide values, with composites run for Mo also. Pulps are retained so that sections might be re-run for additional data, should this be necessary. This sampling and logging are done by the Geologic Department with the assaying being performed in the Mine Assay Office. Check assays are run periodically to assure accuracy. The Geology Department keeps a log of washed cuttings glued to log charts so that even though the five foot samples are thoroughly ground and mixed by the

drilling, a reasonable stratigraphic section of the hole is obtained. We might say this technique produces "Homogenized Stratigraphy".

With the completion of each hole, the Mine Superintendent is supplied with a geologic report. An excerpt from such a report follows:

CHURN DRILL HOLE NO. 281

Location: North side Thornton Pit near General Office
Inspiration coordinates - 5242.11 N. and 8566.71 E.
Collar elevation - 3789.36

Depth: 275 feet. Started 2-21-66. Completed 3-8-66.
Drilling Time - 37 hours at 7.43 ft./hr.

		Total	Oxide
Assays:	0 - 40'	Nil Cu	Nil Cu
	40 - 50'	0.27%	0.08%
	50 - 65'	0.45%	Trace
	65 -180'	0.31%	0.00%
	180 -275'	0.23%	0.00%

Mineralization: Chalcocite, Pyrite, Chalcopyrite, Hematite, and Magnetite with minor Chrysocolla, 40' - 65'.

Rock: Sericitized schist.

Faults: Bulldog fault - 155' - 190'.

Casing: Run - 28' of 15". 170' of 10".
Recovered - 18' of 15" left in hole.

Water: No water reported.

This information is also furnished to the Mine Engineering Department and is plotted on the Bench Assay plans with all other grade data.

BLAST HOLE DRILLING

As has been indicated, early data did not give an "oxide-sulfide" classification and not even an indication of grade in the upper oxide zones. Consequently when mining for dual process is being carried on in an area previously evaluated by per cent Cu only, the data for grading into the five catagories previously indicated can come only from the Blast Hole cuttings. The importance of accurate daily blast assays cannot be over emphasized. The Mine operators must rely on these, plus the results

of "yesterday's run" to dispatch ore and waste to meet the daily plant requirements.

This is how the system works. The Pit Superintendent lays out drilling for two truck-mounted 9" rotary drills on the blast plans of active benches. The Mine Engineers survey and stake these locations in the pit. Drillers collect a one gallon representative sample of cuttings from the bench column only -- although holes generally are put down 3' to 5' below grade to assure good toe fragmentation. These cuttings are collected automatically by the use of a wedge shaped sheet metal pan which sets underneath the drill platform normal to the hole collar. Thus a percentage of the drill cuttings are automatically taken from the cone of cuttings which build up around the collar of the hole. As is customary, the remaining cuttings are left at the collar for a convenient source of stemming, so any hole may be re-sampled prior to blasting, should this prove necessary. Drilling is generally carried out only on day shift, although it has been done on all three shifts. Blasting, however, is reserved for day shift. At the end of "C" shift each day, all the drill hole samples from the previous day are taken to the Mine Assay Office. These samples have priority and are run for both oxide and total copper, immediately. This assaying is by "Slop Cyanide", and although not extremely accurate, it has afforded reasonable mine control in the 0.08% to 2.00% range.

While an attempt is made to schedule all production well in advance, plant requirements can vary from day to day, so the Pit Superintendent must revise any production scheduling on the basis of the most recent blast hole assays. Ideally all drilling would be completed two full days in advance of blasting so that one full day of production might be evaluated before its immediate mining. Practically this may not always be,

so that sometimes the assays of a blast arrive while that particular blast is being dug. Seldom is there a grave error made as to whether material is ore, leaching-waste, or value-less waste, but frequently it is necessary to make adjustments due to high or low oxide or sulfide content. It is this factor which makes evaluation of drill cuttings from the blast holes so important.

One might wonder if it is practical to move shovels or send fleets of trucks from one producing bench to another each time new assays arrive. Certainly it is not. However, Inspiration has enough flexibility due to the amount of live storage in the primary and secondary crusher bins to conveniently sustain changes in grade being made on half shift intervals. This is frequently done.

It should be noted that visual assays, a knowledge of yesterday's run and the facility to obtain "quicky" assays of bench material within 90 minutes, greatly expedites the quality control program. However, in the final analysis, how well the mine is able to supply the consistent mill heads so necessary to efficient metallurgy is dependent almost entirely on how well the drill hole cuttings have been interpreted.

RBM

5/6/66

INSPIRATION CONSOLIDATED COPPER COMPANY
Inspiration, Arizona

HISTORY

Inspiration, like most mines in Arizona, owes its discovery to the old-time prospector and his burro. The beginning of mining operations on the Inspiration property dates back to the turn of the century.

The earliest exploratory working was known as the Woodson Tunnel. This tunnel, driven by hand, went into the hillside for 1000 feet. By 1908, local owners had consolidated claims and groups of claims into a single holding and had induced outside capital to form the Inspiration Mining Company. This name was later changed to that of Inspiration Copper Company. Following this, through a long series of events and negotiations, which saw a merger of the Inspiration Copper Company with the Live Oak Development Company, the Inspiration Consolidated Copper Company came into being in the year 1911. Later the Warrior Copper Company and the New Keystone Copper Company, as well as other properties, were acquired by Inspiration.

Plans were soon formulated to engage in a large-scale copper mining operation. The mine was developed and made ready for operations. A complete surface plant, railroad and concentrator were constructed. This concentrator was the first large-scale plant of its kind to make use of the Flotation Process to recover the copper minerals from the ground-up rock. In all, even at that time, it was necessary to spend close to $20,000,000 before one pound of copper was produced. Construction was completed and Inspiration went into production in 1915.

USES OF COPPER

Copper is one of the oldest known metals. The word "copper" originated many thousand years ago when half-savage tribes living on the Island of Cyprus called it "Cyprian Metal". It has kept the name through all the ages. Our tongues have changed it to "copper". Copper plays an important part in the industry of the United States. In fact, it is the backbone of the electrical industry. Because of this, 60% of the annual output of metallic copper in the United States goes into electrical machinery, power transmission lines and telegraph, telephone, radio and television communication lines and equipment.

Other typical uses of copper include sheet for roofing, tubing for gas, steam, water and oil lines, extruded shapes for industrial equipment, drawn shapes for molding, and all types of brass and bronze. It is also used in the coins of many nations; for jewelry; household articles and architectural designs and shapes.

A recently formed organization, The International Copper Research Association, is doing a large amount of research to find new uses for copper. Ninety-five percent of the copper producers in the Free World, including Inspiration, are members of this Association.

LOCATION

The Inspiration Consolidated Copper Company's operations are entirely in Gila County, Arizona. Inspiration is one of the large copper producers in the State, producing approximately 8.9% of the State's output. In comparison with the nation's copper production, Inspiration produces approximately 4.9% of all copper produced in the United States. The State of Arizona, with its many copper producing districts, accounts for more than 50% of all domestic production. The mine, the town of Inspiration with its U. S. Post Office, and the Company's plant and offices are just north of the town of Miami and are reached by turning off U. S. Highways 60-70, about three-fourths of a mile east of Miami and following the paved road for a distance of about three miles. It is approximately eleven miles around the property.

THE ORE BODY

Inspiration is designated as one of the "Porphyry Coppers". Such an ore body is one in which the copper minerals are widely distributed throughout a large rock mass. At Inspiration the distribution is such that one ton of ore contains less than seventeen pounds of copper. Peculiar to Inspiration is the fact that about half of the copper minerals are present in the oxidized form, the other half being sulphide minerals, mainly chalcocite (Cu_2S). It is the presence of the oxide minerals which gives the green coloration to much of Inspiration's ore.

MINING UNDERGROUND

From the start of operations in 1915, up until 1948, all of Inspiration's production came from underground mining, in which a mining method, known as "block caving", was utilized for the extraction of the ore.

"Block caving" is a method particularly adapted to the mining of large, low-grade ore bodies. The rate of production is high and the cost of breaking and handling ore from the "block" or "stope" can be kept relatively low. Largely, the force of gravity is used, both to break the ore and to deliver it to the ore trains operating on the haulage level under the "block".

Ore trains made up of twelve to twenty-four five-ton cars hauled the ore from the "stope" areas to the shaft, where it was hoisted to the surface in twelve-ton skips.

Inspiration's Main Shafts go to a depth of 850 feet and the Live Oak Main Shaft goes to a depth of 1200 feet, with stations at various levels. From the Live Oak Main Shaft bins, ore was hauled in train loads of sixty-ton railroad cars to the Coarse Crushing Plant at the Main Shaft.

Since 1954, all ore mined has been produced by Open Pit mining.

OPEN PIT MINING

The rapid development of modern methods and equipment for moving earth, coupled with the steady increase in underground mining costs, made it necessary to investigate the possibility of mining much of Inspiration's remaining ore tonnage by Open Pit methods. The decision to go to Open Pit mining followed,

and stripping of overburden was started in 1947. The first Open Pit ore was mined in March, 1948. The adoption of Open Pit methods required the expenditure of several million dollars to meet the cost of construction and equipment of new plant facilities and stripping of waste rock.

Ore and waste are mined by large electric shovels and transported by 40-ton diesel-powered haulage trucks. Considerable equipment, in the way of bulldozers and carryalls, is also required.

Open Pit ore is delivered to a large 42-inch gyratory crusher, where it is crushed down to five-inch size for delivery by train to the main Coarse Crushing Plant.

ORE TREATMENT

Early in Inspiration's operations it was recognized that large reserves of copper were available in the "oxide forms", which could not be recovered by treatment in a concentrator. Years of experimental and test plant work evolved a leaching process which would successfully treat the major portion of Inspiration's ore. A Leaching Plant was erected at a cost of six million dollars. This plant was put into operation in 1926. From 1926 to 1956, inclusive, this process accounted for all but a minor amount of Inspiration's production.

The Inspiration Leaching Plant during the 1926 through 1956 period was the only one of its kind in the world. In this treatment, copper in both the oxide and sulphide form was recovered by Leaching (dissolving). The solvent used was a solution containing both sulphuric acid and ferric (iron) sulphate, with the copper going into solution as copper sulphate. This leaching operation was carried on in large leaching vats, each of which holds 10,000 tons of ore. Nine days of contact time with the solvent solution was necessary to dissolve the copper in the sulphide portion of the ore.

After leaching, the copper dissolved from the ore is recovered from the solution in the electrolytic Tank House. In this process an electric current is passed through the solution, breaking down the copper sulphate and precipitating the copper on thin copper starting sheets suspended in the electrolytic cells. In the course of seven days these starting sheets, made at the plant and weighing fifteen pounds, are built up to a weight of one hundred and forty pounds, then the sheets are withdrawn and shipped as electrolytic copper. Such copper is over 99.9% pure. However, the copper sheets, or cathodes, as they are known, still must be melted and cast into commercial shapes as required by the market. In the electrolytic plant the electric power utilized would supply that needed by a good-sized city.

A vital cog in the Leaching Plant operation is the iron launder system. In these iron launders the last trace of dissolved copper picked up in wash solutions, used to wash ore after leaching, is precipitated out on precipitating material. This iron precipitation material is made up of processed tin cans. The so-called tin can is in reality an iron can coated with a very thin film of tin. Tin cans are cleaned, burned, and shredded and in this process form make an excellent material on which to precipitate dissolved copper from solutions. Most of the tin cans used by Inspiration come from the Houston area in Texas. Total consumption of processed cans amounts to about 1,650 tons per month.

To provide sulphuric acid for leaching, Inspiration operates two sulphuric acid plants which can produce up to 200 tons per day. Sulphur, in molten form, shipped in from east Texas mines, is used in this process.

PRESENT PROCESS

By 1954, increasing copper values in sulphide minerals, not soluble in the ferric iron solution, were noted. The grade of the remaining ore was dropping and the capacity to produce copper was limited by the nine-day leaching time. These factors brought about a study which resulted in a radical change in the metallurgical treatment of the ore. By 1957 the old concentrator had been completely rehabilitated and new, modern machinery installed. In the leaching process only that copper soluble in sulphuric acid, plus the sulphide dissolved in a low ferric iron solution, continued to be sent to the Tank House for electrolytic precipitation, as previously described. The contact time for leaching was cut to four and one-half days. Sulphide copper remaining in the ore is then sent to the concentrator for recovery by the flotation process.

The concentrate so recovered is sent to the Smelter. The concentrate is smelted, fire refined, and cast in the form of copper anodes. These anodes are returned to the Tank House.

Due to the change in process, with less dissolved copper being sent to the Tank House, excess capacity was available. This excess was converted to a Refining Section. Here, copper anodes returned from the Smelter are further refined to electrolytic cathodes. In this process the copper is dissolved from the anode and plated on a starting sheet as electrolytic copper. These cathodes are heavier than the Commercial Section cathodes and weigh as much as two hundred and fifty pounds.

To enable Inspiration to refine all of its own copper, including production from its Christmas Mine, a new eighty-tank refinery section was constructed and is now in operation. The electrolytic refinery at Inspiration is the only one in Arizona.

MOLYBDENUM RECOVERY

With copper concentrate being made in volume under the revised process, it was found that such concentrate contained a small amount of Molybdenum Sulphide (Moly). A section was added to the concentrator, to recover the Moly. This is a difficult and involved process.

SMELTING DEPARTMENT

The Smelter was built in 1915 to handle concentrates from the District's mines and to treat custom ores and concentrates. It was owned by the International Smelting and Refining Company. This plant was purchased by Inspiration in April of 1960. It continues to handle District concentrates and custom business.

At the Smelter, properly mixed concentrates and flux are melted in a reverberatory furnace at a temperature of approximately 2700 degrees. The copper collects in the bottom in the form of "Matte", which is an artificial copper-iron sulphide. Some of the sulphur is burned off. Impurities and waste

material float on top and are skimmed off and discarded as slag.

The matte is tapped off at a point below the slag level and is poured in molten form into a converter. Here, air is blown through the molten material and flux is again added. The air oxidizes (burns) the iron in the charge and the sulphur is burned off. Slag is formed and is poured off and returned molten to the reverberatory furnace. The reaction in the converter provides its own heat. Final product from the converter is known as "Blister Copper".

Blister copper may be cast into cakes for shipment to Eastern refineries or poured molten into the anode furnace.

In the anode furnace it is further refined to fire refined copper by blowing with air and "poling" with oak poles. The copper is then poured into anode molds and the anodes are returned to the Inspiration Tank House for further refining.

It is interesting to note the many steps in the processing of copper and the work necessary to produce a final product.

Inspiration Ore	0.80% Copper
Concentrates	30% to 45% Copper
Matte	40% to 50% Copper
Blister Copper	99.4% Copper
Fire Refined Copper	99.6% Copper
Electrolytic Copper	99.95%-99.97% Copper

POWER PLANT

Requirements for electric power at Inspiration are quite large. To meet the original need, a 25,500-KW Power Plant was constructed. In this plant, natural gas piped from New Mexico is burned under boilers to provide the steam to operate the turbo generators. Waste heat steam from the Smelter boilers is also utilized.

The Inspiration power system is tied into that of the Salt River Project, and most of the power needed is supplied by them. On many occasions the entire Inspiration load has been carried by the Salt River Project system, which derives much of its power from hydro-electric generating stations located below the dams along the Salt River.

SHOPS AND SERVICE

The Company has its own shops, warehouse, and service departments. The shops are fully equipped and are capable of handling almost everything in the way of repairs and maintenance which may be required.

RAILROAD

The Inspiration Company operates seventeen miles of standard gauge railroad. This railroad delivers ore to the treatment plants and concentrates to the Smelter, as well as handling inbound freight and outbound shipments of copper. The railroad connects with the Southern Pacific at the foot of the hill.

TOWNSITE AND STORE

The Warrior Cooperative Mercantile Company operates a general store at Inspiration. This store is operated to serve the needs of Inspiration employees.

Operations are on a non-profit basis and profits earned are returned twice yearly to employee customers in proportion to their purchases throughout the period.

HOSPITAL AND CLINIC

The Miami-Inspiration Hospital is maintained jointly with other companies in the District. Also jointly maintained is the Miami-Inspiration Clinic located on the Globe-Miami Highway. These facilities not only serve industrial cases, but the employee and his family is provided medical care at exceptionally low rates. In all, the families of some 2,600 mining employees in the District are served by these facilities.

STATISTICS

Tons of Ore Mined and Treated - 1915 to 1965		198,050,241
Pounds of Copper Produced - 1965		124,153,059
Pounds of Copper Produced - 1915 to 1965		3,725,271,647
Tons of Ore and Waste Moved from Open Pit in normal 24-hour work day		25-35,000
Wages and Salaries paid in 1965		$13,723,277
Supplies and Equipment Purchased in 1965 . .		15,779,002
State, County, and District Property Taxes. . 1965		2,707,885
Approximate Number of Employees:		
Inspiration	1,465	
Christmas	470	1,935
Number of Stockholders		8,679

In 1965, Inspiration produced approximately 4.9% of the copper produced in the United States.

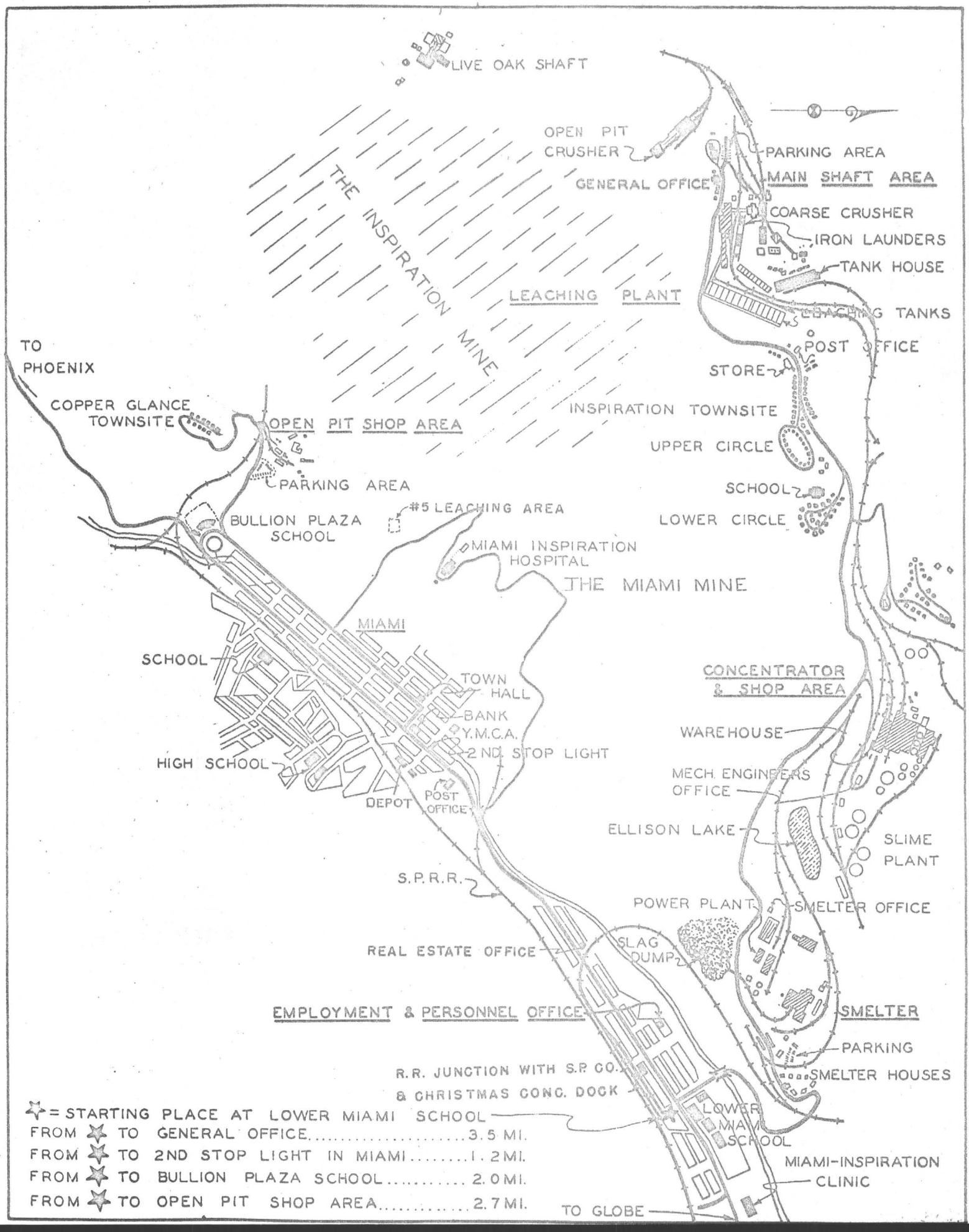
CHRISTMAS MINE

Inspiration is also producing ore from the old Christmas Mine. This is an underground operation. The mine is located some forty miles from Inspiration. It is one mile west of State Highway 77 between Globe and Winkelman, Arizona.

Ore from the Christmas Mine is processed in a new crushing plant and concentrator at Christmas. The concentrates are being sent to Inspiration's Smelter.

ISSUED JANUARY, 1954
ICCCo.
Revised May, 1954
Revised July, 1960
Revised May, 1962
Revised July, 1962
Revised March, 1963
Revised April, 1965

INSPIRATION CONSOLIDATED COPPER CO. INSPIRATION, ARIZONA



5/6/66

MINING GEOLOGY DIVISION - ARIZONA SECTION A.I.M.E.

SPRING MEETING: APRIL 30, 1966

Inspiration, Arizona

"STRUCTURE AND ORE CONTROL AT INSPIRATION"

by Hugh Olmstead, Inspiration Consolidated Copper Company

STRUCTURE

The Globe-Miami District lies near a postulated intersection of four continental lineaments. Locally these are: 1. The Texas lineament (N 74° W); 2. The Arizona Rockies lineament (N 33° W); 3. The Utah-Arizona lineament (N-S); and 4, the Raton-Globe lineament (N 57° E).[9] Regionally, they are traceable as Mesozoic and Tertiary intrusive and extrusive rocks found in mountain ranges and fault structures, which respectively include: 1. The Buckskin, Wickenburg, Pinal, and Pinaleno Mountains; 2. The Swissheim, Galiuro, Dripping Springs, and McDowell Mountains; 3. The Tumacacorie, Picacho Mountains, and the Pleasant Valley fault; 4. The Datil volcanic field and Pinacate volcanic field in Mexico. Inspiration is situated in the lower elevations on the northeast side of the Pinal Mountains. At the mine, the Texas and Raton-Globe lineaments are reflected strongly in the Pinal schist and various faults. The Utah-Arizona and Raton-Globe lineaments are partially delineated by Tertiary Schultze granite and Granite porphyry intrusives.

Physiographically, the district lies in the middle part of the mountain region province.[16] It is in an area of flexing on the southern end of the Mazatzal Mountains. Although a broad deep valley separates the Mazatzals

on the north from the Pinal Mountains on the south they probably were one undisrupted rock mass during the Paleozoic Era. Before and during the Rocky Mountain orogenesis the level ocean beds were uplifted, warped, and distorted to their present position. This and accompanying Tertiary intrusion undoubtedly created a high, sharp-edged mountain range which has been eroded to the present rugged and irregular topography. In the Quaternary, masses of erosion products filled all of the lower elevations. Resulting equilibrium adjustment depressed conglomerate areas which are now the major valley and drainage paths.

Structures in the mine area are somewhat related to the general structure of the Pre-Cambrian schist which trends northeasterly and dips to the southeast.[17] Local granitic intrusives have distorted and obliterated the schist structure, but the schistosity prevails as the major lineation followed by mineralizing solutions.[17] A lack of schistosity in the mineralized zones may be due to more massive beds of schist in which fractures remained open for mineralization and enrichment.

The structural control of the intrusion of the Granite porphyry phase of the Schultze granite, with which the ore bodies are associated, is shown by some of the existing faults. The Miami fault, one of these structures, strikes north 25 degrees east and dips about fifty degrees to the east and drops the Gila conglomerate east of the ore bodies between two and three thousand feet. This fault or the ancient break it followed may have had some pre-porphyry movement which is indicated by the porphyry-schist contact extending along the general fault strike beyond where the present fault zone swings to follow the trend of the Pinto fault for some distance. East of the mining areas the ore bodies appear to be cut off by the Miami fault, although some sections show that the ore did not reach the fault zone or that there is a leached zone near the fault. Some ore in diabase has been

found at considerable depth east of the Miami fault and secondarily enriched sulphides were found below low-grade primary material near a branch of the Miami fault in one drill hole.

The Sulphide fault is also a pre-porphyry structure. It has an east-west trend and dips steeply and irregularly to the north. The contact between the schist and porphyry is in the fault plane for some distance, but the ore bodies do not appear to be displaced by it. The Sulphide fault may have been a conduit for ore-bearing solutions, but cannot be traced in the porphyry for any distance and passes under conglomerate to the west. This fault is paralleled by the Southwestern fault about one thousand feet farther south, which may be a later structure, as it can be traced in the porphyry and appears to displace the ore zone.

The Pinto fault strikes northwest and dips at about forty-five degrees to the northeast. It cuts through the mining areas in both the Miami Copper and Inspiration workings. At the junction of the Pinto with the Miami fault, the Miami swings to follow the trend of the Pinto for more than one thousand feet. The Pinto fault is irregular with a broad crush zone and some fault clay. There are some drag fragments of mineralized material in the zone and also indications of post fault mineralization. Although the ore bodies appear to be down dropped east of the fault, there also appears to be a zone of higher grade primary material in the footwall of the fault in the Inspiration workings, which was probably introduced in a pre-mineral structure. Also, the Miami Copper ore bodies are best developed in the acute angle near the junction of the Miami and Pinto faults, which would indicate pre-mineral structures, but there is no clear indication that the faults were channels for mineralizing solutions.

Other faults in the Miami Copper workings appear to have minor displacement, but where a crosscut from the No. 5 Shaft, which started in

conglomerate, encountered the conglomerate-schist contact east of the Miami fault, there was considerable clay developed at the contact, which might indicate some movement.

The Joe Bush fault is more or less parallel to the Pinto fault, but dips steeply to the southwest or is almost vertical. It passes south of the Miami Copper workings, but is well exposed by the workings on the Inspiration 600 Level. On this level there is an apparant horizontal displacement of the schist-porphyry contact of more than one thousand feet by the fault. This large movement is not shown on the surface and may be, in part, due to a pre-porphyry structure being a guide for the emplacement of the porphyry.

The Bulldog, Keystone, Number Five, and Barney faults are north to northeast trending faults dipping flatly to steeply to the east. The movement on these faults is normal with the east blocks down dropped. The amount of movement on some faults is shown by the displacement of the post-mineral dacite beds.

The Bulldog fault is roughly parallel to the Miami fault and dips at between thirty and forty degrees to the east. This may also be a pre-porphyry structure with some post-porphyry movement, since a dike of porphyry intrudes the fault zone and the fault cannot be traced far in the porphyry or granite. The fault branches to the south and there are a number of steeper hanging wall splits which have about one hundred feet displacement. These are locally called the Colorado faults. Some post secondary enrichment movement on these faults is shown by the displacement of the ore bodies down to the east.

The Keystone fault trends northeasterly and dips at about forty degrees to the southeast, as shown on the Geologic Plan. It probably swings to join the Bulldog fault north of where it is exposed on the surface. There is a displacement of about three hundred feet on this fault, but its location has

never been well established in underground workings.

West of the Keystone there are the Number Five fault, the Barney fault, with about six hundred feet displacement, and other smaller structures, besides the Sulphide and Southwestern faults which have been described. These have a general northeasterly trend and dip to the southeast. Some rotational movement on these structures is shown by tilting of the bedding of the conglomerate and by the slope of the dacite beds and the pre-conglomerate land surface. The Sulphide fault, which guided the emplacement of the porphyry, would have had a southerly dip before this tilting and continued movement on that fault may have caused some anomalous structural conditions, such as schist overlying dacite found in drill holes in the area south of the projection of the Sulphide fault.

From the underground mapping and drill hole information, the ore bodies appear to be cut off on the west by the Porphyry fault, which trends about north fifteen degrees east and dips twenty degrees to the east, although there is considerable decrease in size in the ore zone before it reaches the fault. This fault is either cut off by the Barney fault or there is a considerable steepening of dip towards the surface and the Barney fault is the same structure. Exploration west of these structures has encountered mineralized schist, but no ore bodies.[17]

OREBODY GEOLOGY

The Inspiration orebody is 8,300 feet in length and has a maximum width of about 2,500 feet. The ore attained a total thickness of as much as 700 feet but will average about 300 feet. It has an arcuate, elongated shape which thins in the middle and on its southwest end. (REFER TO MAP). Elongation follows a general trend of N 72° E with a gentle southwest pitch. The original high-grade chalcocite ore was regarded as an irregular sulfide

replacement blanket deposit. Some of the low-grade ore now being mined is regarded as a disseminated or "porphyry-type" deposit. In the early 1900's high-grade chrysocolla was mined from veins in the porphyry; therefore, at least three types of deposits occur, in which each is an integral part of the other.

Primary hydrothermal mineralization is believed to be intimately associated with the porphyritic intrusions of Schultze granite, where faults and relevant crushing created a favorable environment for solution emplacement. There is some evidence suggesting simultaneous action of faulting, crushing, stretching and magma flowage. The primary ore solutions appear to have succeeded the introduction of the Granite porphyry differentiate. These solutions carried pyrite, chalcopyrite, and possibly bornite and chalcocite. They likely were injected into small fractures opened by stretching[1 and 6] which was a resultant of the active diastrophism and volatile pressure. Later differentiates apparently of moderate magnitude carried pyrite, quartz, and molybdenite. Microscopic study indicates a period of orthoclasization and silicification which preceded the flow of ore mineral solutions. It is believed that super-saturated solutions carrying a high potassium content derived from a parent magma and from circulating juvenile solutions altered permeable Schultze granite, and strongly impregnated the granitic wall rock. Data available from a study of the compositional variations of the alkali feldspars states that "the variation in many cases is not independent of the protore distribution."[7] Kuellmer has presented two hypotheses concerning the origin and significance of compositional variations of the alkali feldspars. His second hypothesis postulates the possibility of alkali feldspar compositional difference resulting from a "secondary compositional adjustment of primary crystals during a hydrothermal stage in addition to crystallization during the hydrothermal stage."[7] Microscopic study[10] supports this

hypothesis and strongly suggests secondary orthoclase.

In addition to the alteration products of silicification and orthoclasization, there exists alteration products from kaolinization, sericitization, biotization, hydration, argillization, surface leaching, and enrichment. These are profound within and near the orebodies while epidotization is more prevalent around the north and east fringes.

Over the major portions of the ores, surface oxidation and leaching has imparted a moderate to weak brown coloration. Early prospectors were attracted to this area by conspicuous stains of copper silicate and carbonates, but now strong coloration from ore minerals exists at only four small areas other than the pits. Although not plentiful, residual limonitic boxworks occur near the surface within small quartz veinlets in schist, Granite porphyry and Schultze granite.

In the ore areas dark minerals comprise a small percentage of the whole, partly due to alteration and partly due to an original small supply. Biotite is the main ferromafic and it seldom exceeds five percent of the rock composition. Although restricted, some biotite flakes reveal marginal chlorite alteration. The disintegration of biotite varies but has generally produced kaolin, chlorite, or hydrous micas.

Often in the zones of strong alteration, it is difficult to megascopically distinguish the outlines of the feldspars. Sericite prevails as a hydrothermal alteration product and kaolin is intricately an associate of the alteration cycle. In the crushed zones, near the more prominent faults, there is a weak indication of brecciation[14] but it appears that supergene alteration has etched the fragments (quartz) beyond recognition as breccia. Sericite and kaolin penetrate at least as deep as the enriched zone.

At Inspiration, secondary supergene mineralization is as important as primary mineralization. One without the other would have not produced a mine.

Hypogene metallization penetrated schist, granite porphyry, and Schultze granite. Economic minerals included pyrite, chalcopyrite, bornite, and chalcocite, with later impregnations of molybdenite and pyrite. Pyrite is included as an economic mineral because of its necessity in the supergene cycle. During the metallization a distinct zonal pattern was established, which somewhat controlled the mineral distribution and the mineralization intensity.[15] The ore minerals filled small fractures in the host rock, creating a low-grade protore. Subsequent alteration and erosion by various chemical reactions decomposed the primary minerals and produced copper chlorides, copper silicates, copper carbonates, copper phosphates, and copper rich sulphosalts, which, because of their relative solubility, resulted in chalcocite replacing pyrite, chalcopyrite, and bornite. During this process a protore from hypogene mineralization was oxidized and leached and transformed to an ore of supergene enrichment. Between the leached zone above and the supergene sulfide enriched zone below, there is an intermediate zone of oxidation and hydration containing malachite, azurite, chrysocolla, and ferric hydroxide minerals. Within the zone of oxidation there occurs primary quartz veinlets with chalcopyrite and pyrite which have not completed the enrichment cycle, nor undergone significant oxidation. Since the supergene chalcocite replaced mainly pyrite, ore bodies are localized in zones of primary mineralization regardless of the amount of primary copper present.[17] Colloidal solution activity has not been ascertained, but there is evidence supporting hypogene origin for part of the chalcocite.

The higher grade primary mineralization occurs as bands along the Miami and Pinto faults, and between the Joe Bush and Bulldog faults. (SEE MAP). Very little primary copper mineralization was encountered in the Live Oak mine. However, some primary chalcopyrite was associated with the crushed zone near the Sulphide fault.[17] The deepest zone of ore (supergene) on this property

occurred on the 1200 Level in the southwest corner of the Live Oak Mine. Below the enriched zones there exists a zone of protore of primary mineralization assaying from 0.15% to 0.40% copper. The thickness of this material is not known.

The leached zone of capping varies in thickness from Nil to 1,000 feet or more and will average about 400 feet. Near the upper extremities of the leached zone the copper content is often not more than a few parts per million. The oxidized zone, fairly consistent in thickness, averages about 200 feet, although this varies somewhat in fault zones or other permeable zones. The oxide ore zones resulted from the oxidation of the supergene enriched zones. They are much higher grade than mineralization in protore. At least half of the present ore is produced from this zone. The supergene enriched zone varies but will average about 200 feet and attains as much as 450 feet in thickness.

The age of the ore extends from early Tertiary through the present. It is generally believed that primary metallization first occurred in Late Cretaceous or Early Tertiary. However, secondary enrichment processes were active throughout the Tertiary Period and have been moderately active until today.

The minerals present in the Inspiration orebody should be divided into hypogene minerals and supergene minerals. As the original hypogene mineralization in the porphyry and schist was of low tenor, the orebody, as we know it today, is dependent upon the supergene enrichment. The ore now averages less than one percent total copper, about 0.02 percent molybdenite, with traces of gold and silver.

The hypogene minerals consist of pyrite, chalcopyrite, molybdenite, minor bornite, minor chalcocite,[4] traces of gold and silver, and a few very minor occurrences of galena and sphalerite. In a few places

chalcopyrite-quartz veins cut earlier pyrite-quartz veins, indicating at least part of the chalcopyrite mineralization is later. The molybdenum-quartz veins cut all mineralization and are considered the last phase of the hypogene metallogenetic phase.

The supergene copper sulphide minerals consist of chalcocite, bornite, covellite, and chalcopyrite.[8 and 18] The chalcocite blanket varies from the complete replacement of the original pyrite and chalcopyrite by chalcocite to films of chalcocite on pyrite crystal surfaces. Although argumentative, the consistent occurrences of a chalcopyrite enriched zone beneath the chalcocite blanket is attributed to supergene solutions. Thus, chalcopyrite in the enriched zone[18] occurs as extremely thin films on pyrite crystal surfaces.

Chrysocolla, malachite, azurite with minor copper pitch, brochantite, atacamite, lindgrenite, libethenite, and extremely minor metatorbernite,[20] occur as products of a supergene enriched area in the oxidized zone. An association has been established between the occurrence of granite porphyry, radioactive minerals and ore minerals.[20]

Some notations must be made of the very minor occurrence of native copper and cuprite in a few fault zones. Also, chalchanthite, a product of present mine water seepage, is found in the underground workings.

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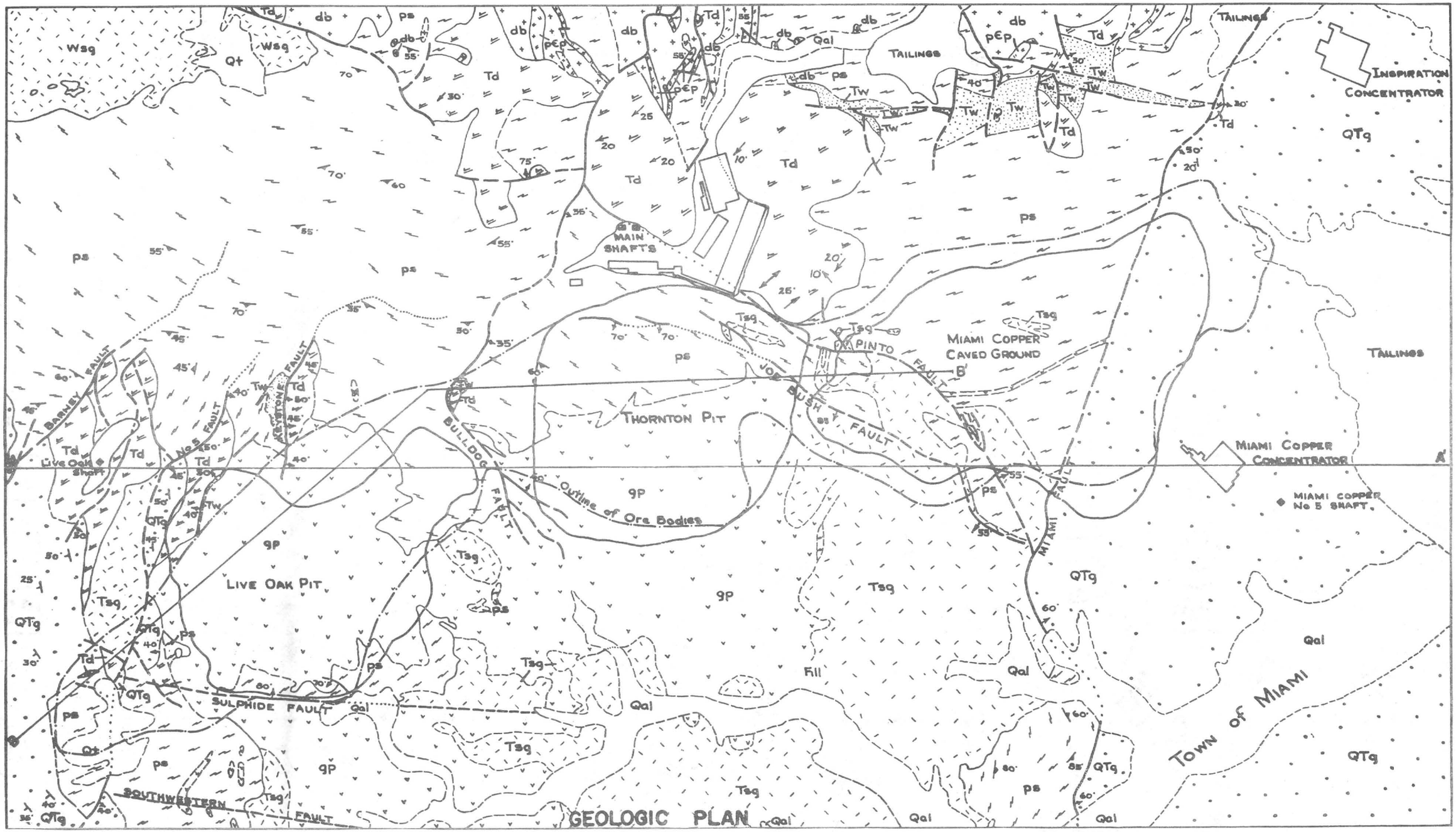
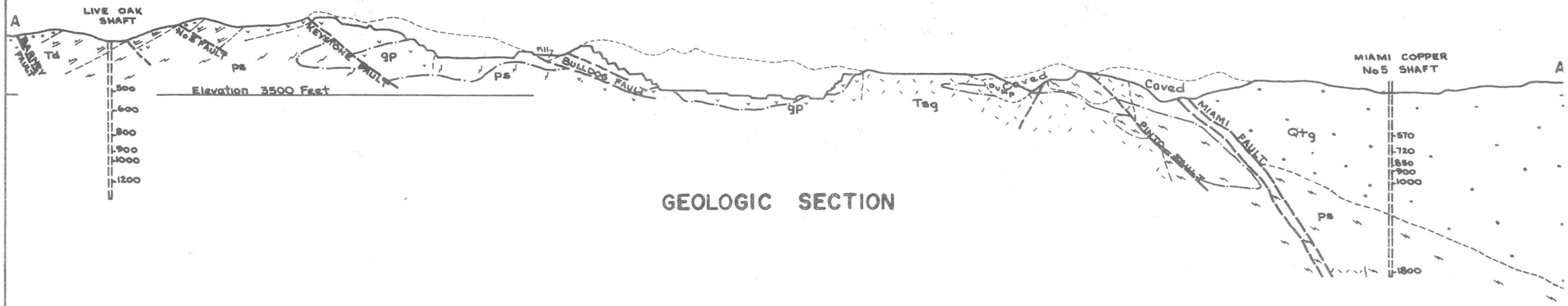
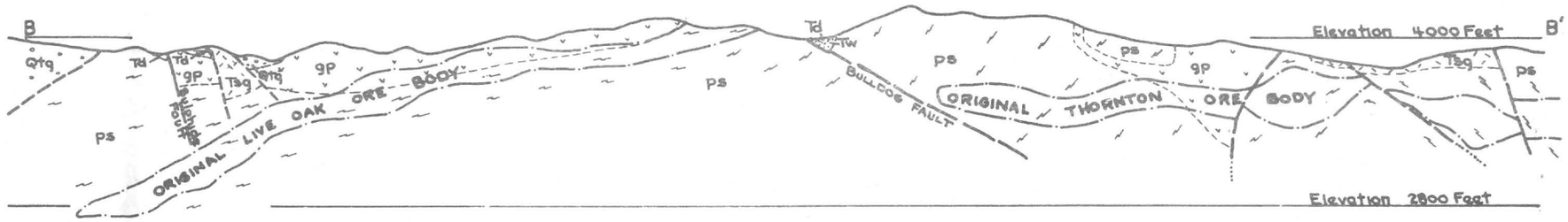


GEOLOGIC PLAN



GEOLOGIC SECTION



GEOLOGIC SECTION ORIGINAL ORE BODIES

EXPLANATION

Quaternary	Qal Qt	Talus and Alluvium	Strike and dip of beds
	QTg	Gila Conglomerate	Strike and dip of foliation
Tertiary	Td	Dacite	Bearing and plunge of lineation
	Tw	Whitetail Conglomerate	Horizontal lineation
	gp	Granite Porphyry	Shaft
	Tsg	Schultze Granite	Outline of ore bodies
	Wsg	Willow Spring Granite	Outline of pits and caved ground
P.	db	Diabase	Faults
Precambrian	p€p	Pioneer Formation	
	Ps	Pinal Schist	

GEOLOGIC PLAN and SECTIONS
INSPIRATION MINE
GILA COUNTY, ARIZONA

1000' 0' 1000' 2000' 3000' 4000' FEET

SCALE

N

SAN FRANCISCO
APR 27 '85
CALIF.

POSTAGE
1.41
METER

FROM

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HOMESTAKE MINING COMPANY
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SAN FRANCISCO, CA 94108

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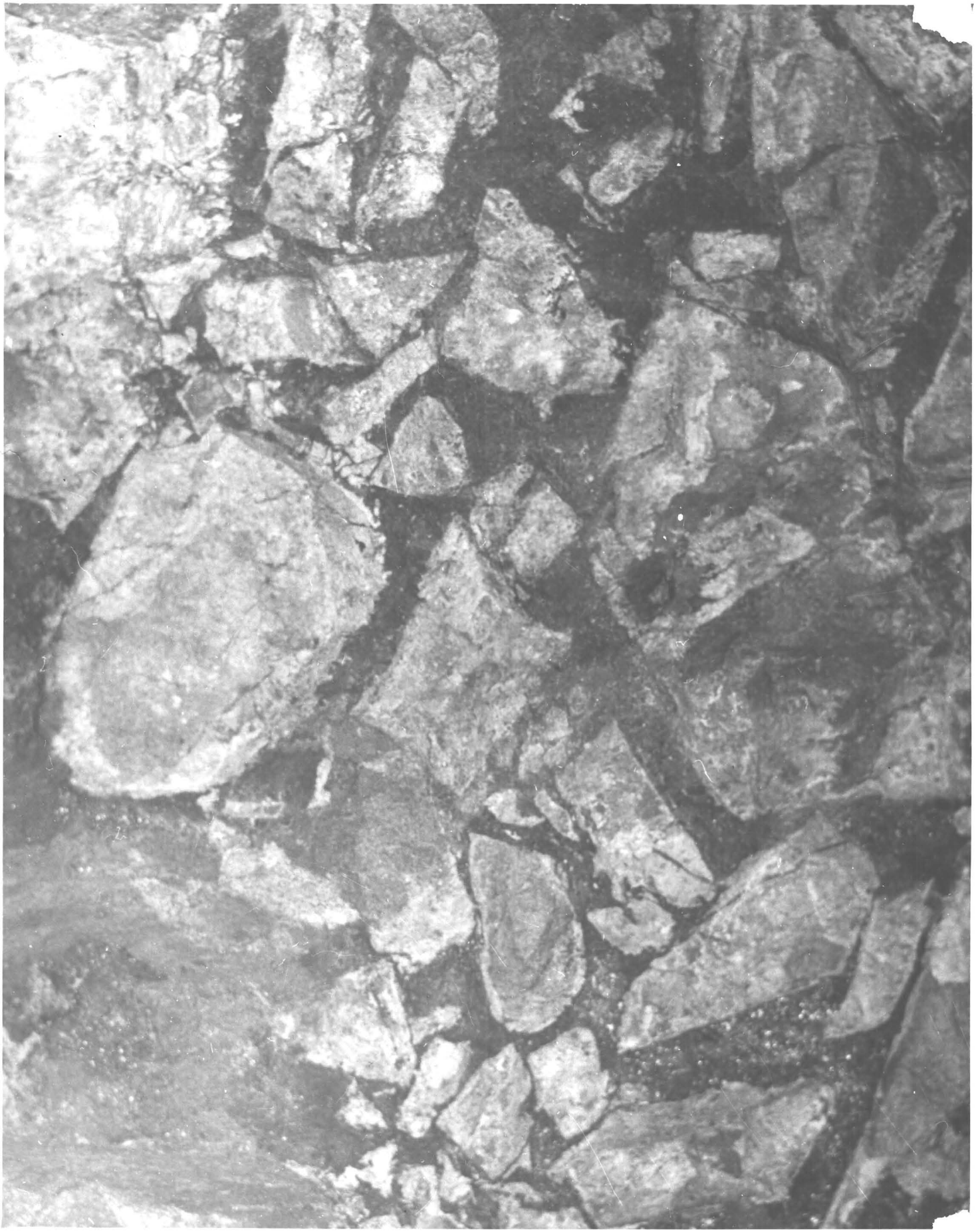
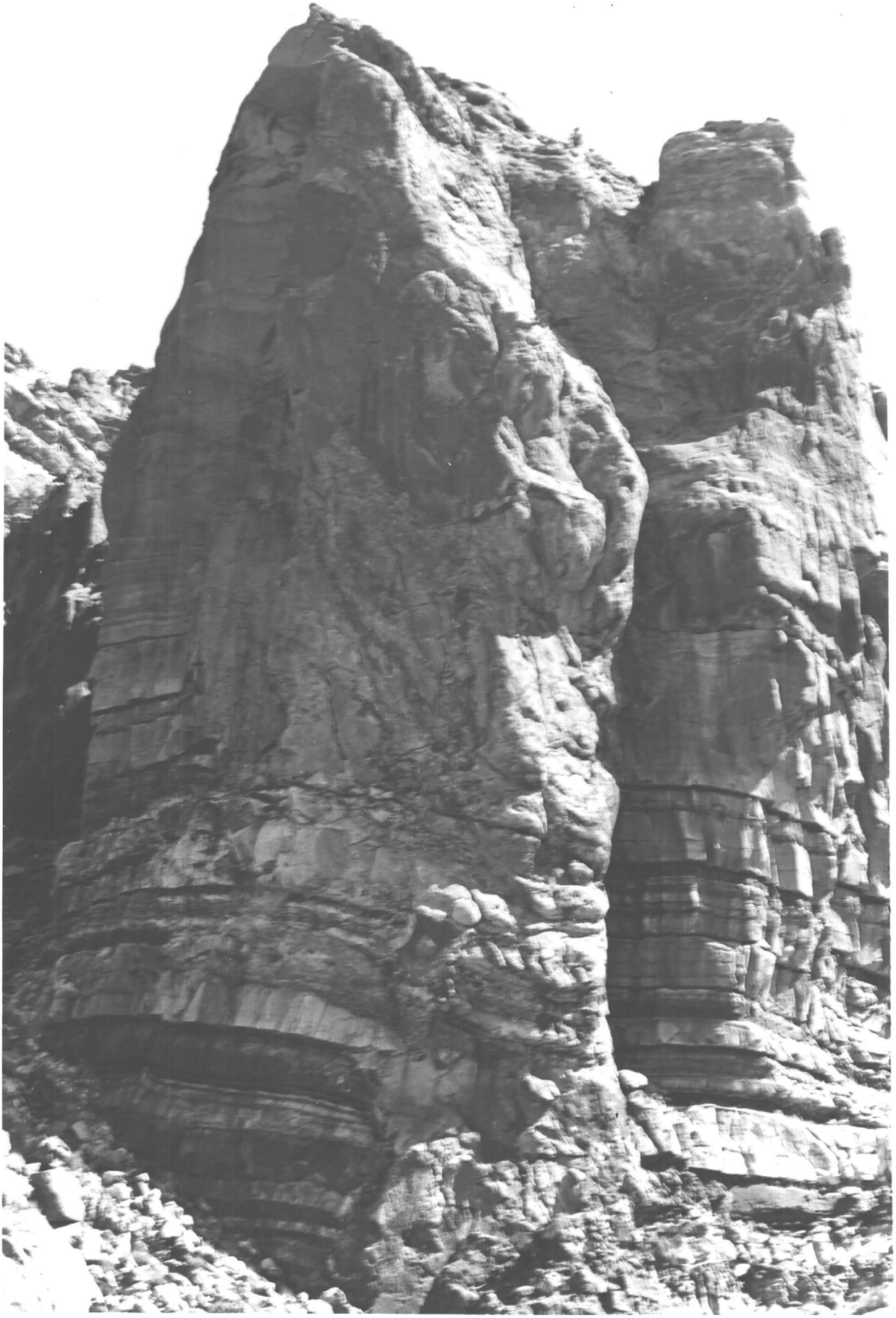
Breccia Pictures
Grants
Canaea
Inspiration Pit
Yosemite



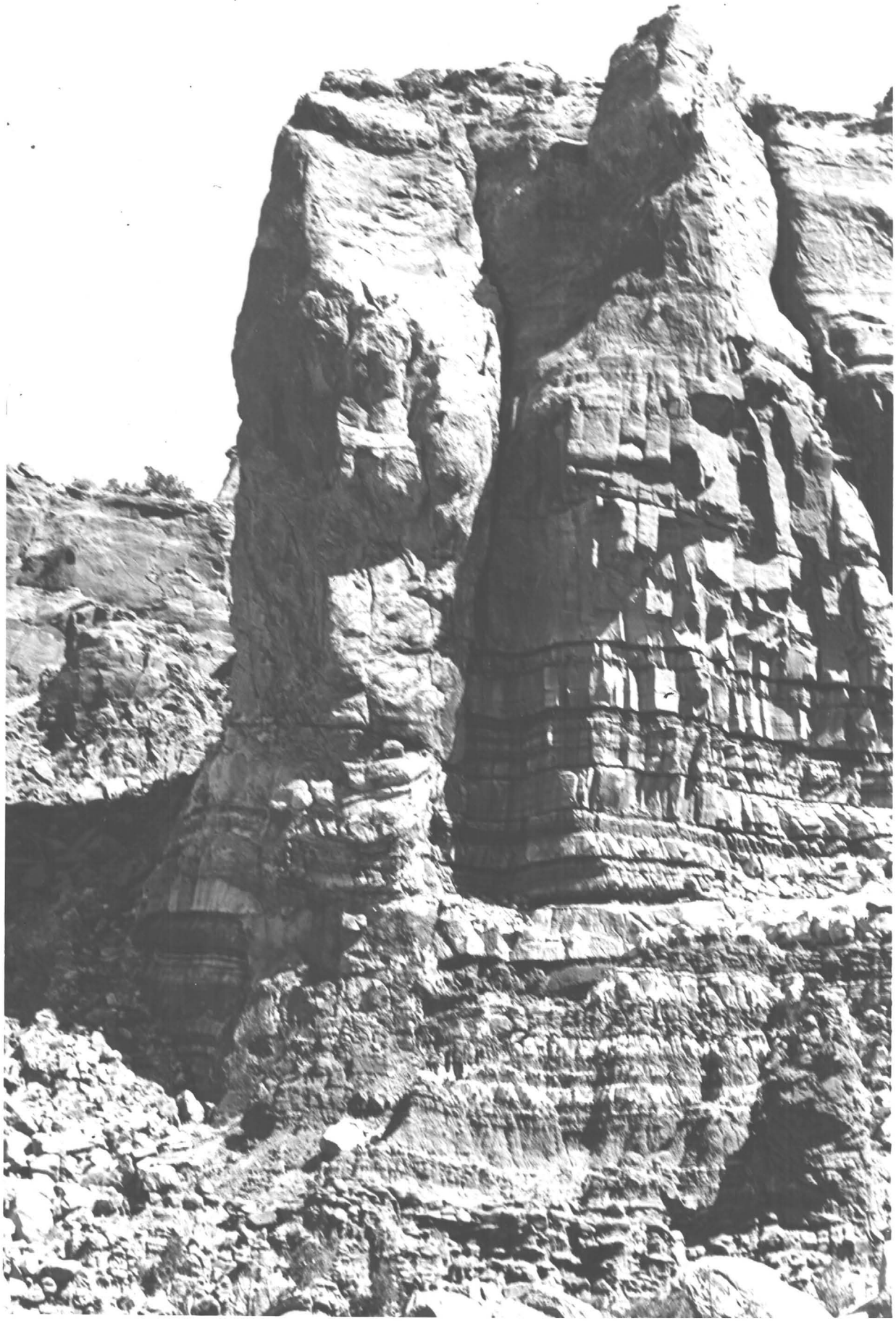
FACE OF 7-121-2-12 INT. XCSE

AT 66 FT. SE OF 7-121-2 RS.

Pyrite and chalcocite surrounding brecciated quartz porphyry fragments.



GRANTS, N.M. 1951



GRANTS, N.M. 1951

YOSEMITE

CANANEA DULUTH BRECCIA

INSPIRATION OPEN PIT BRECCIAS
FROM CAVING 1956

W/ ~~MAPE~~
NEG.

CAHANEA- DULUTH PIPE

LOOKING SE

APPROXIMATELY 200 FT.

ACROSS PIPE

SW WALL

NE WALL