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J.H.C.

MIAMI COPPER COMPANY

DIVISION TENNESSEE CORPORATION

IN PLACE LEACHING MIAMI MINE

MIAMI, ARIZONA

Presented before

MILLING DIVISION, ARIZONA SECTION

A. I. M. E.

April 6, 1962

IN PLACE LEACHING MIAMI MINE

by

J. B. FLETCHER

April 6, 1962

In order to give a background to leaching it is necessary to go back to the mining at Miami. The Miami mine started mining in 1910 and finished in July, 1959. (Slide #1) The orebody was divided as follows:

High Grade	24,400,000	tons	- Top slicing sub-level caving, etc.
Low Grade	102,000,000	tons	- Block Caving
Mixed Ore	9,800,000	"	" "
Low Grade #2	23,000,000	"	" "
TOTAL	159,000,000	tons	

GEOLOGY:

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

MINERALOGY:

The chief mineral is chalcocite, with chalcopyrite, bornite, covelight, malachite, azurite, chrysocola, cuprite, native copper, and molybdenite as minor minerals. The ore minerals occur in seams, veinlets and disseminated particles.

If the ore body was completely mined out in 1959 what was left to leach? (Slide #2) In block caving there is always some dilution and drawing of ore stops when the grade drops past an economical point. This results in a small amount of copper left in the stopes below the capping. There is some copper in the capping, probably below 0.1% or less than 2 lbs. per ton. Over the low grade #2 orebody and portions of the high grade there was an oxide capping which was not mined. The crushed pillars are another source of copper. The leaching at Miami is an attempt to recover this last remaining

copper of a worked-out mine.

The enrichment at the Miami mine had gone more nearly to completion than most ore bodies in the southwest. There is practically no pyrite in the capping and very little in the ore.

HISTORY:

Leaching started on a small scale in Dec. 1941. After 20 years of leaching all the answers should be known. This is far from true. Some of the early problems that were met are as follows:

(1) The first leaching was tried through the conglomerate over the high grade mining. No solution came through the fault. There has been mining below the high grade since the first leaching, which has broken the conglomerate for the second time, but this problem has not been solved to date.

(2) The next leaching was tried over the mixed ore body with water. The solution came through with very little copper. Water was tried over the sulphide stopes with no recovery of copper. Then acid was added to the solution over the mixed ore. The first leaching was started with a 3% H_2SO_4 solution which was immediately dropped to a .6% solution. At the present time we try to control the H_2SO_4 feed by the amount of free acid in the pregnant solution. If the acid in the pregnant solution drops much below 0.5 lbs. H_2SO_4 per ton of solution the pumps, pipe lines, sumps, etc., are clogged with iron salts.

(3) In order to leach sulphides it was realized that Fe''' would be necessary, and several attempts were made to manufacture a high Fe''' solution. It was discovered that by adding acid to the off-solution from the precipitating plant, Fe'' was converted to Fe''' as the solution went through the caved ground.

The underground mine was closed down June 24, 1959, at which time full scale leaching was started.

OPERATIONS:

(1) Miami reports assay values in terms of lbs. per ton of solution. Practically, it is a simple term, because all pumps are rated in G.P.M. and we talk in G.P.M. and shifts. 1 gallon = 8 1/3 lbs., therefore 1 G.P.M. for 24 hrs. $= \frac{8\ 1/3 \times 1440}{2000} = 6$ tons.

(2) Requirements:

The Miami plant was designed to treat 2,000 G.P.M. and to produce 1,500,000 lbs. Cu per month or 50,000 lbs. Cu per day. Therefore

2,000 G.P.M. x 6 = 12,000 tons of solution per day

$\frac{50,000}{12,000} = 4.17$ lbs. Cu per ton of solution

+ 0.04	Tail
4.21	
+ 0.07	Smelter loss
4.28	Required Assay

(3) Leaching can best be thought of in 4 dimensions. The first two dimensions are the surface to be covered. (Slides 3, 4, and 5) We have approximately 5,000,000 sq. ft. to cover. In 20 years we have covered 1/2 the area. 6 years should cover the balance, although that is not the end point of leaching.

Tailing water from the precipitating plant constitutes the most important source of leach solution. Water losses occur principally from evaporation and at Miami they approximate 10%. Fresh water is used as wash water in the precipitating plant and usually supplies the make-up water to maintain the 2,000 G.P.M. from underground. Acid is immediately added to the off-solution at the sump to bring the solution to .5 lbs. H_2SO_4 per ton of solution. This keeps the iron from precipitating out, which gives us a clean sump, pump, and pipe lines to the caved ground. This strength acid does not deteriorate the transite pipe line. When the solution reaches the

caved ground it is measured in weir boxes and additional acid is added to bring the strength up to approximately 10 lbs. per ton. From this point on the solution is carried in polyethylene plastic pipe to the point of application. We have used ponds, sprays, and drill holes to introduce the solution to the caved ground. Due to the uneven surface slope encountered in the caved area, sprays are the most satisfactory. Sprays also allow the solution to be introduced slower than by ponding. Drill holes were used in an attempt to introduce the solution below the conglomerate cover east of the Miami fault.

The 3rd dimension is the amount of ore and waste to be penetrated. This averages about 600 ft. Of this 600 ft. the bottom 150 ft. contains the mixture of ore and waste we are leaching. The capping will average about 0.03% Cu. We try to maintain the underground collecting system so that we have a free flow of water in the main haulage ways and we have a few key sampling points. (Slides 6, 7 & 8)

The 4th dimension is the time factor. It takes from 3 to 4 weeks after a spray is turned on to the surface for it to come through on the 1000 level. The same spray will drop off in about 2 weeks after it is turned off.

(1) The next point is how long can the spray remain at one place before the grade drops.

This depends on the amount of ore in the area being leached. It is practically impossible to correlate the mining extraction records with the material to be leached. So when a new area is to be leached the spray is turned on and there is a waiting period of 3 to 4 weeks before a sample can be taken underground.

The sprays are left in one place until the grade from that area drops below 3 lbs. Cu per ton. The spray is then turned off and the area allowed to lie idle for a period of time. The rest and leach periods are alternated

over and over. The second time over an area we can remain about one-half the time used originally and recover about one-half the amount of copper. The third time over it is one-half the second, the fourth is one-half the third, etc.

There are several explanations for this. During the rest period oxidization of the sulphides may take place, but since we are leaching with an acid ferric sulphate solution and from observed action on the surface it appears that a reverse capilliary action is the best explanation. A copper bearing boulder can be put in a flow of leaching solution, then the solution removed, and as the boulder begins to dry, a copper oxide coating forms on its surface. When a copper bearing rock is broken, it usually cracks on the mineralized seams and shows that the solution is actually penetrating the rock. (Slide #9)

The sulphide areas do not give as high grade a pregnant solution as the oxide areas. The leaching of chalcocite, the predominate ore at Miami, seems to take two steps, which are expressed in the following reactions:

(1) $Cu_2S + Fe_2(So_4)_3 = CuSo_4 + 2\ FeSo_4 + CuS$

(2) $CuS + Fe_2(So_4)_3 = CuSo_4 + 2\ FeSo_4 + S$

The first reaction takes place fairly rapidly. The second reaction is much slower. Chalcopyrite, a minor ore mineral at Miami, does leach but it is very difficult to set up equations for the reactions.

The time to allow for an area to rest is difficult to determine due to the varying depth of the column being leached, the type and amount of mineralization. With a depth of 600 ft. the minimum rest period seems to be three months.

To date the Miami Mine has produced by leaching 140,000,000 lbs. of copper. The iron to copper ratio = 1.3 and the acid to copper ratio = 2.4, both figures based on net smelter returns. It is apparent that the largest

factors affecting the cost per lb. of Cu are the iron and acid consumption. If the acid in the pregnant solution becomes too high the iron consumption per lb. of Cu will go up. (The ferric iron in the pregnant solution also affects the iron consumption.) If the acid strength is too high the gangue consumption of acid will go up. The acid strength is kept below 15 lbs. even in the high grade oxide areas unless the ore is very close to the surface. The time lag between the solution entering the caved ground and the same solution appearing underground where it can be sampled and assayed makes this problem of control difficult. The best way we have found is to keep graphs of the assays of the underground sample points. From these graphs we can check the trend and anticipate a drop in grade or a decrease in acid consumption.

Iron salts do precipitate as the solution goes through the caved material. From the attached typical assay report it can be seen that iron loss from the cave area feed to the precipitation plant feed is 4.2 lbs. Fe per ton of solution. It has been our experience that there is no plugging action of the cave ground from these basic iron salts. It is not known where the iron precipitates, but it is probably high in the column.

The precipitating plant (Slides 10 to 19) at Miami is similar to other wash down plants in the southwest. The only difference is the method of cell loading. De-tinned shredded cans are received by rail and unloaded by a magnet. The magnet drops the cans into a hopper which feeds a belt conveyor. The belt travels over the center of the cells, and by means of a traveling tripper the cells are evenly charged with iron. The precipitated copper is washed from the cans through wooden screens by high pressure water. The precipitates discharge onto a decant slab. Washing is done on the day shift only. Solutions are drained from the precipitated copper and pumped back to the cells on afternoon shift. The copper is moved from the decant

slab to a drying slab early the next morning before washing starts. The copper is moved with a front-end loader. The same loader loads the dry copper into the railroad car. The copper is shipped to the smelter at Douglas.

The plant operates every day in the year. The copper recovery at the plant = 99% or an 0.04 lbs. copper per ton solution tail. The precipitates shipped average 23.5% moisture and assay 79% copper.

The operating crew consists of the following:

Precipitating Plant 7-day operation	(1 operator or pumpman	3 shifts
	(1 equipment operator	day shift
	(1 cave area pumpman	day shift
	(3 cell washers	day shift
	(1 foreman	day shift
Underground 5-day operation	(3 timber crews 6 men	day shift
	(2 hoist engineers	day shift
	(1 foreman	day shift
Maintenance crews from Copper Cities 5-day operation	(1 pipeman	day shift
	(1 machinist	day shift
	(1 mechanic	day shift

The over-all recovery by leaching at Miami will be impossible to determine as we do not know exactly how much copper we had to start. A percent recovery figure for leaching is a misleading figure. A 50% recovery of a 0.2% copper orebody would be considered good, but a 50% recovery of a 1% orebody would be poor.

The first 15% or 20% of the copper in place is very easy to recover. It is only possible through careful attention to details to maintain a steady production and to get a good over-all recovery by this method.

In order to give a complete picture of the leaching at Miami a flow sheet, a typical assay report and a sheet of leaching reactions are attached.

MIAMI COPPER COMPANY
DIVISION TENNESSEE CORPORATION

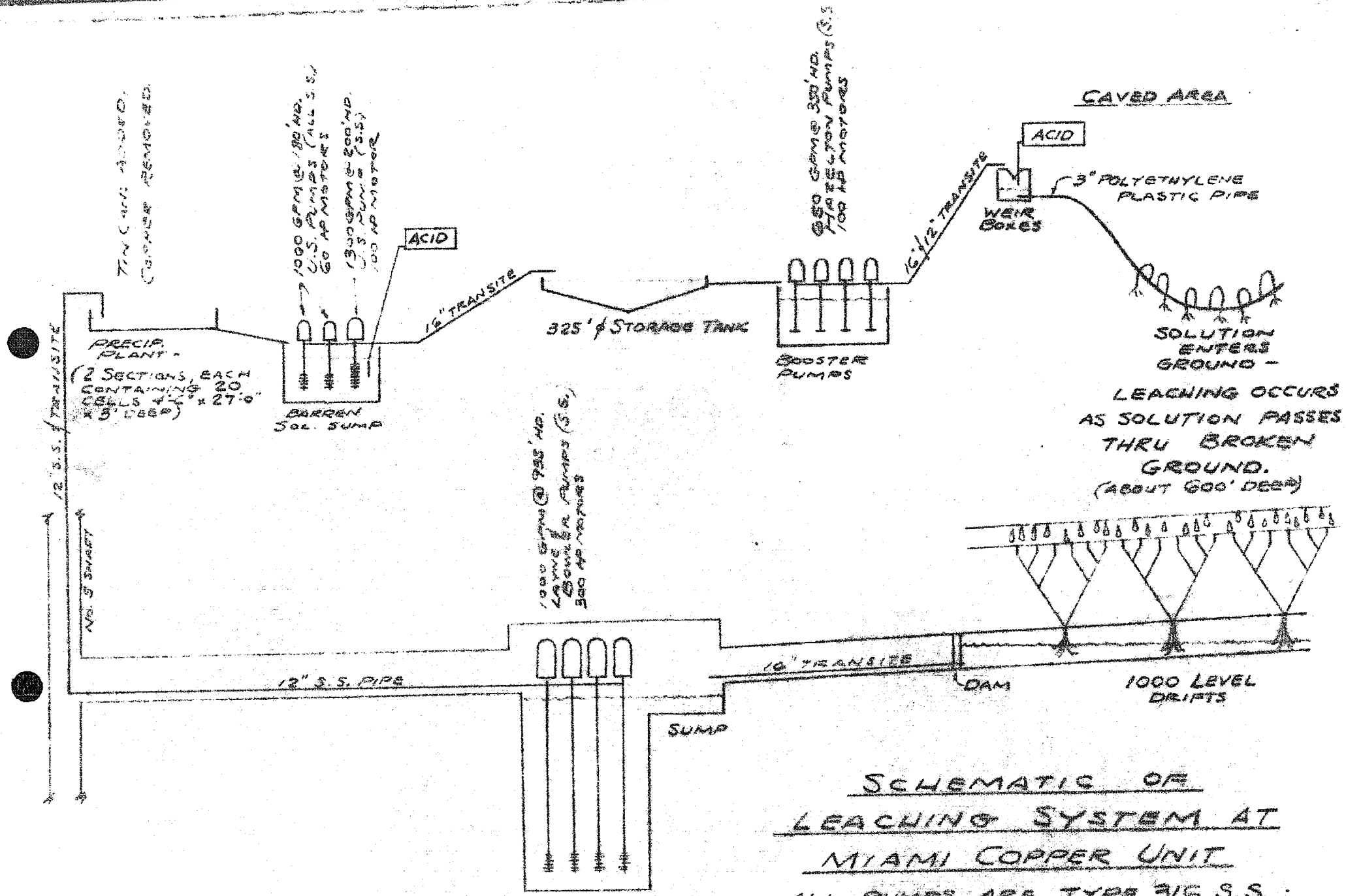
CERTIFICATE OF ASSAY

LEACHING PLANT

TYPICAL 1962

eaker Nos.	MARKS, ETC.	SAMPLE	CU	FE	FE''	FE'''	H_2SO_4	P.H.
		GMS	- POUNDS PER TON -					
	Feed ABC		4.28	4.4	2.3	2.1	0.5	2.4
	Tail A		0.04	9.4	9.0	0.4	Tr.	4.0
	B		0.05	9.2	9.1	0.1	Tr.	4.2
	C		0.03	9.7	9.5	0.2	Tr.	4.3
	C A Feed ABC		0.03		8.6		10.0	1.4
	Sump A		0.04	9.3	8.5	0.8	0.0	2.7
	B		0.03	9.2	8.5	0.7	0.5	2.8
	C		0.06	9.6	8.8	0.8	0.5	2.8

Assayer



SCHEMATIC OF LEACHING SYSTEM AT MIAMI COPPER UNIT

ALL PUMPS ARE TYPE 316 S.S.

LEACHING REACTIONS

$CuO + H_2SO_4 = CuSO_4 + H_2O$
$Cu_2O + Fe_2(SO_4)_3 + H_2SO_4 = 2\ FeSO_4 + 2\ CuSO_4 + H_2O$
$4\ CuO + Fe_2(SO_4)_3 + 5\ H_2O = 3\ CuSO_4 + 2\ Fe\ (OH)_3$
$4\ CuO + FeSO_4 + 6\ H_2O + O_2 = 4\ CuSO_4 + 4\ Fe\ (OH)_3$
$CuO + FeSO_4 + H_2O = CuSO_4 + Fe(OH)_2$

PRECIPITATION

$CuSO_4 + Fe + FeSO_4 = Cu + 2\ FeSO_4$

SULPHIDES

$Cu_2S + Fe_2(SO_4)_3 = 2\ CuSO_4 + 4\ FeSO_4 + S$
$Cu_2S + Fe_2\ (SO_4)_3 = CuSO_4 + 2\ FeSO_4 + CuS$
$CuS + Fe_2\ (SO_4)_3 = CuSO_4 + 2FeSO_4 + S$

CONVERSION - FERROUS TO FERRIC

$4\ FeSO_4 + 2\ H_2SO_4 + O_2 = 2\ Fe_2\ (SO_4)_3 + 2\ H_2O$

OXIDATION OF PYRITE

$4\ FeS_2 + 11\ O_2 = 2\ Fe_2\ O_3 + 3\ SO_2$
$(2\ FeSO_4 + SO_2 + O_2 = Fe_2\ (SO_4)_3$
$(Fe_2(SO_4)_3 + SO_2 + H_2O = 2\ FeSO_4 + 2\ H_2SO_4$
$2\ FeS_2 + 2\ H_2O + 7\ O_2 = 2\ FeSO_4 + 2\ H_2SO_4$

MINERALS

AZURITE - $Cu_3\ (OH)_2.(CO_3)_2 + 3H_2SO_4 = 3\ CuSO_4 + 2\ CO_2 + 4\ H_2O$

MALACHITE - $Cu_2\ (OH)_2.CO_3 + 2\ H_2SO_4 = 2\ CuSO_4 + CO_2 + 3\ H_2O$

CHRYSOCOLLA - $CuSiO_3.2\ H_2O + H_2SO_4 = CuSO_4 + SiO_2 + 3\ H_2O$

CUPRITE - $Cu_2O + H_2SO_4 = CuSO_4 + Cu + H_2O$
$Cu + Fe_2(SO_4)_3 = CuSO_4 + 2FeSO_4$
$Cu_2O + H_2SO_4 + Fe_2(SO_4)_3 = 2\ CuSO_4 + H_2O + 2\ FeSO_4$

CHALCOCITE - $Cu_2S + Fe_2(SO_4)_3 = CuS + CuSO_4 + 2\ FeSO_4$
$CuS + Fe_2(SO_4)_3 = CuSO_4 + 2\ FeSO_4 + S$
$Cu_2S + 2\ Fe_2(SO_4)_3 = 2\ CuSO_4 + 4\ FeSO_4 + S$

COVELLITE - $CuS + Fe_2(SO_4)_3 = CuSO_4 + 2\ FeSO_4 + S$

BORNITE - $Cu_5FeS_4 = FeS + 2\ Cu_2S + CuS$. Reactions are probably as for chalcocite. Iron is also attacked.

CHALCOPYRITE AND PYRITE - Not attacked by acid ferric sulphate solutions.

HYDROLYSIS

$4\ FeSO_4 + O_2 + 2\ H_2O = 4\ FeSO_4(OH)$
$5\ Fe_2(SO_4)_3 + 14\ H_2O = Fe_2\ O_3.4\ SO_3.9\ H_2O + 5\ H_2SO_4$
$Fe_2(SO_4)_3 + 6\ H_2O = Fe_2O_3 \cdot 5\ H_2O + H_2SO_4$
$2\ Fe_2(SO_4)_3 + 13\ H_2O = Fe_2O_3.5\ SO_3.[illegible]\ H_2O + H_2SO_4$

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

March 2, 1948

MEMO TO MR. A. E. RING

ARIZONA, GILA COUNTY
AMICO, Miami
Exploration
Churn Drilling and Sampling

A visit of February 28 to the AMICO project located a short distance southeast of Miami was arranged by Mr. E.N.Pennebaker, consulting geologist. Mr. Burt Reed, Anaconda geologist, provided for an inspection of drilling operations and also discussed several features of drilling, sampling and assaying, a summary of which follows:

Drilling

Two Bucyrus Erie 36L rigs (Aubrey Lyon, Contractor) are in use on the property. Both of the holes now being drilled were collared with 15 inch bits and are now reduced to 5 inch bits at depths approaching 3,000 feet. Casing sizes are 12 inch, 10 inch, 8 inch, 6 inch and 5 inch, (IDs).

Aquajell

The addition of aquajell to the sludge has three purposes, (1) To retard caving or sloughing from the walls of an uncased hole, (2) To thicken the sludge and thus raise the cuttings from the bottom, (3) To facilitate removal of casing by lubrication of the walls of the hole. The need of a sludge thickener to raise cuttings, (2), occurs when the rock being drilled contains a preponderance of quartz and/or sulphides and but a minor amount of altered feldspar or clay.

The Gila conglomerate in which the holes at AMICO are being drilled is a sedimentary rock which produces insufficient slime for efficient churn drilling. A mixture of clay and aquajell in equal proportions is used as a regular practice. The amounts added to the sludge vary from 20 lbs. to 100 lbs. per five foot run.

Aquajell (bentonite) costs about $2.00 per hundred and clay about $1.00 per hundred. The clay is added as a "filler" to reduce the cost.

Miami Copper engineers prohibit the use of drilling muds (aquajell and clay) because of the unknown amount of dilution. This, however, is true only of drilling under water. In drilling above water the amount of aquajell introduced into each sample can be rather closely determined. Also, in drilling a barren formation such as the Gila, there appears to be no valid reason for prohibiting its use. Aquajell was used frequently in the east San Manuel drilling. Mr. Reed estimated the dilution there to be not over 5% in any hole.

MEMO TO MR. A. E. RING
Re: AMICO, Churn Drilling
and Sampling

March 2, 1948

Sampling

Thus far the drilling at AIMCO has been entirely in Gila conglomerate and therefore no sample other than a "grab" or character sample is taken.

Mr. Reed provided some interesting data in connection with Inspiration Copper's sampling on the east extension of the San Manuel deposit. The comparison which follows shows assay results obtained from an Inspiration hole where splits from the same sample were dried over an open fire at the rig (Inspiration practice) and by steam plate at 250 deg. F (Magma practice). The table below shows the average of 30 samples, 150 feet of hole:

Total Copper			Non-Sulphide		
Magma	Inspiration		Magma	Inspiration	
Steam	Steam	Fire	Steam	Steam	Fire
1.208	1.24	1.33	.17	.30	.322

The above Inspiration assays show an increase of .07 total copper and .02 non-sulphide copper in fire drying contrasted to steam drying. This difference, which is roughly 6% in both total and non-sulphide assays, is due, I believe, to the driving off of the water of crystallization present in the clays. This water content would remain in samples steam dried below 250 deg. F, but may be all or in part driven off in samples that were fire dried at temperatures well above 250 deg. F.

The loss in sulphur due to possible roasting in the fire drying was not determined. However, the sulphide content was probably not over 6% in the original sample, therefore the effect of a minor amount of roasting would be negligible.

Magma's steam dried non-sulphide assay is .13 lower than Inspiration's steam dried non-sulphide assay. Mr. Reed believes this difference is due to the effect of the metallic iron (bit cuttings, etc.) that existed in the Magma sample. Tramp iron had been removed by means of an electro magnet from the Inspiration sample. In the non-sulphide method, a dilute sulphuric acid leach, the metallic iron may precipitate some of the copper.

Mr. Reed stated further that experiments at Andes Copper proved that the sulphuric acid leach dissolved copper carbonates, silicates and all oxides but cuprite. 50% of the cuprite was dissolved and 50% converted to insoluble native copper. Also, some of the "sooty" chalcocite was taken into solution.

JHC:W
CC: WHLoerpabel A/M
DJPope A/M
WRLandwehr A/M
LEKitch A/M

J. H. Courtright
J. H. COURTRIGHT

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

March 2, 1948

MEMORANDUM TO MR. A. E. RING

ARIZONA, GILA COUNTY
Miami Copper
Exploration
Churn Drilling and Sampling

COPPER CITIES

A visit to Miami's Copper Cities project, located a short distance north of Miami, on February 27 was arranged through the courtesy of Mr. E. N. Pennebaker, consulting geologist. Mr. Jack Pardee, exploration engineer provided the following information:

Churn Drilling:

Two company operated Bucyrus Erie 22 T (blast hole type) rigs are being used to drill to depths averaging 450 feet. The rigs are gasoline powered, cat track mounted, and have masts reenforced for pulling casing. The maximum depth that can be drilled with these machines under favorable conditions is 750 feet.

Holes are collared with a 10 inch bit and continued until caving necessitates the use of casing. Then 8 inch ID casing is run and the hole continued with an 8 inch bit. The hole is finished with 6 inch ID casing and a 6 inch bit.

Approximately 60 gallons of water is used in 5 feet of 10 inch hole. The average progress is 20 ft. per rig per shift.

Sampling:

Sludge is removed from the hole by means of a flat bottom valve bailer. Holes are bailed dry (4 to 5 bailers) above the water table (around 150 feet below the surface); below the water table the hole is bailed until the sludge becomes extremely thin.

The sludge is passed through a Jones type 3-tier splitter, producing a sample containing 1/8 the original volume. The sample from a 10 inch hole is reduced to 1/64 by pouring through a single tier splitter. The maximum size of any 5 foot sample is 3 gallons. Samples are transported in 3 gallon metal containers to the laboratory and steam dried at 220 degrees F.

A wet sample is sent to the laboratory for metallurgical tests and a sample is panned and dried at the drill for geological examination, sludge board, etc.

The copper occurs as chalcocite and chalcopyrite in about a 3 to 1 ratio. These minerals, including pyrite, occur as disseminated grains and veinlets in a strongly altered monzonite.

CASTLE DOME

In respect to the question of reliability of churn drill samples, the results of open pit operation at Castle Dome are of interest.

Roughly, 20 million tons of .8 copper ore has been mined here at the rate of 12,000 tons per day. Results available at the time of my visit from four different levels showed the tonnage mined varying from 75% to 105% of the expected tonnage and from 90% to 105% of the expected grade. According to local information the average grade of the ore mined has been within 4% (.03 cu) of the grade calculated from the churn drill samples. Most of the holes were drilled open, without casing, and very few reached the water table. The average depth was around 600 feet.

The tonnage extracted on several benches was considerably lower than expected. Tongues of waste which were not recognized or predicted from the drilling which was done on 200 foot centers, accounted for much of the discrepancy.

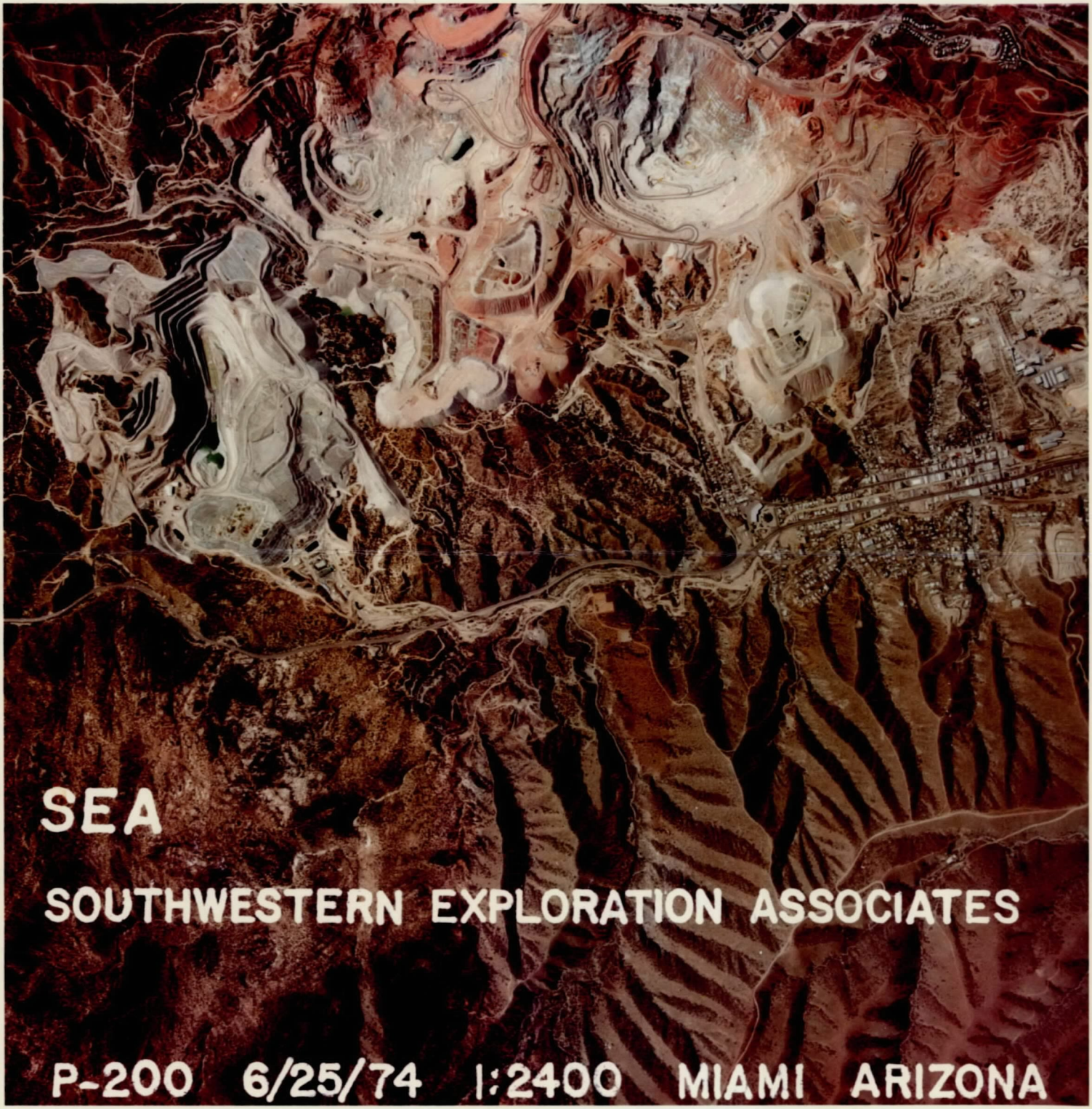
The copper occurs as chalcocite and chalcopyrite (3 to 1 ratio) in a rather hard, lightly pyritized moderately altered quartz monzonite. Over 50% of the copper occurs as discrete grains; the balance as small stringers and veinlets.

JHC:W
CC: WHLoerpabel A/M
DWPope A/E
WRLandwehr A/M
LMKitch A/M

J. H. COURTRIGHT

J. H. Courtright

JHC


SEA
SOUTHWESTERN EXPLORATION ASSOCIATES
P-200 6/25/74 1:2400 MIAMI ARIZONA

JHC

file Miami, Ariz

ARIZONA REPUBLIC
May 14, 1971

In Miami area

Occidental discloses copper find

MIAMI — The Occidental Petroleum Corp. announced yesterday the discovery of a new copper deposit of undetermined size at Miami.

The announcement was made in the annual report released to stockholders. It's the first official word concerning the firms exploratory drilling program that was launched in and around Miami three years ago by Occidental Minerals, a subsidiary of Occidental Petroleum.

Occidental said that copper oxide mineralization was found at depths from 1,100 to 2,000 feet underground. The thickness of the ore varied from 100 to 368 feet, graded from .56 to 1.26 per cent copper, the report stated.

The company said also that 10 of the 18 wide-spaced drill holes completed to date have intercepted copper mineralization.

The company said additional drilling and testing would be required to delineate the limits of the copper ore and to gather data necessary for a feasibility study.

Occidental holds 7,000 acres of mineral rights by lease or by claim in the Miami area including 90 per cent of the town itself and since January of this year the firm has been offering six-year purchase options to Miami property owners.

The Occidental announcement is the third release during the past year by a mining company of the discovery of new copper deposits in the Miami area.

Less than one-half mile east of Occidental's present drilling sites in the town of Miami, the Cities Service Co. earlier announced the discovery of a copper mineralization that ranged from 2,400 to 3,500 feet with an average assay of 1.35 per cent.

A $3.5 million underground shaft is under development for the purpose of evaluating the ore deposit.

About one half mile east, Inspiration Consolidated Copper announced last month to stockholders that it has discovered copper mineralization that ranged from 4,800 feet to 5,500 feet in depth, with a copper content of more than 1 per cent.

Inspiration said further slow, costly deep drilling will be required to analyze that deposit.