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WLK - JDS

ASARCO

Southwestern Exploration Division

November 11, 1992

F.T. Graybeal
New York Office

Ammonia Leaching of
Copper Concentrate
La Escondida Mine

A BHP process engineer said BHP and RTZ are still proceeding with plans to ammonia leach La Escondida concentrate. The engineer gave new insight into the ammonia leach process.

The ammonia leach process removes rims of chalcocite from grains of pyrite, and copper taken into solution is recovered by electrolysis. After ammonia leaching, the remaining concentrate is subjected to flotation which separates out pyrite for disposal. The resulting copper concentrate, reduced in volume by the removal of pyrite but having the same copper grade, is shipped for smelting.

It was not said if the process is profitable or if it is necessary because of volume constraints in the transportation and smelting of the concentrate.

H.G. Kreis

HGK:mek

H.G. Kreis

cc: D.E. Crowell
A.R. Raihl
W.L. Kurtz/J.D. Sell

ASARCO

DEC 10 '91

RECEIVED Mining Department
Tucson, Arizona
JAN 2 1992
EXPLORATION DEPT

MINING DEPT.
RJK
RSR
TCC
DLB
FTG
FILE

December 5, 1991

Mr. R. J. Kupsch, V.P.
Mining Department
NEW YORK OFFICE

R. J. K.
DEC 12 19

F. T. G.

Examination of Leachable Copper Deposits

I have prepared a tabulation of the major leachable copper occurrences that I have examined along with the summarized results of each evaluation. A copy is attached for your files.

Smaller, less significant prospects were not included in the list.

This tabulation will be updated to include future examinations.

S. A. Anzalone

SAA:brw

Attachment

RECEIVED
JAN 10 1992
EXPLORATION DEPARTMENT

Potential Heap Leach SX-EW Properties Examined

Name	Location	Type	Remarks	Future Interest
Emerald Isl.	Arizona	Exotic copper	Low grade. Limited tonnage.	None
Four Metals	Arizona	Breccia pipe, CuOx & chalcocite	Low grade. Limited tonnage.	None
Johnson Camp	Arizona	Porphyry copper skarn	Limited tonnage. Low grade. High waste to ore ratio.	None
Kingman Area	Arizona	Porphyry copper	Low grade. Limited tonnage.	None
Korn Kob	Arizona	Porphyry skarn	Limited tonnage. Low grade. High waste to ore ratio.	None
Lucky Susan (Castle Hot Spring)	Arizona	Porphyry & brecciated zone	Low tonnage. Low grade.	None
Old Reliable	Arizona	Brecciated andresite	Low tonnage.	None
Poston Butte	Arizona	Porphyry copper	Low grade. High waste to ore ratio.	None
Sacaton Mts.	Arizona	Alt porphyry & CuOx veins	Small prospects.	None
Sanchez	Arizona	Porphyry copper	Low grade. Environmental problems.	Review
Zonia	Arizona	CuOx in porphyry, sediments, metamorphics	Limited tonnage. High waste to ore ratio.	None
Zaldivar	Chile	Chalcocite & copper oxide porphyry copper	High waste to ore ratio.	None
Ann-Mason	Nevada	Porphyry system	Low-moderate grade. High waste to ore ratio.	None
Big Mike	Nevada	Exotic copper, Gila Conglomerate	Low tonnage. High waste to ore ratio.	None
Copper Basin	Nevada	Porphyry copper	Low grade. Limited tonnage.	None
Copper Canyon	Nevada	Porphyry copper	Low grade. Limited tonnage.	None

Potential Heap Leach SX-EW Properties Examined

Name	Location	Type	Remarks	Future Interest
Ely (Robinson)	Nevada	Chalcocite & copper oxide porphyry copper	Low grade. Some sulphide ore remains.	None
Lewis Peak	Nevada	Breccia pipes	Limited tonnage. Weak CuOx	None
Mayville Canyon	Nevada	CuOx in quartz veins & weak breccia	Limited tonnage.	None
Yerington	Nevada	Chalcocite & copper oxide porphyry copper	Low grade. Environmental problems.	Review
Bent	New Mexico	Sandstone Copper	Low tonnage/grade	None
Cerrillos	New Mexico	Chalcocite & copper oxide porphyry copper	Limited tonnage. Low grade.	None
Copper Hill	New Mexico	Chalcocite & copper oxide in Precambrian quartzite	Low grade. Limited tonnage.	None
Eureka	New Mexico	Sandstone Copper	Low tonnage. High waste to ore ratio.	None
Gallineas Peak	New Mexico	Alt intrusive	No mineralization.	None
Guadalupita	New Mexico	Sandstone Copper	Low tonnage/grade	None
Hammond	New Mexico	Sandstone Copper	Low tonnage/grade	None
High Rolls	New Mexico	Sandstone Copper	Low tonnage/grade	None
Hillsboro	New Mexico	CuOx in porphyry system	Low grade. Limited tonnage.	None
Ladron Mt. Area	New Mexico	Porphyry area	Reported weak CuOx. Not located.	None
Mt. Baldy-Moreno Ranch	New Mexico	Copper porphyry area	Low grade. Little tonnage.	None
Nacimiento	New Mexico	Sandstone copper	Low tonnage. High waste to ore ratio.	None
Peñas Negras	New Mexico	Sandstone Copper	Low tonnage/grade	None
Pintada Mine	New Mexico	Sandstone Copper	Low tonnage/grade	None
Scholle District	New Mexico	Sandstone Copper	Low tonnage/grade	None

Potential Heap Leach SX-EW Properties Examined

Name	Location	Type	Remarks	Future Interest
Stauber Mine	New Mexico	Sandstone Copper	Low tonnage/grade	None
Zuni Mts.	New Mexico	CuOx in Precambrian and adjacent arkose	Small prospects	None
Kirwin	Wyoming	Porphyry system, CuOx & chalcocite	High waste to ore ratio. Environmental problems.	None

S. A. Anzalone
October 22, 1991

WLR

ASARCO

Exploration Department
Western USA

September 26, 1989

D.E. Crowell
Tucson Office

Heap Leach Test Report
Black Pine, Idaho

For your information enclosed is a report by Kappes, Cassiday & Associates done for Noranda.

W. L. Kurtz

WLK:mek
enc.

cc: R.L. Brown (w/o report)
J.D. Sell (w/o report)
S.L. Lakosky (w/o report)

ASARCO

Exploration Department
Western USA

September 26, 1989

Mr. R.L. Brown
New York Office

September 1988
Feasibility Summary
Black Pine, Idaho

For your general information on projected cost of a heap-leach gold operation, enclosed is part of a September 1988 report by Noranda.

WLK:mek
enc.

W. L. Kurtz

cc: J.D. Sell
S.L. Lakosky/D.F. Skidmore/D.E. Crowell (w/enc.)

✓ copy for: WLK, JDS, FTG, DMS, JCB, PGY

sent 10/12/8[illegible] mek

Circle Reply Ca[illegible]

(*) 50% CANO RESOURCES — HOMESTAKE CAN BUCK-IN

XC: WLR - 8-8-85

ASARCO

Mining Department

August 1, 1985

MEMORANDUM TO: T.E. Scartaccini

SUBJECT: Lakeshore Unit, Noranda Mining Co. - In Situ Leaching

The following summarizes the results of our July 25, 1985 visit to Noranda's Lakeshore Unit to examine their current leaching operations.

The Asarco group included the writer, T.E. Scartaccini, D.E. Crowell and T.D. Henderson. D.E. Crowell's memorandum summarizing the metallurgical aspects of the leach operation is attached. E.H. Ahrens, Chief Geologist and J.T. Kline, Chief Metallurgist for the Lakeshore Unit presented the technical data for Noranda.

Summary

At the suggestion of F.T. Graybeal, a visit to the Lakeshore Unit was arranged to obtain information regarding the current status of Noranda's leaching operation. Lakeshore is being considered as a possible site for in situ leach tests that might receive research funding from the U.S. Bureau of Mines. Since Asarco has competitive mineral zones at Santa Cruz and Sacaton that might also qualify, it was decided to attempt to gather as much information as possible about the Lakeshore operation to assist in presenting Asarco's alternative test site to the U.S. Bureau of Mines. Neither Noranda's current caved zone leaching operation nor their planned 32 hole sub-900' level leach test qualify as pure in situ leach operations. This information should be of importance to the USBM in their search for a suitable in situ leach test site.

Geology - Oxide Zone

Current leaching operations are centered on the old caved areas resulting from previous block caving at Lakeshore. The leachable mineral zone is a draped, tongue-like copper oxide blanket (See plate 1) that resulted from the oxidation of a previously existing chalcocite blanket in a granodiorite porphyry. This oxide zone is bounded on two sides by steep faults, capped by fanglomerate and underlain by unoxidized sulfide mineralization that is considered by Noranda to be impermeable. These zones act as aquicludes to the leach solutions. Copper oxide mineralization is principally chrysocolla with lesser amounts of brochantite, cuprite, tenorite and native copper.

Current leaching operations consist of injecting leach solutions into the old caved areas through 492 drill holes cased with 5 1/8" plastic pipe. Pregnant solutions are collected on the 900' level and pumped to the surface SX plant. Current production is 1.5 million pounds per month at a cost of ± $0.65 per pound. Sulfuric acid consumption is reported

to be high; currently in excess of 20 pounds per pound of copper. Noranda eventually hopes to achieve a figure of 10 pounds of acid per pound of copper (See attached memo by D.E. Crowell for metallurgical details).

Reserves

Ahrens and Kline did not discuss "in situ leachable reserves" in terms of tons of ore. They preferred instead to discuss recoverable pounds of copper at a certain cutoff grade. The "reserve" is based to a large extent on the consumption of sulfuric acid per pound of recovered copper.

Based on other sources, the oxide mineral inventory at Lakeshore is as follows:

Lakeshore Oxide Mineral Inventory - 1975

% Cutoff Grade	Tons	% Acid Soluble Copper
0.10	825,375,000	0.45
0.30	142,583,000	0.72
0.40	103,000,000	0.86
0.50	79,583,000	0.99
1.00	28,250,000	1.52

Of this, the .5% cutoff grade may be amenable to in situ leaching. Lakeshore has been using a .7% cutoff. It is reasonable to assume that there is something in the neighborhood of 60 million tons of approximately 1.00% acid soluble oxide copper available for leaching at Lakeshore. Certain zones within this 60 million tons will assay over 1.5% copper, and it is within one of these high grade zones that Noranda plans to attempt its 32 drill hole "in situ leach" test.

In Situ Test Area

Noranda has delineated a 360,000 ton block that assays 1.63% acid soluble copper immediately below the 900' level. This oxide block is positioned within a basin-like depression in the underlying "impermeable sulfide" zone. Leach solutions will be injected into this zone from the 900' level through 32 drill holes collared on 25' centers. Noranda does not plan to retrieve this pregnant solution by pumping through drill holes. Rather, they expect the pregnant solutions to percolate up through the oxide-bearing rocks in the confined basin structure into the overlying workings on the 900' level. From there, the solutions will be commingled with the pregnant solutions from the existing caved zone leaching operation and pumped to the surface SX plant in two stages. This is not in situ leaching as normally considered. Data from these test wells will not be entirely applicable to in situ leaching of unmined or caved

deposits where solutions are injected through deep drill holes and recovered by pumping to the surface through retrieval wells. This point should be made clear to the USBM.

Noranda has approved funding for the 32 hole test. At this point, they plan to run the test on their own. They believe, however, that they have a good chance of getting some USBM funds because of their "leaching know-how," plus the fact they they have an operating SX plant.

Conclusion

Ahrens and Kline were cooperative in discussing their operation. They did not hesitate to state that Noranda had put approximately $4,000,000 into the current leach operation. They did, however, indicate that Noranda felt it had some proprietary technical data that they did not wish to divulge. During the course of the meeting, they commented that Noranda and Asarco should consider a Joint Venture because Asarco had a reliable source of sulfuric acid and Noranda has an operating "in situ" leach operation and considerable leaching "know-how."

S.A. Anzalone

SAA:llm
attachments
xc: RJKupsch - w/attach.
FTGraybeal - "
DECrowell - w/plate 1

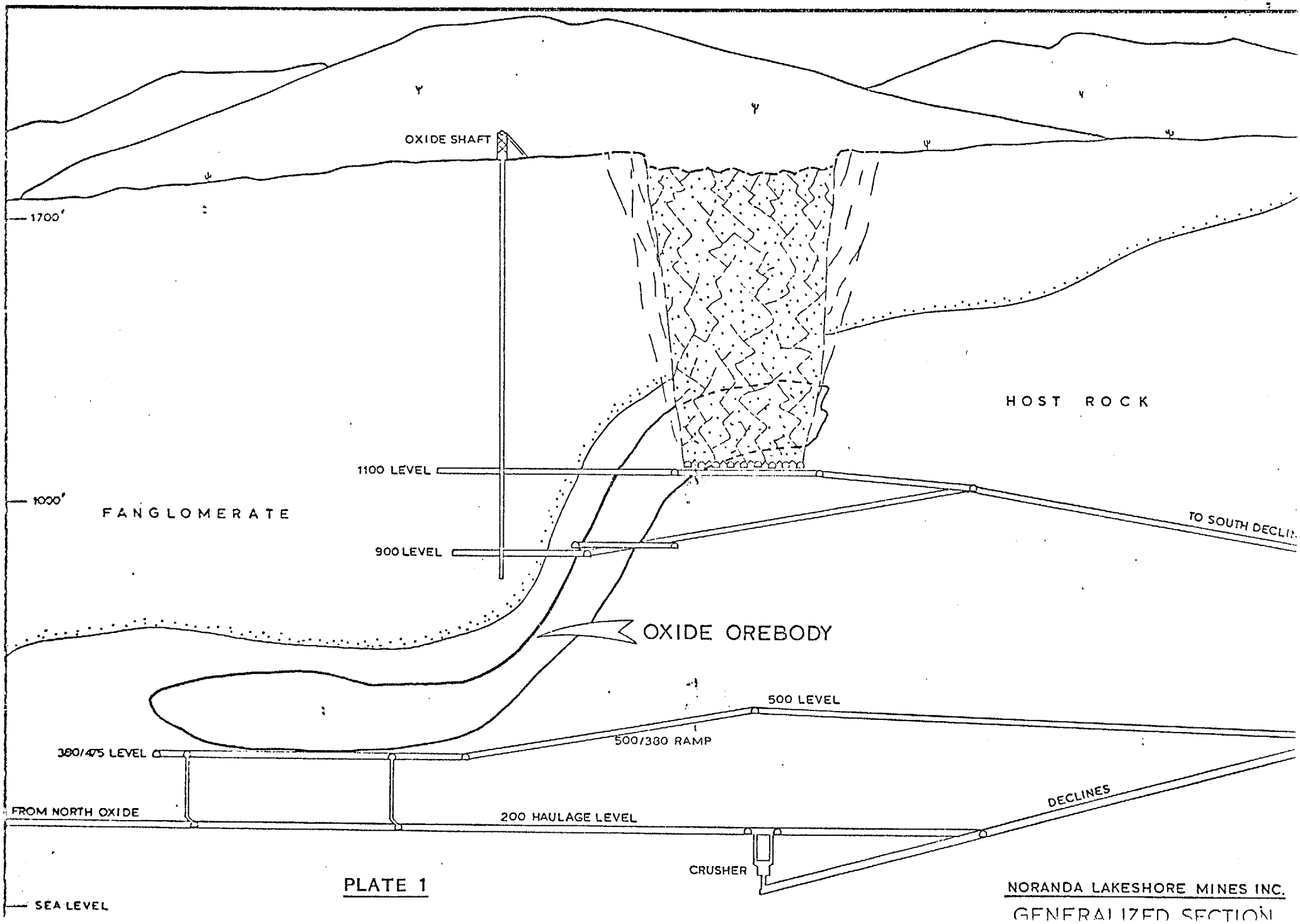


PLATE 1

NORANDA LAKESHORE MINES INC.

GENERALIZED SECTION

ASARCO

Mineral Beneficiation Department

August 1, 1985

MEMORANDUM TO: S.A. Anzalone

SUBJECT: Notes on Lakeshore Plant

SX - EW Plant

1. Original plant built for 2,000 gpm of fairly high grade solution. Now have modified to take 3,000 gpm of lower grade solution (1.6 gm per liter) from present leaching operation.

2. Pregnant solution is very clean (<10 ppm suspended solids) and has a constant temperature of 30° C ± 1° C. This makes for very stable, efficient SX plant operation.

3. There is very little iron in the pregnant solution since only .04 pounds of iron dissolve per pound of copper.

4. The electrowinning plant is running only part of the available cells since the low grade solutions produce less than one-half of the copper formerly produced.

Leaching of Block Caved Ore

1. Leach solutions are injected into the caved area through 5 1/8-inch diameter injection holes drilled on 50 foot centers. There are 492 holes total with about 150 in use at any one time to inject 3,000 gpm.

2. The injection holes are cased with slotted 2-inch diameter Schedule 80 PVC pipe with threaded couplings. The holes were drilled with reverse circulation rigs and some problems were encountered with voids in the caved rock.

3. Before starting the block cave leach, a sample of 550 tons was pulled from a single draw point in an effort to obtain a complete column of ore from the caved area. Laboratory leach tests were run on this sample as well as drill hole samples. The oxide ore is fractured and partially recemented with copper oxides, which makes it ideal leaching ore.

4. Pregnant solution is collected in the old slusher drifts and held in ponds formed by damming main haulageways. Solution is then pumped to the surface in two stages. All pumps and piping underground are stainless steel.

5. Acid consumption is the major operating cost and varies greatly with time. Initial consumption is very high as basic rock constituents are neutralized and then falls off as these constituents are consumed.

Ultimately, they expect a consumption of about 10 pounds acid per pound copper recovered.

6. The cost of setting up the block cave leach is roughly estimated at $4 million, split evenly between the cost of drilling plus casing the holes and the other facilities such as distribution system, collection system and modification of the SX plant.

7. Operating costs were given as $0.60 plus per pound of copper produced including Noranda overhead. This does not include Lakeshore's "locked in" expenses from the previous operations such as Indian royalty, power paid for but not used or long term leases on trucks.

In Situ Test

1. They plan to test a block of ore containing 360,000 tons at a grade of 1.63% acid soluble copper.

2. Leach solution will be injected through 32 2-inch diameter holes (Schedule 80 slotted plastic pipe with 1.5 inch ID) spaced on 25 foot centers. Open area of slots is 9%. The deepest hole is 425 feet.

3. The test block is below the 900 foot level and is essentially surrounded by aquiclude formations. Because of this, they will inject leach solution through all 32 holes and force pregnant solution to rise through natural fractures to be collected in the drift above the ore block. They feel that this method will give better results than the conventional method of injecting in a pattern of holes and pumping from holes in the center of the pattern.

4. They will start drilling the holes in September and plan to finish by January 1, 1986. They will then test with water and "enhancers" until around the end of February when they plan to begin injecting acid.

General

1. The people we talked to were:

E.H. Ahrens	-	Chief Geologist
John T. Kline	-	Chief Metallurgist

We did not see Paul Musgrove, the Manager.

2. Lakeshore used to produce 3.5 million pounds of copper per month with 680 employees when the U/G oxide plant was running. Now they produce 1.5 million pounds per month with 58 employees. There are only 4 people at the plant on swing and graveyard shifts.

3. They determine "ore" by calculating acid consumption per pound copper recovered rather than using cutoff grades. Basically, anything consuming less than 10 pounds acid per pound copper is classified as ore. This criteria can lead to situations where blocks containing .5% acid soluble copper are not ore and blocks containing .2% acid soluble copper are ore.

4. In situ mining comes under the underground injection laws which are now in effect. Test wells or operations at an existing mine come under Class 5 rules. Production wells not associated with existing mines are Class 3 wells which have more stringent rules. Lakeshore is currently unpermitted because they are on Indian land.

5. Although reasonably open and cooperative, the Lakeshore staff feel they have significant proprietary information regarding in situ leaching. One area is in regard to enhancing the transmission of fluids through rock by viscosity control, possibly by pH control and/or chemical additions.

6. Two publications recommended by the Lakeshore staff are listed below. We have ordered both:

   USBM RI 8961
   USBM Report from MSRD - Contract No. JO 295045.

D.E. Crowell

DEC:llm

xc: TEScartaccini
TDHenderson - File L4b

For More Facts Circle 37 on Reply Card

ASARCO

Southwestern Exploration Division

July 2, 1985

File Insets Leach

F. T. Graybeal
New York Office

H. G. Kreis' Report
Solution Copper Mining

Attached is H. G. Kreis' report on Solution Copper Mining in reply to your request.

Kreis has covered the whole ball of wax and I'm not sure this is what you wanted. However, I believe the Figures show the process quite well.

James D. Sell

James D. Sell

JDS:mek
Att.

cc: W. L. Kurtz
H. G. Kreis

Henry G. Kreis
ASARCO Incorporated

June 24, 1985

SOLUTION COPPER MINING

Throughout the twentieth century the American Mining industry has sought and developed lower and lower cost methods of recovering copper from the ground. With the current, severely depressed copper prices, the need for lower cost methods is more critical than ever. The American mining industry needs a radically lower cost method, and it needs it now before such a method is developed and implemented by foreign competition.

Solution mining of copper from soluble copper deposits is such a low cost method, and it has been used to a limited degree in conjunction with conventional mining methods. Solution mining is the only type of domestic copper mining that is profitable at today's severely depressed copper prices. But, for the most part, current day solution mining technology limits its use to extracting soluble copper from rocks broken up but not consumed by conventional mining operations. Thus its present and continued use is strongly dependent on the continuing use of conventional mining methods. For without a continual supply of broken up rock to process, solution copper mining would no longer be feasible using today's technology.

The future of solution copper mining, and perhaps that of the American copper mining industry, rests in developing the technology of applying solution mining to rocks undisturbed by conventional mining methods. That is to say, soluble copper would be extracted from rock in its natural state; and after solution mining ceased, the rock would still be in its natural state less, of course, its former copper content. Developing the technology to solution mine copper from rock in its natural state would free solution copper mining from its dependence on conventional mining techniques that are currently unprofitable. Most importantly, it would allow the development of a large number of soluble copper deposits in the western U.S.A. that are sitting idle because they are not profitable with conventional mining methods, nor are they in a situation that will cause them to be broken up by an adjoining conventional mining operation.

This paper is being written to encourage further interest in the field of solution mining; and, more specifically, to encourage developing the technology to solution mine copper from rock in its natural state. An understanding of the technology and applications of solution copper mining is most easily achieved starting with some of the basic geologic characteristics of a copper deposit.

Geology

The geologic characteristics of a copper deposit are a contributing factor to how the deposit is to be mined. There are, for example, two general classifications of copper minerals that determine how the copper is to be recovered from the rock.

The most important classification, with respect to the topic of this paper, is a group of minerals in which the contained copper metal can be dissolved in water containing 1% acid. The mining industry calls this group of soluble copper minerals--"oxide" minerals. It is from this group of minerals, the oxide minerals, that present day solution mining recovers copper, and this process is called "leaching" by the mining industry. Commonly leached, oxide minerals are chrysocolla, azurite, malachite, atacamite, and chalcocite.

The second copper mineral classification is that of the "sulfide" group. All of the sulfide minerals, with the exception of chalcocite, are insoluble in water containing 1% acid. The most common minerals of this group are chalcopyrite, bornite, and chalcocite. It is from these minerals that conventional mining recovers copper. In the future, technology may find a way to dissolve these minerals so that these minerals too can produce copper by solution mining methods.

Besides the types of copper minerals, there is another geologic feature of importance to solution mining. It is the presence of fractures. Without fractures the copper bearing rocks would be totally impervious to water, and solution mining would be impossible unless the rock was mined and crushed to the size of sand. The fractures allow the ingress-egress of the water solution so that it can make contact with the copper minerals, dissolve the copper, and move the dissolved copper outside the rock to a place where the copper can be recovered from the solution.

All copper bearing rocks are naturally fractured to some degree, and it is this fracturing upon which the future of solution mining rests. Too little is known about natural fracturing in copper deposits, to know if it will allow solution mining of its copper. We do know that rocks fractured, broken, and moved during mining operations readily allow the ingress-egress of leach solutions and, thus, are very amenable to solution mining. Whether or not natural fracturing is suitable for solution copper mining can only be determined by field test work. This will be discussed later in this paper.

Solution Mining Methods

Solution copper mining today is a simple and inexpensive process. It involves putting water with 1% acid on top of copper bearing rock that has been fractured, broken, and moved during conventional mining operations. The solution flows by gravity through the rock pile, dissolving copper as it passes. The copper bearing solutions flow out the bottom of the rock pile where they are collected and sent to a processing plant. The most popular, large-scale processing plant is a solvent extraction-electrowin plant. This type of plant removes the copper from solution and puts it in a directly saleable form of copper, and it does this at low cost.

Adapting this solution copper mining to natural rock, rock not broken by a mining operation, will require developing new techniques of solution application and recovery. Because most copper deposits are 100's of feet beneath the surface of the ground, solution application and recovery would most likely be done with the use of drill holes, say six to twelve inches in diameter. Solutions would be put into the copper bearing rocks with injection drill holes, and the resulting copper bearing solutions would be collected in recovery drill holes.

The actual solution mining process would be as is shown in Figure 1. In this figure the water solution of weak acid leaves the injection drill hole and moves out into the rock through natural fractures, Step A. In Step B the solution comes in contact with the soluble copper minerals (oxide copper minerals), and the copper is dissolved. The solution, now loaded with copper, continues through the fractures, Step C, and collects in the recovery well where it is pumped to the surface of the ground, Step D. In Step E a plant, most likely a solvent extraction-electrowin plant, extracts the copper from solution, and the copper free solution is returned to the injection well, Step F, where it will repeat the process.

The solution mining process is a relativey simple process, but not all of the steps shown in Figure 1 have been proven to work. Both the copper recovery plant (Step E) and the dissolving of copper (Step B) are proven processes widely used in the mining industry today. While injection and recovery wells have not been used in the commercial production of copper, there is sufficient technology in the minerals and petroleum industry to believe they will work satisfactorily. What remains to be proven is: (1) the fractures are such that they will allow commercial quantities of solution to move through the rock (Steps A and C), and (2) the solution will move through the rock in a sufficient abundance of fractures to make contact with most of the copper minerals. These two factors are very poorly known in our current state of knowledge, and only field testing and pilot plants will demonstrate whether they will work on a commercial basis.

Developing the Technology

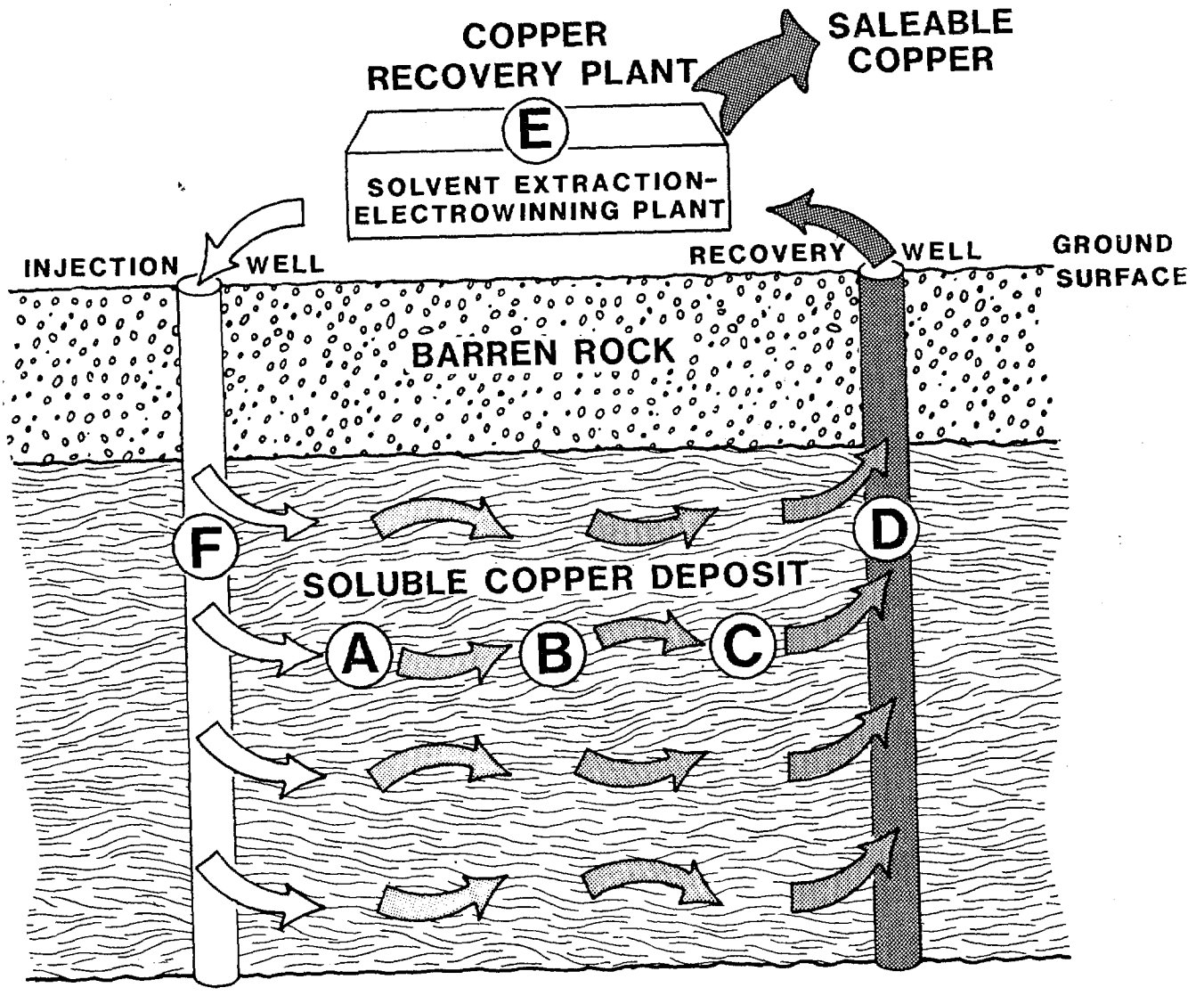
The American copper mining industry has made attempts to develop solution mining of copper from rock in its natural state, but these attempts were never carried out with sufficient work to determine whether or not the process is commercially feasible. The attempts were made by large corporations on big, Arizona copper deposits. In one case, at Miami, Arizona, solutions from an injection well moved through the rock dissolving copper and collected in a recovery well. The test was successful and justified plans for a more comprehensive, fifteen million dollar continuation of test work. However, falling copper prices and an adverse land situation caused the termination of test work in 1980.

Since the termination of that test, there has been little or no follow-up work on that deposit or any other American copper deposit. The reason being the depressed copper prices and the flood of imported foreign copper. American companies have been scrambling to get out of the copper business, and those remaining in it, are struggling to survive. Those surviving in the copper mining business will recognize the inherent difficulties of developing a new mining technique such as solution mining copper from natural rock.

FIGURE 1

WLK

SOLUTION COPPER MINING
SHOWN IN CROSS SECTION



STEP: (A) SOLUTION FROM INJECTION WELL GOES INTO NATURAL FRACTURES.

(B) SOLUTION CONTACTS COPPER MINERAL AND DISSOLVES COPPER.

(C) COPPER-BEARING SOLUTION MOVES TO RECOVERY WELL

(D) COPPER-BEARING SOLUTION GOES TO COPPER RECOVERY PLANT

(E) COPPER IS RECOVERED IN SALEABLE FORM.

(F) SOLUTION IS RETURNED TO RECYCLE THROUGH STEPS A THROUGH F.

One obvious answer to this problem is for a number of companies to pool their soluble copper deposits and some risk capital in a joint venture to develop and operate solution copper mines. But the times are not conducive to initiating such a venture. Alternative answers to this situation would have to include some sort of direct or indirect help from the United States Government. In formulating answers to the problem, industry and/or government will need to know about the step-by-step process involved in developing solution copper mining and the cost of each step.

Development Test Program

Solution copper mining can only be developed and tested on a soluble copper deposit in the field. There is no way it can be simulated on a computer, done in a laboratory, or done in the ground where a copper deposit does not exist. A suitable soluble copper deposit must be selected. Once this task has been accomplished, test work will proceed in two distinctly separate steps with the success of the first step, Phase I, justifying the second, Phase II.

Phase I will establish that commercial amounts of solution can move through the fractures in the natural rock from the injection well to the recovery well. Once this is accomplished, Phase II will evaluate how well the solution is going through the rock and recovering copper. Both Phase I and Phase II will take five years to complete, and the total cost will be $18,000,000. The completion times and costs of each phase are summarized in Table 1. The actual test setup for each phase is displayed in Figure 2 and Figure 3. Success in Phase II would lead directly into a small scale solution copper operation that would gradually develop into a full scale operation commensurate with the size of the copper deposit.

Table 1. Development Test Program

Phase	Objective	Test Setup	Time	Cost
I	Establish and measure the flow of solution from the injection well to the recovery well.	One injection well and two recovery wells set in copper bearing rocks.	1½ yrs.	$3,000,000
II	Demonstrate ability of solution to recover commercial amounts of copper.	Two injection wells and six recovery wells.	2½ yrs.	$15,000,000
		Total	5 yrs.	$18,000,000

FIGURE 2

PHASE 1 TEST OF SOLUTION FLOW

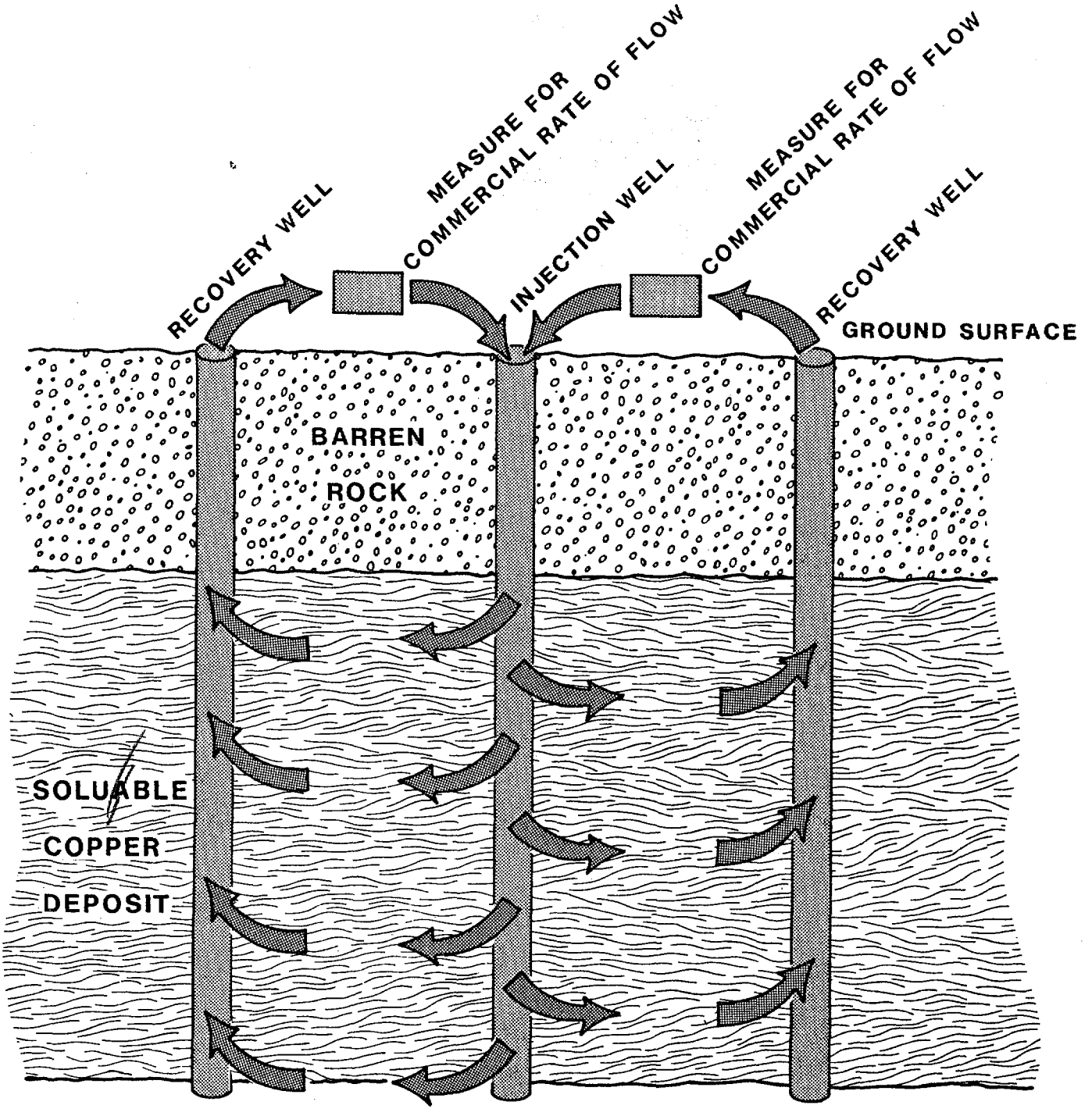
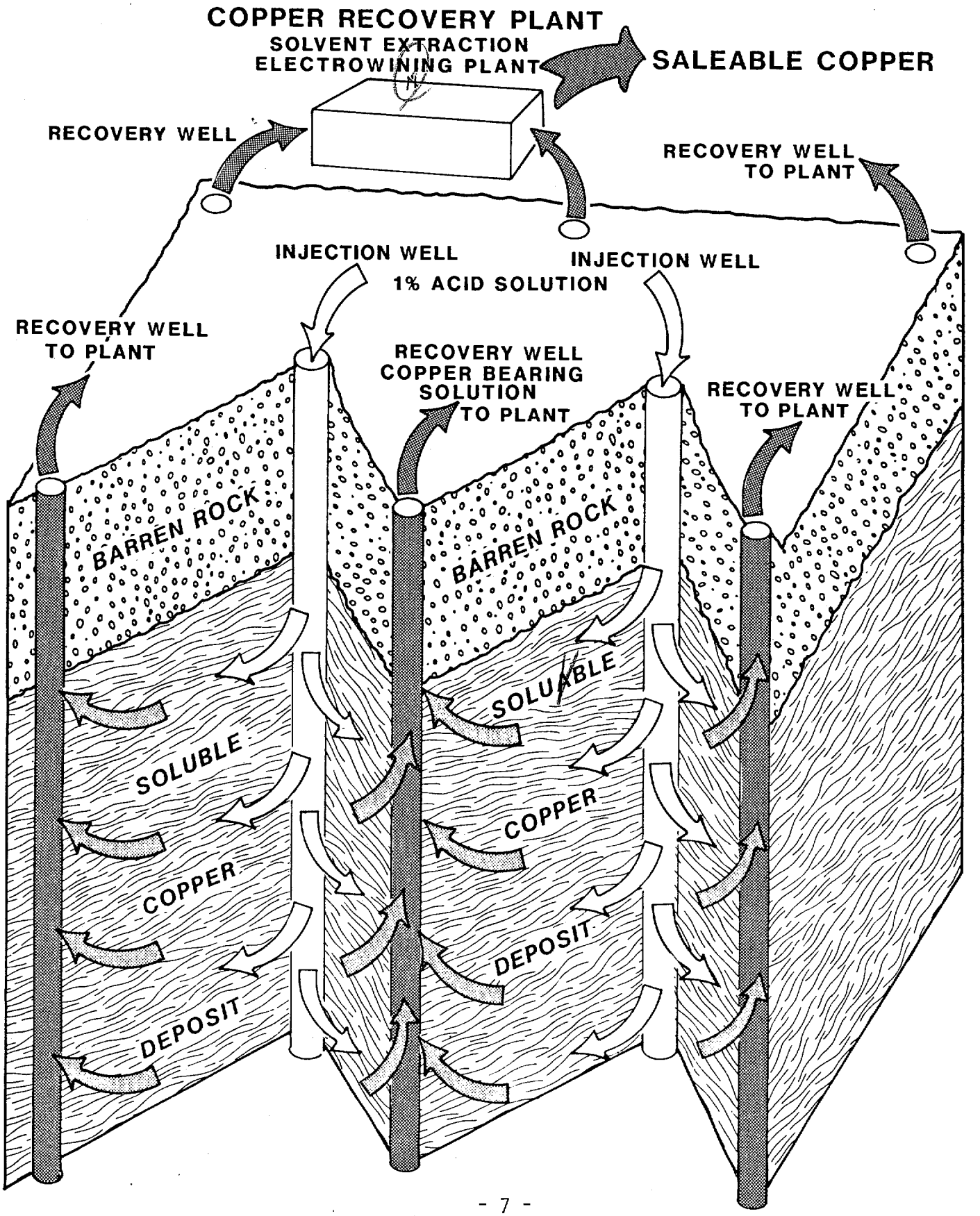


FIGURE 3

TEST OF COPPER RECOVERY



Full Scale Operation

A typical full scale operating solution copper mine is envisioned as having 300 injection and recovery wells. In addition there would be several dozen monitor wells about the periphery of the operation to insure safeguarding the environment and preventing the loss of valuable copper bearing solutions. From 10,000 to 40,000 tons per year of saleable copper would be produced. The actual amount of production would be determined by the size of the deposit and the rate at which the solution can extract the copper from the rocks.

The total capital cost to develop such a mine is estimated at $50,000,000 to $100,000,000. Unlike the all-at-once lay out of capital for developing a conventional mine, a solution copper mine would develop with an expenditure of $20,000,000 at a time as solution copper mining proved itself commercially successful.

The operating costs of a solution copper mine are estimated at 20% to 40% less than the cost of conventional mining. More importantly, solution copper mining offers the potential of operating profitably at today's depressed copper prices.

The cost efficiency of solution copper mining is due to the simplicity of the method. Compared to a conventional copper mine, one ton of saleable copper from a solution copper mine will eliminate the need to mine 750 tons of waste rock and copper ore, the milling of 150 tons of copper ore, and the costly smelting and refining of copper from 3 tons of concentrated copper minerals. Since there are tremendous amounts of soluble copper reserves in the western United States, the total cost savings of utilizing solution copper mining could be enormous.

A Test Deposit

Choosing a soluble copper deposit for the test development of solution copper mining will be a relatively easy task since there are dozens of such deposits in the western United States. In Arizona alone, there are more than a dozen large deposits collectively amounting to well in excess of 15,000,000 tons of copper. Owners of these deposits include the Kennecott Copper Corporation, ASARCO Incorporated, Phelps Dodge Corporation, Freeport Copper Company (Freeport - McMoRan), Newmont, Hudson Bay (acquired Inspiration Consolidated Copper Company), Van Dyke Copper Company, Getty Oil, and Hanna Mining Company.

Amongst all the soluble copper deposits, the best suited deposit is in central Arizona just west of the town of Casa Grande. It has ideal technical characteristics including a highly soluble copper mineralogy (atacamite and chrysocolla), fracturing, a favorable host rock lithology, isolation from acquifers, and an excellent location. Furthermore, it contains in excess of 5,000,000 tons of copper in soluble form which makes it the largest known soluble copper deposit in the world. The ownership of this deposit is divided between two joint ventures composed of Asarco-Freeport and Hanna-Getty. The Asarco-Freeport joint venture is known to be interested in solution copper mining. In fact ASARCO Incorporated allowed the U.S.B.M. to conduct some very preliminary solution mining test work on its soluble copper reserves. So it is

highly likely that most or all of this deposit could be made available for the test development of solution copper mining.

Summary and Conclusion

Solution mining of soluble copper from undisturbed natural rock would be a tremendous technological and economical breakthrough for the mining industry. It has the potential of being economic at today's copper prices, and could make profitable mining operations out of many copper deposits that would never have been mined conventionally, even at high copper prices.

The technology of solution mining copper from natural rock has yet to be totally proven. Steps B and E in Figure 1 have been proven to work time and again in the solution mining of copper from rocks broken during conventional mining operations. Steps D and F are widely used in industrial operations other than solution copper mining, and are expected to be readily adaptable to solution copper mining. Steps A and C have yet to be tested and proven. Steps A and C are simply the process of moving the solution through the fractures in the rock in such a way that commercial amounts of copper become dissolved in the solution.

The only way to test and prove Steps A and C is to conduct test work on a soluble copper deposit. A two phase test is proposed. Phase I (Figure 2) will test the amount of solution that can move through the rock fractures. Phase I will take 1½ years to complete, and it will cost $3,000,000. If Phase I is successful, Phase II (Figure 3) will be conducted to prove commercial amounts of copper can be recovered by the solution mining process. Phase II will cost $15,000,000 and take 2½ years of time to complete. If Phase II is successful, the test work program will be gradually upscaled with continuing success until a full scale solution mining operation is achieved.

The best and largest soluble copper deposit upon which to conduct Phase I and Phase II test work is located near Casa Grande, Arizona. This deposit contains over 5,000,000 tons of copper in highly soluble copper minerals. A joint venture of Asarco-Freeport owns a large portion of this copper deposit and is interested in solution copper mining. The joint venture would consider any proposals to allow Phase I and Phase II to be conducted on its property.

It is recommended that the copper industry and/or the U.S. Government find a way to conduct Phase I and Phase II test work. There are basically three ways for the test work to proceed. First is for one company having sufficient financial resources and a sufficiently large soluble copper deposit to justify taking the risk. But, the chances of this happening are highly unlikely, for no one company or joint venture has undertaken this challenge or has indicated the intent to do so. A second way is for a half dozen companies to pool their soluble copper deposits and financial resources in a large joint venture. Unfortunately, such a joint venture is as difficult and risky to organize and operate as is the task of developing a new mining technique. A third way is for the U.S. Government to develop the technology to a point where industry is willing to invest in the utilization of such technology. This point would

be at the successful completion of Phase II. While it will cost $18,000,000 to reach this point, the Government stands to reap the benefits of this expenditure many times over and over. For, if solution copper mining of natural rock is as profitable as it potentially can be, there will be plentiful amounts of profits for taxation. Additional benefits would include a return to profitability in the domestic copper industry, and lower cost copper for the consumer.

File - Isotopic

WLK

Henry G. Kreis
ASARCO Incorporated

June 24, 1985

SOLUTION COPPER MINING

Throughout the twentieth century the American Mining industry has sought and developed lower and lower cost methods of recovering copper from the ground. With the current, severely depressed copper prices, the need for lower cost methods is more critical than ever. The American mining industry needs a radically lower cost method, and it needs it now before such a method is developed and implemented by foreign competition.

Solution mining of copper from soluble copper deposits is such a low cost method, and it has been used to a limited degree in conjunction with conventional mining methods. Solution mining is the only type of domestic copper mining that is profitable at today's severely depressed copper prices. But, for the most part, current day solution mining technology limits its use to extracting soluble copper from rocks broken up but not consumed by conventional mining operations. Thus its present and continued use is strongly dependent on the continuing use of conventional mining methods. For without a continual supply of broken up rock to process, solution copper mining would no longer be feasible using today's technology.

The future of solution copper mining, and perhaps that of the American copper mining industry, rests in developing the technology of applying solution mining to rocks undisturbed by conventional mining methods. That is to say, soluble copper would be extracted from rock in its natural state; and after solution mining ceased, the rock would still be in its natural state less, of course, its former copper content. Developing the technology to solution mine copper from rock in its natural state would free solution copper mining from its dependence on conventional mining techniques that are currently unprofitable. Most importantly, it would allow the development of a large number of soluble copper deposits in the western U.S.A. that are sitting idle because they are not profitable with conventional mining methods, nor are they in a situation that will cause them to be broken up by an adjoining conventional mining operation.

This paper is being written to encourage further interest in the field of solution mining; and, more specifically, to encourage developing the technology to solution mine copper from rock in its natural state. An understanding of the technology and applications of solution copper mining is most easily achieved starting with some of the basic geologic characteristics of a copper deposit.

Geology

The geologic characteristics of a copper deposit are a contributing factor to how the deposit is to be mined. There are, for example, two general classifications of copper minerals that determine how the copper is to be recovered from the rock.

The most important classification, with respect to the topic of this paper, is a group of minerals in which the contained copper metal can be dissolved in water containing 1% acid. The mining industry calls this group of soluble copper minerals--"oxide" minerals. It is from this group of minerals, the oxide minerals, that present day solution mining recovers copper, and this process is called "leaching" by the mining industry. Commonly leached, oxide minerals are chrysocolla, azurite, malachite, atacamite, and chalcocite.

The second copper mineral classification is that of the "sulfide" group. All of the sulfide minerals, with the exception of chalcocite, are insoluble in water containing 1% acid. The most common minerals of this group are chalcopyrite, bornite, and chalcocite. It is from these minerals that conventional mining recovers copper. In the future, technology may find a way to dissolve these minerals so that these minerals too can produce copper by solution mining methods.

Besides the types of copper minerals, there is another geologic feature of importance to solution mining. It is the presence of fractures. Without fractures the copper bearing rocks would be totally impervious to water, and solution mining would be impossible unless the rock was mined and crushed to the size of sand. The fractures allow the ingress-egress of the water solution so that it can make contact with the copper minerals, dissolve the copper, and move the dissolved copper outside the rock to a place where the copper can be recovered from the solution.

All copper bearing rocks are naturally fractured to some degree, and it is this fracturing upon which the future of solution mining rests. Too little is known about natural fracturing in copper deposits, to know if it will allow solution mining of its copper. We do know that rocks fractured, broken, and moved during mining operations readily allow the ingress-egress of leach solutions and, thus, are very amenable to solution mining. Whether or not natural fracturing is suitable for solution copper mining can only be determined by field test work. This will be discussed later in this paper.

Solution Mining Methods

Solution copper mining today is a simple and inexpensive process. It involves putting water with 1% acid on top of copper bearing rock that has been fractured, broken, and moved during conventional mining operations. The solution flows by gravity through the rock pile, dissolving copper as it passes. The copper bearing solutions flow out the bottom of the rock pile where they are collected and sent to a processing plant. The most popular, large-scale processing plant is a solvent extraction-electrowin plant. This type of plant removes the copper from solution and puts it in a directly saleable form of copper, and it does this at low cost.

Adapting this solution copper mining to natural rock, rock not broken by a mining operation, will require developing new techniques of solution application and recovery. Because most copper deposits are 100's of feet beneath the surface of the ground, solution application and recovery would most likely be done with the use of drill holes, say six to twelve inches in diameter. Solutions would be put into the copper bearing rocks with injection drill holes, and the resulting copper bearing solutions would be collected in recovery drill holes.

The actual solution mining process would be as is shown in Figure 1. In this figure the water solution of weak acid leaves the injection drill hole and moves out into the rock through natural fractures, Step A. In Step B the solution comes in contact with the soluble copper minerals (oxide copper minerals), and the copper is dissolved. The solution, now loaded with copper, continues through the fractures, Step C, and collects in the recovery well where it is pumped to the surface of the ground, Step D. In Step E a plant, most likely a solvent extraction-electrowin plant, extracts the copper from solution, and the copper free solution is returned to the injection well, Step F, where it will repeat the process.

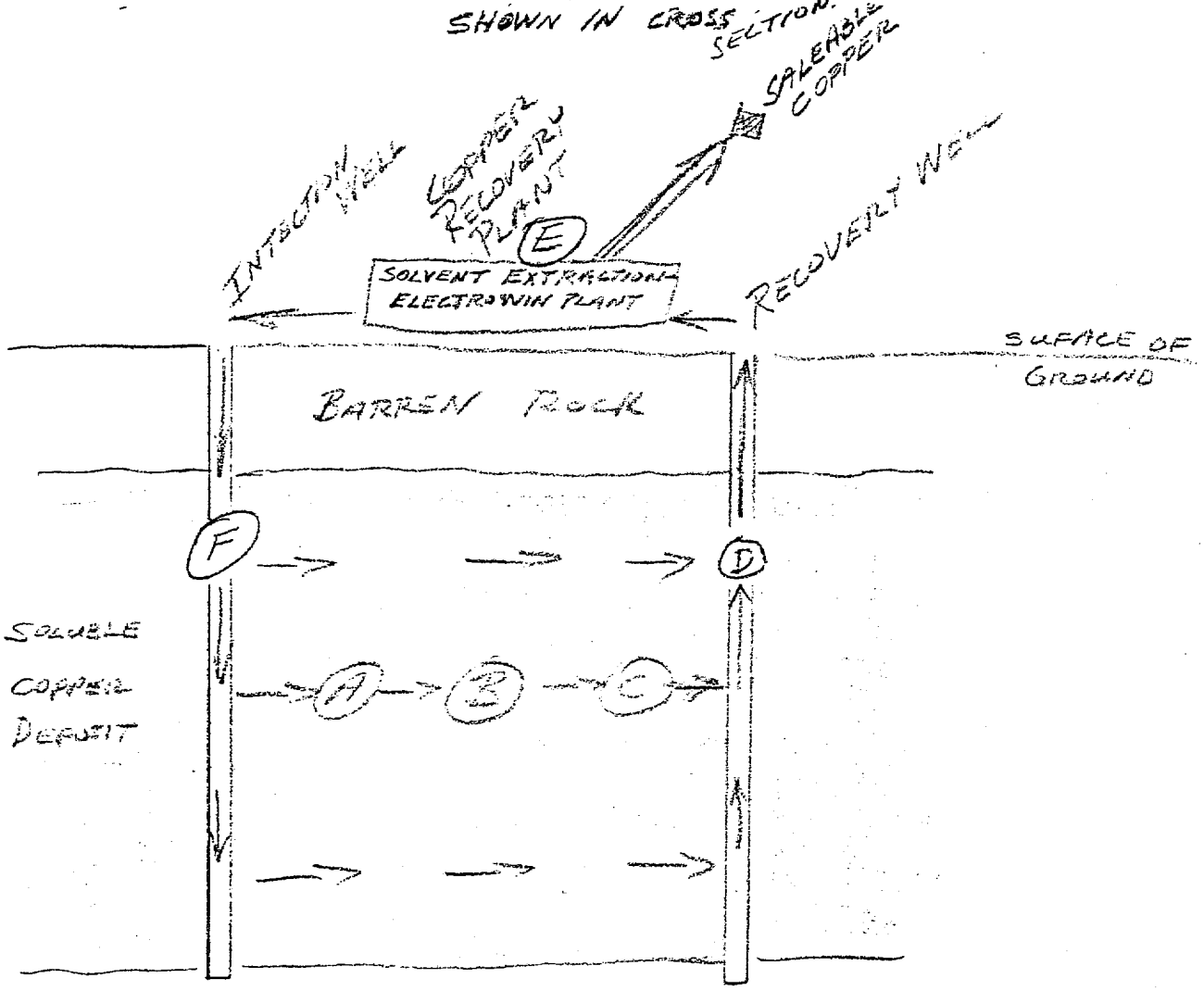
The solution mining process is a relativey simple process, but not all of the steps shown in Figure 1 have been proven to work. Both the copper recovery plant (Step E) and the dissolving of copper (Step B) are proven processes widely used in the mining industry today. While injection and recovery wells have not been used in the commercial production of copper, there is sufficient technology in the minerals and petroleum industry to believe they will work satisfactorily. What remains to be proven is: (1) the fractures are such that they will allow commercial quantities of solution to move through the rock (Steps A and C), and (2) the solution will move through the rock in a sufficient abundance of fractures to make contact with most of the copper minerals. These two factors are very poorly known in our current state of knowledge, and only field testing and pilot plants will demonstrate whether they will work on a commercial basis.

Developing the Technology

The American copper mining industry has made attempts to develop solution mining of copper from rock in its natural state, but these attempts were never carried out with sufficient work to determine whether or not the process is commercially feasible. The attempts were made by large corporations on big, Arizona copper deposits. In one case, at Miami, Arizona, solutions from an injection well moved through the rock dissolving copper and collected in a recovery well. The test was successful and justified plans for a more comprehensive, fifteen million dollar continuation of test work. However, falling copper prices and an adverse land situation caused the termination of test work in 1980.

Since the termination of that test, there has been little or no follow-up work on that deposit or any other American copper deposit. The reason being the depressed copper prices and the flood of imported foreign copper. American companies have been scrambling to get out of the copper business, and those remaining in it, are struggling to survive. Those surviving in the copper mining business will recognize the inherent difficulties of developing a new mining technique such as solution mining copper from natural rock.

FIGURE 1 SOLUTION COPPER MINING
SHOWN IN CROSS SECTION.



STEP (A) Solution from recovery well goes into natural fractures.

(B) Solution contacts copper mineral and dissolves copper.

(C) Copper-bearing solution moves to recovery well.

(D) Copper-bearing solution goes to copper recovery plant.

(E) Copper is recovered in saleable form.

(F) Solution is returned to recycle through steps (A) to (E).

(Rough draft)

One obvious answer to this problem is for a number of companies to pool their soluble copper deposits and some risk capital in a joint venture to develop and operate solution copper mines. But the times are not conducive to initiating such a venture. Alternative answers to this situation would have to include some sort of direct or indirect help from the United States Government. In formulating answers to the problem, industry and/or government will need to know about the step-by-step process involved in developing solution copper mining and the cost of each step.

Development Test Program

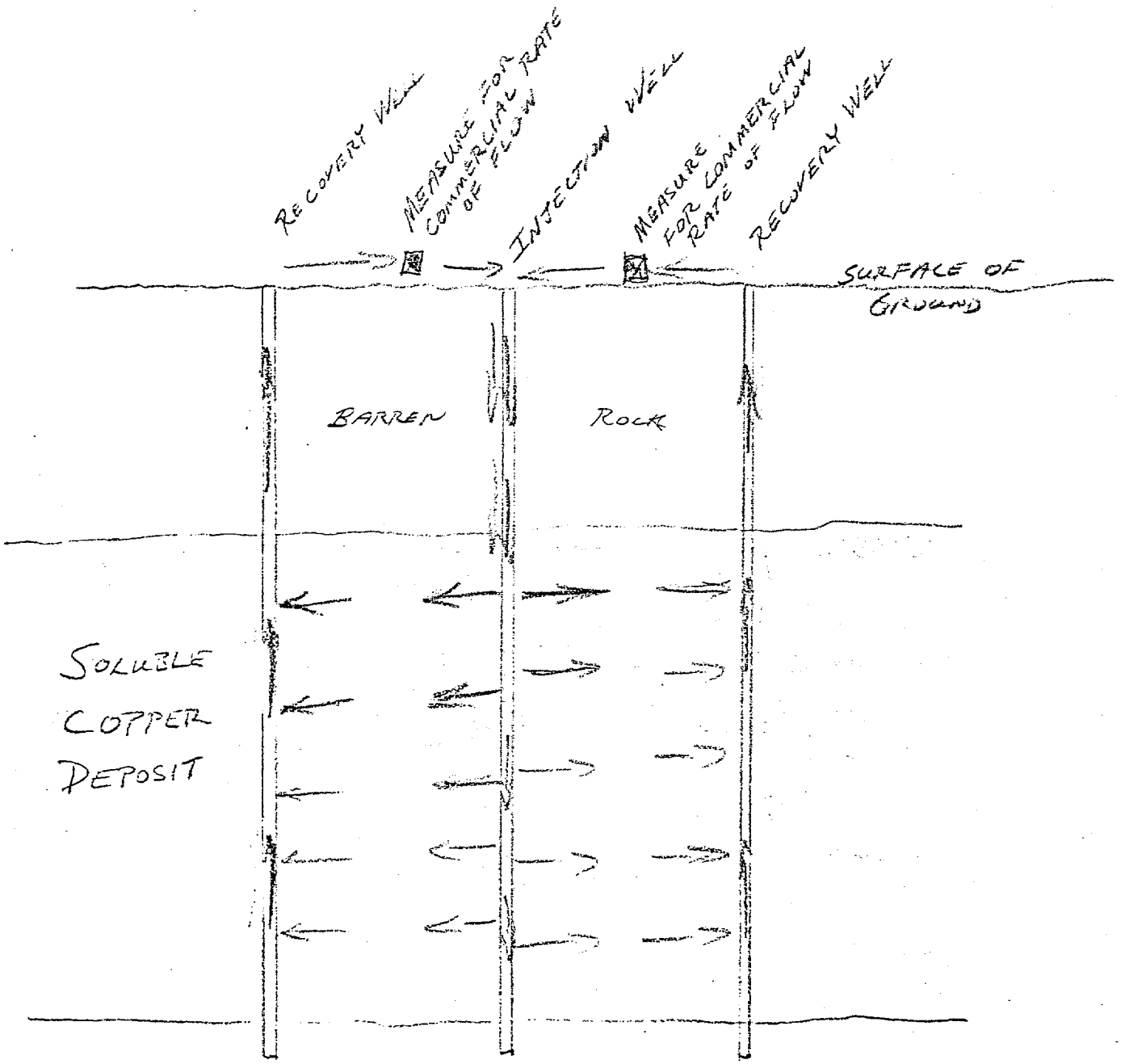
Solution copper mining can only be developed and tested on a soluble copper deposit in the field. There is no way it can be simulated on a computer, done in a laboratory, or done in the ground where a copper deposit does not exist. A suitable soluble copper deposit must be selected. Once this task has been accomplished, test work will proceed in two distinctly separate steps with the success of the first step, Phase I, justifying the second, Phase II.

Phase I will establish that commercial amounts of solution can move through the fractures in the natural rock from the injection well to the recovery well. Once this is accomplished, Phase II will evaluate how well the solution is going through the rock and recovering copper. Both Phase I and Phase II will take five years to complete, and the total cost will be $18,000,000. The completion times and costs of each phase are summarized in Table 1. The actual test setup for each phase is displayed in Figure 2 and Figure 3. Success in Phase II would lead directly into a small scale solution copper operation that would gradually develop into a full scale operation commensurate with the size of the copper deposit.

Table 1. Development Test Program

Phase	Objective	Test Setup	Time	Cost
I	Establish and measure the flow of solution from the injection well to the recovery well.	One injection well and two recovery wells set in copper bearing rocks.	1½ yrs.	$3,000,000
II	Demonstrate ability of solution to recover commercial amounts of copper.	Two injection wells and six recovery wells.	2½ yrs.	$15,000,000
		Total	5 yrs.	$18,000,000

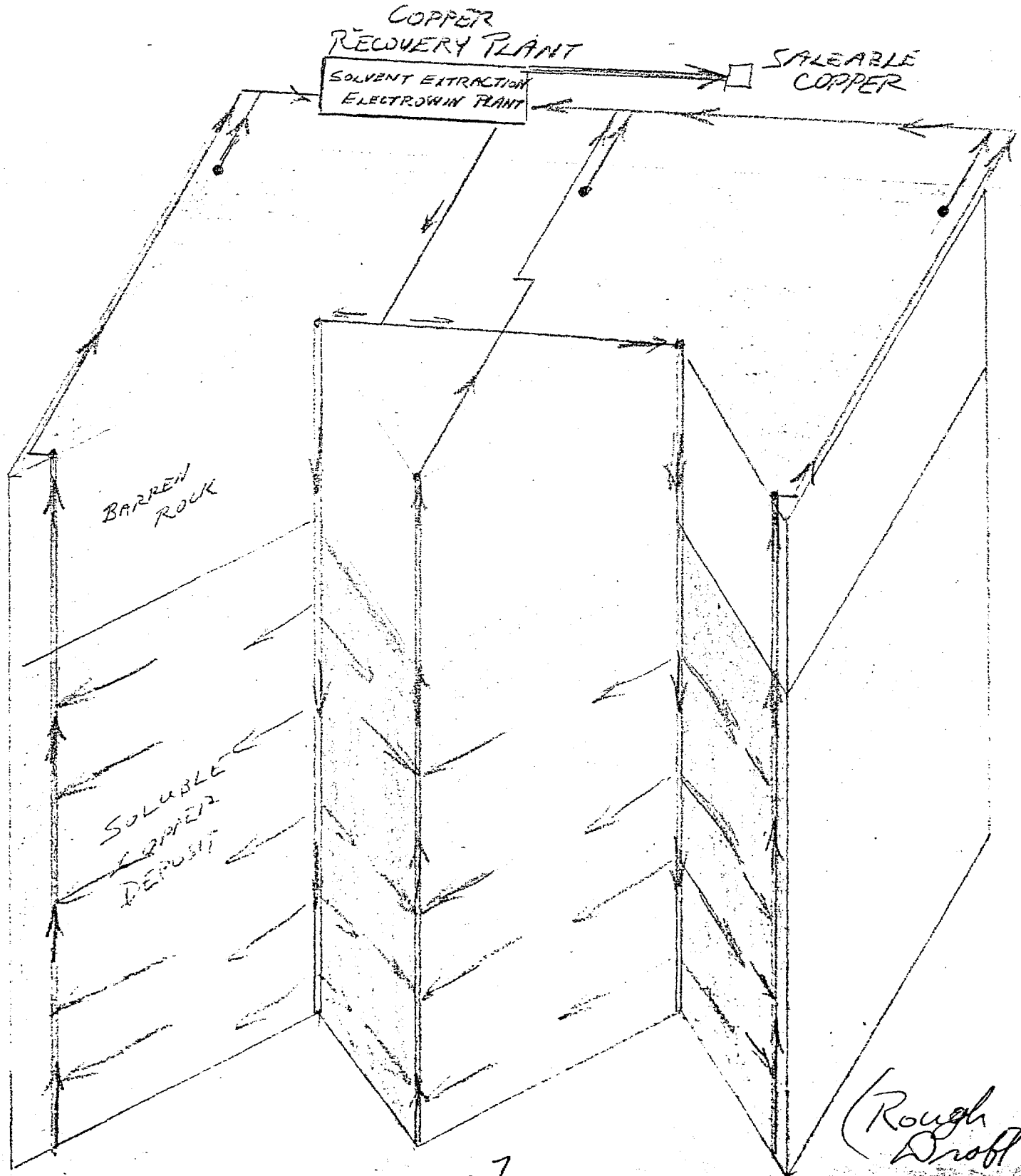
FIGURE 2 PHASE I TEST OF SOLUTION FLOW



(Rough Draft)

FIGURE 3 TEST OF COPPER RECOVERY

• I INJECTION WELL
• R RECOVERY WELL
→ COPPER BEARING SOLUTION
→ WATER WITH 1% ACID



(Rough Draft)

Full Scale Operation

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be at the successful completion of Phase II. While it will cost $18,000,000 to reach this point, the Government stands to reap the benefits of this expenditure many times over and over. For, if solution copper mining of natural rock is as profitable as it potentially can be, there will be plentiful amounts of profits for taxation. Additional benefits would include a return to profitability in the domestic copper industry, and lower cost copper for the consumer.

copy for R C Brown ✓ sent 12/13/85 mek

ASARCO

Southwestern Exploration Division

December 12, 1985

To: J. D. Sell

From: H. G. Kreis

Copper Oxide Reserves

During this period of depressed copper prices, heap leaching coupled with SX-EW copper processing is becoming the only way to produce copper with a respectable profit.

Phelps Dodge's new $35 million SX-EW plant at Tyrone has resulted in substantial production cost reductions, and a $90 million SX-EW plant proposal at Morenci is under consideration. The real eye opener is Newmont's $71 million investment at San Manuel.

At San Manuel, Newmont is starting an open pit mine based solely on heap leaching and SX-EW (memo 12/12/85). There is nothing novel about the operation; the encouraging fact is that it is being done during modern history's most severely depressed copper prices.

These few examples are part of an accelerating trend toward solution copper mining. Coupled with the trend is the USBM's tentative proposal to invest $15 to $20 million in the research and development of in-place leaching of undisturbed, soluble copper reserves.

Considering these events, there is ample reason to believe we should maintain our inventory of soluble copper reserves (namely, Santa Cruz, Sacaton, and Poston Butte) and find or acquire new reserves, such as Hanna-Getty's Casa Grande West Deposit.

The best open pit-heap leach potential reserve that I know of is the Parks-Salyer target on Asarco's Sacaton Mine land. As you will recall the Parks-Salyer target is a potential for 10 to 30 million tons of open pitable ore at 1% to 1.5% Cu (Kreis, 11/29/84). The dominant copper mineralogy is expected to be chalcocite, but there is a reasonable potential for 10 million tons at 1% to 1.5% Cu as atacamite and chrysocolla close to the surface of the ground. This potential could be drill tested for $11,000.

each year we have recommended drilling this.

Conoco's (Dupont) Florence Deposit had an open pit plan that included mining 248 million tons of oxide ore (chrysocolla) at 0.45% Cu (0.23% cutoff grade). Conoco's data should be reviewed to check for an open pitable copper oxide reserve of, say, 100 million tons at 0.55% Cu with a 1.5:1 strip ratio.

H. G. Kreis

HGK:mek

H. G. Kreis

cc: W. L. Kurtz

FROM: W. L. KURTZ

FES 7, 1964
Gonzalez

TO: DECrowell

KNOW OF THIS ?

New York, January 14, 1983

FILE MEMORANDUM

In-Place Leaching
Copper and Precious Metals

Ray Huff (an associate of Don Davidson) called today and advised that CODELCO is moving forward with their test work for in-place leach recovery of copper at Exotica. Huff plans another trip to Chile in February when equipment arrives on site.

Huff has also been closely involved with Bob Schumaker of San Francisco Mining Associates (Joe Wargo's firm) regarding in-place leaching of low-grade gold. Huff has been collecting all relevant legislation which would deal with in-place leaching with cyanide and I advised him that we would appreciate being kept advised of future developments.

Original signed by
F. T. Graybeal

F. T. Graybeal

cc: WLKurtz
JDSell
DMSmith, Jr.
PGVikre
JCBalla

RECEIVED
JAN 20 1983
EXPLORATION DEPARTMENT

ASARCO

File In Situ Leaching

Southwestern Mining Department
T. E. Scartaccini
General Manager

January 12, 1982

Mr. James Muhm, Chairman
Arizona Mining Technical Advisory Group
Occidental Minerals Corporation
Irongate Building # 4
777 South Wadsworth Boulevard
Lakewood, Colorado 80226

Dear Sir:

Your request for cost input in respect to the Arizona Underground Injection Control was received on December 28, 1981. Asarco's Southwestern Mining Department is not involved in the underground injection control regulatory process at this time due to the operation of sulphide plants and an above ground leaching facility on an impermeable surface. However, in discussions with our Exploration Department, it becomes obvious that even though Asarco's Southwestern Mining Department is not involved at present, insitu leaching of underground oxide deposits may develop in the future. As an example, two potential project areas using the following criteria, allow an estimate of costs of these regulations in the future:

Perimeters - 3 miles (min) to 5 miles (max)
Depth of leach - 2,000 feet
Interceptor wells - 500 feet apart
Monitor wells - 500 feet apart

At 3 miles - 32 interceptor and 32 monitor wells are needed.
At 5 miles - 53 interceptor and 53 monitor wells are needed.

Using rotary or cable rigs:

A 2" monitor (PVC lined) @ $10/ft x 2,000 feet = $20,000 each

A 20" interceptor (8" column) (S.S. lined) @ $60/ft x 2,000 feet = $120,000 each

ASARCO Incorporated P. O. Box 5747 Tucson, Az 85703
1150 North 7th Avenue (602) 792-3010

then:

32 monitors @ $20,000 = $640,000 + $320,000
(pumps and lines) = $960,000

32 interceptors @ $120,000 = $3,840,000 + $16,768,000
(pumps and lines) = $20,608,000

53 monitors @ $20,000 = $1,060,000 + $530,000
(pumps and lines) = $1,590,000

53 interceptors @ $120,000 = $6,360,000 + $27,772,000
(pumps and lines) = $34,137,000

Environmental considerations on these projects for the least cost using a 50% factor for contingencies, pipe lines and power is estimated to be $32,400,000; the maximum scenario envisioned would be about $54,000,000.

Please keep in mind that these are very rough estimates based on present day prices.

It has been stated to me that there are presently thought to be twenty two such project areas in the state of Arizona. Using the same suppositions as used above: (22 x 2 miles perimeter @ 2,000 foot depth)

465 monitor wells @ $20,000 = $9,300,000 + $4,650,000 = $13,950,000 x 1.5 =	$ 20,925,000
465 interceptor wells @ $120,000 = $55,800,000 + $243,660,000 = $299,460,000 x 1.5 =	449,190,000
Total estimate for Arizona	$470,115,000

These projects would cost one billion dollars extra in eight years due to the regulations and escalation at 10% per year.

I hope this exercise is of some assistance to you in our participation in the UIC process.

Yours very truly,

Scott L. Burrill

Scott L. Burrill

SLB/kh

cc: TEScartaccini
RBCrist
WDPayne
WLKurtz
JRStringham

ASARCO

Southwestern Exploration Division

WLK file
Lowrey
Leaching

September 2, 1981

TO: W. D. Payne

FROM: H. G. Kreis

Van Dyke Copper Deposit
Pinal County, Arizona

I asked Mr. Don Davidson of TRW about his opinions of the size and grade of the Van Dyke deposit. Mr. Davidson feels that the Van Dyke deposit has a marginal amount of recoverable copper. Ideally a candidate for possible in situ leaching should be of the size of the Santa Cruz deposit or the Safford deposit in order to meet minimum production requirements. Furthermore, Mr. Davidson expresses concern that copper loading may be significantly low in areas of low grade copper (0.2% Cu).

Mr. Davidson says there is no problem with the chemistry of the Van Dyke copper leaching process. Occidental achieved reasonable loading (3 gpl) over a period of time, and there was no problem injecting solutions into the formation. Although no specific solution recovery determinations were made, it was thought that 75% of the injected solutions were recovered. Solution loss may have been related to difficulties with the casing.

Mr. Davidson has set up a computer program to run ROI's on in situ leach proposals. To evaluate the Van Dyke he needs a week's time and some up-to-date operating costs. This could be done at an estimated cost of $5,000. I told Mr. Davidson such an evaluation would be considered if the property looks suggiciently encouraging.

H. G. Kreis

H. G. Kreis

HGK:lb

cc: WLKurtz ✓

June 24, 1980
Tri-State Leaching

1.

Look for In-Situ Leach (In Place Leach)

KCC 1970-79 $30 million objection to S tho lot of work on the oxides

Demonstrated Feasibility of ISL
adequate flow
fluid communication 100-200
1-2 g/L copper loading over 2 yr. period
Petroleum eng. applicable.

ISL Best: deep, low grade, water saturated

Principal approaches — 1. natural flow
2. pressurization — hydraulic — explosive

Sulfides require oxidizing agents.

ISL Technology
Experience — U Texas — in production within 500' surface water well technology
U pilot — Wyo
Colo
Co — Miami Occidental

Economic —
rates fluid injection + production
rate mineral solubilization
volume vs swept by solution
vol. solution processed thru system

Deposit Characterization — Depth, ore grade, total contained metal
or Thickness
permeability
void space
mineral distribution vs. permeable void
Ground water flow

Selection of Solvent

composition
relative metal solubility
interaction with gangue
impact permeability
" construction materials
" surface metal recovery
environmental

Specs for Operation

Effluent metal conc.
wells
well spacing
frequency of well patterns.

Capital Investment

Surface facilities – prod. capacity
vol. sol. processed

wells – depth; flow rate, metal conc.

Pumps – vol solution processed
depth
permeability > pumping head

Startup costs –
operating costs
time to exchange pore fluids
well life
porosity
grade > volume of pore fluid
total plant gpm

place than copper leaching gpd next —
highest not necc. best.

Overall program evaluation

1) economics for site specific
2) Lab experiments, modeling to make predictions.
3) field operations to verify and modify scale up operations.

Surface facility require more than 50% of $

Initial stages – test – do not want to recycle because of time/expense / uncertainties of original leaching.

Exploration holes – can run permeability; can run leach tests.

Sol. min will be at +100 gpm – will need submersible pumps – surface pumps at 3-5,000' will not handle the volume.

need at least 1-2 gpl. → 6 might be the best
from chemistry know will need gas + liquid (for sulfide) to get enough O2 in solution at even 5,000'

1. Porosity – logs from well logging
– issues for between hole communication

2. Porosity High pressure injection test – by not to exceed fracture pressure of rocks.

All test run thus far have had excess O_2 therefore may not have much scrubbing (oxygen) supply being formed.

Not much change in DCFER on gpl (example

3 gpl 17.4 5 gpl 18

10 # acid / lb Co if over bad for costs

50% rec. Livermore test when Co within 2" fracture over one year.

KCC patent 2.5 - 25 md in igneous rocks.

KCC patents provide good literature source.

ASARCO

Southwestern Exploration Division

October 31, 1978

TO: W. L. Kurtz

FROM: F. T. Graybeal

Leaching at the Miami Mine
Gila County, Arizona

The attached report by H. G. Kreis is a carefully researched resume of available literature supplemented by information gained from Mr. J. B. Fletcher, former Leaching Superintendent at the mine. Although I have not checked Mr. Kreis' calculations, his estimates indicate that over roughly a 35-year period the mine has obtained a 64% recovery of mixed copper oxide-chalcocite mineralization from ground previously broken by block caving.

The data are of general interest and may be useful should ASARCO become involved in a similar operation, such as at Sacaton if the underground reserve is ever mined.

F. T. Graybeal

F. T. Graybeal

FTG:lb
Att.

cc: TCOsborne - w/att.
DECrowell - w/att.
SAAnzalone - w/att.
HGKreis - w/o att.

ASARCO

Southwestern Exploration Division

October 24, 1978

TO: F. T. Graybeal

FROM: H. G. Kreis

Leaching of Block Caved Ground
at the Miami Mine
Miami, Arizona

To date the Miami Mine has produced about 360 million pounds of copper by leaching ground broken by former block cave mining operations. The technology involved is very simple and the leaching process operates relatively trouble free. The production costs are said to be low and the profits are rumored to be high. Thanks to Jim Fletcher's publications and communications I have been able to collect a meaningful amount of information on leaching at the Miami Mine. This information is included in the attached report. Similar leaching of underground workings were performed at Ray, Arizona and the Ohio Copper Company mine at Bingham Canyon, Utah, but the technology developed by these operations has been very poorly documented in the literature.

The leaching of oxide-chalcocite copper reserves prepared by underground mining methods probably will be employed at other underground mining operations in the future. Asarco's underground orebody at Sacaton and Hanna-Getty's new deposit are likely candidates. Furthermore, as new copper oxide deposits are found, such as Santa Cruz, there may be sufficiently high-grade portions to justify block cave mine preparation of the reserve for in-place leaching.

The economics of initiating a mine based solely on block cave preparation of leachable copper reserves should be evaluated. For example, given 100-200 million tons of leachable copper reserves, what would the copper grade-thickness versus copper price have to be in order to be profitable?

H. G. Kreis

H. G. Kreis

HGK:lb
Att.

LEACHING OF BLOCK CAVED GROUND
AT THE MIAMI MINE
MIAMI, ARIZONA

Henry G. Kreis

ASARCO Incorporated

October 24, 1978

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Photographs of Miami Leach Operation

INTRODUCTION

The Miami Mine is located at Miami, Arizona and is operated by the Miami Copper Operations (formerly Miami Copper Company) of the Cities Service Company. Underground mining commenced in 1911 and ceased in June 1959 (see Table 1). In-place leaching of block caved ground began in December 1941 and continues to this date. The purpose of this report is to summarize the in-place leaching operation so that other copper oxide and/or leachable chalcocite deposits can be evaluated using similar mining techniques. No attempt is made to report on the technology of the pumping and metallurgical treatment of the leach solution or the economics of the overall operation.

ACKNOWLEDGMENTS

Most of the information in this report must be credited to Jim Fletcher's experience as known from published articles and in personal conversations on a consulting basis. Fletcher was the Chief Mine Engineer of the Miami Mine at the time of its shutdown in 1959 and then became Leaching Superintendent. He retired in about 1970 and has been actively consulting on leaching operations and proposed leach mines since that time. Undoubtedly Fletcher is one of the most experienced people in block cave mining followed by leaching of the block caved ground.

Also FTG Bob Crist, Don Crowell, Tom Henderson, and myself were given a tour of the Miami leach operation in August 1978. They are due thanks for having read and commented on this report while it was in rough draft form.

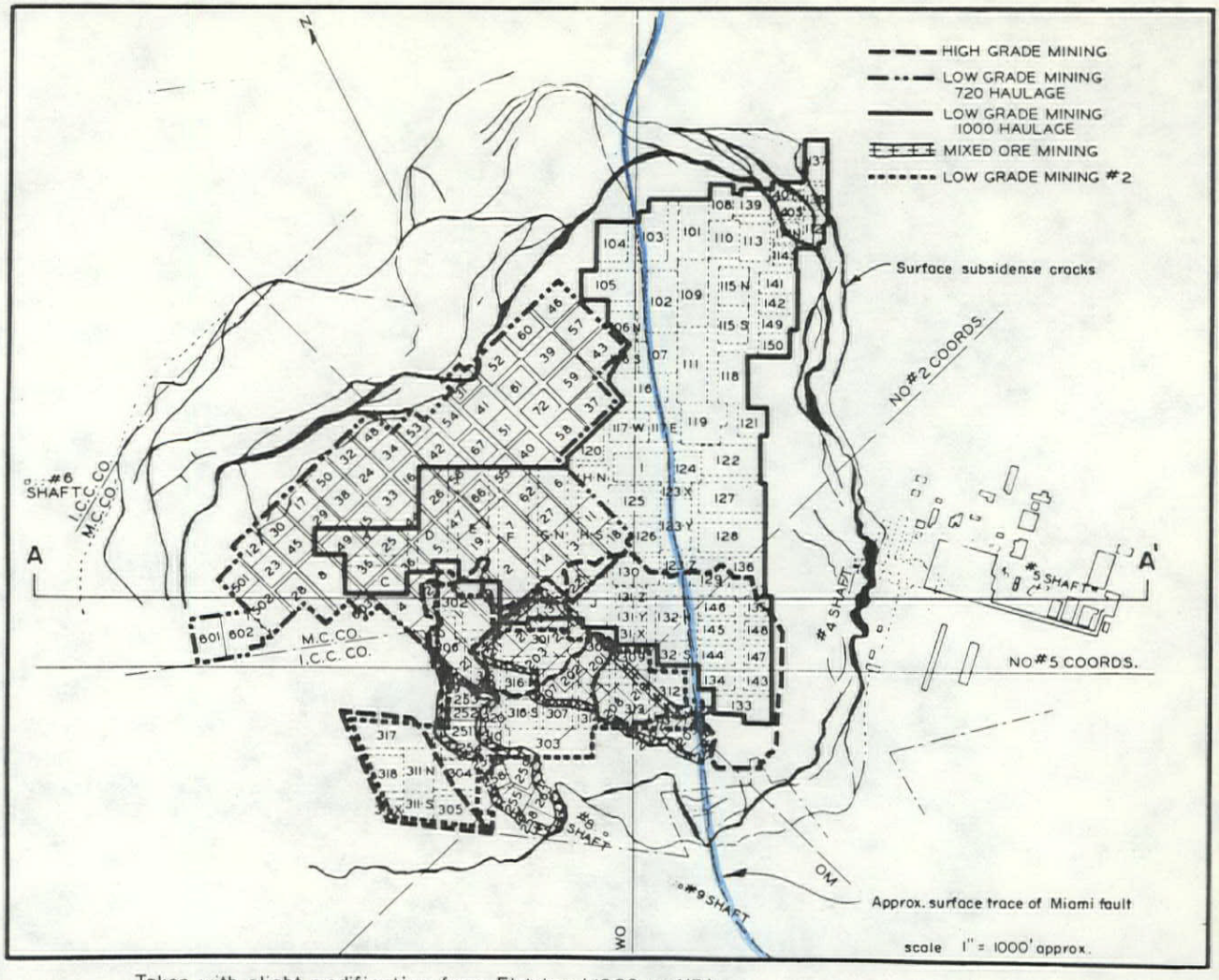
UNDERGROUND MINING OPERATIONS

The underground mining operations are summarized in Table 1 and located in Figure 1. Block cave mining was done using the 720 and 1,000 haulage levels and the Number 5 shaft. The Number 5 shaft, another shaft for ventilation, and the 1,000 haulage level have been maintained and used in conjunction with the leaching of the block caved area. A few years ago the Number 5 shaft was sunk to a depth of 3,450 ft. for eventual mining of the Miami East deposit. deepened

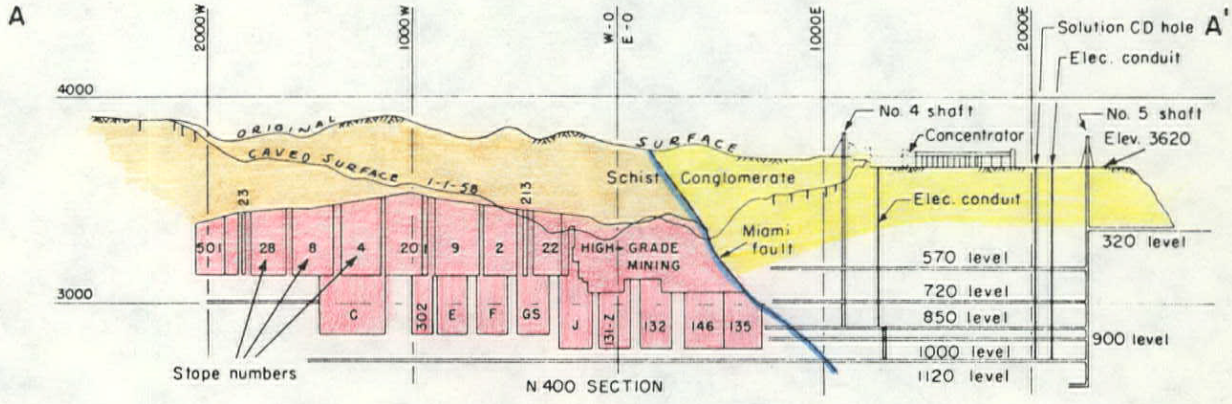
TABLE 1. Summary of Underground Mining Operations

Year	Tonnage	Cu Grade	Mining Method
1911-1925(+?)	24.4mm(1)	Slightly more than 2%(2)	Top slicing, shrinkage stoping, sublevel caving(1)
1925-1959	128.0mm(1)	Approx. 0.80%(3)	Block caving(1)
Total mining	152.4mm	Approx. 1.0%	

(1) Fletcher, 1971, p. 310.
(2) Maclennan (1930, p. 3) reported this grade for 22.1mm tons mined between 1911 and 1925.
(3) Still (1955, p. 90) reported this grade for 108.1mm tons of the 128.0mm tons. In 1930, Maclennan (p. 85) reported a reserve of 108.5mm tons at 0.88% Cu (0.79% as copper sulfide).



Taken with slight modification from Fletcher (1960, p. 413)



Taken from Hardwick (1965, p. 25)

scale 1" = 1000' approx.

Gila Conglomerate

Leached Capping

Mined Ore

FIGURE-1 UNDERGROUND MINING OF MIAMI OREBODY.

mn 5033 dam

GEOLOGY

The Miami copper deposit, prior to mining, was a supergene enriched chalcocite blanket with minor copper oxides on the top. The chalcocite blanket was covered by outcropping leached capping. The host rock was Precambrian Pinal schist with minor Laramide intrusive rocks. Maclennan (1930, p. 4) described the schist as varying from hard, highly silicified schist to soft, kaolinized schist and as free caving rock after being dewatered and dried by ventiliation. In plan view approximately 2 million square feet of the block caved area (totalling 7 million square feet) along the east side is covered by the Miami fault and its hanging wall of Gila conglomerate (see Fig. 1).

Chalcocite was the dominant ore mineral (Fletcher, 1971, and Maclennan, 1930). Chalcocite and chalcocite-coated pyrite occurred in veins, seams, and disseminations and were associated with minor amounts of chalcopyrite, bornite, and covellite (Fletcher, 1971, p. 310).

The copper oxide minerals were chrysocolla, malachite, azurite, and cuprite; and native copper was also present (Fletcher, 1971, p. 310). During the periods 1934 to 1943 and 1957 to 1958 copper oxide mineralization was mined in mixed oxide-sulfide ore and treated by leach-precipitation-flotation recovery (Fletcher, circa 1973, p. 14). Chrysocolla was the most common copper oxide mineral in mixed oxide-sulfide ore mined in 1957 and 1958 (Fletcher, circa 1973, p. 15). Copper oxide mineralization that was not mined occurred over portions of the high-grade ore (pre-1925 mining) and over the 13mm-ton Low Grade #2 orebody (Fletcher, 1971, p. 310).

Originally there was a 500-ft. to 600-ft. vertical thickness of ore to be mined by block caving methods (Still, 1955, p. 89). The ore was overlain by a 250-ft. to 500-ft. vertical thickness of leached capping averaging about 0.03% Cu (Still, 1955, p. 89, and Fletcher, 1971, p. 310). According to Fletcher (1971, p. 311) the leach column of block caved ground as it now exists has an average thickness of 600 ft., consisting of 450 ft. of leached capping underlain by a 150-ft. thickness of copper-bearing rock (to be discussed in detail in the following sections).

BLOCK CAVED GROUND

Except for pillars and crushed pillars the entire 600-ft. leach column has been well broken by block caving. From stope boundary drifts (Fig. 2) Fletcher has observed the process of the block caving of solid rock. He has described (1978) the formation of an open space above the broken ore in a stope as shown in Figure 2. The space is formed by the pulling of broken ore through the finger raises. Above this space a dome (an arch in cross section) is formed at the base of the unbroken rock. Rock spalls off the dome shaped base of unbroken rock. The falling pieces of rock collect on top of a broken rock pile that fills the stope and feeds the finger raises with broken rock. Because the broken rock occupies a greater volume than the unbroken rock, ore must be continually drawn from the stope. This maintains sufficient open space at the base of unbroken rock to allow spalling and the subsequent upward migration of the block caving process. A sufficient thickness of ore at Miami was extracted to allow the block caving process to migrate through the overlying barren leached capping and reach the surface.

The tonnage factor (cubic feet per ton) of the caved ground decreases with depth and approaches that of unbroken ground as shown in Figure 3[1] (Fletcher, 1960, p. 420). The tonnage factor varies from 12.5 cubic feet per ton for unbroken rock to 20.0 cubic feet per ton for broken rock at the surface.
* Fletcher has driven drifts through block caved ground with no difficulty (Fletcher, 1978). Because wet caved ground has a tendency to flow, it is doubtful that one could drift through block caved ground being leached (Fletcher, 1978).

COPPER REMAINING IN BLOCK CAVED AREA

The quantity and the mineral form of copper remaining in the block caved ground have not been evaluated in detail. However, the major reserves of copper in the block caved area have been identified and a reasonable quantitative estimate can be made.

The pillars left during block cave mining represent a major copper reserve. Probably about 310 million pounds of copper were left in pillars having a copper grade of 0.86% (720 level) to 0.75% (1,000 level; Still, 1955, p. 90). The pounds of copper in pillars were calculated by extrapolating Still's pillar data (1955, p. 90) for 108 million tons to the entire 128 mm. tons that was block cave mined (Table 1). Fletcher is certain the pillars are being leached (circa 1972, personal communication).

Extrapolating Still's data on copper losses during mining (1955, p. 90) one can estimate that 74 million pounds of copper was left in the stope areas in addition to the pillar losses discussed above. Most of this copper probably represents copper reserves that became diluted to subeconomic grades during the block cave operation.

Copper oxide mineralization of unknown tonnage but having a grade of about 1% (Fletcher, 1978) occurred over portions of the high-grade ore (pre-1925 mining) and over the 13 mm.-ton Low Grade #2 orebody. This mineralization was never mined as previously discussed. In addition a small mixed oxide-sulfide reserve averaging about 0.9% Cu was probably left in the ground when low copper prices forced the shutdown of the leach-precipitation-flotation plant in 1958. Since the leached capping averages less than 0.03% Cu (Fletcher, 1971, p. 310) it is not considered a possible source of leachable copper.

In 1969 1.3 mm. tons at 0.78% Cu (0.59% Cu as oxide and 0.19% Cu as sulfide), or 21 million pounds of copper, was dumped into the block cave surface depression by open pit mining operations (Fletcher, 1971, p. 312). In the future additional leachable copper reserves will be dumped into the block cave depression (Sosh, 1978).

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1. Some data points may not be technically correct, but the general sense of the graph is acceptable.

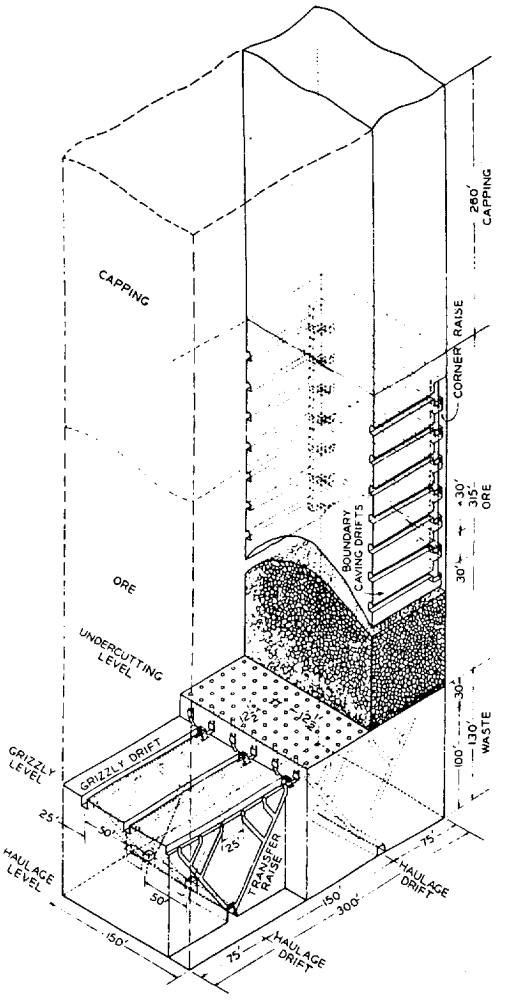
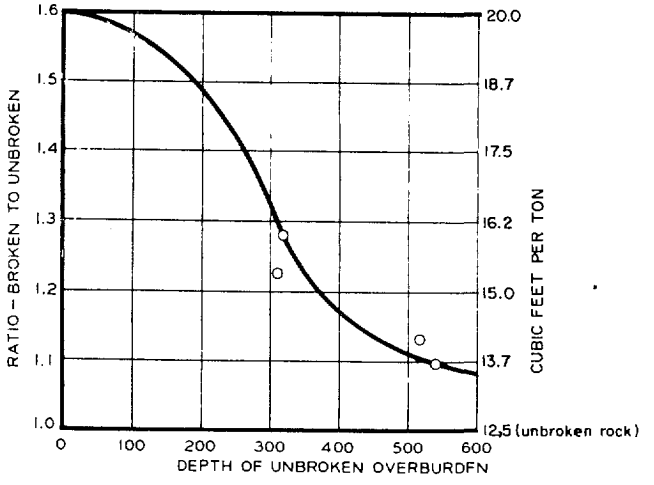


FIGURE · 2
DIAGRAM OF A BLOCK CAVE STOPE IN THE YEAR 1929
Taken from Fletcher, (1960, p. 414)



FIGURE·3 TONNAGE FACTOR VARIATION WITH DEPTH OF BURIAL

Taken from Fletcher, (1960, p. 420)

An estimate of the pounds of copper in the block cave area, prior to leaching and excluding the copper in leached capping, is as follows:

Source	Pounds of Copper (millions)	% of Total
Pillar losses	310 (reasonable estimate)	55%
Other mining losses	74 (reasonable estimate)	13%
Copper oxides not mined	160* (guess)	28%
Open pit leach material	21	4%
Total	565	100%

* - guessed at 5% of total lbs. of Cu mined.

COPPER LEACHING

An acid solution of 6 pounds of sulfuric acid per ton of solution is sprayed on the surface of the block caved area (Sosh, 1978; Fletcher, circa 1973, p. 20, reported 10 pounds of sulfuric acid per ton of solution). The surface area amenable to spray application amounts to 5 million square feet (Fletcher, 1971, p. 311). The acid solution is sprayed from 2" diameter plastic pipe perforated with an ice pick. In 1962 only half of the area had been covered and by 1968 the entire area was covered (Fletcher, circa 1973, p. 20).

Sosh (1978) reported that drill hole application of the leach solution is now becoming more desirable than spray application. In the past drill holes were used to apply leach solution beneath the impervious Gila conglomerate and beneath the solution deflecting Pinto fault. The holes are drilled to a depth of about 200 ft., and some are as deep as 450 ft. (Sosh, 1978; Fletcher, 1978). Recent drill holes are lined with 6" diameter plastic pipe perforated at the bottom, but old drill holes were not cased. Acid solution is fed into the drill hole with 1" diameter plastic pipe. Caving, siltation, or other types of problems do not occur in the drill holes. A 25-ft. radius of effective leaching occurs about the drill holes so the drill holes are spaced on 50-ft. centers (Fletcher, 1978). The radius of effective leaching was determined by Fletcher from his observations of the influx of drill hole applied solution into finger raises 400 ft. and 1,000 ft. below the surface of the ground.

Full-scale leaching operations were initiated in 1959, and apparently the solution was being applied at the rate of 2,000 gallons per minute (gpm) at this time or sometime shortly thereafter. The rate of the application of leach solution was increased to compensate for decreasing head grades. By 1970 a maximum design rate of 2,600 gpm was achieved (Fletcher, 1971, p. 313) and probably maintained until the SX-EW plant was built in 1976. In 1976 the flow rate was probably increased to 3,000 gpm, the present rate of application.

In 1971 the caved ground took 0.052 gpm per 100 square feet of area (2,600 gpm over 5 million square feet) from the spray system (Fletcher, 1971, p. 313). The drill hole application system applies 10 to 20 gpm per drill hole (Sosh, 1978; and Fletcher, 1978) and this calculates to 0.4 to 0.8 gpm per 100 square feet of area about the drill hole. Open pit material dumped into the caved area took sprayed solution at the maximum rate of 0.45 gpm per 100 square feet before ponding occurred (Fletcher, 1971, p. 313).

The leach solution moves through the block caved ground in a uniform, unobstructed manner (Fletcher, 1978). No channeling problems have ever been reported, and there is no evidence of ponding or surging of the leach solution as it moves through the 600-ft.-thick leach column. It takes 3 to 4 weeks for surface applied solution to create a discharge on the 1,000 level; and, if the solution application is stopped, it takes about two weeks for the water on the 1,000 level to taper off (Fletcher, circa 1973, p. 20-21). Fletcher (circa 1973, p. 21) estimates the solutions move through the caved material at the rate of 2 feet per hour.

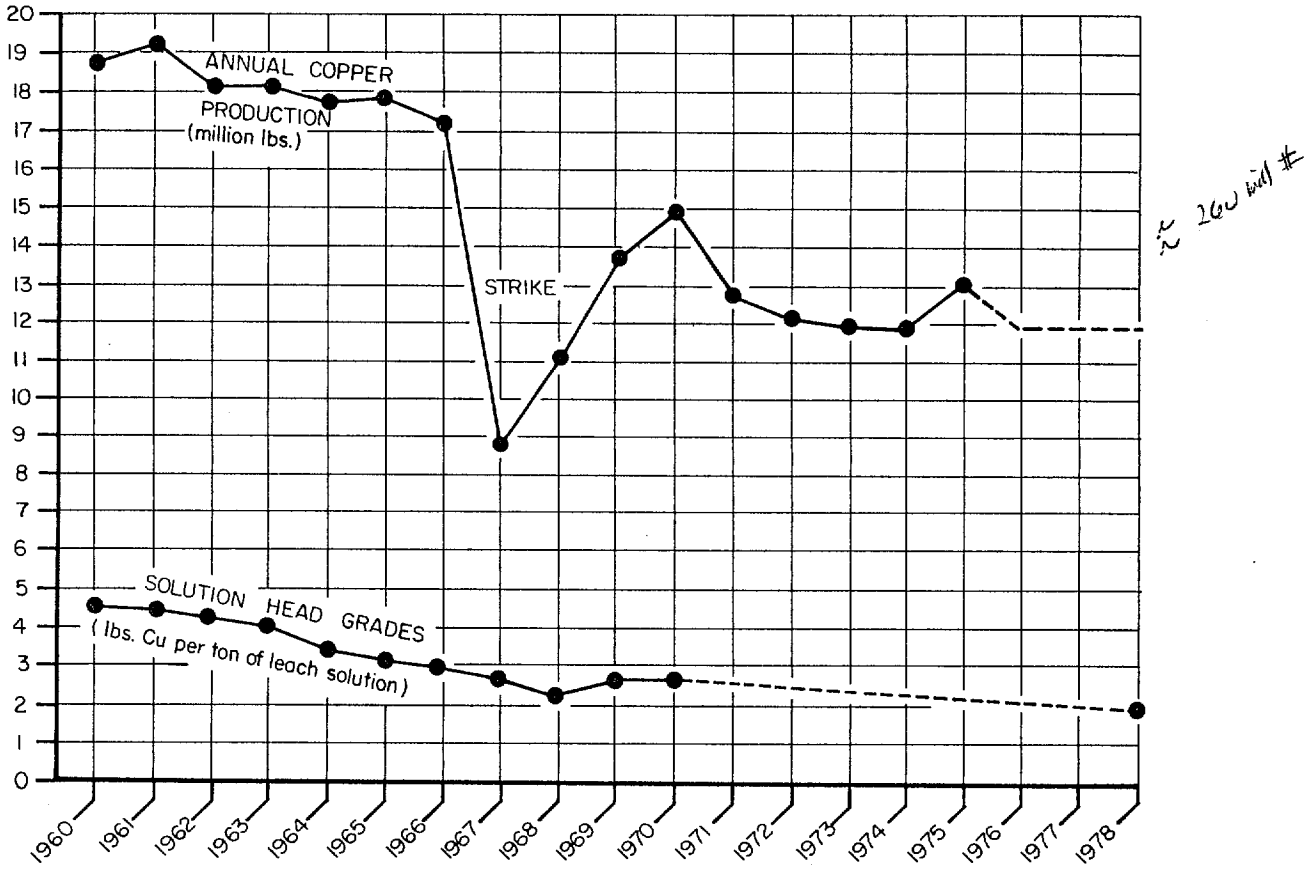
At the bottom of the block caved ground, leach solutions move into the underground workings and are eventually collected by the haulage drifts on the 1,000 level. In 1971 Fletcher reported that there was no solution loss in the ground; however, 10% of the solution was lost to evaporation and make-up water was required. Sosh (1978) stated that presently there is 95% solution recovery with the loss due to evaporation. Furthermore, groundwater (possibly ICC leach solution) is contributing to the extent that a very small part of the tail solution from the SX-EW plant must be disposed of on the tailings pile. Fletcher (1978) said the Miami Mine makes (made?) 50 gpm of natural groundwater. Drill hole application of the leach solution decreases evaporative losses (Fletcher, 1978). Fletcher (1978) drifted in natural rock below block caved ground being leached and found no leach solution in the natural rock.

Most of the 1,000 level haulage drifts serve as a collection reservoir for the leach solutions, and the timbers are periodically replaced to keep the drifts open. The drifts are heavily timbered but there is no problem in keeping them open. Finger raises (and/or transfer raises) are boarded over to keep muck from running into the mine. One side drift was permanently abandoned because a raise was not boarded over and muck ran into the drift. Solutions collecting in the 1,000 level haulage drifts carry no suspended materials so the drift reservoirs are free of silt.

As with all leaching operations the head grades to the plant decrease with time as shown in Figure 4. The head grade has dropped from 4.5 pounds per ton of solution in 1960 to 2 pounds per ton in 1978.

The surface application of leach solution, the pumping of leach solution from underground, and the maintenance of the underground workings and equipment are done by the following work force (Fletcher, 1971, and 1978), i.e., the work force exclusive of the SX-EW plant:

6 men for timbering (3 crews), day shift, 5 days
1 pumpman, day shift, 7 days
2 hoist engineers, day shift, 5 days
1 foreman, day shift, 5 days
3 maintenance men, day shift, 5 days
13 total



Data from a similar graph in Fletcher (1971, p. 313) and Sell (1977, p. 2)

FIGURE·4
HEAD GRADES AND PRODUCTION
AT MIAMI

H.G.K. 10/78

mp 5035 dgm

COPPER RECOVERY

Prior to May 1976 copper was recovered by scrap iron precipitation in a plant designed to produce 50,000 pounds of copper per day from solutions amounting to 2,000 gallons per minute (Fletcher, 1973). Fletcher (1971, p. 311) reported that 1.3 pounds of iron and 2.4 pounds of acid were consumed for each pound of net smelter copper (based on the production of 261 million pounds of copper produced to January 1970). Approximately one pound of iron for each pound of copper recovered was precipitated in the block caved ground; however, no plugging of the dumps was experienced (Fletcher, 1971, p. 312).

In May 1976 a new SX-EW plant was used to recover the copper from solution (Sell, 1977, p. 3), and the iron precipitation plant was phased out sometime between then and 1978. The following description of the plant was from a recent mine tour (Sosh, 1978).

The SX-EW plant was built by Homes-Narver at a cost of $5 million and was designed to produce 32,000 pounds of electrowon copper per day (one million pounds per month). The maximum plant capacity is 3,200 gallons per minute. The electrowon copper product is sold on the COMEX.

Heads to the SX plant presently average 2 pounds of copper per ton of solution (1 gram per liter), and the solution has a pH of 2.3. About 95% of the copper is stripped from the leach solution by the SX process and the resulting pH of the tails is 1.7. The leach tails have 4 pounds of acid per ton of solution and this is raised to about 6 pounds of acid per ton before being dispersed on the caved ground.

Leach solution collected on the 1,000 level and pumped to the surface SX-EW plant has some superior qualities that make the solution more amenable to SX-EW recovery of copper. The leach solution delivered to the plant maintains a uniform 70-71°F temperature year round. Consequently the SX plant is not troubled with liquid separation problems during the colder winter months. Furthermore, the leach solution to the SX plant is free of suspended solids that have caused excessive amounts of scum to form in some of the other operating SX plants. The SX-EW plant has experienced minor scum formation in slightly more than two years of operation. This scum has formed from dust and bugs collecting in open tanks that are not protected by any roofing whatsoever.

The leaching operation is currently operating with a profit. The operating costs were said to be half of that experienced at Pinto Valley (nearby open pit copper mine operated by Cities Service) and were said to be less than 35¢ per pound of copper produced.

The percentage of copper leached from the caved area cannot be determined with a high degree of certainty because the pounds of copper in the leach material are not known beyond the degree of certainty previously discussed. Based on available data Sell (1977, p. 1) reported 338 million pounds of copper was recovered by the very end of 1975. Assuming that 1.9 pounds of copper was recovered from an average of 2,850 gpm since the very end of 1975, one can estimate that a total of 360 million pounds of copper has been

recovered from the caved ground. Realizing approximately 565 million pounds of copper was in the caved ground prior to leaching (see previous discussion) one can estimate a copper recovery of 64% to date. The annual copper production is graphed in Figure 4.

PERSPECTIVE

The following discussion puts the geology and leaching operation into a generalized perspective after the completion of all underground mining and before any leaching has taken place.

As previously estimated, 565 million pounds of copper was left in the block cave area prior to leaching operations. This copper can be thought of as an estimated 35 million tons with an estimated grade of 0.8% Cu. Over an area of 7 million square feet the average thickness of this 35 million tons would be 60 ft. The mining operations, however, have diluted and expanded the 35 million tons to such a degree that it spans a 150-foot thickness and has a tonnage factor of about 14 cubic feet per ton. Fifty-five percent of the copper represents pillar losses, 13% as other mining losses (mostly dilution), 28% as copper oxides not mined, and 4% as leach material dumped onto the surface of the caved area. The copper mineralogy is estimated to be such that two-thirds of the total copper is in the form of chalcocite and one-third is in the form of copper oxides, dominantly chrysocolla.

Overlying the 150-ft. thickness of diluted copper mineralization is an estimated 200 million tons of waste (0.05% copper) having an average thickness of 450 ft. and a tonnage factor of 20 (surface) to 14 (450' depth) cubic ft. per ton. On a tonnage basis the ratio of waste to ore is about 6:1. The entire thickness of the leach column of block caved rock averages 600 ft.

Without considering an optimum production schedule and plant design it is of interest to review the rate of copper recovery from the block caved ground (based on past production records, solution application rates, and head grades) and apply this rate to a full-scale leaching operation of the MCO caved ground as though the ground were being leached for the first time. Solutions would be delivered to a plant at the constant rate of 3,000 gpm. Initially the head grade would be greater than 4.5 pounds of copper per ton of solution. With time the head grade would decrease to 4.5 to 3 to 2 pounds of copper per ton of solution by the time the recovered copper amounted to 100, 200, and 300 million pounds, respectively (18%, 35%, and 53% respective copper recovery). With these figures a non-detailed calculation shows 370 million pounds of copper would be recovered in the first 20 years of operation for an estimated copper recovery of 65%. In comparison MCO has recovered 360 million pounds of copper (64% recovery) over a period of 37 years consisting of 18 years of less than full-scale leaching and 19 years of full-scale leaching with the rate of solutions to the plant increasing with time from 2,000 gpm to 3,000 gpm, its present rate.

CONCLUSION

MCO's leaching of copper from block caved ground is a technologically sound operation, operates relatively trouble free, and apparently produces copper with very low operating costs. The operation has been well tested by time, and the recently constructed SX-EW plant reflects MCO's confidence in the leach operation.

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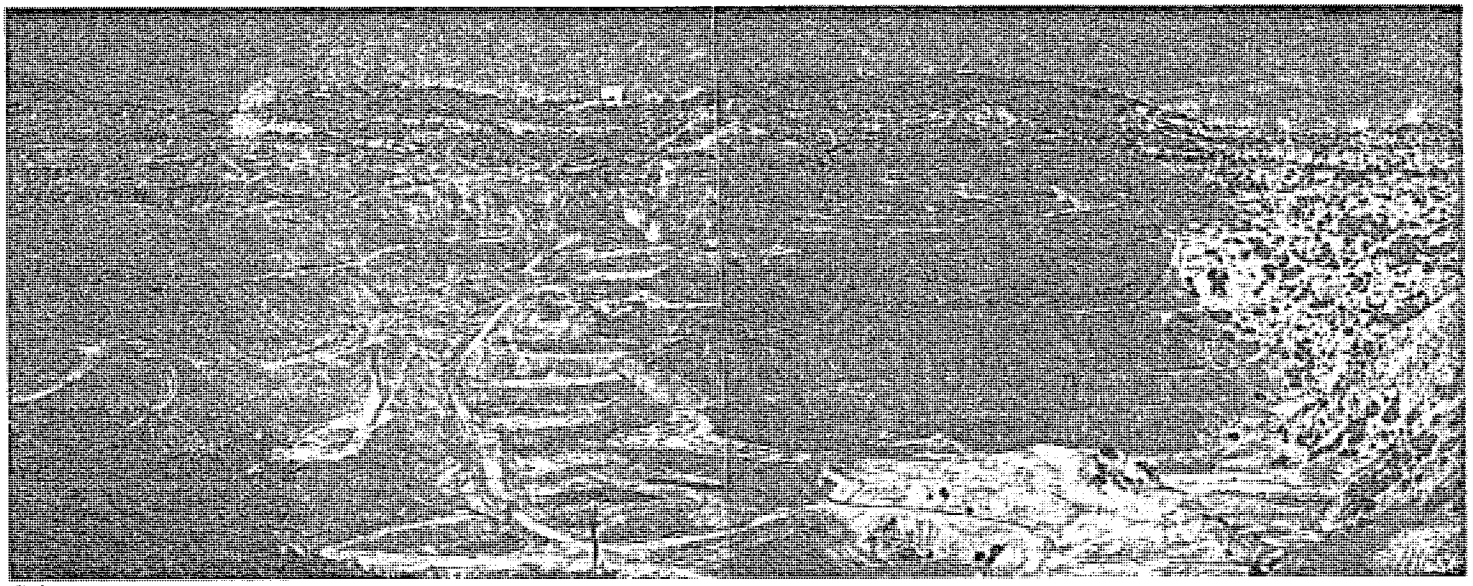
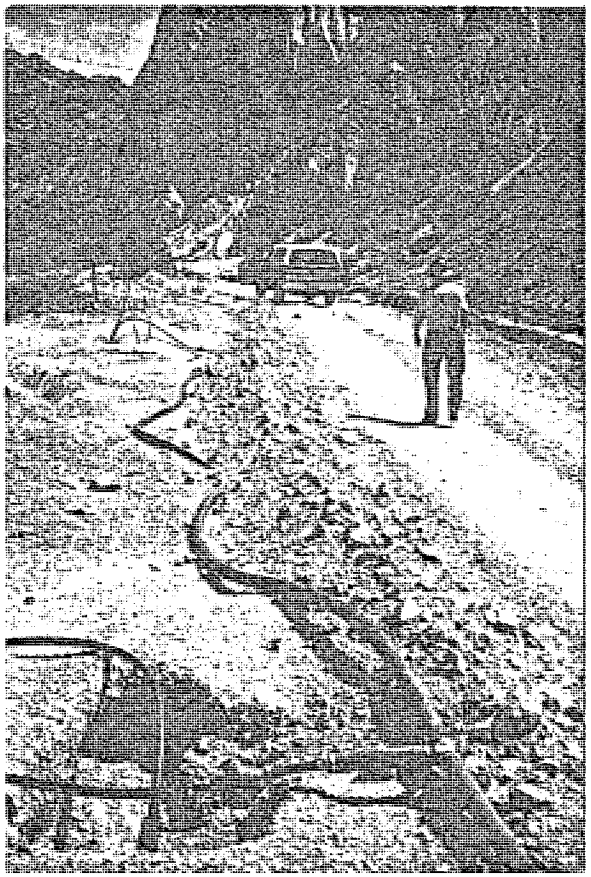
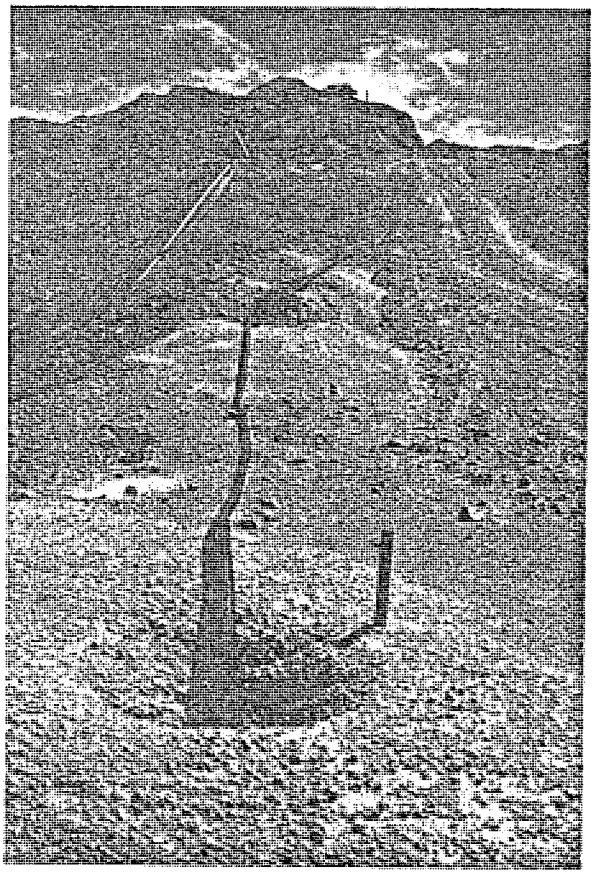
APPENDIX



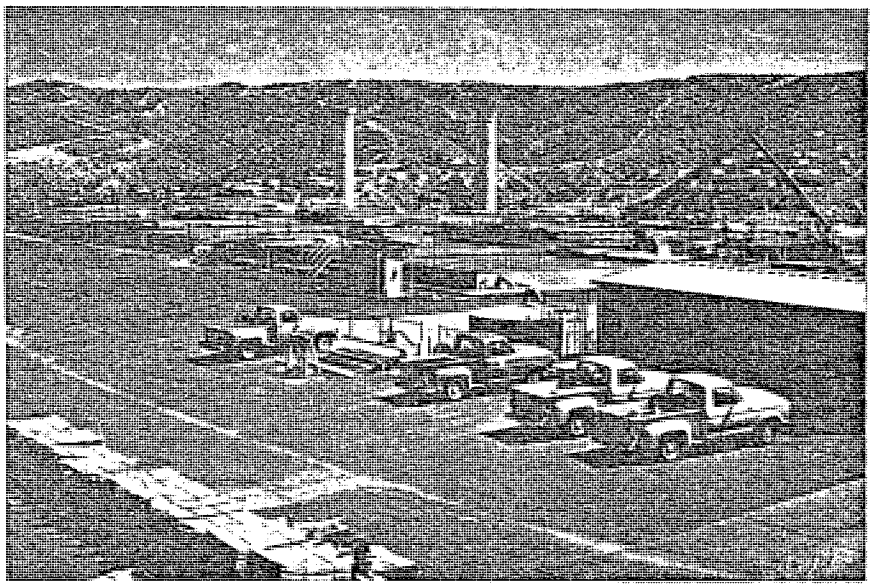
Block caved ground over the Miami orebody (looking NW'ly). The surface of the ground has been terraced for solution application. The red color is leached capping, the gray color (right side of photograph) is Gila Conglomerate, and the Miami Fault trace separates the red and gray colors.





Application of leach solution by drill holes on 50' centers. The solution is gravity fed into the drill hole.

Application of leach solution by the spray method. Gravity fed solution is sprayed from holes made with an ice pick.



Miami Copper Operations SX-EW plant. The EW portion of the plant is just off the right edge of the photograph. The smoke stacks are part of an abandoned power plant.

Oct 23, 1978

FROM: W. L. KURTZ

TO: TC Osborn

RE: Simon D. Strauss
MEMO TO RLH
KENNECOTT LEISEMONT LABS

The in situ copper test at KCC's Safford deposit was on chalcopyrite. I suspect the grade in the test area was probably 0.6% Cu.

cc. DE Crowell.
FT Graybeal

Mr. T. C. Osborne

cc (WLK/PJG) Tucson.

cc Don Crowell Tucson

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file
~~In Site~~ Leaching

AUG 29 1978

Mr. R. L. Hennebach
B U I L D I N G

Dr. J. C. Agarwal of Kennecott's Ledgemont Laboratories was at the Michigan Technological Minerals Symposium last week and flew back with me on the Jet Star.

Dr. Agarwal gave the following interesting information on two Kennecott programs which are more or less on the shelf:

With respect to the exploration work at Safford, Arizona, he stated that about $30,000,000 has been spent so far in drilling and testing of the solution mining in situ of this deposit, calculated at over 600,000,000 tons of 0.4% copper. Two drill holes about 60 feet apart have been put down to more than 3,000 ft. in depth and the solution has been circulated through these two holes resulting in a product containing about 1 lb. of recoverable copper per thousand gallons of solution. Originally economic studies had been made on the basis of a recovery of 6 lbs. of copper per thousand gallons of solution.

Dr. Agarwal now believes the estimates are too high but that if the drill holes had been spaced further apart with consequent longer period of underground percolation, the content of the solution could be increased to 3 lbs. per thousand gallons. He described their studies as indicating that capital and labor costs for solution mining would be distinctly lower than for extraction of the ore through actual mining but that on the other hand, the cost of chemicals would be the chief item of expense and could be quite high.

Because of the market situation Mr. Milliken has put this entire project in the suspense category, even though Kennecott considers this, together with the Phelps Dodge Safford property, to be the major undeveloped copper deposits in the United States.

With respect to Kennecott's deep sea manganese nodule projects, Dr. Agarwal characterized himself as being among the least enthusiastic in the Kennecott management group. He sees at least two serious unresolved problems with respect to the nodules. The first of these has to do with getting the nodules from the ocean bottom up to the mining vessel. It has been assumed that the most interesting of these nodule deposits may average about 10 kilos of nodules per square meter of ocean bottom surface. Dr. Agarwal doubts that more than two kilos per square meter can actually be recovered and due to the uneven topography on the ocean bottom the big question is how much non-mineralized material will also be raised at great expense to the deck of the mining vessel.

The second problem is that of transferring the nodules from the mining vessel to the shuttle vessels which will be transporting the material to the land-based treatment plants. From an engineering standpoint Dr. Agarwal feels that this is a very serious problem on which insufficient work has yet been done. He does not at all accept the concept that mining of the nodules on a commercial scale may begin as early as 1985 -- even if the Congress were to enact legislation protecting U. S. operators who might seek to begin operations prior to international agreement on "The Law of the Sea". Dr. Agarwal also shares my skepticism about the impact of the nodules on the metal markets, assuming the legal and technical problems are resolved.

Thus, as few as two ventures, each processing as much as 3,000,000 tons of nodules a year, could in theory supply most of the world demand for cobalt and manganese. The effect of this on prices for these materials would be so drastic as to greatly distort present estimates of potential revenue.

In sum, Kennecott's work on these two projects appears unlikely to yield returns in the foreseeable future.

S. D. S.

SIMON D. STRAUSS

cc: Messrs. C. F. Barber
R. de J. Osborne
N. Visnes
T. C. Osborne

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2 copy Occi (Oxy min) attach
with file - 1) occidental 2) Intercontinental JHC WKK FIG BJD

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PAY DIRT

A PUBLICATION DEVOTED TO THE INTERESTS OF ALL THE ARIZONA MINE OPERATORS

Number 460 P.O. Drawer 48, Bisbee, Arizona 85603 October 1977

of the Pueblos.

ASARCO

Exploration Department

OK - RCG

while file

USBM interested in coop program at Poston Butte?

June 7th, 1977.

FILE MEMO

COPPER LEACHING, IN SITU

In the past few years with the advent of sharply increased capital and mine operating costs, along with evermore severe environmental restrictions, in situ leaching has received considerable attention. In an attempt to assess actual progress, I have made a review of literature at hand.

Leaching in place was conducted in Hungary during the 15th century. Some 400 years later, copper was extracted by this method from the Eureka mine at Ducktown, Tennessee, and in 1923 at the Ohio mine in Bingham, Utah. Later, it was used in worked out portions of the Ray mine in Arizona, the Rio Tinto mine in Nevada, and in Butte, Montana(1).

In 1941 leaching in the Miami block-cave mine was started and this highly successful operation continues today (mining ceased in 1959). Other more recent operations include: The Emerald Isle, the Old Reliable, Zonia, and the Big Mike which reportedly is back in operation after a brief shutdown. Kennecott's Safford project is still active and Oxymin has started a 2 hole test on the Van Dyke property in Miami(2).

In place leaching operations were undertaken in Japan(3) during the last few years. Reportedly, substantial amounts of copper have been recovered there from low grade (.39% Cu) mineralization (chalcopyrite, chalcocite and pyrite) around mined-out Kuroko-type deposits.

In summary, most in situ leach projects have involved deposits that had been mined to some extent, either by open pit or underground methods. With the exception of a few case history type articles published, the literature reviewed is in general long on theory and short on supporting data.

No bore hole type leach process data appears to have been published on deposits comparable to Poston Butte or Santa Cruz. Thus, appraisal of the financial risk that would be incurred in test work on either deposit must be largely conjectural. However, considering the magnitude of the potential return in the event of success, as well as the Company's position in H_2SO_4, an expenditure of say, $500,000 for a pilot test project is warranted, in my opinion. The gambling odds here are, I believe, equivalent to, or better, than those riding on our average exploration project.

SOUTHWEST USA PROJECTS

Miami Block Cave Leach

Jim Sell has provided a summary of this operation, with published data attached(a), which produced 168,900 tons of copper during the period 1941 through 1975. From 1970 through 1975, the average yearly rate was 6,410 tons. Cement

copper was produced until 1976, when a solvent extraction-electrowinning plant was put on stream.

The copper occurs principally as secondary chalcocite (with very little pyrite) plus minor silicates and oxides in altered schist and porphyry. The average depth of overburden (Gila conglomerate in part) is 450 feet. The "ore" blanket averages 150 feet thick.

The acid-water solution is applied by surface spray; however, experimental work with drilling indicated that a 50 foot diameter zone could be leached through one vertical hole (diameter not given). No iron salt precipitation plugging problems have reportedly been encountered in the leach columns.

Cost data history were not provided, but an estimate was given of production costs for 2,000 tons/year projected to the year 1980, as follows:

Assumptions:	Acid (H_2SO_4)	1.25¢/lb.
	Iron	3.20¢/lb.
Cost:	Acid	2.875¢/lb.Cu
	Iron	6.400¢/lb.Cu
	Fr., Smlt., Ref.	7.000¢/lb.Cu
	Labor, Power, Taxes, etc	27.375¢/lb.Cu
	TOTAL COST -	43.650¢/lb.Cu

Current prices of iron are considerably higher and acid somewhat lower than used in the above estimate, but this would result in only a minor upward adjustment ----- totalling well under 50¢ per pound of copper.

Old Reliable, Copper Creek, Arizona. Ranchers Exploration[b]

Ore body: A near vertical pipe in altered volcanics and granodiorite, containing 4 million tons of .80% Cu (chalcocite, chalcopyrite, malachite, chalcanthite, and chrysocolla). Acid soluble copper in samples tested range from 15% to 60% of total copper.

Fragmentation: 4 million pounds ammonium nitrate detonated in 6,000 feet of coyote tunnel.

Leach solution: Water, sulphuric acid, ferric iron.

Production: (Start December '72) First 10 months cell operation: 4 million pounds Cu. In August '73 plant design of 20,000 pounds Cu/day reached. Plant operated to July '75 ----- 7,300,000 pounds cement Cu, or 5,800,000 lb.Cu total. (Production dropped between June '74 and July '75 when operation stopped ----- only 466,000 lb. cement produced during this period). Reportedly, plugging up by iron salts and clay was the main problem; however, Geo. Van Etten, Manager, stated (via phone) that more fragmentation (by drilling and blasting) was the principal requirement, and that they may resume operations this summer. He also stated that operations at Big Mike, Nevada were re-started in January '77.

Big Mike, Pershing County, Nevada. Ranchers Exploration [c]

After open pit mining of 100,000 tons of high-grade sulphide ore (10% Cu), low grade (±2.0% Cu) mixed oxide and sulphide in pit walls and bottom (475,000 tons) were drilled and blasted. Acid solution was applied to the broken ore by sprinklers and production started at the rate of 250 gpm (2 gms.Cu/litre) from one 180 ft. 12" well in the pit bottom. Pad leach of mixed ore sorted out during mining (300,000 tons crushed to minus 2") started in late 1971.

No data was found on recovery or on the profitability of these operations, but as noted above, they are currently in production.

Emerald Isle Mine, near Kingman, Arizona [d]

Ore body: Chrysocolla in flat-lying, 70' thick bed of Gila conglomerate (copper, probably derived through weathering and leaching of the Mineral Park porphyry deposit (in mountains to the east).

History: El Paso Mining and Milling Company mined open pit 1.4 million tons of 1.0% Cu ore, treated by vat leach and in a leach-precipitate-float plant. Open pit mining ceased in 1973.

In Situ leach: Ore remaining in pit bottom was drilled, blasted and subjected to acid solution leaching with one 9" diameter recovery well. The pregnant solution contained around 0.5 gm/litre. Also, seven 12" diameter holes were drilled on 50 ft. spacing in the pit bottom and acid solutions circulated without blasting. Solutions recovered contained 0.6 gm/litre.

Ore under 200 ft. of overburden was drilled on 20 ft. centers (Phase II) and blasted, but no solution circulation was implemented. The entire operation was abandoned in November 1975.

Zonia Mine, Yavapai County, Arizona (McAlester Fuel Company) [4]

Ore body: Copper carbonates and silicates occur in schist as a zone 300 ft. wide and 1,500 ft. long.

Operated as an open pit with heap leaching from 1966 to 1973. In place leaching was initiated in '73 after drilling and blasting over 1,700 surface drill holes. Production from '66 to March '75 (at shut down) amounted to 17,000 tons of copper contained in the cement product. This represents 6 pounds of copper recovered per ton of rock leached. Separate data for the in situ leach were not provided; however, operating costs, including amortization, to leach in place the 20 million ton reserve of 0.30% Cu (oxide) were estimated at around $.33 per pound of copper contained in cement. Adding freight, smelting and refining, would bring the total to about $.42/lb. This calculation assumes a 30% recovery of the copper over a 10 year period ----- presumably based on results of the '73-'75 in place leaching operation.

Van Dyke Property, Miami, Arizona, Occidental Minerals[5]

Plus 100 M tons 0.55% (80% oxide, 20% chalcocite). Occurs as 100 ft. to 600 ft. thick blanket in schist and porphyry at depths of 1,200 ft. to 1,600 ft. Bench scale leach test recovered 72% of total copper.

Oxymin has recently undertaken a test leach program with two holes, 70 ft. apart, which had been blasted by slurry injection after hydraulic fracturing. Using 600 psi pressure, acid-water circulation has been established between the two holes. The test area is near the Miami fault, on the edge of the drilled orebody, and contains only minor non-oxidized copper minerals.

Safford, Arizona, Kennecott.

An experimental program involving in-place leaching of a chalcopyrite deposit (±0.60% Cu) was announced in 1975[6].

Recently it was reported[7] that Kennecott had completed five, 10" diameter holes (in 1975 or '76) to depths of over 4,000 ft. Solutions of unknown character, but including liquid oxygen, were injected at 400 to 600 psi through one hole and recovered by pumping from the other 4 holes.

A new pilot plant with 20 tons/day copper capacity is being, or has been constructed. Approximately half of the $30 million budgeted has been expended.

DISCUSSION

Leaching the copper remaining in chalcocite enriched deposits after cave type mining operations has been perhaps the most successful, due to (1) the abundance of readily acid soluble minerals, (2) permeability of the rock mass, and (3) an effective solution collection system provided by mine workings.

The deep Sacaton East ore body should be considered a logical candidate for leaching after cave mine operations are completed.

Limited success has been achieved in drilling, blasting (including "Coyote") and leaching of some very shallow, or outcropping oxide copper deposits. Existing reserves of this type are relatively small; however, considerable useful data on fragmentation and solution technology has been accumulated.

Very few leaching tests on moderate to deeply buried deposits ----- which constitute by far the largest reserves ----- have been undertaken, and none of these have progressed beyond the "embryo" stage, except perhaps Kennecott's Safford project, on which no information has been released. Assuming a deposit is composed of readily acid soluble minerals, the critical question to be answered first obviously is: can sufficient solution permeation of the rock be achieved to profitably extract copper? The general attitude of people in the industry is one of skepticism, the concept being that solutions will "channel" through existing fractures and that the required permeability cannot be obtained even by hydraulic fracturing and/or blasting. Also, the generally slow rate of copper recovery in dump leaching tends to influence

thinking. However, the average leach dump which contains around 0.30% Cu with 50% or less acid soluble (in contrast to deposits available for in place leaching, ranging from 0.45% to 1.5% copper with 75% to 100% acid soluble) produces low-cost copper ($.204 reported operating cost per pound of copper in cement at Silver Bell in 1976). But dump (or heap) leaching as an independent operation would be non-economic in most cases due to the low rate of return on capital. An exception is the Bluebird Mine (heap leach) which has operated profitably for the past 9 years on 0.50% Cu heads with 95%, or more, acid soluble.

and the lack of a sulfide orebody to pay for mining the oxide material

Theoretically, less recovery of the copper in a deposit than generally assumed is sufficient for a profitable in situ operation. Preliminary economic studies of Asarco's Poston Butte property in 1973 indicate that 33.5% recovery of the copper in a 175 million ton body of 0.46% Cu over a 15 year period would provide a DFC-ROR of 22% after taxes (50¢ Cu price). This calculation was based on equidistant well spacing (120 ft.), solvent extraction-electrowinning and production of 17,500 tons of cathode copper and consumption of 91,000 tons of acid per year. While the capital cost of this process is much higher than iron-precipitation-cement copper, the cost per pound of cathode copper produced is much lower, and iron salts ----- the principal cause of "plugging"(8) ----- are eliminated rather than added to the stripped solution which is recycled(9).

In respect to the question posed in the foregoing, the chances for success appear reasonably good in enriched porphyry-type deposits that have undergone cave mining. Otherwise, in "virgin" deposits beneath several hundred feet of overburden, the answer can be obtained only by conducting extensive tests by means of injection and recovery wells, using various methods of increasing permeability, such as "sand cracking" and/or blasting. The cost of carrying out an experimental in situ leach test at Poston Butte was estimated to range between $400,000 and $700,000 by Dowell in 1973(10).

In Mr. Graybeal's recent memo(e) on Poston Butte, the cost of a two-well test was estimated at $250,000 by George Laflin (Technical Service Dept. of Dowell). Reportedly, Conoco's solvent extraction-electrowinning pilot plant should be available for use in such a test program.

The partially explored Santa Cruz oxide copper deposit (roughly estimated at 60 to 150 million tons of 0.80% ----- essentially 100% acid soluble) represents a possible alternative for an in place leach test. Using the economic criteria developed on Poston Butte, extraction of 22% of the Santa Cruz copper would be required over a 15 year period for an equivalent outcome (22% ROI) on the larger tonnage, or 55% on the smaller.

J. H. Courtright
J. H. Courtright.

JHC:jlh
c.c. T.C.Osborne, (2)
W.L.Kurtz, F.T.Graybeal
C.W.Campbell, S.A.Anzalone
D.E.Crowell
file

Attachments

(a) Memo on Block Cave Leaching, Miami, 1-13-77, by J. D. Sell, accompanied by J. B. Fletcher's AIME paper, 11-6-70.

(b) In Place Leaching of a Mixed Copper Ore Body, R. L. Longwell. From Solution Mining Symposium, AIME, February 1974.

(c) Surface Blasting Followed by In Situ Leaching, M. H. Ward. From Solution Mining Symposium, AIME, February 1974.

(d) In Situ Copper Leaching Research at the Emerald Isle Mine, D. V. D'Andrea and S. M. Runke. World Mining and Metals Technology, MMIJ-AIME, September 1976.

(e) In-Place Leach Testing, Poston Butte Deposit, File Memo by F. T. Graybeal, 5-25-77.

References

(1) Rosenbaum, J. B., and McKinney, W. A., "Dissolution of Copper Sulphide Minerals from Fractured Ore Bodies", AIME Preprint 70-AS-329, October 1970.

(2) JDS memo to FTG, Oxymin Project, 5-20-77, Asarco file.

(3) Ito, Ichiro, Present Status of Practice and Research Works on In-Place Leaching in Japan. World Mining and Metals Technology, MMIJ-AIME, September 1976, Vol.I.

(4) Zonia Mine, Asarco file.

(5) Occidental, Miami East, WLK to TCO, 11-7-73, and In-Place Leaching, Oxymin Oxide, 4-5-77, Asarco file memo by JDS.

(6) EM & J, page 37, September 1975.

(7) Personal communication, Jim Loughry, Tucson.

(8) St.Peter, A. L., In Situ Mining, A Design for Positive Results. AIME Preprint No.76-AS-36, page 30, February 1976.

(9) Ranchers' Recovers Bluebird Oxide Copper by Solvent Extraction, 1969 April World Mining.

(10) Poston Butte Proposed In Situ Leaching, R. B. Crist. Appendix I, in Poston Butte Project Report, W. L. Kurtz, 2-2-73. Asarco file.

JHC → WLK

ASARCO ATTACHMENT A Southwestern Exploration Division

January 13, 1977

TO: W. L. Kurtz

FROM: J. D. Sell

Block Cave Leaching
Miami Copper Company
(Cities Service Company)
Miami, Arizona

Leaching of copper values from the block caving operations at Miami Copper was initiated in December 1941 and carried on concurrently with cave mining until June 1959 when underground mining was terminated. Since June 1959 increased leaching solutions have been maintained over and adjacent to the block cave area and leaching continued. In 1969 a portion of the Miami ore body that was too low grade to mine by caving was mined by open pit and dumped into the caved area adjacent to the mining area.

Based on available figures, 337,924,007 pounds of copper have been produced by leaching the Miami ore body from 1941 through December 1975. The breakdown for the three periods of operation is shown in Table 1.

Table 1 - Leach Production and Averages Per Year During Operation Periods

Period	Pounds of Copper	Copper Per Year Average (lbs.)
* 1941-Dec. 1959	100,203,817	5,566,879
1960-Dec. 1969	160,796,183	16,079,618
1970-Dec. 1975	76,924,007	12,820,668
1941-Dec. 1975	337,924,007	9,938,941

Yearly production figures are partially available from 1954 through 1975. The figures are recorded in Table 2 and during the block cave mining period also show the tons of ore mined, the pounds of mined copper recovered (or the total), the pounds of leach copper recovered, and the apparent grade of the tons mined.

The production figures are primarily from the reports of the Arizona Department of Mineral Resources. The total production to 1970 is from a paper by J. B. Fletcher (In-Place Leaching at Miami Mine, Miami, Arizona: AIME Tran. vol. 250, p. 310-314, 1971). In a paper by J. W. Still (Block Caving at Miami: Mining Congress Journal, v. 41, no. 4, p. 89-92, 1955), where the block caving is described prior to shut down, he reports the pillars which were left on the 720 level averaged 0.859% copper. None of these pillars were recovered during mining on the level. The 1000 level pillars averaged 0.747% copper and only a minor amount of

* J.W. Still - 1942 - mid '54 28.34 tons/each = 56,68[illegible] pounds

them were recovered during mining. However, stope production was higher than expected and part of the pillars were undoubtedly drawn with the stope. Still also states that of a ten-year average the total tonnage grade was 0.687% with a net sulfide content of 0.601%. Fletcher notes that the plus 1 million tons of ore mined in 1969 and dumped into the caved area and subsequently leached assayed 0.78% total copper, of which 0.59% was oxide copper and 0.19% sulfide copper. The apparent grade as shown in Table 2 for the last six years of block caving operations suggests a grade range of 0.5-0.6% recovered copper.

Table 2 - Production Figures - Miami Block Caving and Leaching

Year	Tons Mined Ore	Pounds Mined Copper	Pounds Leach Copper	Apparent Grade
1954	3,413,914	44,940,161	(Includes M & CD Leach)*	0.66%
1955	3,721,675	33,690,037	5,476,284	0.53
1956	3,821,165	38,564,000	(Includes M Leach)	0.50
1957	3,455,120	40,896,046	(Includes M Leach)	0.59
1958	1,870,865	17,919,616	7,481,126	0.68
1959	998,659	21,229,033	(Includes M Leach)	1.06

Underground block cave terminated 6/26/59
*M - Miami, CD - Castle Dome
Underground leaching continued in the block cave and adjacent areas.

1960	18,930,454
1961	19,102,143
1962	18,077,492
1963	18,195,285
1964	17,757,353
1965	17,905,982
1966	17,168,489
1967	8,726,235
1968	11,076,950
1969	13,755,800
1970	14,965,326
1971	12,806,085
1972	12,170,335
1973	11,987,674
1974	11,968,789
1975	13,075,798

It is of interest to note that single level pulls rarely exceed the expected pull whereas multiple level pulls generally have an overdraw on tonnage and the amount of copper produced, as shown in Still's figure (p. 91) on the mining results of a 97 million ton draw.

Miami has precipitated the copper on shredded tin cans from the inception to the present. During 1976 Miami converted to a solvent extraction system which was placed on stream in May 1976 producing cathode copper.

The two papers by J. W. Still and J. B. Fletcher are attached and provide details into the block caving and in-place leaching. The paper by F. W. MacLennan referred to by Still and covering the earlier development of the block caving system is available in the Asarco library as Technical Publication No. 314 of the AIME.

James D. Sell

James D. Sell

Atts.

cc: FTGraybeal

Chapter 16

IN PLACE LEACHING OF A MIXED COPPER ORE BODY

Ronald L. Longwell

Ranchers Exploration and Development Corporation
Mammoth, Arizona

PROPERTY LOCATION

Located in the Copper Creek area of the Galiuro Mountains, the Old Reliable is on ground first claimed for mineral value during the Civil War. The deposit was mined sporatically from 1880 to 1919 and again in 1953 and 1954, with a total of about 30,000 tons of ore removed.

The ore body is a near vertical breccia pipe containing approximately 4,000,000 tons of 0.80% copper with mineralization occuring as chalcocite, chalcopyrite, malachite, chalcanthite, and chrysocolla. The host rocks are extrusive lavas of cretaceous age which were intruded and mineralized by Copper Creek Granodiorite, which is Laramide in age.

The ore zone, which occurs on the toe of a ridge, is about 300 feet by 600 feet in plan, with the bulk of the deposit extending from the surface to a depth of 500 feet.

DEVELOPMENT AND BLAST

Previous mining operations were conducted on two levels. The bottom level or "200" level is located at the base of the ore zone at 3,750 feet above sea level with the other level or "100" level located 100 feet above. In addition to extending these old workings a third, the "A" level, was established 165 feet above the "100" level. After the development work there existed approximately 8500 feet of 6 foot by 6 foot adits and crosscuts.

Seismic and sample blast tests were conducted prior to final planning of the blast. It was imperative that damage to the ecology be minimal. Three test blasts, ranging from 100 to 4,000 pounds of explosive, were conducted. From these tests it was determined that little if any damage would occur, and this was the actual case. Observers at a point three miles from the blast felt only a minor tremor.

The loading for the blast consisted of the emplacement of approximately 4,000,000 pounds of ammonium nitrate explosives into 6000 feet of coyote tunnel. This operation was designed and assistance provided by E.I. DuPont de Nemours & Co. As the specified areas were filled with explosive, approximately 80,000 cubic feet of sand stemming was blown in place through a stowing machine and by hand stacking sand bags. The detonation was instantaneous on the top level with the middle level delayed 100 milliseconds and the bottom level delayed an additional 150 milliseconds. Initiation was from a firing bunker one mile from the mine. Three days prior to the blast date, the area was cleared of all livestock and personnel except for the arming crew and others directly involved with the blast. Aerial surveillance as well as ground security patrols were maintained until time of the blast.

The blast was detonated on schedule, March 9, 1972, and it appears that it was a complete success. The projected fracture line and the actual fracture line are quite similar. Structures and water flow in the area were unaffected. Surface disturbance, as expected, was minimal with only an area of about 500 feet in diameter being affected. It is believed that a mean rock size of 9 to 10 inches in diameter was achieved in the ore zone.

RECOVERY PLANT

Production from the mine will be achieved by precipitating the copper from a pregnant liquor using shredded iron. This product being generally referred to as cement copper.

The blasted area has been terraced by crawler tractors with the benches about 20 feet high and varying from 15 to 180 feet in width. To provide the quantity of water required to soak a leach area this size, a 17.5 inch diameter well was drilled at a location about six miles from the plant site. Fresh water is pumped to the mine, by utilizing a diesel driven well pump and two booster stations, through a six inch steel pipeline. The water is acidified with sulfuric acid at the plant in the barren solution pond before it is applied to the ore.

The leach liquor is gravity fed to two 200-hp, two stage centrifugal pumps. Each pump provides liquor to a discrete distribution system that services one-half the terraced area. Both systems are comprised of one six inch main transmission line and several two inch distribution lines. Spaced at 40 foot intervals along the distribution lines are rotating sprinkler heads. The sprinklers are plastic bodied with stainless steel pins and springs. The front and rear spray nozzles cover a circular area with a radius of 60 feet.

The pregnant liquor exits the ore body at the base of the hill in the vicinity of the "200" level portal. A concrete core catch dam was constructed down stream in Saloon Wash to collect the pregnant liquor flow and direct it, by means of two 10 inch plastic pipes, to the pregnant liquor pond at the plant site.

In Saloon Wash, above the blasted area, an earthen diversion dam was constructed to channel storm run-off through 600 feet of 72 inch culvert. This culvert carries run-off around the pregnant liquor collection pond and returns it to the natural drainage system below the area of the mine.

From the pregnant liquor pond the copper laden solution flows through a magnetic flow meter into the precipitation cells. There are six cells which have an active volume with dimensions of 24 feet long, 8 feet wide and 4 feet deep. The cells may be operated with all cells in series or in a parallel-series arrangement. The latter method of operation is the one

most used. Cells 1 and 2 operate in parallel with their combined discharge feeding cells 3 and 4, which feed cells 5 and 6. Make-up acid is added to the barren solution in the tails race of the cells to provide adequate mixing prior to discharge into the barren solution pond.

Copper precipitates are washed daily from the cells into a decant sump. The wash water is returned to the feed end of the cells by decanting from the top of the liquid level. The precipitates are then moved from the sump, using a front-end loader, to a concrete drying area where moisture is reduced to about 15% prior to shipment.

METALLURGICAL TEST WORK

In order to determine the amenability of acid leaching of the Old Reliable ore, both Ranchers and Hazen Research conducted separate studies. These tests used both diamond drill samples and bulk samples obtained from the different levels of the mine. In addition, tests were run comparing Old Reliable ore to Ranchers' Bluebird ore. It was felt that the leaching experience available from the Bluebird ore was a good yardstick with which to estimate the rate of recovery from the Old Reliable. These tests indicated that Old Reliable ore is quite amenable if leached under proper conditions. Ferric iron must be present to achieve significant recovery from the ore.

Four different samples were utilized in the test work;

Composite 1: core reject material which assayed 1.03% total and 0.162% acid soluble copper.

Composite 2: run-of mine materials which assayed 1.12% total and 0.601% acid soluble copper.

Composite 3: deep hole core rejects which assayed 2.29% total and 0.06% acid soluble copper.

Composite 4: Bluebird ore which assayed 0.31% total and 0.22% acid soluble copper.

To collect data on the acid consuming characteristics and preliminary data on copper extraction, constant pH agitation leach tests were conducted on composites 1 and 2 for 24 hours.

TABLE 1

Constant pH Agitation Leach Tests, 24 Hours

	Leach pH	%Cu Dissolution	H_2SO_4 Consumed Lbs/Ton	Lbs/Lb Cu
Composite 1.	0.5	25.9	19.7	5.2
	1.0	21.1	9.4	3.7
	1.0	20.9	7.9	3.3
	1.5	17.8	11.9	4.7
Composite 2.	0.5	61.0	8.6	2.1
	1.0	58.0	6.7	2.0
	1.0	57.7	5.6	1.9
	1.5	55.6	5.5	1.9

Composite 1 yielded 18% of the copper at pH 1.5 and 26% at pH 0.5. Composite 2 yielded 56% of the copper at pH 1.5 and 61% at pH 0.5. Although the copper dissolution characteristics of the composites differed markedly, neither consumed much acid. Based on the acid consumed by materials other than copper, consumption in 24 hours was 8 to 20 pounds H_2SO_4 per ton of rock for composite 1, and 6 to 9 pounds H_2SO_4 per ton of rock for composite 2. Both materials, because of their low acid consuming characteristics, appeared suitable for prolonged periods of leach contact, such as would occur during heap, dump, or in-place leaching.

Constant pH agitation leach test at pH 0.5 conducted on composite 3 showed, that without ferric iron, only about 19% of the copper was dissolved in 24 hours.

TABLE 2

Constant pH Agitiation Tests, Composite 3 (24 hrs)

pH	Initial Fe+3-g/l	%Cu Dissolved	H_2SO_4 Consumed Lbs/Ton	Lbs/lb Cu
0.5	0	18.8	40.2	4.7
0.5	6.5	49.2	20.5	1.3
0.5	16.4	52.1	22.6	0.9
2.0	16.4	52.2	0	0

When ferric iron was added during the agitation tests, the percentage dissolution increased to 52% in 24 hours. This occurred whether acid was added or not, indicating an autogenous generation of acid during the dissolution reactions.

Percolation tests were then conducted on composites 1, 3, and 4 to compare the leaching characteristics of Old Reliable ore to that of Bluebird ore. Two columns were run containing material from composite 1, one with acid alone (10 grams per liter) and one with ferric iron added (6.5 grams per liter). Composite 3 was contacted with solution containing about 10 grams per liter H_2SO_4 and 5 grams per liter ferric iron. Composite 4, Bluebird ore, was contacted with 10 grams per liter H_2SO_4.

TABLE 3

Old Reliable Composite 1, Column Test with Acid Only

Time Days	Assay, Gram/Liter Cu	H_2SO_4	% Cu Recovered	H_2SO_4 Lbs/Ton	Consumed Lbs/lb Cu
1	4.93	1.2	13.1	11.1	4.7
2	0.63	6.8	17.3	15.1	4.8
3	0.28	7.8	19.2	18.0	5.2
4	0.24	7.6	20.8	21.2	5.6
5	0.24	7.5	22.3	24.6	6.1
6	0.23	7.5	23.9	27.5	6.4
7	0.21	7.7	25.2	30.4	6.6
8	0.18	8.1	26.4	32.7	6.8
9	0.17	8.2	27.5	34.8	7.0
10	0.16	8.2	18.6	36.8	7.1
12	0.15	8.3	30.6	41.1	7.4
Wash	0.02	1.3	30.8	38.1	6.8

Composite 1, in the absence of ferric iron in solution was, at best, only moderately amenable to acid leaching. With ferric iron present, however, the rate of dissolution and the absolute amount of copper dissolved increased significantly. After 12 days the column in contact with ferric iron solution yielded 63% extraction while the column without ferric iron only yielded 31%.

Composite 3, also indicated 63% extraction after 12 days of contact. Leaching of the column was continued for a total of 27 days at which time 80% of the contained copper was extracted.

Copper recovery from Composite 4, Bluebird ore, was 72% in 8 days. This comparison indicated that, in terms of copper dissolution rate, Bluebird ore was more amenable than Old Reliable. To reach the 70% copper recovery level, for instance, Bluebird required about one-third the time required by Old Reliable.

TABLE 4

Old Reliable Composite 1, Column Test

Time Days	Assay, Gram/Liter Cu	H_2SO_4	Fe+3	% Cu Recovered	H_2SO_4 Consumed Lbs/Ton	Lbs/lb Cu
1	4.93	2.7	1.27	24.9	9.2	1.9
2	1.99	7.5	2.93	37.0	12.3	1.7
3	0.94	7.3	4.10	42.8	14.3	1.7
4	0.64	8.7	4.65	46.7	15.5	1.7
5	0.54	8.7	5.21	49.9	16.9	1.7
6	0.43	8.9	5.12	52.5	18.2	1.7
7	0.35	9.2	5.27	54.5	19.2	1.8
8	0.33	9.3	5.33	56.5	19.8	1.8
10	0.28	9.1	5.33	60.0	22.1	1.8
11	0.25	9.1	5.22	61.5	22.9	1.9
13	0.25	9.7	5.49	64.4	22.9	1.8
15	0.22	9.3	5.51	67.1	25.6	1.9
Wash	0.03	1.6	0.81	67.5	21.2	1.6

TABLE 5

Old Reliable Composite 3, Column Test

Time Days	Assay, Gram/Liter Cu	H_2SO_4	Fe+3	% Cu Recovered	H_2SO_4 Consumed Lbs/Ton	Lbs/lb Cu
1	3.17	4.6	0.00	10.0	10.8	2.5
2	3.12	8.0	0.26	21.4	15.1	1.7
3	3.00	8.8	2.32	33.0	18.4	1.3
4	1.96	9.0	1.98	40.4	21.4	1.2
5	1.13	9.1	3.29	46.5	24.2	1.2
6	1.11	8.9	3.20	50.7	28.0	1.3
7	0.81	9.1	3.79	53.7	30.6	1.3
8	0.66	9.3	3.59	56.0	32.8	1.4
9	0.53	9.2	3.85	58.0	35.2	1.4
10	0.47	9.5	4.03	59.7	37.1	1.5
11	0.49	9.5	4.17	61.6	39.3	1.5
12	0.58	9.4	3.49	63.8	41.6	1.5
13	0.61	9.2	3.31	66.0	44.0	1.6
15	0.36	9.6	4.42	68.7	47.9	1.6
17	0.30	9.8	4.80	70.9	50.9	1.7
20	0.30	9.8	4.87	74.3	55.2	1.7
22	0.19	10.0	4.48	75.8	57.8	1.8
24	0.18	10.0	4.86	77.1	60.6	1.8
27	0.19	10.0	4.84	79.3	63.5	1.9
Wash	0.07			80.0	63.5	1.9

TABLE 6

Bluebird Ore Column Test, Acid Only

Time Days	Assay, Gram/Liter Cu	H_2SO_4	% Cu Recovered	H_2SO_4 Consumed Lbs/Ton	Lbs/lb Cu
1	3.02	0.3	45.8	12.8	4.0
2	0.99	7.3	62.8	16.0	3.6
3	0.20	9.2	66.3	17.0	3.6
4	0.11	9.2	68.2	18.1	3.8
5	0.073	8.9	69.3	19.3	3.9
6	0.057	9.0	70.3	20.4	4.1
7	0.046	9.1	71.1	21.5	4.3
8	0.033	9.1	71.6	22.5	4.4
Wash	0.0033	1.1	71.8	19.9	3.9

Neither ore consumed much acid. With ferric iron present, composite 1 ultimately consumed about 1.6 pounds of H_2SO_4 per pound of copper recovered. Composite 3 ultimately consumed 1.4 pounds of H_2SO_4 per pounds of copper recovered, and composite 4 ultimately consumed 3.9 pounds of H_2SO_4 per pound of copper recovered.

From this data Ranchers proceeded with the project. It is believed that 50 percent of the copper value at Old Reliable can be recovered during a five year period with an acid consumption of no more than two pounds of acid per pound of copper recovered and an iron consumption of 1.5 pounds of iron per pound of copper recovered

OPERATIONAL DATA

On August 23, 1972 the first leach liquor was pumped to the ore. No acid was added during the two week start-up period to minimize the problems associated with the testing of the moderate pressure (300 psi) pumping system.

The leach liquor was acidified to pH 2.0 at the end of the first week in September. Thirty days following the date of initial application, underflow solution appeared in the area of the old "200" level portal. This was the anticipated point of return. The underflow solution had a pH of 6.0 and was precipitating copper hydroxide in the collection area.

The underflow pH remained unchanged until mid-October when the pH of the leach liquor was reduced

to 1.8. Shortly thereafter the underflow pH dropped to 5.5 and remained at that level of acidity. Again the pH of leach solution was lowered, this time to pH 1.5. The acid concentration of the leach liquor has since been maintained at this level. The result has been to achieve an underflow of pH 2.4 with copper concentration as high as 2 grams per liter.

It appeared that soakage of the ore body was achieved by early February, 1973. In the first two-thirds of the month, 97 percent of the liquor applied to the ore was returned to the plant.

Leach liquor is being distributed over an area of 245,900 square feet at the rate of about 1100 gallons per minute. This is equal to about 4.5 gallons per minute per thousand square feet. After one year of leaching there is no ponding on the surface and no evidence of lateral migration of solutions.

Ferric iron first appeared in the underflow solution in April, 1973, and increased to about 0.6 grams per liter in August, 1973. This is evidence of the existance of a bacterial colony in the ore body which will aid in the dissolution of the sulfide portion of the mineralization.

In the first ten months of cell operation approximately 4,000,000 pounds of copper were produced. Acid and iron consumption for the production of this copper was 5.17 pounds of acid per pound, and 1.39 pounds of iron per pound. The acid consumption is expected to decrease with time as the acid consuming gangue materials in the ore body are depleted and acid generation by the bacterial action is initiated.

Copper production increased steadily from the initial production in December, 1972, until the plant design level of 20,000 pounds of copper per day was reached in August, 1973. The above average rainfall during the first winter of operation delayed reaching design capacity much sooner.

CONCLUSION

The process can be fully evaluated only by the ultimate recovery of the copper values. At this time it does appear that the facturing of a suitable ore body by conventional explosives for the purpose of leaching-in-place is a reasonable approach to provide low cost copper recovery. In the case of the Old Reliable the fracturing was sufficient to increase the porosity of the ore to allow good penetration of leach solutions and expose mineral surfaces to contact.

Chapter 17

SURFACE BLASTING FOLLOWED BY IN SITU LEACHING THE BIG MIKE MINE

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INTRODUCTION

Historically, planning a mine involved the economic decision of whether to mine by open-pit methods or underground methods. The method selected was influenced by a number of factors, including size, tenor, and location of the deposit. If the project could not be justified by either underground or open-pit practices, it remained in fallow until economic conditions changed or technology advanced. Technology has advanced and the mining industry now has another option, that of in situ leaching. This method is on the threshold of becoming a commonly used alternative for production in many types of metal and mineral deposits. It will be utilized as the primary method for developing an ore body or as an adjunct to open-pit or underground operations. The following considers application of the technique to a deposit that had been partially exploited by open-pit methods.

HISTORY

The Big Mike Copper Mine is located approximately 30 miles south of Winnemucca, in Pershing County, Nevada. Interest in the property dates from the turn of the century, but little exploration was undertaken until the mid '60s. In 1967 Cerro Corporation acquired the property and initiated a major drilling effort. Ranchers Exploration obtained the property after this exploration program in late 1969 and confirmation drilling was undertaken. The deposit was delineated and it was determined that reserves amounted to about 100,000 tons of 10.0% massive copper sulfide ore and 700,000 tons of 2.0% mixed oxide-sulfide ore. The ore body was a lenticular deposit, dipping at 50°. The massive sulfide mineralization was chalcocite and

chalcopyrite and was surrounded by a halo of the lower-grade mixed oxide-sulfide ore. The deposit was approximately 600 feet long and 300 feet wide and extended to a depth of 300 feet from the outcrop on the surface.

Phase I of the exploitation plan for the orebody involved mining of the high-grade massive sulfide ore and direct shipping to foreign ports. Stripping commenced in January, 1970 and by August over 3,500,000 tons of waste and low-grade ore had been mined along with about 100,000 tons of high-grade sulfide. The objective of this phase was to get the high-grade material to market as rapidly as possible, first, so that pricing could be established in a favorable market and, secondly, so that the advantages of large-volume stripping could be realized. Daily mine production exceeded 40,000 tons at its peak.

Pit design incorporated an over-all slope of 53° with a 12% 45-foot-wide spiralling haul road. Safety benches of 25-foot-widths were spaced throughout the pit, some on a planned basis and others at various intervals as needed. The ore was selectively mined, crushed to minus two inches, and blended to the required grade prior to shipping. The project proceeded as planned, and this portion was considered quite successful.

During the mining of Phase I, some 300,000 tons of the low-grade mixed oxide-sulfide ore was mined and stockpiled. In late 1971 this ore was crushed to minus two inches, stacked on an impermeable asphalt pad, and placed under leach. A precipitation plant, decanting area, and drying pad were constructed and are currently being used to produce cement copper.

PIT ORE

Mining of the 100,000 tons of high-grade sulfide ore was designed to maximize extraction of this material and to minimize the time that the pit would remain active. As a result, about 475,000 tons of the low-grade mixed oxide-sulfide ore was left in the walls and bottom of the pit. After extensive study, the decision was made in mid-1972 to blast this remaining ore into the pit and to leach it in place. Open pit mining had been considered, but it would have required a stripping ratio of 6.5 to 1 and the economic rate of return on such a venture was unacceptable.

Planning

The advantages of blasting and in situ leaching are several: it minimizes development time, disturbance to the environment, and cost. Additionally, it offers an opportunity to exploit lower-grade material and therefore preserves our natural resources. Ranchers gained the benefit of all these factors, but its decision was dictated

primarily by costs. Text work and feasibility studies indicated a profit could probably be realized if in situ leaching was utilized, while the economics of conventional mining and processing projected a loss situation.

The four requirements for successful in situ production were reviewed:

(1) Could the metal be dissolved?
(2) Could the deposit be made permeable?
(3) Could the pregnant solutions be collected?
(4) Could the valuable product, copper, be recovered?

The answers to these questions were not as difficult to obtain as they might be for some deposits. Numerous leach tests had been performed on the material and it appeared to dissolve satisfactorily. Test work had been undertaken by Hazen Research Institute and was confirmed by Ranchers' Bluebird laboratory. It appeared that over 80 percent of the copper could be removed by adding higher quantities of acid (13 pounds per pound of copper). Further testing had indicated that an acceptable recovery of about 70 percent could be obtained by using much lower quantities of acid and maintaining the leach solution at a pH of 2.0. The latter approach appeared most favorable.

Additionally, the low-grade mixed oxide-sulfide ore that was removed during the initial phase had been crushed and was being leached. The recovery from this ore was about as anticipated, so it was apparent that an acceptable amount of copper could be dissolved. The question of what product to produce required no particular study because cement copper was currently being produced and marketed. Facilities for precipitating, decanting, and drying the cement copper were available and additional investment would not be required for this phase of the project.

Pit Blast

The problems of creating permeability in the deposit and collecting the solutions after they had dissolved the valuable product required special study. A coyote blast was considered for fracturing the ore body but surface drilling and loading were selected, primarily because of the steep-wall configuration of the pit. The Du Pont Company was requested to provide assistance in planning and executing the blast. Hopefully, the material could be broken to a size averaging less than 12 inches. Study of explosive factors and pit terrain indicated that the desired fracturing could best be obtained by utilizing large closely-spaced angle and vertical blast holes. The pit was de-watered prior to drilling. The walls were scaled and all loose material pushed into the bottom. The backfilling aided in drying up the pit floor and provided a flat accessible area for drilling.

Approximately 21,000 feet of 5 3/4-inch, 9-inch, and 9 7/8-inch-blast hole drilling was required. Drilling was performed by a contractor using a rotary cyclone PH 60 top-head drive angle-hole rig plus a similar rig without the angle-hole capability. About 60 percent of the drilling was angle-hole of 5 3/4-inch and 9-inch diameter with the angles ranging from 3° to 35° off vertical. The drill pattern varied with holes drilled on a 20-foot by 23-foot spacing for the 9 7/8-inch and 9-inch holes, and 10-foot by 10-foot spacing for the 5 3/4-inch holes. Hole depths ranged from 40 feet to 200 feet. Penetration rates of the rotary rigs averaged 11.7 feet per hour for the entire job. A total of 1800 feet of 9 7/8-inch hole were drilled in the bottom of the pit, using a standard churn drill with an average penetration rate of 5.3 feet per hour.

The bottom of the pit yielded about five gallons of water per minute, and while a minor quantity, it required the use of a high-density water-resistant explosive or water-gel. Holes above the water table were primarily loaded with ammonium nitrate but were supplemented by the heavier density water-gels in areas of high rock burden. Drill cuttings were used for stemming. The loaded holes were connected with primacord for an instantaneous blast initiated at the hole bottom by electric caps. All holes were double primed and multi-feeder lines of primacord were strung into the pit to eliminate any chance of a mis-fire. About 400,000 pounds of explosive were used to fracture approximately 600,000 tons of material, giving a ratio of 1.5 pounds of explosive per ton. The blasting station was located approximately 2,000 feet from blast center. The blast area was cordoned off and aerial surveillance used to clear the area. Two-way communication was established between air and the blasting station, and the countdown did not commence until the area was safe. The blast was executed on schedule without incident. Broken material filled the pit bottom to approximately 50 feet from its original level, as indicated in Figure 1. Fracturing was considered excellent, having an average rock size of about 9 inches.

Solution Collection

Control and recovery of the pregnant leach solution is an important consideration in any leaching operation. A water table was present near the bottom of the Big Mike pit, and it was thought that this plus tight pit walls would act as a barrier to the percolating liquors. The consolidated rock had a porosity which ranged from 5 to 20 percent, but permeability of this material was quite low. Secondary permeability associated with faults and fractures was generally greater than the primary permeability. Study indicated that the hydraulic gradient in the vicinity of the mine was fairly high, but as noted earlier, rate of water movement was slow due to the low average permeability. The height of the water in the pit during inactive periods was constant at a certain level, and it was thought that after the water reached this level, it exited laterally through the structure. Plans called for controlling ground water contamina-

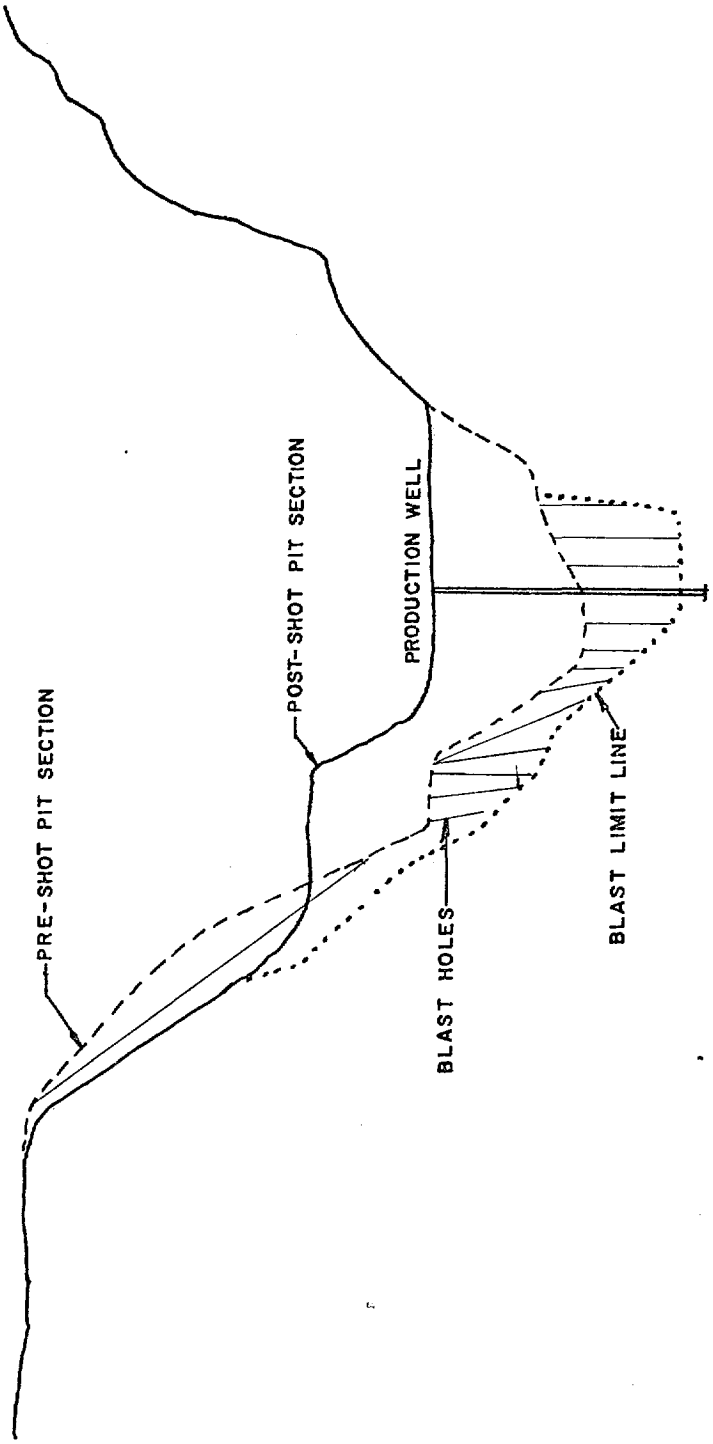

FIGURE I- PIT CROSS-SECTION BEFORE & AFTER BLAST

tion and loss of pregnant solution by monitoring the fluid level in the pit. The saturated level in the pit is maintained at a level below the original water level and this establishes a gradient of fluid flow toward, rather than away from, the pit. Several monitoring wells were drilled near the pit along the most likely avanue of lateral water loss. Surveillance of these wells has indicated that no loss or contamination is taking place.

Distribution System

After blasting, a new access road was cut into the pit, and terracing of four large benches of varying dimensions was initiated. All terraces were leveled and ripped in preparation for receiving leach solution. Distribution of such solutions is through sprinklerheads attached to two-inch PVC plastic branch lines. These pipes are connected to a six-inch header line that receives acid liquor from the barren solution pond adjacent to the precipitation plant 3000 feet away. Leach solution acid strength is maintained at a pH of 1.5 to 2.0, and this is accomplished by adding concentrated acid to the barren solution storage pond. A single-stage stainless steel centrifugal pump picks up the solution from the storage pond and transports approximately 200 gallons per minute to the pit distribution system (Fig. 2). The leach liquor is sprinkled on the broken ore and percolates by gravity to the main collection well pump located 160 feet below the surface of the rubble pile in the pit bottom.

Collection Well

Solution recovery involves one production well in the pit bottom. Several methods for establishing a collection point were considered since penetration of the broken rubble was expected to be difficult. A conventional churn drill well was selected, which appeared to offer the greatest chance for success at the lowest cost. The well was drilled from the lower terrace to a depth of about 180 feet. A steel casing 16 inches in diameter was inserted as the hole was drilled. The purpose of this casing was to assure no movement of the broken rubble until the permanent casing was installed. After the drilling was completed, this shell was perforated for 50 feet off the bottom of the hole, as shown in Figure 3. The permanent 1/4-inch wall, 12½ inches inside-diameter stainless steel casing was set to a depth of 160 feet. A 40-horsepower 10-stage stainless steel vertical turbine pump was then set inside the casing for pumping the pregnant leach solution back to the precipitation plant. Electric probes were installed in the well for automatic shutdown and start-up purposes.

Production

In an effort to expedite the project, sprinkling of the broken ore was started prior to completing the drilling of the recovery well. This assured a saturated orebody and immediate production once the

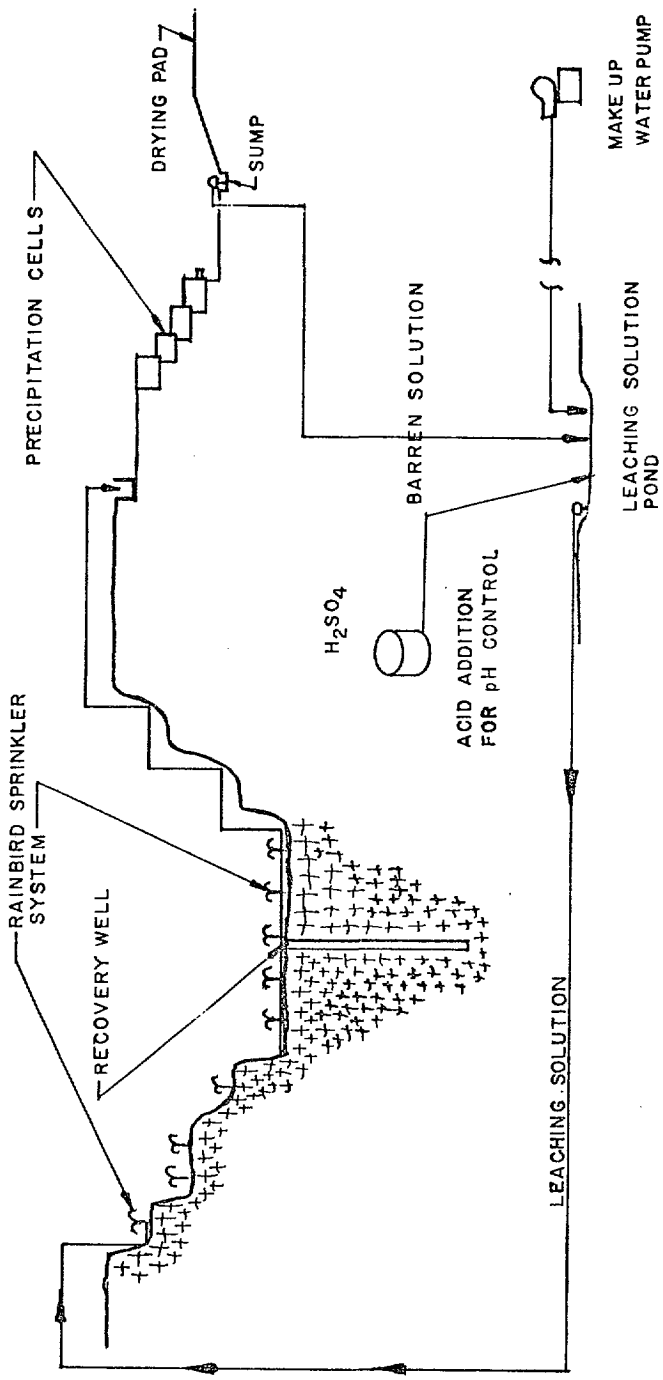

FIGURE 2 – FLOWSHEET-PIT LEACH OPERATION

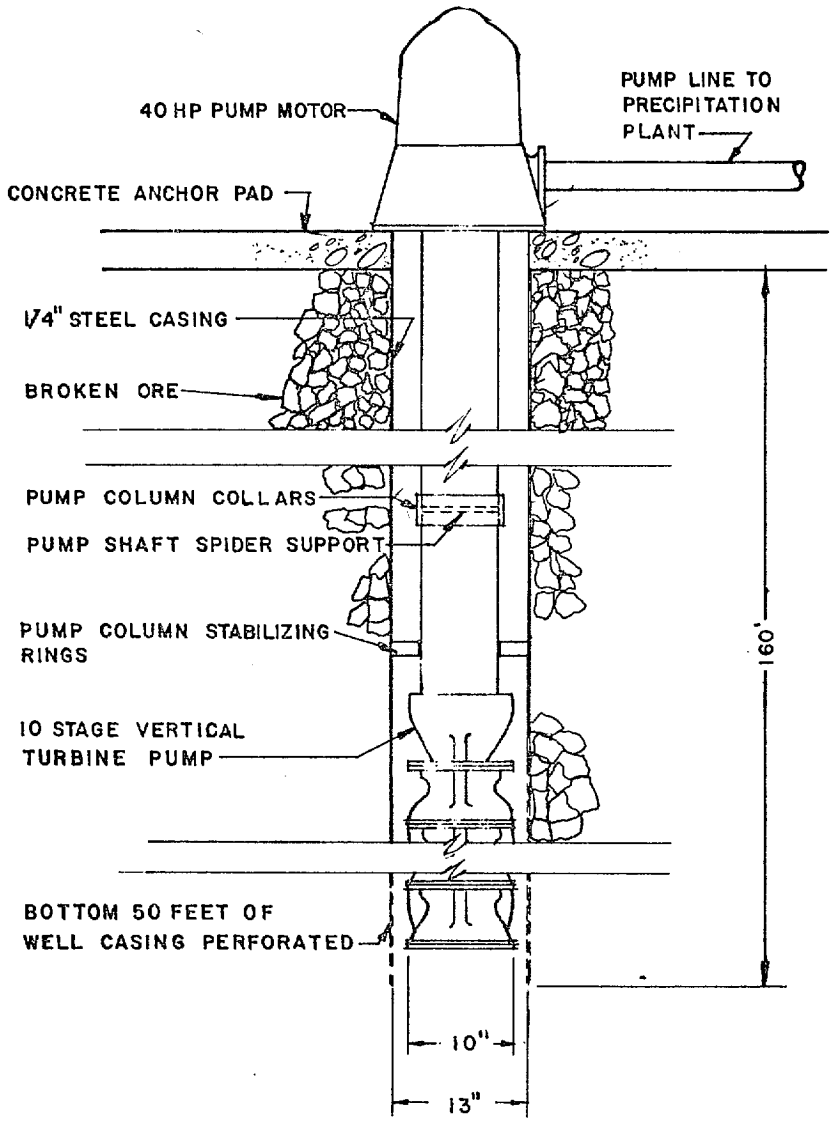

FIGURE 3- RECOVERY WELL- PUMP COLUMN & CASING

recovery well pump was made operational. Pumping commenced about mid-October, 1973 at a rate of 250 gallons per minute of 2.0 grams per liter copper. It was anticipated that production for the first year would average 200 gallons per minute at two grams per liter, and it appears that this goal will be attained.

The pregnant copper-bearing liquors are pumped from the recovery well to a standard iron launder precipitation plant. Cement copper is washed from the concrete cells into a decant area. The copper is then placed on a concrete drying pad where it is turned periodically until the moisture is reduced to an acceptable level. The marketable product, cement copper, is trucked to the railhead in Winnemucca and then shipped by rail to a Nevada smelter.

Project Costs

The cost for a project similar to the Big Mike pit blast and leach operation will depend on a number of items and would be different for each application. The existing facilities and utilities, the physical layout, the type of rock, and the location of the deposit will greatly influence the economic viability of the venture. Ranchers already had a leaching facility adjacent to the pit and, therefore, did not have to invest in a water source, precipitation plant, or office and laboratory facilities. The following cost figures do not cover Ranchers' entire costs in the project, but they do represent costs for particular areas.

Item	Cost
Dozer Work (before and after blast)	$ 16,500
Blast Hole Drilling (21,000 feet)	106,500
Explosives (water gel and nitrate)	69,500
Recovery Well (drilling and casing)	21,500
Recovery Pump, column and controls	29,500
Distribution System and Misc. Costs	56,500
	$300,000

Copper recovery from the available ore at this project, as in any in situ leach operation, is uncertain. It appears that the effort was a success, but only time will yield specific answers. Fortunately, a recovery of less than 10 percent was all that was required for the project to pay out.

Chapter 24

IN SITU COPPER LEACHING RESEARCH AT THE EMERALD ISLE MINE

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ABSTRACT

This paper describes the Bureau of Mines-El Paso Mining and Milling Company cooperative research program at the Emerald Isle mine[3] near Kingman, Ariz. This research program was directed toward developing in situ leaching methods for 200,000 tons of ore exposed in the pit bottom and also 1,500,000 tons of ore under 200 feet of overburden adjacent to the pit. A test area in the pit bottom containing 15,000 tons of ore was blasted and leached. The results of this leach test were encouraging, and the entire pit bottom was leached in-place for about 7 months. Two test blasts under 200 feet of overburden and extending to 290 feet were detonated in an area near the open pit. The results of these blasts were evaluated by core drilling and permeability measurements. An in situ leaching system was designed, but not implemented, to recover copper from the 1,500,000-ton area near the pit. The company decided to terminate operations at the Emerald Isle mine in November 1975.

INTRODUCTION

The Emerald Isle mine located 15 miles northwest of Kingman, Ariz., is owned by El Paso Mining and Milling Company. This copper deposit (principally chrysocolla) was owned by several companies over the

[3] The work upon which this report is based was done under a cooperative agreement between the Bureau of Mines, U.S. Department of the Interior, and the El Paso Mining and Milling Company, El Paso, Tex.

years and was mined by underground and open pit methods. The ore has been heap leached and most recently treated in a leach-precipitate-float plant. El Paso has mined 1,400,000 tons of 1-percent-copper ore since it acquired the property. Approximately 200,000 tons of 1-percent-copper ore remains exposed in the bottom of the pit, and about 1,500,000 tons of 0.5-percent ore remains under 180 to 250 feet of overburden.

Open pit mining at the Emerald Isle mine ceased in July 1973 because increased stripping requirements made continued operation uneconomical. El Paso decided to develop an in situ leaching system to recover as much of the remaining copper as possible. The Bureau of Mines entered into a cooperative agreement with El Paso Mining and Milling Company in November 1973 to conduct research at the mine related to the development of this system. Both the Twin Cities Mining Research Center and the Salt Lake City Metallurgy Research Center of the Bureau of Mines became involved in this cooperative program. El Paso agreed to pay for all parts of the program that were required for the actual production of copper such as blasthole drilling, explosives, solution distribution and recovery systems, acid, and scrap iron, etc. The Bureau conducted core drilling, drill core analysis, physical property measurements, ground water studies, permeability measurements, laboratory leaching tests, seismic measurements, and assisted El Paso with the design of the blast and the in situ leaching system.

A research program for the Emerald Isle mine was prepared. Phase I of the research program began in January 1974 with a test blast followed by in-place leaching of 15,000 tons of ore in the pit bottom. The information developed from this Phase I test was used to design an in situ leaching system for the remaining 200,000 tons of ore exposed in the pit bottom. Leaching of this ore began in December 1974 and continued for about 7 months. A Phase II test site with again about 15,000 tons of ore was selected in an area under 200 feet of overburden. A Phase II test blast was detonated in July 1974 with seven 280-foot-deep blastholes. Circulation tests with water showed that this blast did not create sufficient permeability for leaching. In August 1975 a second blast was detonated in the Phase II test area with three blastholes. The effects of this second blast were evaluated using core drilling and permeability techniques. In November 1975 El Paso Mining and Milling Company decided to terminate operations at the Emerald Isle mine, and plans to leach the Phase II test area were abandoned.

This paper presents the research program at the Emerald Isle Mine and describes an in situ leaching system which was designed to recover copper from a 1,500,000-ton area under 180 to 250 feet of overburden.

EMERALD ISLE DEPOSIT

Figure 1 is a cross section through the Emerald Isle mine showing the Phase I and Phase II test areas. The deepest portions of the open pit have been excavated to about 200 feet exposing the top of the ore body (Gila Conglomerate). The Gila Conglomerate averages about 70 feet thick and dips approximately 10° to 15° to the southwest. The Gila Conglomerate is overlain by alluvium (quaternary) which is 150 to 250 feet thick and underlain by a granite gneiss. The dominant copper mineral in the Gila Conglomerate at the Emerald Isle mine is chrysocolla.

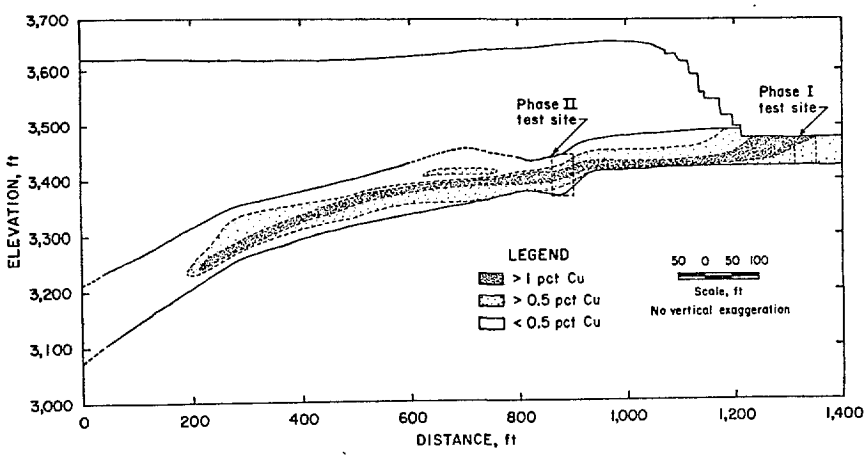


Figure 1. - Cross section through Emerald Isle deposit

Figure 2 is a plan view of the Emerald Isle mine with contours of the copper grade of the Gila Conglomerate. During open pit operation El Paso mined 1,400,000 tons of ore that averaged 1.0 percent copper. The copper mineralization continues beyond the pit in the downdip direction in a channel type distribution that decreases in grade downdip. At about 1,000 feet from the pit crest the grade is about 0.1 percent. Total ore reserves greater than 0.1 percent copper are estimated to be 3,000,000 tons.

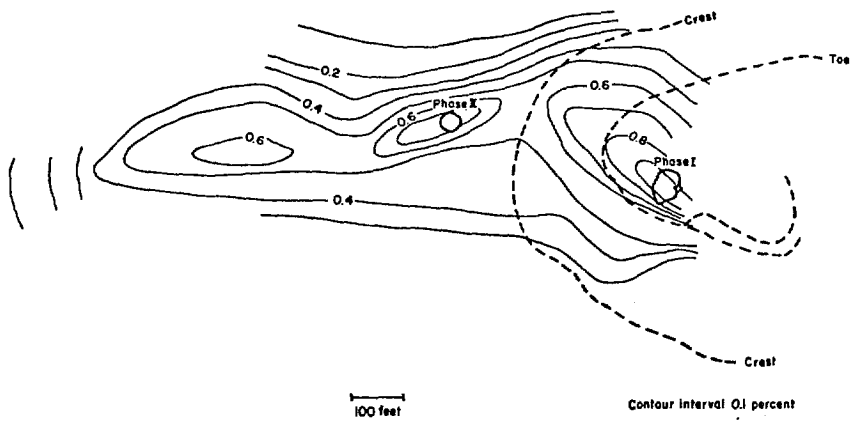


Figure 2. - Contour map of copper grade

Table 1 lists the physical properties of Gila Conglomerate core

TABLE 1. - Physical properties of the Gila Conglomerate

Property	Phase I	Phase II
Porosity.....pct..	20.6	16.3
Density.....g/cu cm..	2.29	2.28
Permeability.....darcy..	-	.65
Pulse Velocity.....ft/sec..	8,100	9,500
Torsional Velocity.....do..	4,200	5,100
Compressive Strength.....psi..	2,600	-
Tensile Strength.....psi..	72	-
Young's Modulus.....10^6 psi..	0.85	-

samples measured in the laboratory. The porosity, permeability, and strength values probably reflect the characteristics of the cementing material in the conglomerate and not the actual in situ values.

Ground water entering the pit, about 14 gpm, was pumped from a sump area for about 30 hours each week. The ground water table was controlled by sump pumping at about 5 feet below the surface of the pit floor.

PHASE I TEST BLAST

Figure 3 shows the Phase I test blast design. Seven 8-3/8-inch-diameter blastholes, about 50 feet deep were spaced 25 feet apart in a seven-spot pattern with one central blasthole. These blastholes had an average 22-foot powder column with 25 feet of stemming. A total of 4,500 pounds of slurry was detonated without delays. Table 2 lists the blast design data for the Phase I blast and the two Phase II blasts.

TABLE 2. - Blast design data

	Phase I	Phase II 1st Blast	Phase II 2nd Blast
Number of blastholes.....	7	7	3
Blasthole spacing.....ft..	25	20	18
Average hole depth.....do..	47	277	277
Average top powder column.do..	25	205	192
Average powder column.....do..	22	72	85
Average stemming.....do..	25	205	192 (90 ft gravel)
Blasthole diameter.....in..	8-3/8	9	9
Explosive diameter.....do..	8-3/8	7	7-in bags cut
Explosive.....	nonaluminized slurry	smokeless powder slurry	smokeless powder slurry
Total explosives.....lb..	4,500	12,000	7,450
Loading density.....lb/ft..	29.2	23.7	29.3
Powder factor[1].....lb/ton..	0.30	-	-
Powder factor[2].....do..	0.78	0.95	1.47
Delays (between each hole)....	Instantaneous	17 ms	25 ms

[1] Powder factor (lb/ton) includes ore above top of powder column

[2] Powder factor (lb/ton) = $1814 \frac{Pe}{Pr} \left(\frac{D}{S}\right)^2$

Pe = specific gravity, explosive
Pr = specific gravity, rock
D = blasthole diameter, ft, and
S = blasthole spacing, ft, for infinite triangular pattern, ore zone only.

for ore in powder column zone only

Topographic surveys were run before and after the Phase I blast so that the blast swell could be calculated. The maximum surface rise was only about 1.5 feet, and the blast swell was 1.4 percent. The volume of ore broken by this blast was estimated to be 15,000 tons.

PHASE II TEST BLASTS

Two blasts were detonated in the Phase II test area under 200 feet of overburden. The first blast had seven blastholes drilled to an average depth of 277 feet. These holes were 9 inches in diameter and 20 feet apart in a seven-spot pattern. Figure 4 shows the blasthole

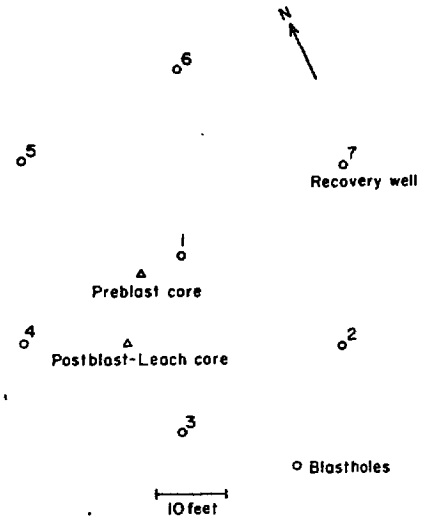


Figure 3. - Phase I test blast design

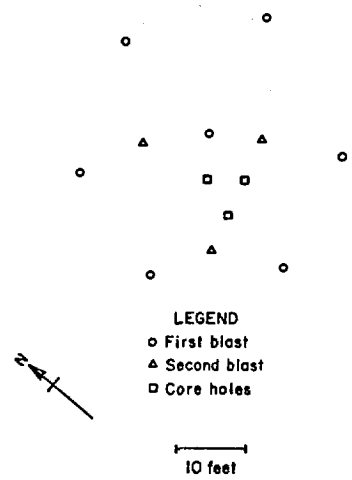


Figure 4. - Phase II test blast design

patterns for the two Phase II blasts. The second blast had three blastholes, 9 inches in diameter, 277 feet deep, and they were spaced 18 feet apart. Table 2 lists the blast design data for the Phase II blasts.

The first blast was loaded with 7-inch-diameter, 50-pound-bags, of smokeless powder slurry in 9-inch-diameter blastholes. The second blast was loaded with the same explosive, but the bags were cut before they were lowered into 9-inch holes. The cutting procedure increased the loading density from 23.7 lb/ft for the first blast to 29.3 lb/ft for the second.

The second blast in the Phase II area was considered necessary after circulation tests with water revealed that the first blast had not created sufficient permeability for successful leaching. After the first blast a recovery well pump was installed at a depth of 290 feet, and drawdown tests were conducted with and without water injection at the 200-foot level. Flow rates from these tests averaged less than 1 gpm over a 24-hour period and were not considered adequate for leaching. In addition the water injected at the 200-foot level did not work its way into the fractured zone and into the recovery well but apparently flowed across the top of the alluvium-conglomerate contact. The water table in the Phase II area was at a depth of 230 feet.

After the second blast the recovery well pump would have been reinstalled and water circulation would have been tested but this plan was not implemented before El Paso closed down the Emerald Isle operation. However, a core hole was drilled after the second blast for fracture analysis. Permeability tests were also run in this core hole and in one of the blastholes that was open to a depth of 250 feet. The core fracture analysis and in-place permeability tests indicated that the second blast substantially improved the breakage and flow.

Detailed topographic surveys before and after each Phase II blast did not reveal any significant surface rise. Elevation increases of 4 to 6 inches were observed near the collars of some of the blastholes that vented.

PHASE I LEACH TEST

The Phase I leach test began in March 1974 and continued for 117 days until July 1974. Leach solutions were distributed over the surface of the broken ore through perforated pipes. Solutions were recovered in a well located on the east side of the blasted zone as shown in figure 3. The recovery well hole was drilled with a churn drill using one of the blastholes that remained open after the blast. The recovery well hole was about 11 inches in diameter, 50 feet deep, and had a 9-inch-diameter casing. The bottom 20 feet of this casing was perforated.

During the 117-day leaching period flow rates averaged 57.4 gpm. Because of ground water dilution of the leach solutions and the desirability of drawing down the water table, a bleed of 9 gpm was established in the flow circuit after discharge from a cementation system and before recharging with acid. The water table was about 5 feet below the surface before leaching, but during the leach test it went to about 20 feet below the surface and reached a maximum of 23 feet drawdown.

Table 3 lists the results of the Phase I leach tests and also data from the pit bottom leaching. The pH averaged 1.18 for the sulphuric acid influent solutions and 1.71 for effluent solutions. Effluent copper grade averaged 0.562 gram/liter, calculated copper production was 245 lb/day, and a total of 29,000 pounds of copper was produced. Copper was recovered by cementation on scrap iron. Iron consumption was 4.7 lb Fe/lb Cu. The acid consumption figure of 15 lb H_2SO_4/lb Cu was high, but this was attributed to the 9 gpm bleed in the solution circulation system, the small size of the test area with important fringe effects, the short duration of the leaching test, and the small difference between the pH of influent and effluent solutions. Figure 5 shows the cumulative copper recovery for the Phase I leach test.

TABLE 3. - Leaching results

	Phase I Leach Test	Pit Bottom Leaching
Ore leached.....tons..	15,000	100,000
Grade of ore.....pct..	1.0	1.0
Duration of leaching.....days..	117	190
Average running time.....hr/day..	20.5	23.2
Leach influent:		
pH.....	1.18	1.10
Copper.....g/l..	0.147	0.091
Iron.....g/l..	8.5	-
Leach effluent:		
Flow rate.....gpm..	57.4	115.9
pH.....	1.3	-
Copper.....g/l..	0.562	0.646
Iron.....g/l..	6.2	-
Precipitation effluent:		
pH.....	1.71	1.62
Copper.....g/l..	0.067	0.076
Acid consumption...lb H_2SO_4/lb Cu..	15.0	10.0
Iron consumption.....lb FE/lb Cu..	4.7	2.75
Copper production.....lb/day..	245	748
Total copper production.....lb..	29,000	142,000

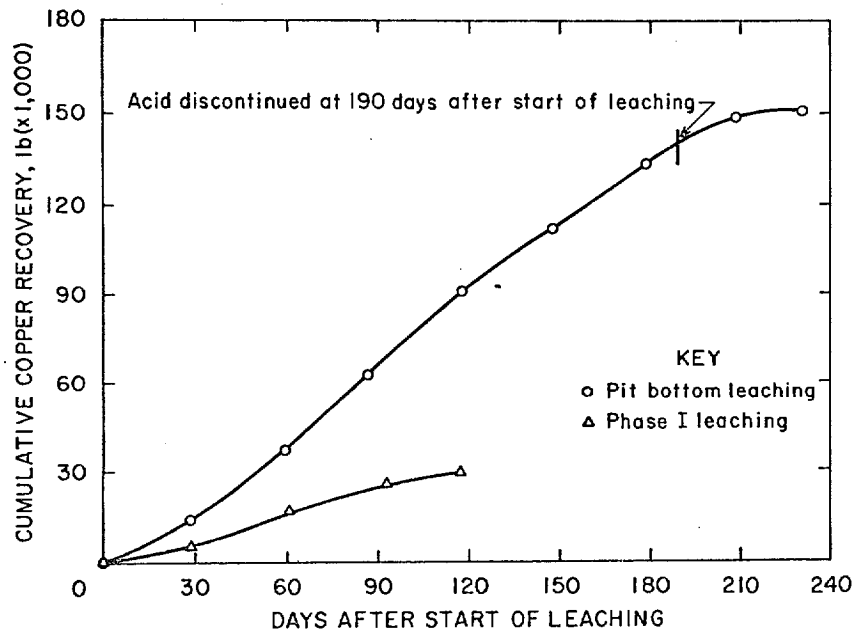


Figure 5. - Cummulative copper recovery - Phase I test and pit bottom leaching

PIT BOTTOM LEACHING

Pit bottom leaching of about 100,000 tons of ore began in December 1974 and continued for 190 days. A system of seven recovery wells was installed. These wells were installed in 12-inch-diameter holes to an average depth of 50 feet and were spaced 50 feet apart. The ore in the pit bottom was not blasted before leaching in the hope that natural permeability was adequate for successful leaching.

Pit bottom leaching results are listed in Table 3. The average flow rate was 116 gpm, pH of influent solution was 1.10, and the copper grade of effluent solutions was 0.646 gram/liter. Calculated copper production averaged 748 lb/day, and a total of 142,000 pounds of copper was produced. Solutions were circulated for 41 days after acid addition ceased to bring up the pH of the solutions remaining in the pit bottom. During this period an additional 9,000 pounds of copper were produced. Figure 5 shows the cumulative copper production for the pit bottom leaching.

During pit bottom leaching acid consumption averaged 10 lb H_2SO_4/lb Cu, and iron consumption averaged 2.75 lb Fe/lb Cu. This was an improvement over the Phase I leach test but was still high. However, acid consumption was dropping as the test continued.

Pit bottom leaching was stopped because the flow rates of leach solutions were not as high as desired. Pit bottom leaching was halted, and the ore was to be blasted to improve permeabilities and flow rates. The drilling and blasting program began, but the Emerald Isle operation closed before the pit bottom ore was leached further.

DESIGN OF FULL-SCALE OPERATION

Based on the experimental blasting and leaching tests, a full-scale in situ system was designed to recover the majority of the remaining copper. However, this plan was not implemented because the mine closed. This plan involved blasting and leaching of first a higher grade area followed by blasting and leaching of a second lower grade area. The first area would include some ore in the pit bottom, some ore under the pit walls, and ore under 180 to 250 feet of overburden along a channel extending 700 feet from the crest of the pit. The 700-foot-channel would follow the high-grade copper mineralization as shown in figure 6. The second area would be along the flanks of the 700-foot-channel and would include lower grade ore. A third area parallel to the second area could be added if leaching of the initial areas were successful.

Fragmentation would be accomplished primarily with vertical blastholes, although some inclined holes from the pit crest and some horizontal holes in the pit bottom would be required to break the ore under the pit walls. The ore in the pit bottom and under the pit

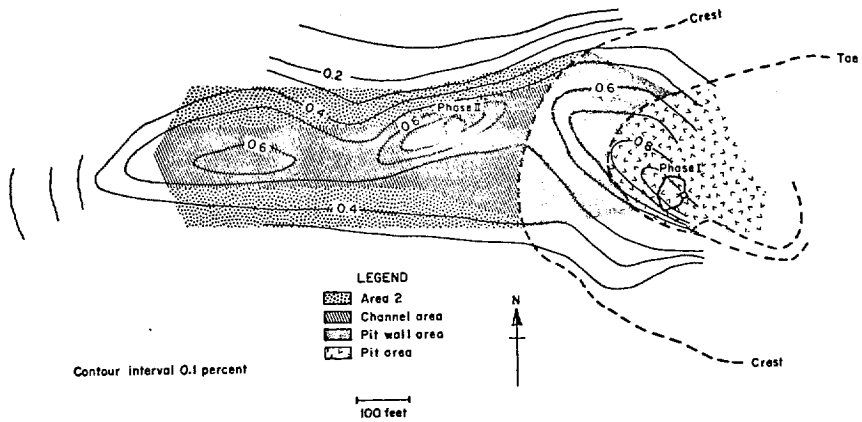


Figure 6. - Blast design for full-scale operation

walls would be broken in one blast while the channel area would be broken in a series of seven-hole blasts beginning at the pit crest and proceeding in the downdip direction. Each of the seven-hole blasts would be detonated as soon as drilling was completed so that this drilling and blasting program would continue for several months. A series of blasts would also break the second area. Table 4 lists the full-scale blast design data.

TABLE 4. - Full-scale blast data

	Area 1			Area 2
	Pit	Walls	Channel	
Area.....sq ft..	59,000	56,000	96,000	88,000
Thickness.....ft..	40	69	78	76
Volume.....cu yd..	87,000	143,000	276,000	246,000
Weight.....tons..	167,000	274,000	530,000	472,000
Grade.....pct..	0.70	.58	.55	.41
Total copper.....lb..	2,300,000	3,200,000	5,800,000	3,900,000
Drilling depth.....ft..	40	125	287	297
Powder factor....lb/ton..	0.5	1.0	1.5	.75
Total drilling.....ft..	2,200	12,900	76,300	36,000
Total explosives.....lb..	83,000	274,000	795,000	354,000

Leach solution injection would occur primarily in the pit bottom with some vertical injection holes near the pit crest. Solutions would be recovered downdip with a series of recovery wells. It is

estimated that four recovery wells would be required at about 700 feet from the pit crest. The first area would be leached for about 2 years depending on the grade of solutions before the second area would be added. Leaching of both the first and second areas would then continue for another 2 years.

The major advantages of this in situ leaching system are:

1. Initial leaching is in the higher grade ore, and the second lower grade area need not be added unless the first area yields good results.

2. Leach solutions are in contact with the ore for a longer time since the solutions migrate downdip from the pit bottom to the recovery wells. This should result in higher grade solutions if the pH can be controlled along the 700-foot channel.

3. The second blast which fragments lower grade ore can be detonated at a lower powder factor than the first, because the initial channel has been weakened by the action of leach solutions and provides a volume into which material from the second blast can expand.

4. Blasting of the second area should shake up and restimulate copper production from the first area.

This full-scale leaching system design could be implemented and would be profitable if the price of copper were to increase. The Emerald Isle mine is presently closed but could be reopened if the copper situation improves.

ASARCO ATTACHMENT E Southwestern Exploration Division

May 25, 1977 J. H. C.

MAY 27 1977

FILE MEMORANDUM

In-Place Leach Testing
Poston Butte Deposit
Pinal County, Arizona

On May 24, 1977 I met with Mr. George Laflin, Regional Mining Engineer with the Technical Service Dept. of Dowell, and Messrs. D. E. Crowell and T. D. Henderson of ASARCO Incorporated, to discuss possible in-place leach testing at Poston Butte. Asarco had previously met in Nov. 1976 with Mr. D. DeMott of the New Ventures group of Dowell for the same purpose. Their work has been stalled by their insistence that the efficacy of chelates as leach compounds be researched contrary to our clearly stated request that sulfuric acid be used for leaching. The purpose of the meeting with Mr. Laflin was to discuss Dowell's interest to serve as a contractor for pump tests.

Mr. Laflin proposed a program which would involve two wells approximately 100 ft. apart drilled at 9-5/8 in. diameter in the conglomerate and 7 in. diameter in bedrock. These wells would be cased and the area between the casing and the hole would be cemented back to the collar. A 2-in. injection stainless steel or plastic pipe would be placed inside the casing. Acid would be injected directly into the casing ahead of the injection pump to eliminate the need for a stainless steel pump. He indicated that it would require six-month lead time to contract an oil well rig of sufficient size and that a 2000 ft. hole would cost about $50/ft. He thought we might get a water well drill out of Phoenix much sooner and at much less cost. Apparently the design of the injection well and the pump is relatively simple.

The recovery well is not so simple. Although it would be drilled and cased in the same way, the problems involve wear and corrosion on pumps when pumping the pregnant solution to the surface. He suggested that an airlift system would be most suitable and involves pumping air down the outer annulus of the hole to force the pregnant solution up the interior portion of the pipe. The construction cost for the two-well program is roughly estimated by Mr. Laflin at $250,000. To this must be added operating and pilot plant costs. Mr. Laflin emphasized the need to be alert for various environmental problems. The production wells and all existing and future exploration holes should be cemented to the surface to eliminate contamination of the near-surface alluvium.

He pointed out that injection would occur over a vertical height of approximately 10 feet and would be pulled into the bottom of the recovery well at the base of the casing. The maximum width of the flow lines between the two wells at 100 ft. spacing would be approximately 40 ft. and the flow lines would approximate the shape of a football. Geologic information

which would be most critical as far as assisting with the permeability testing would be fracture information. Iron oxide mineralogy might also be important because of the tendency for iron to accumulate in the pregnant solution and then clog the rock pores when the barren solution is pumped back in the hole. Clay mineralogy is not critical because montmorillonites do not swell in an acid solution.

Mr. Laflin has had considerable experience with in-place copper leach testing at Ely, Safford, and currently at Occidental's Van Dyke deposit in Miami. His opinion is that the Poston Butte deposit is the most ideally suited he has seen for in-place leaching.

Mr. Laflin, although currently with Dowell, is planning on taking an early retirement in approximately May 1978. At that time he plans to join the firm of Emery Nielson and Associates, also in Denver, and would be available to us at that time as a consultant. He pointed out that our money might be better spent with a consultant because there are numerous political problems within Dowell involving competition between groups. In addition, the amount of service which would be required from Dowell would be so small as to probably be of no interest to Dowell. Mr. Nielson has informally suggested he make a two to three-week detailed cost estimate of a 2-hole program possibly sometime later this fall. This would be done during his vacation when he would bill us as a consultant at $250 a day plus expenses. Messrs. Crowell, Henderson, and I were generally impressed with Mr. Laflin's knowledge of Poston Butte, his general practical approach to the problems, and his experience with drilling and fluid circulation in other in-place leach programs. We think the approach suggested by Mr. Laflin is reasonable -- that is using a consultant -- and that he could supply the requisite expertise for the pumping tests. It might nevertheless be advisable to consult with another firm such as Core Laboratories before making a detailed cost estimate of the pumping program.

Before a pilot program could be started it would be necessary to confirm the Poston Butte ore reserve by drilling 5 additional exploration holes at an approximate cost of $150,000. If previous reserve estimates are verified I would recommend that an in-place leach test program be initiated.

F. T. Graybeal

F. T. Graybeal

FTG:lb

cc: DECrowell/TDHenderson
WLKurtz

ASARCO

Southwestern Exploration Division

May 25, 1977

FILE MEMORANDUM

In-Place Leach Tests
Ely, Nevada and
Miami, Arizona

During a meeting with Mr. George Laflin of Dowell on May 24, 1977 he noted some interesting tests that he had been involved with at Ely. One program involved a five-spot test on a 25-ft. spacing. This program ran for 7-8 months and maintained a fairly steady 0.8 to 3 pounds of copper per thousand gallons of water. A second test was conducted along the edge of the Ruth(?) pit with the intent of pumping water down the hole and gathering it in the pit. Two holes were drilled on a 250-ft. spacing 50 ft. from the pit perimeter; however, water which was pumped into one well did not move into the pit but instead communicated with the other well 250 ft. away. The explanation was thought to be due to concentric faulting which channelled the flow of the injected solution.

He also mentioned some work currently going on at Occidental's Van Dyke deposit. They are currently attempting to leach copper oxides from schist, but apparently the schistosity causes a major problem in the flow direction of the injected water. It requires 1-2 days to communicate between wells several hundred feet apart parallel to the schistosity, but movement of water is virtually nil at right angles to the schistosity.

F. T. Graybeal

F. T. Graybeal

FTG:lb

cc: JHCourtright
WLKurtz
JDSell

Copy to DK Crowell
WL Kurtz

ASARCO

Southwestern Exploration Division

May 20, 1977

F.T.G.

WLK file In-Place Leaching

TO: F. T. Graybeal

FROM: J. D. Sell

In-Place Leaching
Oxy-Min Project - Miami
Gila County, Arizona

In conversation with C. Caviness he stated that in their in-place leaching test the initial hydrofract pressure was 1600 psi which then dropped rapidly as circulation was established between the two bore holes (70 feet apart).

They are using a straight acid-water leach at a pressure of 600 psi to force sufficient fluid into the hydrofract ground.

Solution of chrysocolla, malachite, and azurite is excellent. Except for several barrels of "high grade liquor" (to be sent to their research department to make "first production" trinkets) the production solution of up to 6000 gallons per test is being given to ICC which operates a new solvent extraction plant nearby (after they take out control samples).

Oxy-Min originally hydrofracted the basal portion of the 1000-foot drill holes and they are now planning on moving up the hole in increments and repeating the hydrofract and leaching tests.

Caviness says the technique is relatively easy and productive, with the maintenance of pumps, valves, etc. the largest bottleneck to date.

James D. Sell

JDS:Lb

cc: JHCourtright

Leach fill In Place Leaching (In Situ)

MIAMI - from JDS Rpt Jan 13, 1977 with additions by WEK

Overall tonnage in stockcars and adjacent areas

1942 - mid 1954 2834 tons = 56,680 pounds

	# Copper	Tons
1960	18,930,454	9,465
	19,102,143	
	18,077,492	
	18,195,285	
	17,757,353	
	17,905,982	
	17,168,481	
	8,726,235	
	11,076,950	
	13,755,800	
	14,965,326	
	12,806,085	
	12,170,335	
	11,987,674	
	11,968,789	
	13,075,798	
	237,670,140	~~4,100,835~~ 118,835
	56,680	1942 - mid '54
	237,726,870	
est.	42,000,000	54-59
	279,000,000	vs JDS rpt. 337,924,007

4 = 33

Rock Cave closed July 1959

Minami

J.W. Steel April 1955 Mining Congress Journal

1925 — mid 1954 97,000,000 tons just low grade mined

25,000,000 high grade

12,500 tpd

122,000,000

152,400,000 total to July 1959

In-Place Leaching

Mon., Jan. 24, '77 The Arizona Republic

Miami fears pollution of water supply

By VINCE TAYLOR

MIAMI—Concerned citizens, jarred by the testing of a new underground copper leaching process, are looking to the governor's office to help solve Miami's latest mining problem.

The object of their concern is a development program of Occidental Minerals Corp. that residents claim has the potential for disturbing the basic life-support system of the community — its water.

Miami is reported to be sitting on top of an ore body estimated at 100 million tons of .52 per cent copper.

The estimate was made by Occidental, which has drilled about 30 exploratory holes deep into the bedrock under Miami to ascertain the extent of the low-grade ore body.

To recover the copper economically, Occidental has projected what it calls its "in situ pilot test program," which involves blasting underground and leaching the broken rock with sulphuric acid.

A detonation test that rocked the town last Monday signaled the beginning of Occidental's program. The blast cracked windows in homes near the site just north of town and "shook the buildings' in the downtown area.

Citizens stormed into a meeting of the Town Council that night to protest and Town Attorney Donald C. Premeau was instructed to seek legal ways to prevent further blasts. Councilmen joined with the Miami Chamber of Commerce in asking Gov. Raul Castro to intervene.

Robert Duber, law partner of Premeau, said Thursday that the town cannot, under Arizona law, obtain a temporary restraining order to stop the blasting. He said, however, the council could file for a permanent injunction, take action under zoning ordinances or conduct condemnation proceedings against property owned by Occidental.

Councilmen agreed to meet again today on the matter.

Mayor Katie Weimer, a native of Miami and not one to mince words, declared, "They (Occidental) are trying to swallow us up."

"Our big concern is the way they want to leach the copper, destroy our water and endanger our property," she said.

Meanwhile, Occidental, which has been quietly negotiating for options on Miami residential property for 10 years, has stepped up its development plans in what a spokesman termed "a test program."

In a statement issued after Monday's blast, the company said:

"Occidental Minerals Corp., in furtherance of its in situ pilot leach test program, conducted a detonation test at its test site near the VanDyke shaft on Jan. 17.

"The purpose was to slightly increase the permeability of the underground ore body to allow a leaching test to dissolve the copper ore and pump the copper solution to the surface.

"Results of the test are currently being studied."

The company statement said the blast "was the first detonation test on the property. No decision regarding further tests can be made until an evaluation of the test is completed."

Mayor Weimer said Occidental has options on 80 per cent of the property in Miami.

"But they're not buying, she said. In their original option agreement they offered one property owner $20,000 for four houses and five lots. Now, just the other day, they offered another man $50,000 for three houses and four lots, on a 7-year option."

Ed Dunavant, who lives with his wife Alta at 904 Smith St., high on the hill overlooking Miami, said he would have no part of it.

"All they want is the mineral rights," he said. "They told us if we optioned our property we could do anything we wanted with the house, move it or tear it down.

"But we're not interested. I'm convinced their exploration won't be good for our water supply.

"They were here about a month ago and offered us $8,000 for the place," Dunavant related. "They said they would pay us only $160 a year for the option. What good is that? All it would do is tie up our property for years."

Mrs. Dunavant said several of their neighbors also had turned down Occidental's proposition.

Mayor Weimer explained that a majority of Miami residents own their homes "and many have lived here all their lives."

Miami officials generally feel that Occidental's program will hinder the community's progress.

"They've got a clause in their mineral rights that says they're not liable for damages to structures on the surface," said Vice Mayor Ed Maxwell. "We want them to remove that and give us protection to 500 feet."

Maxwell, who spent 45 years as a miner and smelter worker here, pointed out that the council has already restricted test drilling to 12 hours a day.

Mayor Weimer said "both the (Veterans Administration) and (Federal Housing Administration) have hesitated to give loans" for building or remodeling.

Gene Miller, town coordinator and president of the Chamber of Commerce, said Occidental's program also may affect plans to revitalize the downtown area through federal grants.

"We've applied for nine separate grants to improve the town, including housing, sewer improvements, streets and sanitation," he said. "We've already begun a program of removing some of the old vacant buildings in town."

In a letter to Castro, Miller said, "Miami has been fighting for its very life for a number of years, and the Miami Chamber of Commerce is vitally concerned about the future of the town.

"We are writing to ask whatever help you may be able to give us in our fight."

9/14/76

FILE: POSTON BUTTE

FILE: In Place Leaching ✓

EM/J August '76

OIL COMPANIES ACTIVE MINERAL SEARCH

✓ ARCO
British Pet.
CONOCO
✓ Exxon (think licenses 100% negative)
✓ Getty
✓ Gulf
OCCI
✓ Shell
✓ Chevron
✓ Amoco

WLK - FTG JHC → f.u.

ASARCO

Southwestern Exploration Division

August 6, 1976

Should the water prove too big a problem — in situ leach might be the only hope for Santa Cruz

JHC

TO: W. L. Kurtz

FROM: F. T. Graybeal

WLK

In-place leaching of copper deposits in Arizona

Exploration in Arizona by Asarco has resulted in the discovery of several copper deposits which present formidable problems to conventional mining and milling techniques, largely due to excessive depth and non-sulfide mineralogy. These problems make it questionable whether some of these deposits will be competitive unless trends in current cost-price curves are reversed or new extractive technologies are developed. To illustrate the amount of copper involved I have compiled estimates for these deposits as follows:

Deposit	Estimated Size		Mineral
Poston Butte	175 million	0.46% Cu	copper oxide, chalcocite
Superior East	500 million	0.8% Cu	native copper
Sunnyside	400 million	0.45% Cu	chalcopyrite
3R	100 million	0.3% Cu	chalcocite

In addition to the deposits listed above, hole A-2 in the Superior East area intersected 623 ft. averaging 0.31% Cu as chalcocite in a zone separate from the native copper. At Sacaton approximately 200 million tons averaging 0.43% Cu as chalcopyrite will remain under and adjacent to the open pit after mining is complete and additional material will remain adjacent to the underground orebody. At Santa Cruz significant volumes of copper mineralization will remain after any physical mining operation.

The development of a method to extract the copper profitably would significantly increase Asarco's copper ore reserves. One technique involves various combinations of hydraulic or explosive fracturing and in-place leaching of the copper minerals. As compared to conventional methods copper recoveries are thought to be less with in-place leaching, although this deficiency may be more than offset by lower capital and operating costs, the absence of the risk inherent to underground mining operations, and by negligible environmental impact. I am in complete agreement with your memorandum of Jan. 26, 1976 that further research on in-place leaching should be closely watched. However, the potential importance of the above-mentioned deposits to Asarco's copper reserve position suggests that an active research program is warranted.

F. T. Graybeal

F. T. Graybeal

FTG:lb

cc: TCOsborne

cm: JHC 8/1 XO

wk file / leaching

ASARCO

Mineral Beneficiation Department

W. L. K.
AUG 17 1976

August 5, 1976

Mr. W. L. Kurtz, Manager
Western Exploration Department
Building

In-situ Leaching

Dear Mr. Kurtz:

Reference is made to your memorandum of January 26 on this subject in which you requested that someone do some preliminary research on the "state-of-the-art" of in-situ leaching. Mr. Raihl of this department has done some research on the subject and a copy of his preliminary report is attached.

Although Mr. Raihl has not been able to spend much time on this project, I believe that his conclusions are sound: we will not learn very much about in-situ leaching from the literature. Considerable information can be found regarding in-place leaching of broken or caved rock, but very little regarding leaching unfractured rock. Kennecott at Safford has done a considerable amount of work along this line but have published very little. They have applied for several patents and I assume they intend to sell their information via royalties or some other arrangement.

It appears that if we are seriously considering this type of process, we should be prepared to spend some money on research on our own or possibly in a joint venture with another company. Another alternative would be to wait until Kennecott has perfected (and patented) their technology and then pay royalties for the use of it. This second alternative could prove to be by far the most expensive if in-situ leaching proves feasible.

Mr. Raihl projects costs of millions of dollars for this research. I am not sure that it need be that expensive, at least to obtain enough information to determine if a major effort seems to be worthwhile. The initial test program (5 wells) for Posten Butte was estimated at $700,000. in 1973.

If it is decided to proceed with this research, we have two choices. The first choice would be to set up a test program at a relatively shallow oxide deposit such as Posten Butte where there is a reasonable chance of expanding the test program into a profitable operation. The second choice would be to make a pure research effort and test a deep sulfide deposit where all of the problems are greater and the economics are much less favorable. Although we would ultimately want to test deep sulfide deposits, our best approach would probably be to test Posten Butte or now possibly the Santa Cruz oxide deposit.

Yours truly,

D. E. Crowell

vh
Attachment
cc: NVisnes CWCampbell
TScartaccini FGraybeal ARRaihl/file

ASARCO

Mineral Beneficiation Department

August 2, 1976

Memorandum To: Mr. D. E. Crowell

Subject: In-situ Leaching Survey

Several months ago it was requested that a limited literature survey be done on in-situ leaching. The purpose of the survey was to determine whether or not to maintain options on several properties where copper values were insufficient to mine by present methods, but may be sufficient to mine by in-situ leaching.

The chemistry involved in leaching the majority of the important copper minerals, with the exception of chalcopyrite, is well established. Considerable work has been done by Kennecott and Lawrence Livermore Laboratory on the oxidation and consequent leaching of chalcopyrite with water saturated with oxygen. The oxidation of chalcopyrite with oxygen is expensive and apparently can only be used under conditions when excess oxygen is recovered.

Each ore body where in-situ leaching would be considered is a complete study in itself. However, it is the opinion of the undersigned that the major criteria for in-situ leaching is particle size. In areas where the average rock size can be reduced to less than 12 inch, present knowledge and technology is sufficient to strongly consider a leaching operation. Examples of this type of in-situ leaching are Ranchers Exploration and Development Corporation's Old Reliable and Big Mike operations. An example within ASARCO would be the possibility of in-situ leaching the oxide blanket overlaying the Sacation underground ore body.

The type of in-situ leaching that is mainly of interest for this report is the relatively deep low-grade mineralization that has not been previously mined. Knowledge and technolgoy are not sufficiently advanced on this type of in-situ leaching as to make any satisfactory outcome predictions. If ASARCO is interested in this type of endeavor, it should be considered as a research project and not as an operational entity. This type of research could be conducted in one of two ways:

1. Joint venture with either another mining company, such as Kennecott who is probably the most advanced in this area, or perhaps an oil company that has knowledge in drilling techniques.

2. Start from scratch and assemble a research group that has knowledge in the required areas. The purpose of this group would be to gain knowledge both in the laboratory and in the field. If an in-situ research field project became economically feasible, it could be turned over to an operating crew.

   The in-situ leach research project would be of such a magnitude that it should be approached in the same manner as Kennecott's

approach. A team was assembled to work almost exclusively on the project and have been working on the project for several years. Kennecott is presently conducting limited field testing; however, results of this test work is unavailable and are apparently closely guarded.

If the company is interested in in-situ copper leaching of a deep ore body, the company should be preapred to spend millions or, perhaps, tens of millions of dollars on research in order to develop the required technology.

Only a limited amount of work was done on a literature survey of in-situ leaching. If additional information is required, a more detailed literature survey will be conducted. However, it is the opinion of the undersigned that the same conclusion will be reached.

Yours truly,

A. Raihl (vh)

A. R. Raihl

vh

From the Desk of –
R. B. Meen

W. L. Kurtz

Tom L has written for "In-situ Leaching of a Nuclear Bubblized Copper Air Body" etc June 1975, U.C.R.A.A. Report. We will review it and then decide how we will cover it, maybe directly by someone from our office.

We discussed it briefly with Ken Mathison, Gen Mgr., KKC, to get the lay of the land.

R. B. M.
MAR 3 1976

RBM – 3/3/76 D. E. C. MAR 3 1976 Ref.

Good. Tom Osborne suggests that we — you, me, Crowell keep abreast of in-situ leaching. He doesn't feel Central Research is the one to do it.
Kurtz

LOFTIN'S

Exploration Department
Western USA

January 26, 1976

RECEIVED
JAN 28 1976
EXPLORATION DEPT.

As usual, will in this matter report back to you and to TEO
Dick Brown

In-Situ Leaching

hat someone or some organization
ning experiments, developments,
deposits — oxides, chalcocite,
leaching could have profound
deposits.

best organization to closely
cott Copper Corp., through its
pany studying In-Situ leaching,
ation at Safford.

W. L. Kurtz
W. L. Kurtz

F. T. G.
MAR 3 1976

ASARCO

Exploration Department
Western USA

January 26, 1976

Memorandum for T. C. Osborne

In-Situ Leaching

Mr. Graybeal has pointed out, and I agree, that someone or some organization in Asarco should closely follow In-Situ leaching experiments, developments, and research especially as related to copper deposits — oxides, chalcocite, and chalcopyrite-bornite. Obviously In-Situ leaching could have profound effect on the economics of deep primary ore deposits.

I suggest that Central Research might be the best organization to closely follow this subject. To my knowledge, Kennecott Copper Corp., through its Ledgemont Labs, is the most active major company studying In-Situ leaching, with physical tests of deep sulfide mineralization at Safford.

W. L. Kurtz

W. L. Kurtz

WLK:lb

cc: JHCourtright
FTGraybeal
DECrowell

2/13/76

From: T. C. OSBORNE

To: W. L. Kurtz

By all means I think the co. should be following in-situ leaching. However I don't believe Central Research could do the job. Rather, I think the Tucson office, collectively (that means yourself + Meen, + Crowell etc) is best equipped thru the mix of various disciplines, to do it.
Please discuss with Bob Meen.

ASARCO

Exploration Department
Western USA

January 26, 1976

JAN 30 1976
J. V. D.

RECEIVED
JAN 28 1976
EXPLORATION DEPT.

Bill
I will ask around here to see what can be done. In the mean while if you know of any Arizona consultants who work on these problems and can report to us how time to time it would be another way to go. But in any case I will find out what Asarco capability in this matter is and report back to you and to TCO

R.L.B.
JAN 28 1976

Memorandum for T. C. Osborne

In-Situ Leaching

Mr. Graybeal has pointed out, and I agree, that someone or some organization in Asarco should closely follow In-Situ leaching experiments, developments, and research especially as related to copper deposits — oxides, chalcocite, and chalcopyrite-bornite. Obviously In-Situ leaching could have profound effect on the economics of deep primary ore deposits.

I suggest that Central Research might be the best organization to closely follow this subject. To my knowledge, Kennecott Copper Corp., through its Ledgemont Labs, is the most active major company studying In-Situ leaching, with physical tests of deep sulfide mineralization at Safford.

W. L. Kurtz

W. L. Kurtz

WLK:lb

cc: JHCourtright
FTGraybeal
DECrowell

CIRCLE 137 ON READER SERVICE CARD

CIRCLE 138 ON READER SERVICE CARD

Copy to FTG - 4/24/75

wk file © Silver Bell
© Leaching

W. L. K.
APR 28 1975

AMERICAN SMELTING AND REFINING COMPANY
TUCSON ARIZONA

April 24, 1975

FILE MEMORANDUM

COPPER LEACHING IN-SITU
E.I. DUPONT DE NEMOURS

On Friday, April 19, I accompanied Messers. Newton, Bergman, and Beeder of DuPont to an area in North Silver Bell, some 3500' north of the El Tiro pit, which is underlain by a partially oxidized blanket of chalcocite that might qualify for an in-place leaching test.

The area, about 500' by 700', contains 6 drill holes with variable thickness and grade, but averaging around 75' at .80% Cu. The mineralization occurs in dacite porphyry beneath leached capping averaging 85' in thickness. Topographic relief is about 150'; the water table varies between 200' and 300' depth. Probably at least 80% of the copper is in readily acid-soluble form.

The DuPont engineers showed considerable interest in the deposit and will conduct a study to determine how it ranks with several others they have investigated.

The DuPont proposal (copy attached) involves drilling and blasting a group of holes followed by circulation of dilute sulphuric acid solutions. The first phase objective will be to establish circulation in a shallow deposit and then to test copper recovery over a period of 6 to 12 months (cost: $400,000 to $600,000). According to their calculations the cost of energy in leach production per pound is only 30% of that for conventional methods. Capital and operating costs are projected at around 50% less on .80% ore based on a 50% recovery of the total copper by leaching. This (50%) appears high considering recoveries obtained in most dump leach operations. However, dump grades are generally low (± .30%) with perhaps no more than 50% of the copper in readily soluble form (copper oxides and chalcocite). In contrast, at the Blue Bird Mine where over 80% of the copper is oxide, 40% to 50% recovery is obtained by heap leach in around three months.

The critical question is whether sufficient permeability can be achieved by blasting, pressure injection and/or other techniques to permit recoveries even approaching the economic level. The odds against success are generally regarded to be heavy; however, in view of the low energy requirement relative to conventional extraction and the environmental advantages, a modest investment by Asarco in a test program is warranted, in my opinion. Reportedly Kennecott is well along on a several million dollar project at Safford which involves leaching of .60% Cu as chalcopyrite at depths below 3000'.

J.H. Courtright
J.H. Courtright

JHC:vmh
cc: J.J. Collins w/enc.
N. Visnes w/o enc.
R.B. Meen w/o enc.
W.L. Kurtz w/o enc.

Note: Enclosure in J.H.C. "Leaching" file.

Cpy to J HC 4/18

W. L. K.

APR 17 1975

Du Pont Proposal for IN SITU LEACHING of Deep Copper Deposits

PREPARED BY:

MINERALS SECTION
POLYMER INTERMEDIATES DEPT.
E. I. DU PONT DE NEMOURS

OBJECTIVE

ECONOMICAL EXTRACTION OF METAL VALUES (E.G. COPPER) AT DEPTH FROM LOW GRADE DEPOSITS BY IN SITU LEACHING.

INCENTIVE

THE OPPORTUNITY TO DEVELOP A NEW MINING PROCESS THAT WOULD HAVE WIDE-SPREAD APPLICATION THROUGHOUT THE UNITED STATES FOR ECONOMIC RECOVERY OF COPPER FROM PRESENTLY MARGINAL DEPOSITS.

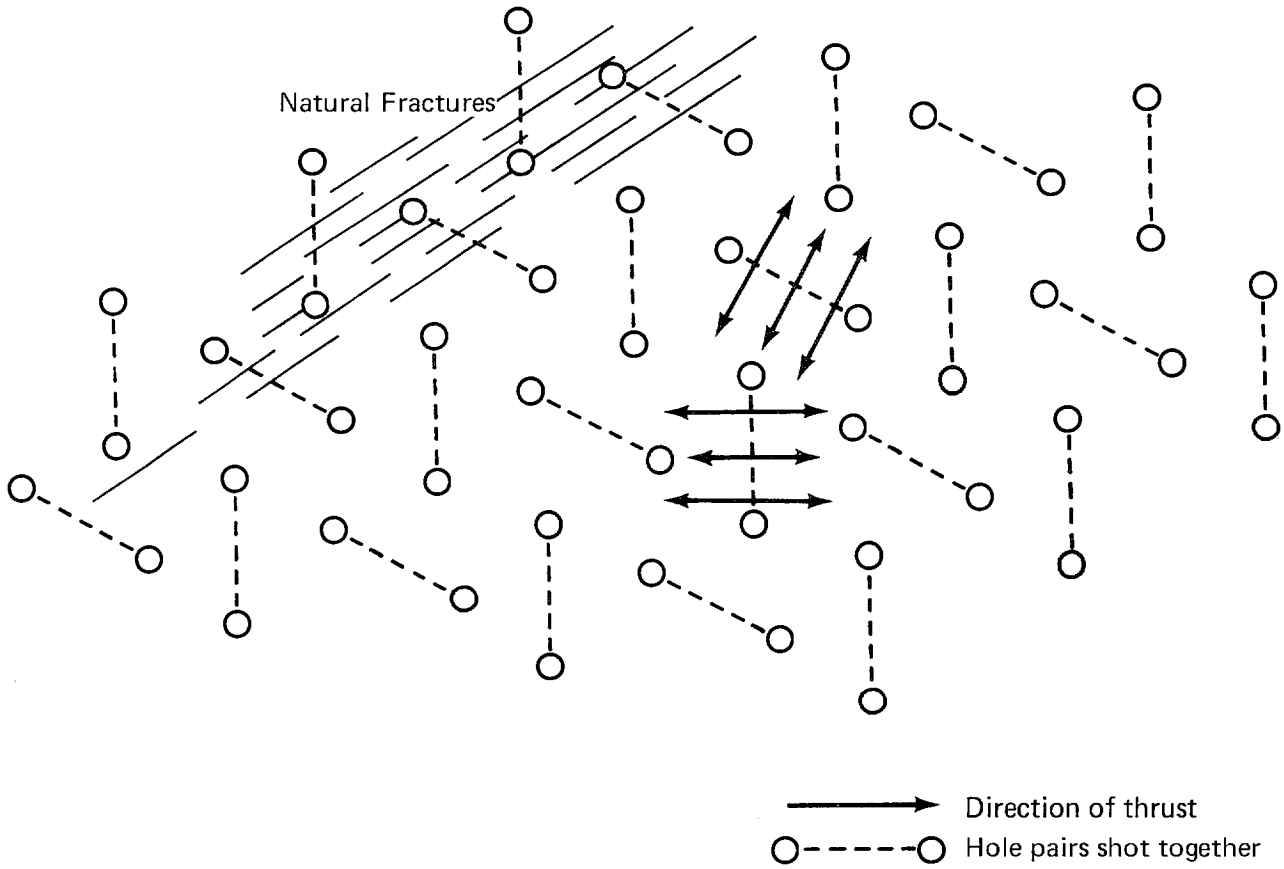
CONCEPT

THE METHOD INVOLVES EXPLOSIVELY FRACTURING OREBODIES IN PLACE TO ALLOW SUBSEQUENT *IN SITU* LEACHING OF METAL VALUES AT DEPTH.

THE CONCEPT CONSISTS OF DRILLING A PATTERN OF BOREHOLES THROUGH THE OVERBURDEN INTO THE OREBODY, LOADING THE HOLES WITH EXPLOSIVES AND DETONATING PAIRS OF LOADED HOLES SUCCESSIVELY. THIS GENERATES DIRECTED THRUSTS IN THE ROCK IN THOSE DIRECTIONS IN WHICH THE ROCK IS PARTICULARLY VULNERABLE TO FAILURE DUE TO PREEXISTING NATURAL FRACTURES AND TECTONIC STRESSES. (SEE FIG. 1)

FIGURE 1

SHOT-HOLE PATTERN AND RESULTING DIRECTIONAL THRUSTS

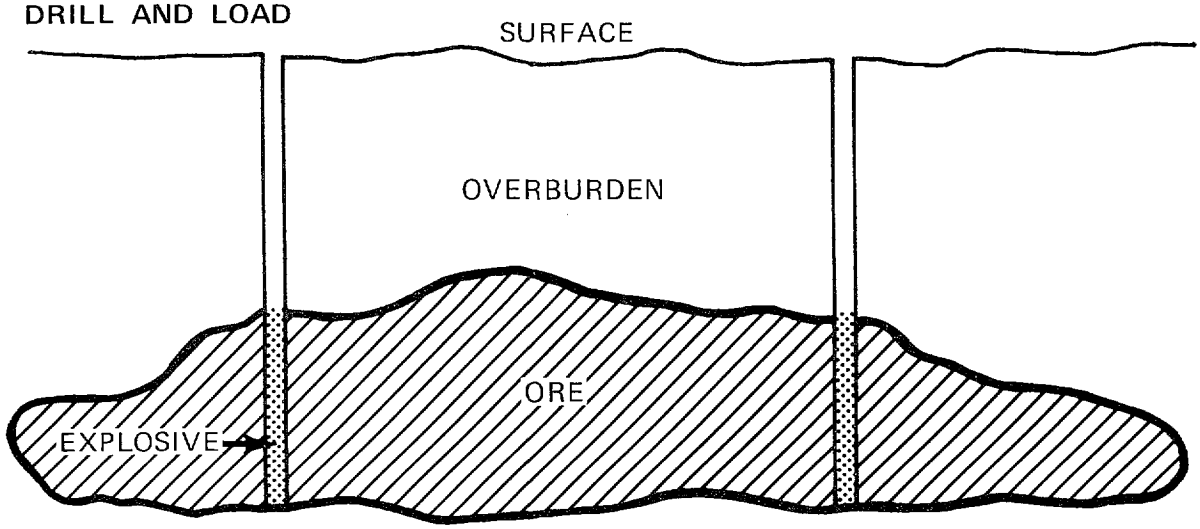


IN ORDER TO REDUCE THE NUMBER OF HOLES REQUIRED FOR FRACTURING (PARTICULARLY FOR OREBODIES AT GREAT DEPTHS WHERE DRILLING COSTS BECOME A LARGER FRACTION OF BLASTING COSTS) IT MAY BE DESIRABLE TO ENLARGE THE BOREHOLE BOTTOMS BY MECHANICAL CHAMBERING OR BY EXPLOSIVE "SPRINGING" BEFORE LOADING AND DETONATING THE MAIN CHARGE. THE LATTER TECHNIQUE IS SHOWN IN FIG. 2.

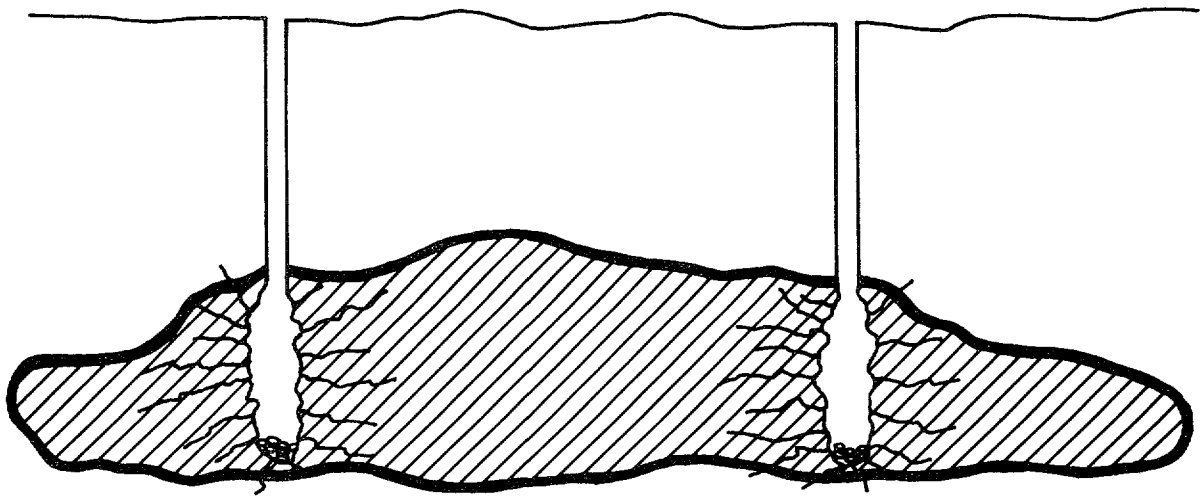
FIGURE 2

EXPLOSIVE "SPRINGING" AND FRACTURING

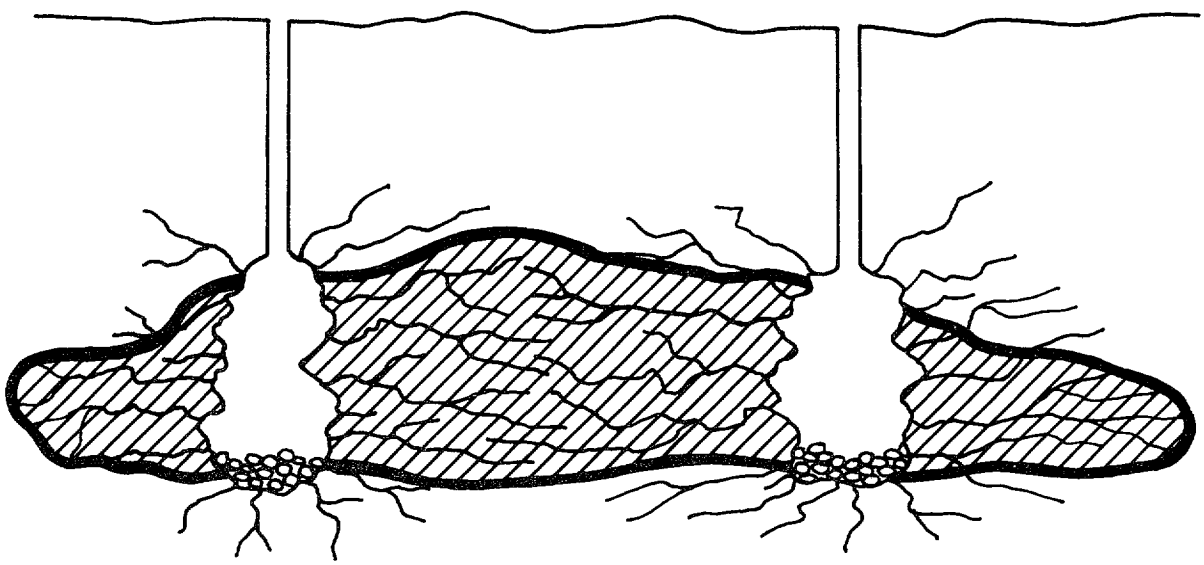
DRILL AND LOAD



SHOOT TO ENLARGE HOLES ("SPRINGING" SHOT)



RE-LOAD AND RE-SHOOT (FRACTURING SHOT)

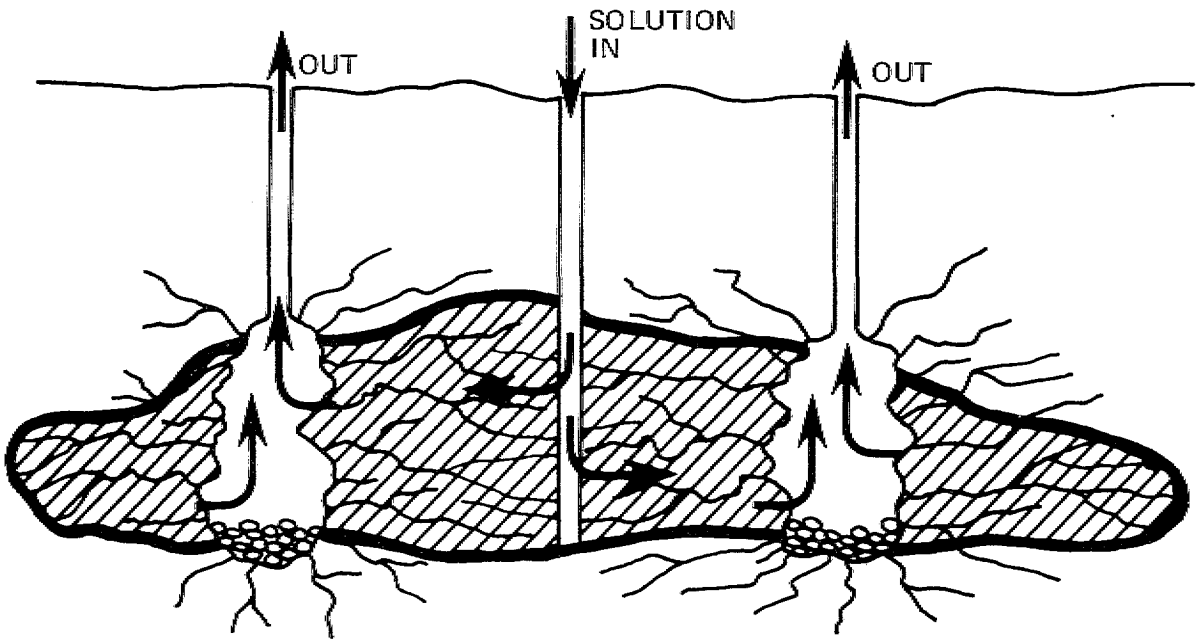


AFTER BLASTING, THE OREBODY IS LEACHED BY INJECTION OF LEACH SOLUTION INTO AND RECOVERY OF PREGNANT SOLUTION FROM THE HOLES (SEE FIG. 3).

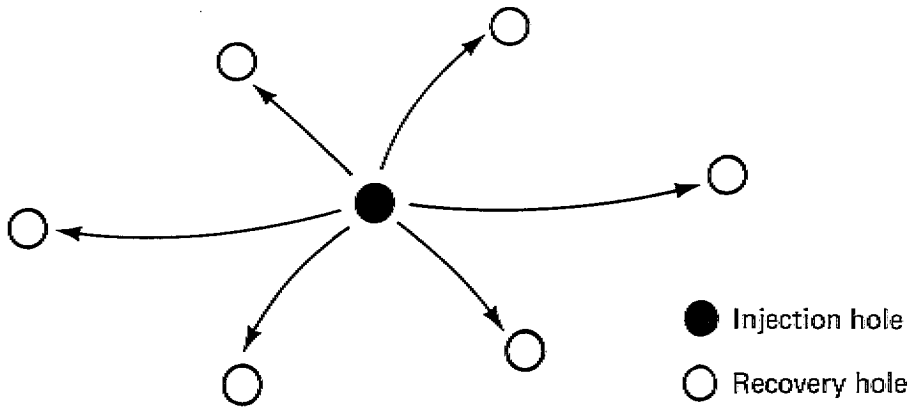
FIGURE 3

LEACHING OF FRACTURE OREBODY

CROSS SECTION OF INJECTION AND RECOVERY HOLES



PLAN VIEW OF INJECTION AND RECOVERY HOLES



THE EXACT NATURE OF THE LEACHING SYSTEM TO BE USED DEPENDS ON THE TYPE OF ORE AND THE DEPTH OF THE OREBODY.

IN SHALLOW OREBODIES CONTAINING ACID SOLUBLE ORES SULFURIC ACID IS A SUITABLE LIXIVIANT.

AT DEPTH WHERE SULFIDE MINERALS PREDOMINATE THE PRESENCE OF AN OXIDANT (FERRIC ION OR OXYGEN) IS REQUIRED TO SOLUBILIZE THE SULFIDES. CHALCOPYRITE IS PARTICULARLY DIFFICULT TO LEACH BUT WORK AT LAWRENCE LIVERMORE LABORATORIES HAS SHOWN THAT ADEQUATE RATES AND EXTRACTIONS CAN BE ACHIEVED BY LEACHING WITH H_2SO_4 AND O_2 AT SEVERAL HUNDRED PSI PARTIAL PRESSURE AND A TEMPERATURE OF ABOUT 90° C. WE BELIEVE IT IS FEASIBLE TO ESTABLISH AND MAINTAIN SUCH CONDITIONS IN A FRACTURED OREBODY AT DEPTH.*

* RECOVERY OF COPPER FROM THE PREGNANT LEACH SOLUTION CAN BE DONE EITHER BY CEMENTATION OR, PREFERABLY, BY LIQUID ION-EXCHANGE AND ELECTROWINNING.

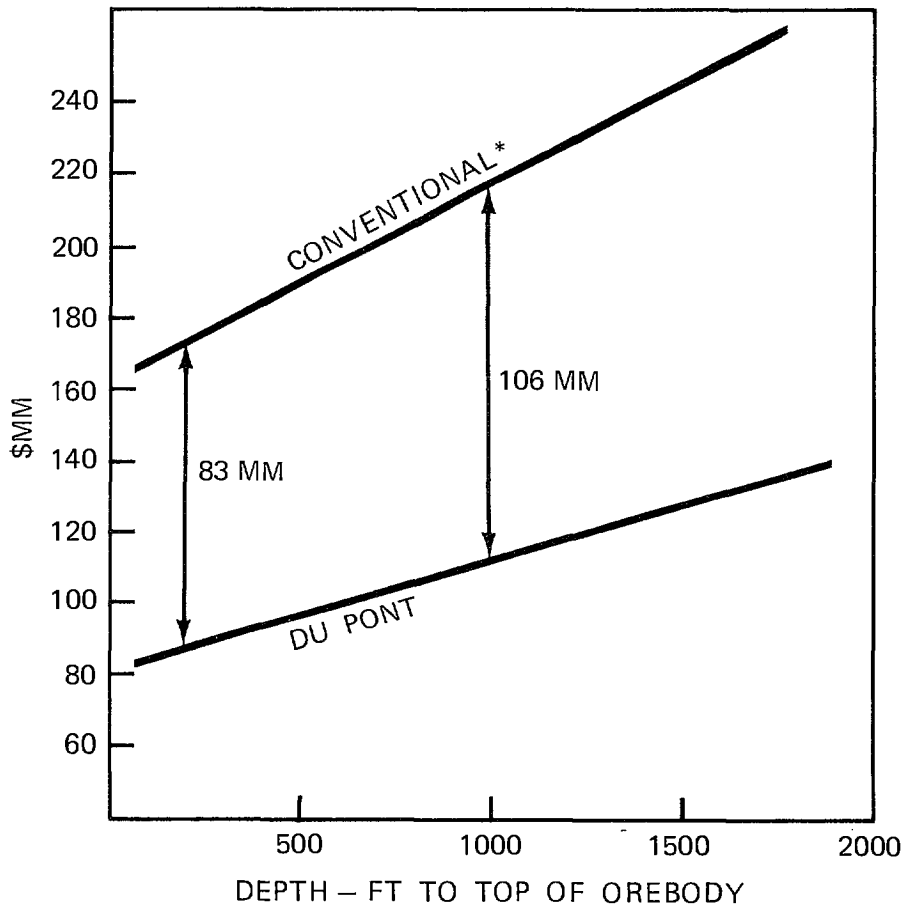
ECONOMIC AND ENERGY COMPARISON

ECONOMIC ESTIMATES FOR RECOVERY OF COPPER HAVE SHOWN THAT THE DU PONT CONCEPT HAS SUBSTANTIALLY LOWER CAPITAL COST AND OPERATING COSTS THAN CONVENTIONAL PROCESS - ING. THIS ADVANTAGE INCREASES WITH INCREASING DEPTH (SEE FIG. 4).

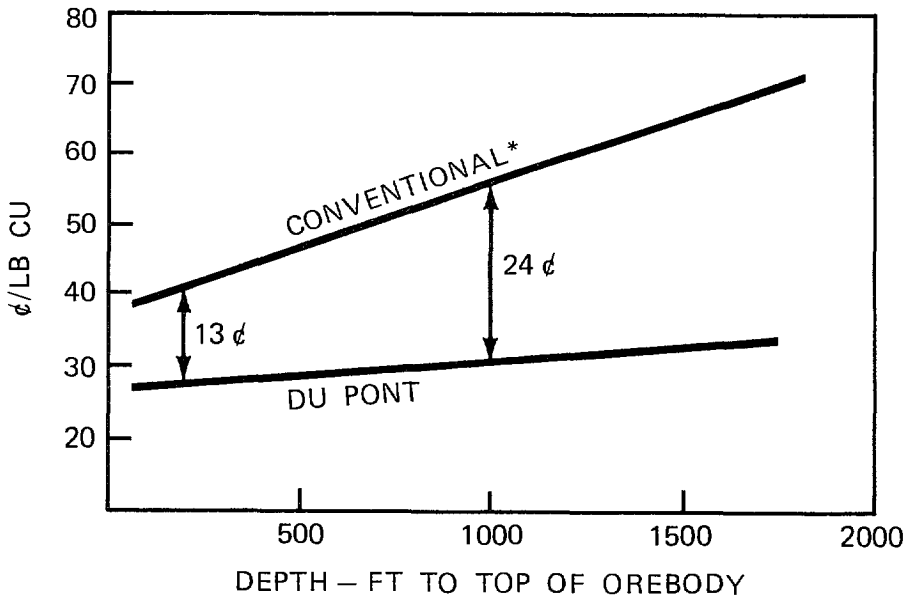
FIGURE 4

ECONOMIC COMPARISION 100 MM LB CU/YR, 0.8% ORE

CAPITAL COSTS



OPERATING COSTS



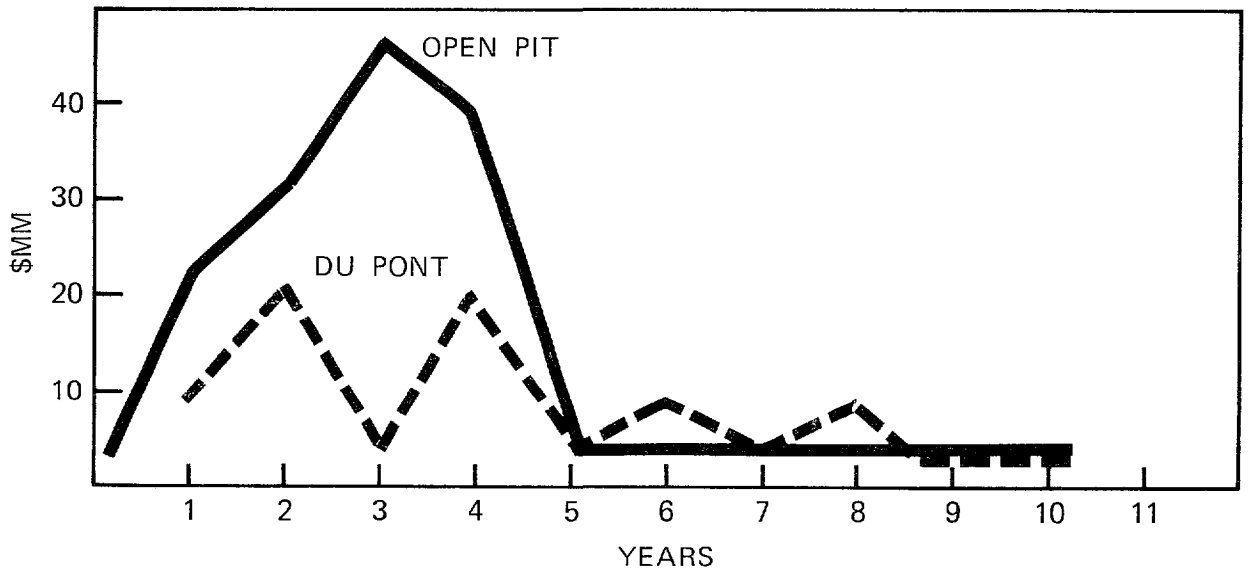
*Open pit/underground mining, mill, smelt, electrorefine

THE DU PONT APPROACH HAS THE FURTHER ADVANTAGE OF HAVING CONSIDERABLY LOWER FRONT - END CAPITAL REQUIREMENTS AND OF COMING ON-STREAM SOONER THAN CONVENTIONAL MINING (SEE FIG. 5.). THE PEAKS AND VALLEYS FOR CAPITAL COST IN THE DU PONT CURVE REFLECT EXPANSION OF THE OPERATION IN DISTINCT PHASES DURING THE LIFE OF THE PROJECT. THE PEAKS ARE LOWER IN THE LATER PHASES AS THE NEED FOR ADDING RECOVERY FACILITIES DISAPPEARS.

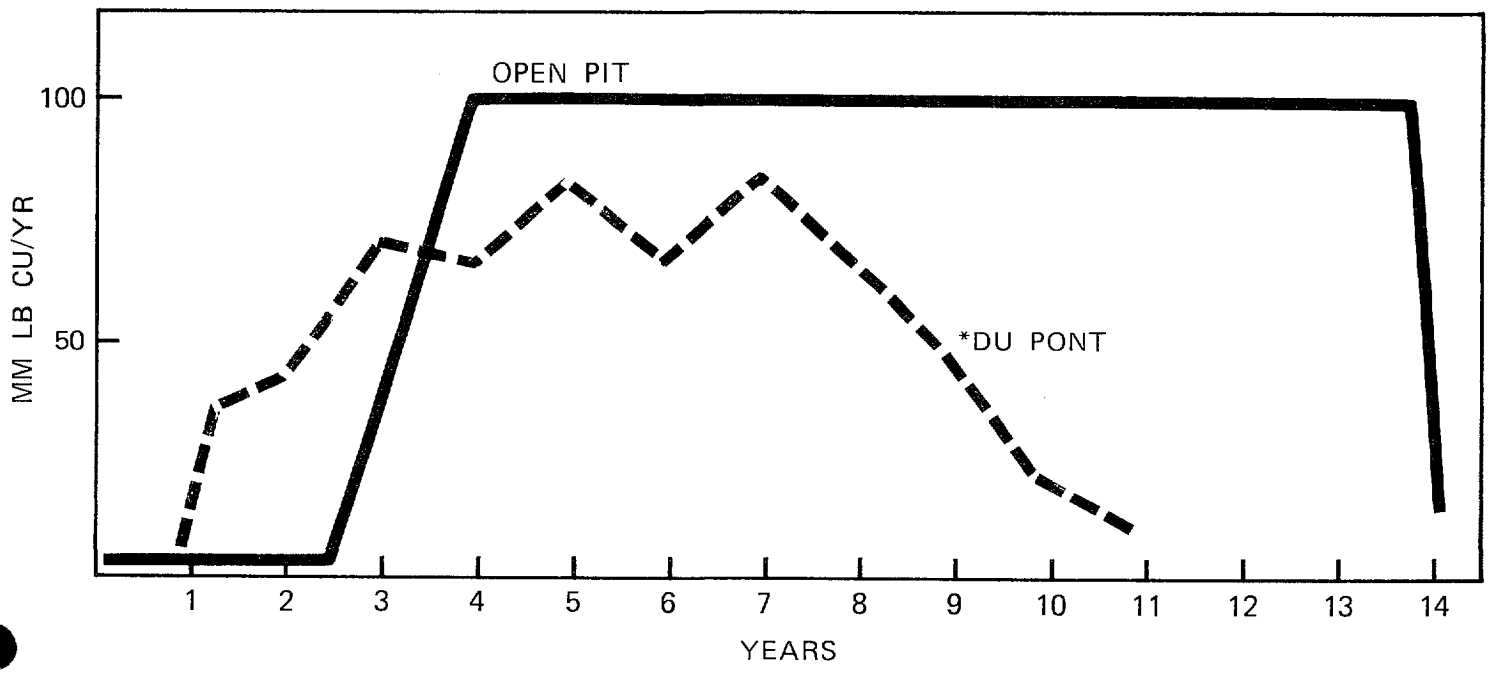
ENERGY REQUIREMENTS FOR THE DU PONT CONCEPT WERE CALCULATED AND FOUND TO BE ONLY 30% OF THOSE FOR CONVENTIONAL METHODS TO PRODUCE 1 LB. OF COPPER.

FIGURE 5

CAPITAL COSTS FOR DEVELOPMENT



PRODUCTION PROFILE



*Based on 50% recovery

DEVELOPMENT PROGRAM

UNDER THE GUIDANCE OF DR. D. L. COURSEN, DU PONT HAS WORKED ON THIS PROCESS FOR THE LAST FOUR YEARS AND INVESTED ABOUT $500M IN EXPLORATORY RESEARCH TO DETERMINE THE TECHNICAL FEASIBILITY OF THE CONCEPT.

PRELIMINARY DEVELOPMENT

WORK TO DATE BY DU PONT HAS BEEN IN BARREN ROCK AT DEPTHS BETWEEN 80 AND 450 FT. TO STUDY IMPORTANT FEASIBILITY QUESTIONS SUCH AS FRAGMENTATION IN THE ABSENCE OF A FREE FACE, PRODUCING PREMEABILITY IN THE ABSENCE OF SIGNIFICANT SWELL, SOLUTION CHANNELLING, ENLARGEMENT OF HOLES AT DEPTH AND LOADING COMMERCIAL BLASTING EXPLOSIVES THROUGH SEVERAL HUNDRED FEET OF WATER WITHOUT AFFECTING DETONATION PROPERTIES.

SIGNIFICANT RESULTS TO DATE

- DEVELOPED A SUCCESSFUL TECHNIQUE FOR EASY AND INEXPENSIVE LOADING OF DEEP BOREHOLES WITH COMMERCIAL WATER GEL EXPLOSIVES.
- SHOWED SUBSTANTIAL ROCK FRACTURING TO CONSIDERABLE DISTANCE FROM THE SHOT HOLE.
- SHOWED SUBSTANTIAL INCREASE IN PERMEABILITY AS A RESULT OF BLASTING.
- TRACER TESTS AFTER BLASTING SHOWED UNIFORM SOLUTION FLOW THROUGH BROKEN ROCK AND NO SIGNIFICANT CHANNELLING.
- DEMONSTRATED SUCCESSFUL SPRINGING OF HOLES DOWN TO 250 FT. DEPTH.

FUTURE DEVELOPMENT

THE NEXT STEP IN THE DEVELOPMENT PROGRAM IS TO TEST THE DU PONT CONCEPT IN A COPPER OREBODY. WE VISUALIZE A THREE PHASE PROGRAM:

PHASE I - IDENTIFY SUITABLE OREBODY. TEST CONCEPT AT A DEPTH IN THE RANGE OF 300-600 FT.; 12-Hole PATTERN OF BLASTS (18-24 MONTHS) FOLLOWED BY LEACHING AND COPPER RECOVERY TESTS (6-12 MONTHS).

PHASE II - TEST CONCEPT AT A DEPTH IN THE RANGE OF 700-1000 FT.; 12-HOLE PATTERN OF BLASTS FOLLOWED BY LEACHING AND COPPER RECOVERY. (TIMING - ABOUT 21 MONTHS).

PHASE III - APPRAISE RESULTS, RE-ESTIMATE ECONOMICS AND IF FAVORABLE, PROCEED WITH COMMERCIALIZATION WHILE EVALUATING APPLICATION TO EVEN DEEPER DEPOSITS. (TIMING - ABOUT 3 MONTHS).

SUMMARY AND PROSPECTS

OUR RESULTS TO DATE HAVE BEEN QUITE PROMISING; HOWEVER, FOR THE FURTHER DEVELOPMENT AND EVENTUAL COMMERCILIZATION OF THE CONCEPT WE BELIEVE IT IS ESSENTIAL THAT WE WORK WITH AN EXPERIENCED MINING COMPANY. BY COMBINING OUR CONSIDERABLE EXPLOSIVES EXPERTISE AS WELL AS OUR MORE RECENTLY ACQUIRED SOLUTION MINING EXPERIENCE WITH THE SKILLS OF AN ESTABLISHED MINING COMPANY, WE BELIEVE WE WOULD HAVE THE TOTAL CAPABILITY REQUIRED FOR SUCCESSFUL DEVELOPMENT OF A NEW MINING PROCESS.

APPENDIX

ECONOMIC COMPARISON

(Conventional vs. Du Pont Deep Blast/Leach, 100MM lbs/yr, 0.8% ore)

Capital Costs - $MM

	Conventional		Du Pont	
	Open Pit*	Underground**	Shallow*	Deep**
Mine	28	76		
Mill	85	85		
Smelter & Refinery	58	58		
Total	171	219		
Drill & Shoot			60	84
Leach			5	5
LIX-EW			23	23
Total			88	112
			160 holes, 10" diam. 800 ft deep on 100 ft centers $22/ft	160 holes 10" diam. 1600 ft deep on 100 ft centers $25/ft

Operating Costs - ¢/lb Cu

Mining	9	24		
Milling	13	13		
Smelting & Refining	18	18		
Total	40	55		
Leaching			13	17
LIX-Electrowinning			14	14
Total			27	31

* 200 ft to top of orebody

** 1000 ft to top of orebody

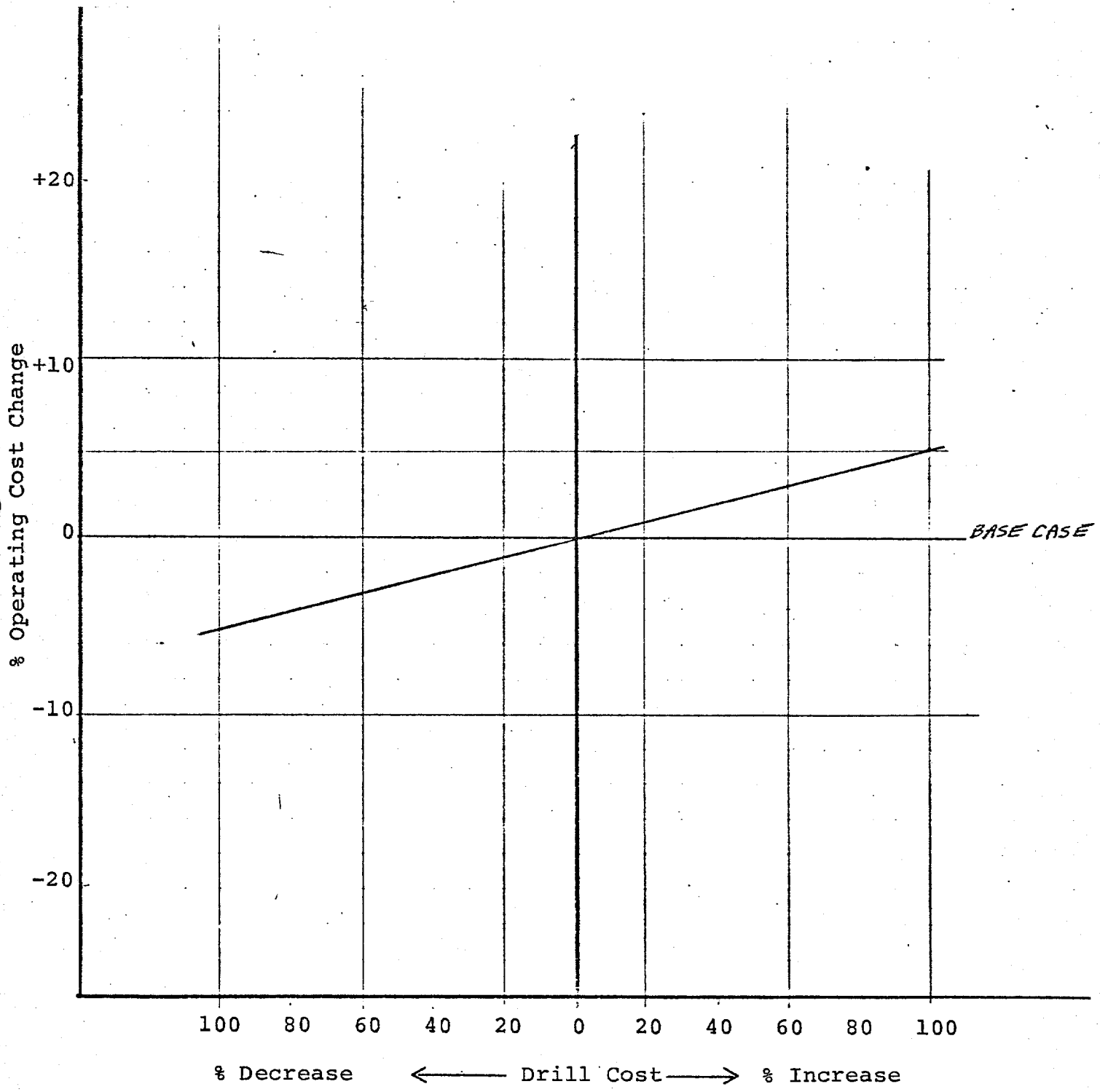
CAPITAL COSTS VS. TIME - DU PONT DEEP BLAST/LEACH CONCEPT ($MM)

Year	1	2	3	4	5	6	7	8	Total
Property Study and Monitoring Holes	7								7
Shoot 1 Section	4		4		4		4		16
Piping for injection and recovery holes		10		10		10		10	40
Leaching and LIX/EW facilities		14		11					25
	11	24	4	21	4	10	4	10	88

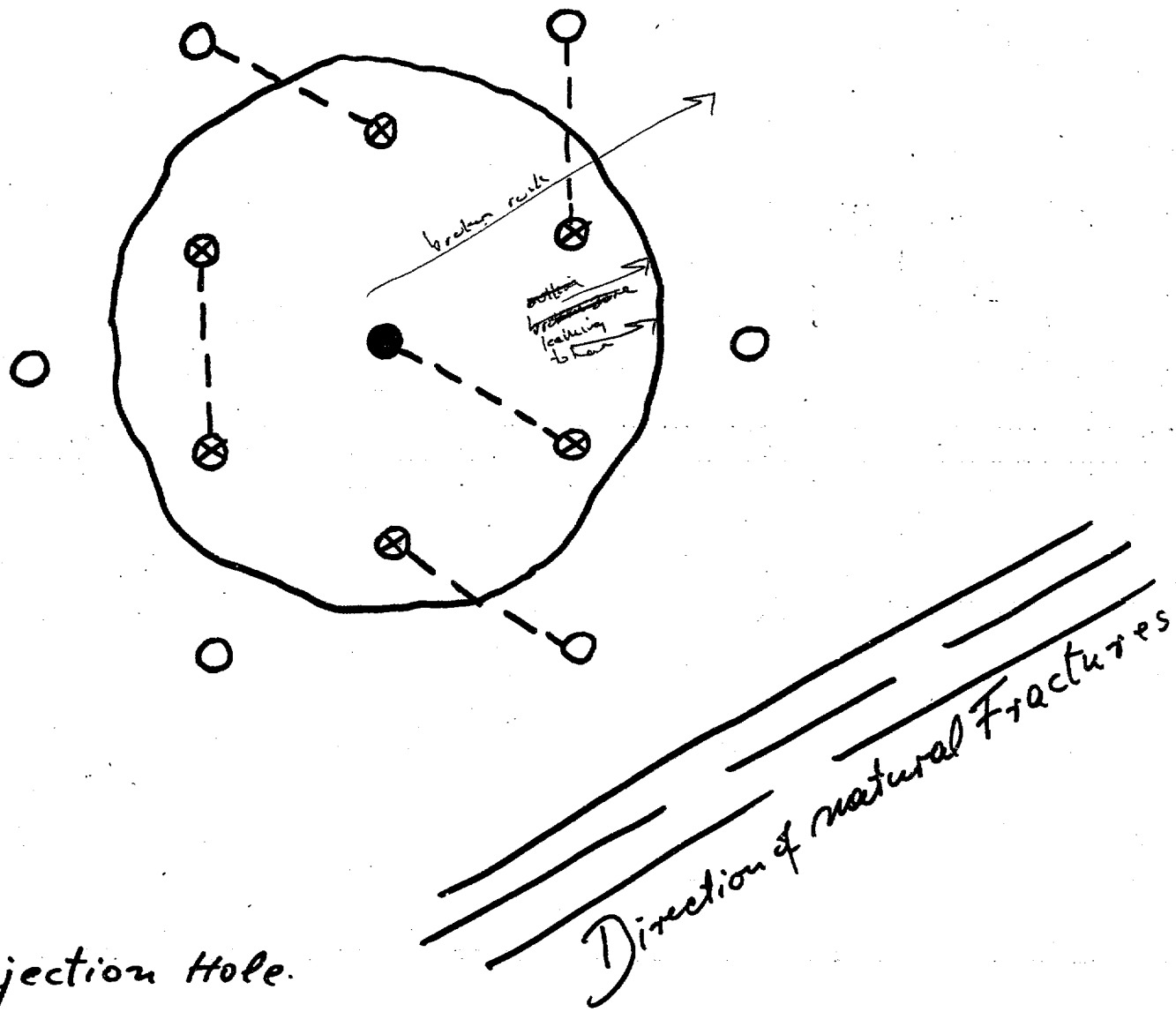
SENSITIVITY OF OPERATING COST TO DRILLING COSTS

FOR

DU PONT DEEP BLAST/LEACH CONCEPT



Twelve-Hole Test



● Injection Hole.

⊗ Recovery Hole

○ Satellite Hole

~~cy for JHC~~

AMERICAN SMELTING AND REFINING COMPANY
SILVER BELL UNIT
Silver Bell, Arizona

November 2, 1970 ?

W. L. K.
JUL 24 1974

file – leaching

MEMORANDUM TO: Mr. D. R. JAMESON, Superintendent

Subject: VISITS TO DUMP LEACHING OPERATIONS

On October 6 and 7, 1970 ? Mr. B. L. Rickman and the undersigned visited several dump leaching operations. The main purpose of the trip was to inspect solution distribution systems and obtain possible methods of revitalizing older leach dumps. Presented below are summaries of each of the properties visited and possible applications to Silver Bell's leaching program.

KENNECOTT COPPER CORPORATION -Ray Mines Division

At Kennecott we were escorted by Mr. C. K. Vance, Mine Superintendent and Mr. Carl Hartless, leaching shift foreman.

The production from Kennecott's pit is classified into one of three areas: Concentrate ore, silicate leach ore, and dump leaching material. All of the material sent to the dumps is leached; no waste is segregated. Copper content of the dumps is approximately .17-.18% copper. Quantity of barren solution pumped to the dumps is about 15,000 gpm.

Kennecott concentrates heavily on dump preparation and working of dumps for their leaching program. Two dozers (D8 or of comparable size) and one "Cat" scraper work 12 hours daily on the leach dumps. This equipment is used on dump preparation and workings.

Approximately every 2 years the top 4 to 8 feet of the dumps is ripped thorougly and removed. This tonnage is pushed over the end of the dump, having the effect of very gradually lowering and broadening the dump. On all dumps ripping of surfaces is done approximately every 6 months between leaching cycles. The constant working of the dumps is designed to keep good air circulation and an open dump surface. Mr. Vance stated the work on the dumps had doubled the production.

Kennecott employs several methods for introducing barren solution to their dumps. A continual trial and error system is used to determine their method of distribution to a certain area. The methods used are sprays through 2-1/2" John-Manville PVC pipe with 3/8" holes, ponding in which furrows are created in the bottom of the ponds by ripping, furrows without ponds, and deep trenches of 12 to 15 feet which are flooded. The deep trenches are made by using a cat and digging the width of the blade about 12 to 15 feet deep. These trenches enable sampling of top layers of leaching dumps and provide excellent

observation for past solution flows through the top part of the dumps. Leaching cycles are determined by grade of solution coming from the dumps. Areas are rested when solution grades fall to the range of .35-.45 grams per liter of copper. Fresh water make up is provided by sumps in bottom of the pit. Solution return is good, about 10% loss.

All pumps are constructed of 316 stainless steel. Allis Chalmers horizontal pumps are used. Concrete sumps are used for barren solution while holding dams similar to ours are employed for pregnant solution.

Feed to precipitation consists of 14,000 to 15,000 gpm of approximately 0.7-0.75 gram per liter copper solution. Approximately 600 gpm of pregnant solution from the silicate leach plant is combined with the solution coming from the dumps. The silicate solution contains 20 to 30 gram per liter of H_2SO_4 and supplies all the acid for the circuit; no other acid additions are made. The pH of various solutions remains fairly constant around 2.4. This pH maintains scale free lines and prevents formation of iron salts.

Fourteen stainless steel cones are used for precipitation but some iron launders still remain. The cones or launders are charged with scrap cans from either Proler steel or L. A. By-products. The cones are loaded from the top by means of a conveyor belt. The cones handle about 1000 gpm each and are flushed every 12 hours from the bottom until clear solution is discharging. The precipitate and cans flushed from the cones enter one common sump. The precipitate-can mixture is retrieved by clam shell and introduced to a 3/16" vibrating screen, the undersized from the screen (83% copper) is dried to 20-25% moisture on a gas fired hot plate in 24 hours. The iron consumption during precipitation is 1.6 to 1.9 pounds of iron per pound of copper. Recovery is approximately 93%. Daily production of precipitate varies between 100,000 and 150,000 pounds of copper.

INSPIRATION - Miami, Arizona

At Inspiration we talked with Messrs: Anderson, Superintendent, Sorsen, Leaching Superintendent, and Bilsen, Superintendent of Ox-Hide Unit.

At Inspiration all of the dumped material from two pits, Thornton and Live Oak, is leached. The average dump grade is 0.3% copper with the major mineralization being chrysocolla. Quantity of barren solution pumped to the dumps is 2700 gpm.

Mr. Sorsen stated that dump preparation consists of ripping and leveling. Some zones impervious to solution penetration needed to be drilled with an auger to allow solution flow down through the dump. Inspiration uses the figure 1 gallon per minute per every 333 square feet of dump surface to control solution additions.

Solution distribution is performed by flexible plastic pipe. Rather than spraying like Kennecott, small plastic valves are used to trickle the solution onto the dump. An acid level of 10 grams per liter in the barren solution prevents any scale formation.

Iron launders (10 cells) are employed for precipitation of the copper. The pregnant solution runs 1.2 grams per liter of copper. The tail from the launders is .01 to .02 grams per liter copper. Acid consumption and iron consumption figures are 6-12 pounds and 1.2 to 2.5 pounds per pound of copper respectively. Precipitation grade usually runs in the area of 85% copper.

A laboratory testing program by Inspiration on samples taken from leaching dumps indicated that a considerable amount of copper was non-recoverable by leaching.

Total leaching operation - no ore sent to mill

Mr. Bilsen accompanied us on our tour of the Ox-Hide Unit of Inspiration. The ore at Ox-Hide is heap leached in 20 to 25 foot heaps. Assay of the heaps is .3 to .4% copper with major mineralization being chrysocolla. Mr. Bilsen stated that about 70% of the copper to the heaps was acid soluble.

Heap preparation consists of extensive ripping at every three feet when building new heaps. Ripping is done several times perpendicular to the previous ripping direction. At Ox-Hide the barren solution is distributed at the rate of 1 gallon per minute to 500 square feet of dump surface.

Solution distribution is accomplished by flexible 2" pipe. The pipe is placed on 8 foot centers with holes every 4 feet. Plastic valves are used to distribute the solution. The acid level of the barren solution is about 3 grams per liter and maintains scale free pipe lines. Heaps are leached in 3 month cycles and about 33% recovery is obtained in this period. The base of the leaching heaps was prepared by asphalting to create an impervious zone. Solution return is approximately 92%.

All pumping of solution is done by Worthington vertical high speed pumps. Mr. Bilsen stated these pumps were quite a bit of trouble and required considerable maintenance.

Feed to precipitation launders is 2000 gallons per minute of 2 gram per liter copper solution. Iron consumption is 1.7 pounds per pound of copper. Precipitate usually contains 20% moisture and 80 to 85% copper. Production of precipitate is about 40,000 pounds of copper per day.

COPPER CITIES - Miami, Arizona

At Copper Cities we were entertained by Wayne Gould, Director of Metallurgy, Burt Frazier, Assistant Leaching Superintendent, and Jim Bean, Chief Metallurgist.

The leach dumps contain 0.2 to 0.3% copper. The major mineralization is chalcopyrite. Quantity of barren solution pumped to the dumps is 2500 gallons per minute. Ninety percent of the solution is returned to the iron launders.

Dump preparation is similar to Inspiration. The dumps are leveled and throughly ripped. Leaching cycles of 6 months are used. While dumps are idle they are reripped and cut. Barren solution is added at the rate of 1 gallon per minute per 100 square feet of dump surface. The barren solution contains about .25 grams per liter of acid (which is added to barren solution). The acid content prevents iron scale formation. Equipment for working the dumps is under the mining department and is loaned out for leach dump use.

Solutions distribution methods is spraying through flexible pipe. Both Carlon and Swanson 2" pipe is used. The piping is placed every 33 feet apart for feed lines. There is no set distance for the spray holes on the feed lines. Besides horizontal piping, piping in 15 to 17 foot vertical drill holes is being tried toward the edge of the dumps. This piping is also 2" and contains 3/4" holes every 12 to 18 inches for solution distribution. Copper Cities also spray the dump slopes in hopes of recovering additional copper. Mr. Frazier stated that at Copper Cities Castle Dome operation vertical holes 10 feet apart and 6 to 8 feet deep were used for distributing the solution. The Castle Dome operation is not presently active.

Iron launders using scrap cans fed by an overhead conveyor are employed for precipitation of copper. Pregnant solution contains 0.5 grams per liter copper and 0.35 gram per liter total iron at a pH of 2.6. Recovery in the launders varies between 96 and 98 percent. Iron consumption is 1.5 to 1.6 pounds per pound of copper. Average monthly production is 350,000 pounds of copper.

RANCHER'S EXPLORATION - Blue Bird Mine - Miami, Arizona

Total leaching operation

At Bluebird we were escorted by Messrs: Ken Power, General Manager and Lamar Kinnard.

All mining and stripping is accomplished by D-8 and 9 dozers and scrapers. The scrapers vary in capacity from 28 to 50 tons. Dozers rip the area to be mined and then push the scrapers to enable them to load. The present stripping ratio is .3 of tons of waste to a ton of ore. Future reserves indicate a 1 to 1 ratio.

Bluebird leaching operation is very similar to Inspirations Ox-Hide Unit. Heaps of 18 to 20 feet are constructed from 0.5% copper ore containing mainly chrysocolla with small amounts of azurite and malachite. About 1600 gallons per minute of barren solution (0.5-0.6 grams per liter of copper) is pumped to the heap with a return of 1450 gallons per minute. Solution is added at the rate of 1 gallon per minute per 333 square feet of surface. Slower rates are used on older heaps.

Dump preparation consists of ripping and leveling. One test dump is being constructed of ore which has been crushed to minus 2 inch. Leaching cycles are completed every 135 days. New heaps are cut in and out every 15 days. Acid levels to the heaps vary with time. For the first 10 days of leaching 50 gram per liter H_2SO_4 is used, for 10 to 20 days of leaching 30 gram per liter acid is used, for 20 to 30 days of leaching 20 gram per liter acid is used, after 30 days of leaching 10 grams per liter or below is used (raffinate).

Solution distribution is through 2" PVC flexible pipe. Plastic trickle valves are used. The pipes are 8 feet apart with a valve every 2 feet. Pipes are placed perpendicular to the last direction of ripping.

Feed to the solvent extraction circuit (1200 gpm) contains 3 grams per liter of copper and 2.8 grams per liter of iron. The organic solvent contains 92% kerosene (Napoleum 470) and 8% LIX 64 and LIX 64N. Original solvent was LIX 64 but now all make up is LIX 64N.

Three stages of mixing and settling are used to load the organic with copper. The raffinate contains .2 to .3 grams per liter copper and is recycled as barren solution. The organic is stripped with 150 grams per liter H_2SO_4. A bank of flotation cells separate any extrained organic out of the feed to electrowinning. This feed contains 40 gram per liter copper. Napoleum 470 is used to reduce misting in the tankhouse. Spent electrolyte containing 35 gram per liter copper is recycled. Production is 30,000 to 40,000 pounds of 99.9% copper per day.

RECOMMENDATIONS FOR SILVER BELL'S LEACHING PROGRAM

After examination of other dump or heap leaching operations several possible variations in Silver Bell's dump preparation and solution distribution should be tried. The Oxide #2 leach dump would provide an excellent test area. The pregnant solution grades have fallen off and it can easily be isolated for sampling. The other operations visited all spent considerable more time and money on their leaching programs. Our older dumps have not been given the chance to produce effectively.

One immediate variation to be tried should be reducing the rate of solution addition per square foot of dump surface. Leaching more ponds with a smaller amount of solution to each pond may give the solution a better chance to percolate through the dump rather than wash through. All of the iron salts should be removed and the dump surface should be completely ripped. A deep trench, similar to Kennecott's, may prove beneficial in examining solution flows through the top part of the dump. Flooding the trench may be tried for a method of adding barren solution. Horizontal piping should be tried with trickling and/or sprays. In order to use distribution piping a sump and pump would have to be installed to create enough pressure for the flow through the pipes. Acid would be needed in order to keep scale from plugging the lines and valves. Vertical holes with flexible piping is another method which could be tested which would utilize the sump and pump.

Fred H. Lightner

Fred H. Lightner
Leach Plant Metallurgist

FHL:jca

cc: GWBossard
BLRickman
MTMason
TDHenderson
FHLightner
File

WES WHC Rich Leaching

SOCIETY OF MINING ENGINEERS of AIME

345 EAST 47TH STREET, NEW YORK, N. Y. 10017

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IN SITU LEACHING OF COPPER - PILOT PLANT TEST

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Research Director

E. E. Malouf

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Kennecott Copper Corporation
Salt Lake City, Utah

This paper is to be presented at the AIME CENTENNIAL ANNUAL MEETING - New York, New York - February 26 - March 4, 1971.

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INTRODUCTION

Sociologists and economists indicate that our current standard of living requires 28 pounds of copper per capita annually. In comparison, Japan consumes 22 pounds, oceania countries 11, Russia 7, and the underdeveloped countries, constituting better than two-thirds of the world's population, using less than one pound of copper per capita annually. Normal economic growth of industrialized nations, in association with an expected acceleration in the standard of living in underdeveloped nations, can only lead to an increasing demand upon the world's copper supply.

This increasing demand for copper, coupled with rising costs and depletion of readily accessible and richer ore deposits, requires development of economical techniques to recover copper from progressively leaner ores.

Consideration has been given to the use of nuclear devices to fracture low-grade mineral deposits with the subsequent leaching of the metal values. However, on closer examination this approach appears to generate more problems than it solves. In addition to the problems of processing radioactive solutions, is the problem of completely healing all previous fracturing of the mineralized rock by the plastic flow created in the tremendous release of energy from a nuclear device.

Recovery of copper from low-grade deposits has since turned to a consideration of the techniques employed in nature's natural process of secondary (supergene) sulfide enrichment. Numerous individuals in many companies have given thought to this process utilizing some of the techniques developed in the petroleum, potash, and sulfur industries.

In Place (Leaching)

7-30

From: J. J. Collins

To: Mr W L Mantz - Tucson

ia, Ndola.

Send copy to JSC — John, more and more people taking look at in situ leaching

(Continued from previous page)

FIELD NOTES FROM THE ARIZONA BUREAU OF MINES

Vol. 1, No. 4 | Earth Science and Mineral Resources in Arizona | Dec. 1971

W. L. R.
FEB 14 1972

LEACHING OXIDE COPPER

by
G. H. Roseveare

The leaching of copper oxide ore was first started in Arizona at Ajo in 1918. It is now practiced in many areas of the state and accounts for approximately 10 percent of the present copper production in Arizona.

Four types of leaching operations are used in Arizona—vat, agitation, heap and underground.

In vat leaching, the ore is ground to about one-quarter inch, put in tanks or vats and flooded with weak sulphuric acid. In about seven days the acid leaches out about 80 percent of the copper contained in the ore. The solution is removed from the vat, the copper is recovered by electrolisis and goes to market as 99.95 percent copper.

Fine ground ore treated in an agitator tank requires a short time of contact to obtain good recovery. The copper is removed from the pulp by counter current decantation in thickeners. The solution goes to iron precipitation and the copper mud is processed in smelters the same as copper concentrates.

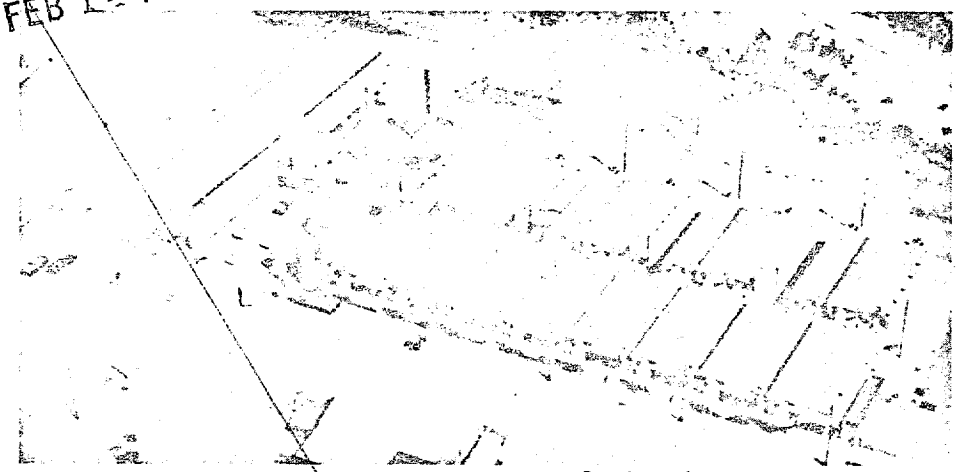
In heap leaching, the ore, with or without crushing, is piled in large heaps preferably on bedrock and acid is spread over the top by irrigation or sprays. The solution percolates through the heap and is collected at the bottom in ponds ready for precipitation. Recoveries vary from 35 percent to as high as 60 percent, but the method is slow—usually taking years. One metallurgist estimated that ten years is required to make a 51 percent recovery.

Several mines in the state are leaching cappings of old mines. Acid solutions are sprayed on the surface and trickle down through the oxide cappings and old filled stopes into old drifts where it is collected and pumped to the surface for precipitation.

Although most of the copper is precipitated by scrap iron or shredded tin cans and sent to a smelter for processing, the liquid ion exchange (LIX) method is being used at some mines and is producing metallic copper of 99.95 percent purity. The Blue Bird and Bagdad leaching plants are using this method and

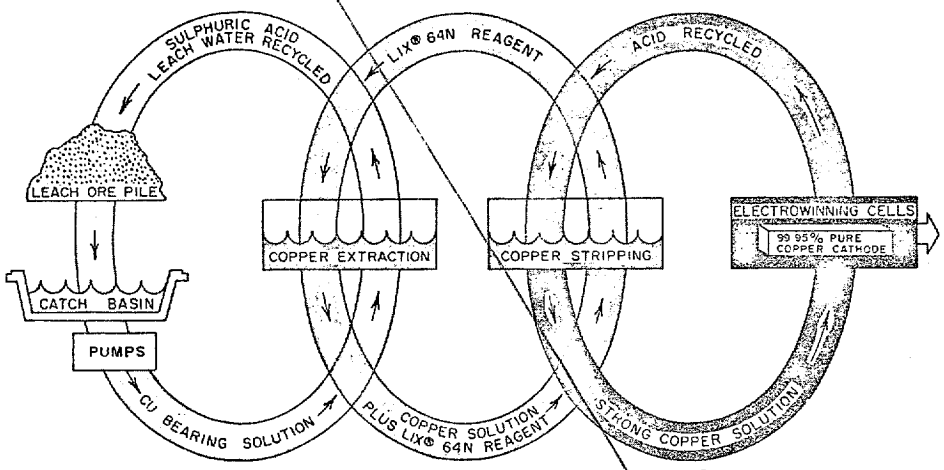
continued on page 2

LIQUID ION EXCHANGE



Courtesy of Bagdad Copper Corporation

Liquid Ion Exchange and Electrowinning Plant, Bagdad Copper Corporation.



Courtesy of General Mills

Copper recovery at Bagdad and Blue Bird begins with sprinkling of weak acid solutions on the copper oxide ore dumps. Acid dissolves copper as it trickles through the ore pile and the resulting copper bearing solutions are pumped to a catch basin. Next, the weak copper solution from the catch basin is mixed with the General Mills' LIX 64N reagent which has a high degree of affinity for copper ions and a low affinity for other contaminating ions, including iron. The reagent operates on a hydrogen ion cycle, which in general proceeds as follows:

The reagent carried in an organic medium, is intimately contacted with the aqueous leach solution in the extraction stages where hydrogen ions are exchanged for copper ions. Thus the sulfuric acid content of the leach solution is regenerated while the copper is extracted. The organic solution containing the copper passes to the stripping stages where it is contacted with sulfuric acid. The LIX 64N is thus regenerated and recycled to the extraction system. The enriched copper sulfate solution is an essentially impurity-free concentrated electrolyte from which high quality cathode copper is produced by electrowinning.

An electric current is passed through the solution and this deposites the copper on thin copper sheets called "starter sheets." The electrowinning circuit produces the sulfuric acid required in the stripping of copper from the organic medium. In about a week the starter sheets grow to approximately 130 pounds of copper. Bagdad and Blue Bird produce about 35 tons of electrolytic copper per day.

Leaching (continued)

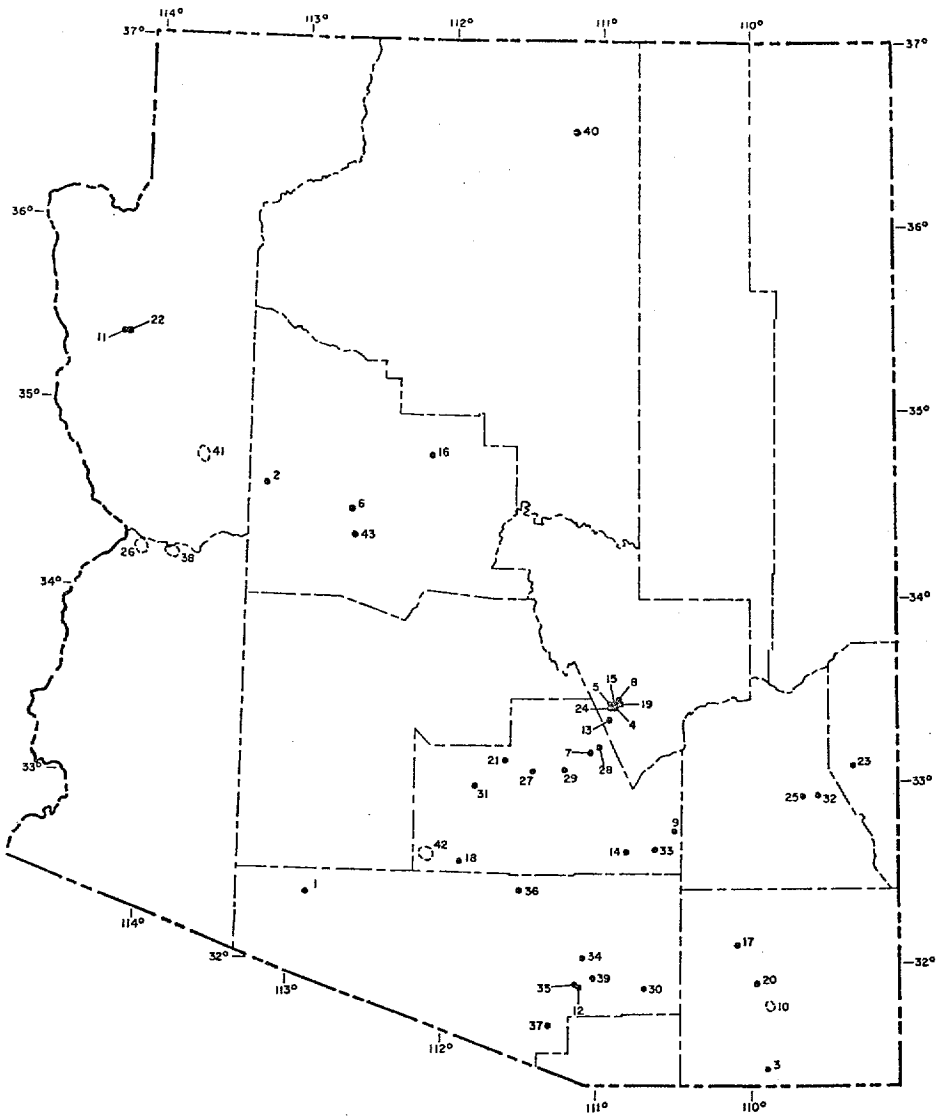
producing about 35 tons of copper per day. Although the future of this process looks promising for large operations, it is too expensive for the small producer whose property has only a short life expectancy.

Oxide copper ores in limestone or containing calcite have not been treated by leaching in the past because they require large quantities of acid to neutralize the lime before any leaching of the copper takes place. This type of ore may require from 150 to 300 pounds of acid per ton of ore to obtain a good copper recovery. However, since there will be considerable cheap sulphuric acid available in the near future resulting from the large production due to smelters converting sulphur dioxide to acid, high lime ores will become attractive as a source of copper.

The high lime ore consumes an enormous amount of acid and cannot be leached in heaps because in neutralizing the calcite, gypsum is formed and this soon blocks off the solution circulation. The Arizona Bureau of Mines has made several tests on ore ground to minus 20-mesh that required 160 pounds of acid per ton and was leached successfully in a pachuca tank. Approximately 85 percent of the copper was dissolved in 3 to 5 hours.

The high lime ores will require up to 8 times as much sulphuric acid as the non-lime bearing ores and will use much of the cheap acid which will be available at the smelters putting in sulphuric acid plants to reduce the sulphur dixide polution problem.

LEACHING IN ARIZONA



FLOTATION OF CONVERTER SLAG

Metallurgists are interested in a new use of flotation. Smelters produce considerable slag from their converters containing from 1 to 6 percent copper which is poured into the reverberatory furnace. This slag contains considerable magnetite that forms a mushy layer between the slag and the matte which hinders the copper-iron sulphide, or matte, from settling from the slag.

The U.S. Bureau of Mines recently published Report of Investigation 7562 on the flotation of copper-bearing material from converter slag, and currently an Arizona company is experimenting with the separation of the copper-bearing material from convertor slag in the mill by means of fine grinding and flotation. Additionally, Cia Minera de Cananea, S.A. de C.V., has made several runs using converter slag. It has been found that treating the slag separately from the ore gives the best results. The slag is crushed and ground at about half the feed rate of the ore. The amount of collector used is abnormally high, approximately 0.6 pounds per ton of slag. The concentrate thus far produced from slag assays approximately the same as concentrate produced from ore and the tailing assays about half that of the smelter slag. Smelters which may employ flash smelting will be able to use converter slag concentrate along with regular concentrate from ore as feed to the new type of furnace.

CHEMICAL ENGINEERS AND MINING INDUSTRY

The University of Arizona is fortunate to have its Department of Chemical Engineering within its College of Mines. Here the Chemical Engineers can work closely with the Department of Metallurgical Engineering in research of particular benefit to the mining industry. Techniques developed originally for the chemical or petroleum industries are being applied to mining and milling processes.

For example, beginning in 1967, cooperative Chemical-Metallurgical studies directed by the Chemical Engineering staff resulted in two unique advances in the state-of-the-art of large tonnage grinding circuits.

In 1967, Dr. Edward J. Freeh, (Chemical Engineering faculty) realizing the importance of grinding in efficient concentrator operation, recognized the existence of a lack of understanding of the mechanics of comminution. As advisor to Dr. W. E. Horst (then a graduate student in Metallurgical Engineering), Ed Freeh initiated a research program to evaluate the mechanisms and effectiveness of alternative grinding processes. Computer modeling and subsequent experimental verifications of a model by Horst and Freeh resulted in a "best paper award" when published in a technical journal. Upon graduation Dr.

LEACHING IN ARIZONA

Name of Property or Location Number Keyed to Map	Tank	Heap	Liquid extraction	Proposed	Potential	Name of Property or Location Number Keyed to Map	Tank	Heap	Liquid extraction	Proposed	Potential
1. Ajo	O				+	23. Morenci		●			
2. Bagdad		●	●			24. Ox-Hide		●			
3. Bisbee		●				25. Peacock		●			
4. Blue Bird		●	●			26. Planet area	O	O			
5. Castle Dome		O				27. Poston Butte					+
6. Copper Basin		O			+	28. Roy	●	●			
7. Copper Butte					+	29. Red Hill		●			
8. Copper Cities		●				30. Rosemont					+
9. Copper Creek area					+	31. Sacatone					+
10. Courtland-Gleason area	O					32. Sanchez					+
11. Emerald Isle **	●					33. San Manuel				+	
12. Esperanza		●	O			34. San Xavier		●			
13. Gibson Mine		●				35. Sierrita		●			
14. Gold Hill					+	36. Silver Bell		●			
15. Inspiration *	●	●				37. Silver Ray		O			
16. Jerome					+	38. Swansea area		O			+
17. Johnson Comp					+	39. Twin Buttes				+	
18. Lake Shore				+		40. White Mesa	O				
19. Miami *		O				41. Wickieup area					+
20. Middle March		O				42. Vekol area					+
21. Mineral Butte					+	43. Zonia		●			
22. Mineral Park		●									

● operating
O not operating
* underground
** floating precipitated copper

Horst continued his studies of grinding and comminution mechanics, and now lectures regularly at the College of Mines as well as consults in the fields of concentrator design and computer modeling.

Second, and concurrent with the work of Dr. Freeh, Dr. R. M. Edwards (Chemical Engineering) initiated an experimental and computer simulation of hydrocyclone operation. This study coupled computer models of the open-loop comminution process with those of the hydrocyclone for closed-circuit grinding. Dr. Edwards had previously interacted with Dr. T. M. Morris (Metallurgical Engineering) in the study of applications of special purpose analog computers and modeling to froth flotation. These studies outlined techniques which could provide mill operators with low cost on-site guidance, not available in current conventional operations.

Later, Dr. Alan D. Randolph (Chemical Engineering) used a new approach to the study of comminution; i.e., formulation of fundamental functional relationships (adapted from population birth and death statistical functions) as applied to particle-size distribution in crystallization. The predictive capability of this theoretical approach has been validated by an M.S. graduate student in Chemical Engineering, Mr. Tony Durando, by comparing open-circuit grinding data with the computer simulation results. This work in the College of Mines Metallurgy Department is being expanded by two of Alan Randolph's students, Jose Larios and Ricardo Cornejo. Computer modeling is being employed to validate theoretical projections and to expand the spread of basic data on ball mill parameters and variations in the characteristics of concentrator feed or mill heads.

In addition to his pioneering work of theories of comminution, Alan Randolph this year initiated studies of alternate processes for the removal of iron oxides from recycled acidic copper leach liquors; e.g., iron launder OFF-solutions. One process uses a high-slurry-density reactor–crystallizer to eliminate the need for acid neutralization. Since the liquor remains acid, the copper is not lost by entrainment in the flocculated basic-iron precipitates which ensue if the solution is neutralized. A Chemical Engineering doctoral candidate, David Milligan, will study the parameters of this process using experimental apparatus now under construction.

The simulation of single process units for specific operating conditions and changing ore characteristics is but the first step toward a practical application of this research. In the design of a new concentrator or in the operation of any given concentrator, the interaction of grinding and flotation circuits with variations in mill feed compositions and flow are of primary concern to the mining industry. Low cost techniques to evaluate proposed control philosophies are important and this is where a system approach is useful. A newly-appointed member of the College of Mines faculty, Dr. James W. White (Chemical Engineering), is working with both chemical and metallurgical engineers to define the requirements for useful systems for data analysis. The goal is model building and computer simulation of the various steps in ore processing flowsheets plus the regulatory philosophies involved. Dr. White currently has a Chemical Engineering graduate student, Bill Root, studying grinding circuit performance and cost/performance factors. Another Chemical Engineering graduate student, Mark Richardson, and Bill Root, are looking at the flotation/grinding circuit operations which are of interest to Alan Randolph. Dr. White and the College of Mines Computer Committee are planning the expansion of the Mines Computing Laboratory to cover metallurgical instrument analysis and, especially, the analysis of data from measuring devices in operating mines, mills and smelters in the field.

The results of these computer modeling studies of metallurgical processes appear very encouraging. The useful data developed thus far justifies the effort. Consequently, the interdisciplinary aspects of the research program and the communication with industry are being stepped up. The College of Mines at the University of Arizona hopes the result will be a maximum utilization of the new knowledge being generated by these studies.

ENGINEERS BREAKFAST

At the annual Engineers' Breakfast on October 28, 1971, George E. Atwood, Executive Vice-President of Duval Corporation, and Lawrence K. Cecil, retired chemist and consultant on water, environmental, and pollution problems, received the University of Arizona Alumni Associ-

Continued on page 4

ark Leaching

~~JCB, GBS, JDS~~ → W L Kurtz

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S. W. U. S. EXPL. DIV.

ASARCO

AMERICAN SMELTING AND REFINING COMPANY
ROCKY MOUNTAIN EXPLORATION DIVISION
1805 SOUTH BELLAIRE STREET. DENVER. COLORADO 80222

TELEPHONE 303-757-5107

September 9, 1971

Mr. John J. Collins
Director of Exploration
American Smelting and Refining Company
120 Broadway
New York, New York 10005

Leaching Ore

Dear Sir:

In our copper investigations in Texas and Oklahoma, we have passively considered the possibilities of acid leaching.

We are quite confident that we can develop significant tonnages of copper bearing rock, which could be mined and leached at a profit, on the basis of acid having only a transportation cost. These would be strip operations, ranging from about one to three thousand tons a day of ore tonnage, with copper grades varying, depending on the area, from .5% to about 3.5% Cu. (There are very limited tonnage potentials of 6. to 11.% Cu).

Several questions arise for active consideration of leaching.

(1) Will Hayden be the only producer of excess (free) acid?
(2) If not, what other smelters will produce acid?
(3) How much acid will be available?
(4) Can we consider cheap (in the range of .10 - .20¢/cu yd.) strip coal mining costs as realistic mining costs?

If Amarillo or El Paso should produce excess acid, acid costs to a mine in Oklahoma or central Texas would be much less than acid coming from Hayden.

The following is a brief note outline of some of the various leaching type deposits available, and their exploration outlook.

Leachable copper in igneous rocks:

(1) Copper King, Wyoming

5 - 10 million tons of open pit material of mixed oxides and sulphides at ± 1.00% Cu, near railroad. May be another similar deposit a few miles north.

Disadvantages of leaching operation:

a. Rock will require blasting.
b. Lose gold values,

(2) Oro Grande, New Mexico

May have exploration potential for leachable copper, and is about 50 miles from El Paso smelter.

Disadvantages:

Rock will require blasting, grade and tonnage potential essentially unknown.

(3) Torpedo Fault Zone, Organ District, New Mexico

About 50 miles from El Paso Smelter. Has probable leachable tonnage potential, copper as chrysocolla.

Disadvantages:

Area now subdivided into expensive housing sites, and crossed by Interstate Freeway, which precludes mining.

Copper or other metals in sandstone:

(1) Baca County, Colorado, Northeastern New Mexico, and Oklahoma Panhandle.

Copper occurs primarily as disseminated carbonates and chalcocite in a quartzose, friable, medium to coarse grained porous sandstone, from 10 to 20 feet thick. Copper is known to occur over a number of miles of strike length, and a number of miles of "width". Grades run from a few tenths to over one percent Cu.

Outlook:

Exploration potential exists for several 5 - 10 million ton or larger open pit chalcocite ore bodies. Grades would be better than 1% Cu, with minor silver, in a porous rock which would not require grinding for leaching. Much of area is open for staking, and private land owners are reasonable.

Disadvantages:

Rock may require blasting, long acid freight haul.

(2) High Rolls, New Mexico

Leachable potential exists for (+ 10% Pb) lead carbonates with sulfates and low grade copper oxides, in an arkose, close to the El Paso smelter, some ground probably available for staking.

Disadvantages:

Lead oxide will change to sulphides down dip; copper potential uncertain; near resort area.

(3) Numerous relatively unknown sandstone coppers occur in New Mexico, but tonnage potentials for individual deposits vary widely, from poor to excellent. As an example, the up dip, dip slope, outcrop extension of the copper deposit now being mined at Cuba, New Mexico, has low grade disseminated oxide copper mineralization over a dip length of about two miles. Source of acid and nearness of the deposit to railroad transportation will determine the significance of particular areas.

(4) Sandstone coppers in Texas and Oklahoma.

Clear evidence of leaching in clean beds, with copper, etc., is observable. Similar contraints as in (3) above apply.

Copper or other metals in siltstones and shales:

Copper in grades from .75 to about 3.5% Cu occurs in persistent beds with known thicknesses ranging from about 8 to 18 inches. Prospects can be selected so that the essentially flat beds are only a relatively

few feet below the ground surface over large areas. Substantial stike and dip persistance can be demonstrated. As an example of persistance, samples for 10 miles in down dip direction in one area show an average thickness of about .7 foot at .83% Cu. No obvious bounds on aerial potential in such terraine is presently known.

Outcrop areas controlled by the Lobaris Copper Company north of Creta, Oklahoma, show 12 to 18 inches of oxide copper-bearing siltstones and shales, with an interbedded brown silty sandstone. Copper grades may be on the order of 3% Cu. The presence of the permeable sandy unit may make the area quite attractive for heap or vat leaching purposes. There is no known significant down dip exploration in this area, although chalcocite is known to occur at distance, down dip.

Several other different copper bearing beds are known to occur in the Texas-Oklahoma area. Persistance of individual beds can be measured in tens of miles along strike. Blasting in these soft rocks would not normally be a part of the mining process, and mining operations would follow strip coal mining practice.

If we assume that we can achieve mining costs comparable to coal mine stripping, the general area has substantial exploration potential for significant tonnages of open pit copper bearing rock, with the copper minerals amenable to acid leaching.

We have been proceeding with stratigraphic studies in the Texas-Oklahoma area to try and determine where, in the down dip direction, thicknesses and grades will increase, so that the copper bearing beds would support normal mining and milling processes. For acid disposal purposes, thicknesses and grades presently evident should be seriously considered as showing potential for a number of small to medium size mining operations, each of which would have a relatively low capital investment, and a corresponding high rate of return. It is our opinion that with low cost acid, such a large copper resource, as is evident in these states, will be mined.

We should point out, in addition, that if there is any interest in high acid consuming rocks, such as Anaconda is considering at Butte, thin copper bearing dolomites generally overlie copper bearing siltstones in Oklahoma and Texas. With careful study, it should be possible to find such dolomitic rocks in thicknesses great enough to be of interest.

Very truly yours,

Stephen Von Fay

Stephen Von Fay

SVF/jlk
cc: WLKurtz ✓

Note: High Rolls, p. 3

Exploration for lead sulfides between High Rolls and the porous Abo reef environment down dip and not far to the southeast deserves more consideration then it has received.

SVF

[redacted]

Pb Cu Technical Learned Cuttings Mr. Sorgen! S-10.10

Australia - WES

AMERICAN SMELTING AND REFINING COMPANY
SILVER BELL UNIT
Silver Bell, Arizona

W.E.S.
APR 20 1971

April 14, 1971

MEMORANDUM TO: Mr. J. H. Courtright, Chief Geologist

Re: FREIDA RIVER PROSPECT MINERALOGY REPORT

By: J. W. Cameron - Dated January 26, 1971

INTRODUCTION

In our telephone conversations, several ideas in the report were questioned. These ideas include:

A. The pyrite content and its involved implications.
B. Oxidation below the water table, in general.
C. Bornite, chalcocite and covellite as products of oxidation replacements; in general and in the Freida River samples.
D. "Subsequent oxidation has taken place."
E. "Supergene enrichment is not indicated in this 18 gram sample.", or oxidation in situ dominates.

Each of these queries will be discussed and justified considering theoretical and emperical knowledge correlated with the mineralogical facts collected during the initial microscopic investigation and subsequent electron beam microprobe investigation. Personal communications with other individuals concerning the mineralogical implications will also be mentioned. One must keep in mind the author was working with only one sample, rather than an entire suite; therefore, statements made directly involve only this sample even though some of the implications from this one sample could and very likely encompass a much larger volume of rock. Examination of a complete suite of rocks would be desirable to both broader the exact mineralogical information and correlate the possible economic/exploration factors involved (these will be discussed later).

DISCUSSION

Known mineralogy data in the 18 gram sample included:

1. Large amounts of magnetite (iron and oxygen) with moderate amounts replaced by hematite.
2. Chalcopyrite commonly rimmed by bornite, chalcocite and covellite, in some areas the chalcopyrite showed no replacement by later minerals.
3. Some primary, but more secondary bornite, bornite replaced by chalcocite and covellite.

4. Covellite was in contact with chalcocite as inclusions, stringers and rims.
5. Very low ($<0.1\%$) amounts of pyrite.
6. The gangue mineralogy was quartz, biotite, chlorite; therefore, chemically neutral.

A. PYRITE CONTENT:

The pyrite content was extremely low in the two polished section areas. Pyrite occurred only along one vein. This low pyrite content along with high total content of iron and oxygen (magnetite) suggests:

7. The hydrothermal solutions were deficient in sulfur relative to the total iron and copper content. The result was that large amounts of magnetite and chalcopyrite and only negible amounts of pyrite and bornite were precipitated as primary minerals.
8. If there is a deficiency in pyrite, then there is a poor change that leaching and secondary enrichment would take place. "Iron sulfides are essential, and deposits lacking them rarely contain supergene sulfide zones. Chalcopyrite alone does not yield sufficient solvents, and deposits like Ajo, Arizona" (second oxidizing cycle)", where this mineral predominates in the primary ore, have negligible supergene enrichment" (Bateman, p 275). Blanchard does state (p 48), "Where the air-water oxidation processes operate above the water table — especially the semi arid-arid regions" (which is not the Freida River case) "... The oxidation and hydrolysis of chalcopyrite in dilute solutions" (neutral) "may yield its own equivalent in free acid".

B. OXIDATION BELOW THE WATER TABLE:

Blanchard states (p 43), "The reader however, must guard against a misconception which prevails among some geologists, namely, that an acid solution percolating down from above and reaching the water table, immediately spreads out laterally, and through dilution or diffusion dissipates itself rapidly to the point of impotence." Also, "the observations showed that where the normal fracturing has a general vertical continuity, with gangues of weak neutralizing power such as quartz-sericite schist, siliceous shale, or veins with dominantly quartz gangues, the solutions derived from sulfides directly below the water table may descend more than 300 or 400 feet with their acidity only slightly decreased, especially in pyrrhotite or sphalerite ores which are easy to oxidize."

Limited circulation with limited quantities of oxidizing agents does not hinder development of oxide products such as ones that were seen in the 18 gram sample. Indeed, this type of environment is critically necessary for their development. If a strongly oxidizing environment existed, then the copper or copper-iron (or other metals) minerals would be completely oxidized and/or leached producing oxides, silicates, carbonates or mobile copper-rich solutions. The point here is that the Freida River Prospect sample does show that redox conditions in the sample are, or always were, more similar to primary conditions than is reflected in an arid surface environment such as Silver Bell.

C. BORNITE, CHALCOCITE AND COVELLITE AS PRODUCTS OF OXIDATION REPLACEMENT:

Several factors give the approximate Eh-pH conditions of the environment involved in the Freida River sample. All of the factors suggest a "depth environment" (Sato, I) with conditions somewhat similar to the ground water zone of a typical Southwestern supergene enrichment environment.

The occurrance of the phase associations present in the sample, magnetite-chalcopyrite-bornite-pyrite and hematite-bornite-chalcocite-covellite defines the redox conditions as given by Garrels' and Sato's diagrams. Figures 1, 2, and 3, show the approximate position of some natural environments as characterized by Eh and pH. Figure 1 does not give the total range of all noted environments. The area of interest is labeled "ENVIRONMENTS ISOLATED FROM THE ATMOSPHERE" as will be pointed out below. Figure 2 (Sato, I) gives the probable ranges of Eh-pH values for both the weathering and the depth environment. Figure 3 (Garrels) gives the redox region for hydrothermal acitvity. Figure 4 (Garrels) shows the system Cu-Fe-S-O-H (in part) at 25°C and 1 atmosphere total pressure (total dissolved sulfur = 10^{-4}m). Garrels (p 162) states, these figures "serve to show expected stability fields of a large number of compounds under conditions corresponding to those encountered in nature during the processes of oxidation and secondary enrichment of copper ores."

Four areas have been marked on Figure 4: A, that which represents the primary minerals environment; B, that which represents the minerals in the Freida River sample; C, that which represents copper in solution; and D, a typical Southwest supergene enrichment zone. Note the comparison between the hydrothermal region of Figure 3 and the "A area" on Figure 4; also, "Transitional" areas on Figures 1 and 2 with the "B and C areas" on Figure 4. In moving from "area A" to area B" (oxidation in situ) on Figure 4, one moves vertically or in the direction of increased odixation. Supergene enrichment would involve moving from "A" to "D" via "C", therefore, also an oxidation process. To make this oxidation trend more obvious, Figure 5 has been added. Figure 5 (Garrels) shows log P_{S_2} and log P_{O_2} conditions for the primary and secondary (area A and B of Figure 5, respectively) minerals in the Freida River sample. In moving from area A to area B, one moves in the direction of increased oxidation.

Bateman and Blanchard gives chemical reactions for the replacement of chalcopyrite, bornite and chalcocite in a supergene environment:

9. $Cu^{+2}S^{-2}Fe^{+2}S^{-2} + 4O^{0}_{2} = Cu^{+2}S^{+6}O^{-2}_{4} + Fe^{+2}S^{+6}O^{-2}_{4}$, at 25°C, 1 atmosphere and

10. $Cu^{+2}S^{-2}Fe^{+2}S^{-2} + Cu^{+2}S^{+6}O^{-2}_{4} = 2Cu^{+2}S^{-2} + Fe^{+2}S^{+6}O^{-2}_{4}$.

Upon examining the positive and negative valences one observes that only the sulfur and oxygen in equation 9 change valence states (+2 to +6, and 0 to -2, respectively); therefore the sulfur is being oxidized. The copper and iron sulfates are initially in solution and may be transported. When proper conditions exist the Cu in the $CuSO_4$ will precipitate out replacing an Fe site, e.g. in chalcopyrite. This reaction is equation 10. One observes that the Cu in the $CuSO_4$ is combining with sulfur (as CuS) or is being reduced even though no valence change occurs. The Fe in the chalcopyrite (CuSFeS in equation 10) is combining with oxygen (as $FeSO_4$) and is being oxidized. Therefore, the mineral chalcopyrite undergoes oxidation in the production of supergene covellite, e.i., the chalcopyrite is being oxidized to covellite and the $CuSO_4$ is the oxidizing agent.

Garrels

by Sato (Motoaki Sato, 1958, A study of the electrochemistry of self-potentials associated with sulfide ore bodies. Paper presented at American Institute of Mining and Metallurgical Engineers, Annual

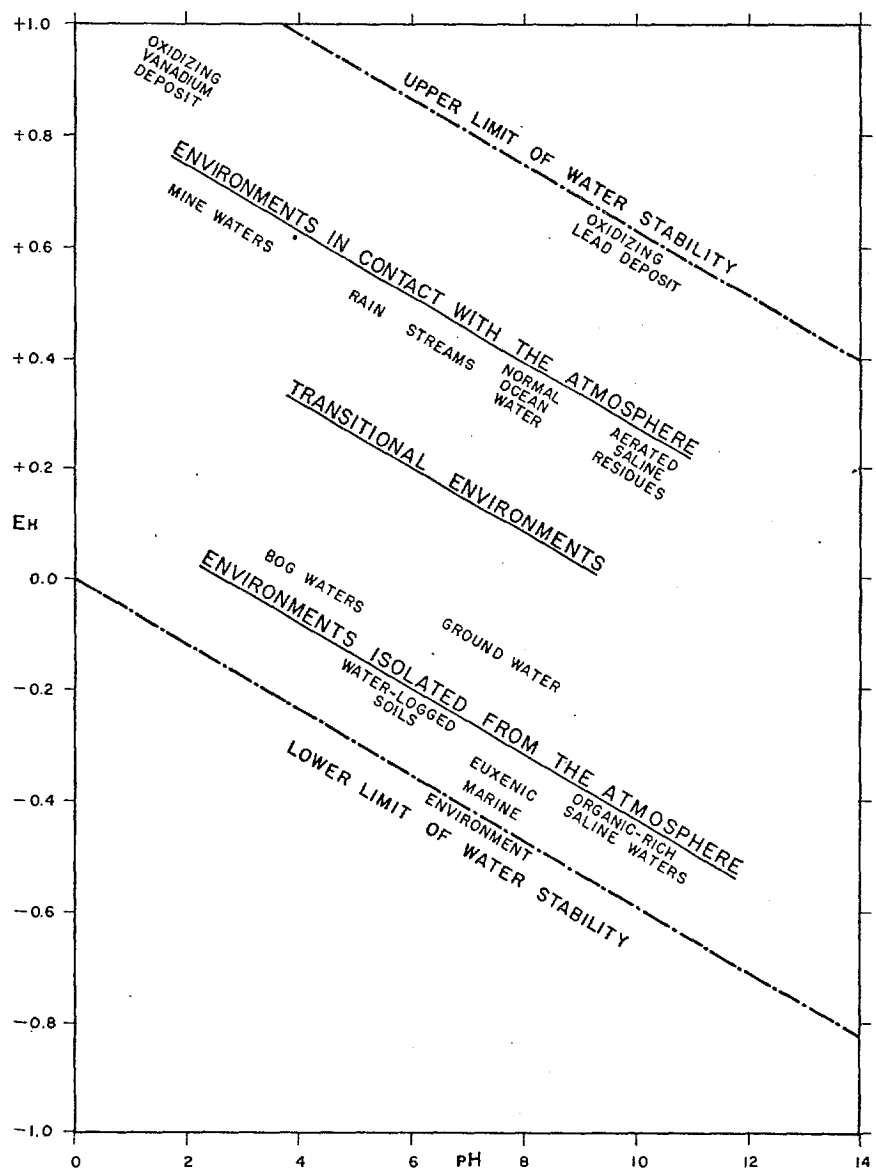


FIGURE 7.1. Approximate position of some natural environments as characterized by Eh and pH.

M. Sato

is reduced:

$$2Fe_3O_4 + H_2O = 3Fe_2O_3 + 2H^+ + 2e, \quad Eh = 0.221 - 0.0591\ pH \quad (9)$$

may be taken as the lower marginal value of Eh for the depth environment. Anomalous low values may exist locally due to active reduction by H_2, CH_4, and other strong reducing agents produced during organic decomposition.

The same principles for defining the range of pH as those discussed for the weathering environment may apply to the depth environment, namely, the equilibrium pH value for the calcite-water system without an external CO_2

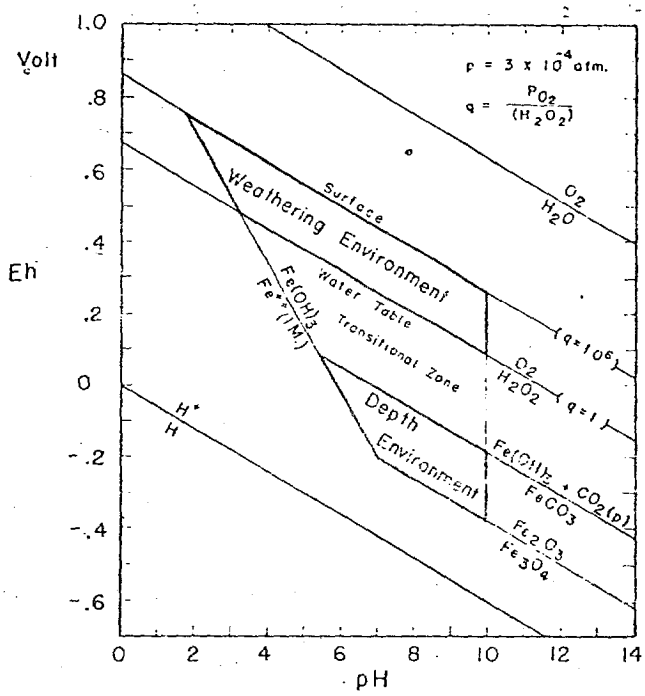


FIG. 9. The probable ranges of Eh-pH values for both the weathering environment and the depth environment.

pressure at the basic side, and that for the ferric hydroxide-ferrous ion (1 molal) system at the acidic side. The validity of the former is established by very common occurrence of calcite in various rocks. The latter is given as a function of Eh. Although equation (12") is used to define the lower marginal value of pH for both environments, the equilibrium pH value for the depth environment as given by this equation is much higher because of more negative Eh value. The pH value at which the dissociation and oxidation of siderite take place simultaneously under a given partial pressure of CO_2 also corresponds to this value.

The marginal values of Eh and pH thus defined for both the weathering environment and the depth environment enclose two separate regions on a

Washington, D.C., 1957, 161–164) has done a beautiful piece of work on stability relations of iron oxides and silicates as a function of P_{O_2}. But we are almost totally lacking in data that can be used to obtain standard free energies of formation of ions in aqueous solution. In many instances the temperature coefficient of the standard free energy can be obtained at 25°C, but extrapolation upwards is dangerous business.

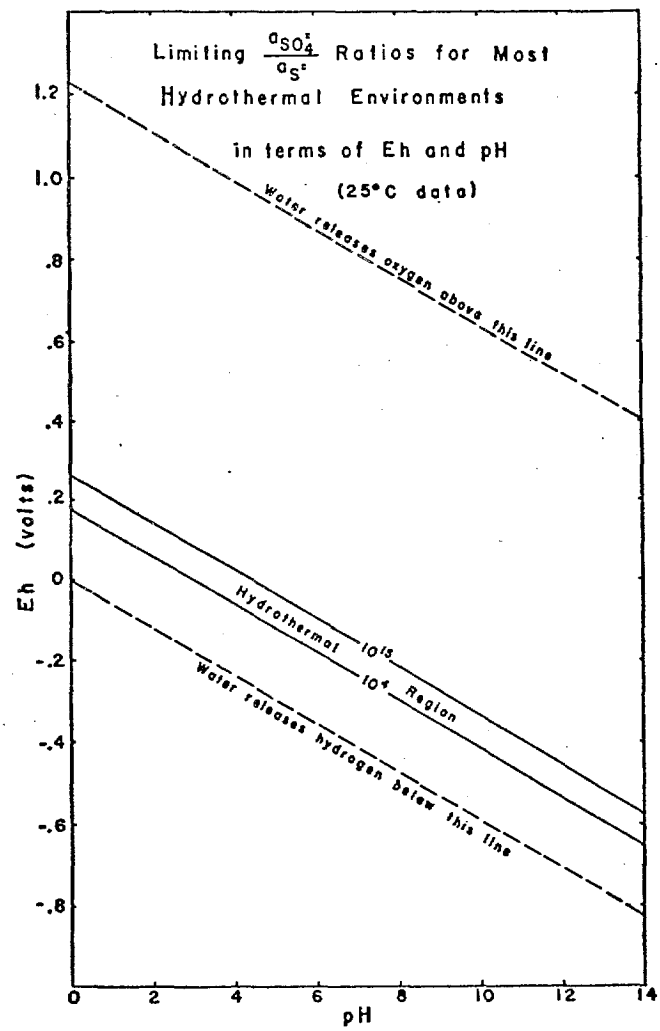


FIGURE 7.9. Environment of hydrothermal ore deposition as interpreted by Barton. (From Paul B. Barton, Jr., 1957, Some limitations on the possible composition of the ore-forming fluid. *Econ. Geol.*, *52*, 345.)

Garrels

3

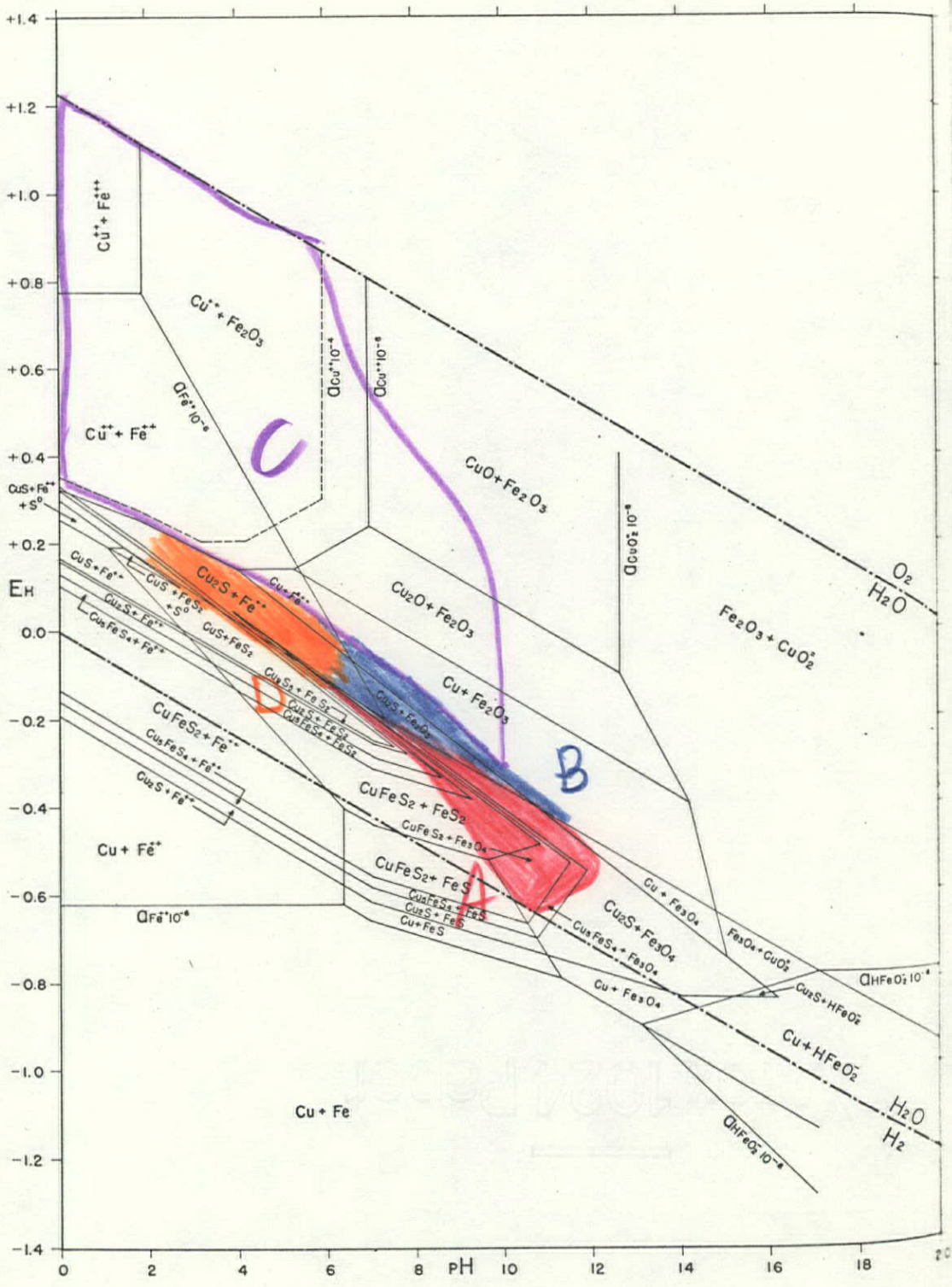


FIGURE 6.25 (~~opposite page~~). The system Cu–Fe–S–O–H (in part) at 25°C and 1 atmosphere total pressure. Total dissolved sulfur = 10^{-4} m. (Courtesy R. Natarajan and R. Garrels.)

4a

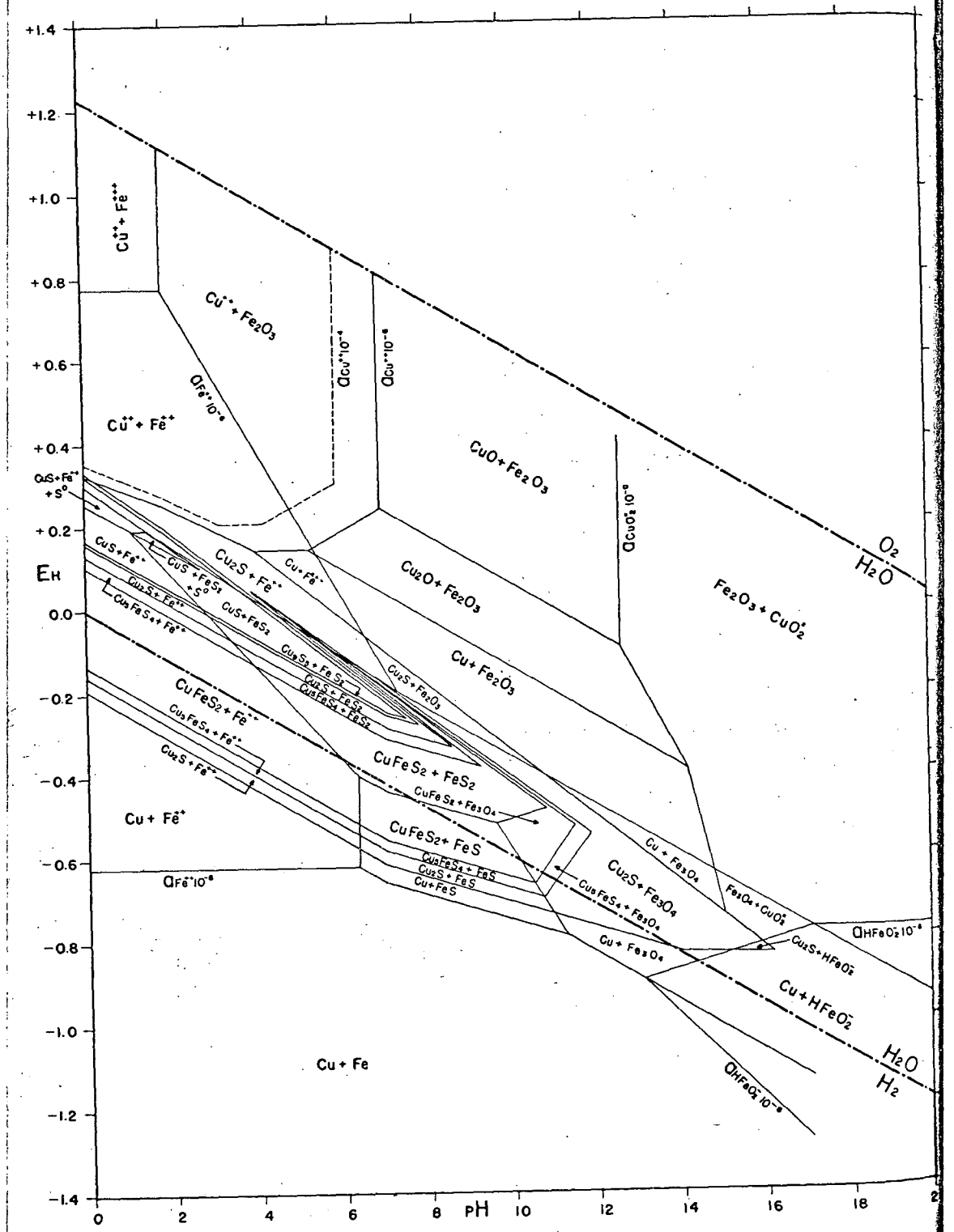


FIGURE 6.25 (*opposite page*). The system Cu–Fe–S–O–H (in part) at 25°C and 1 atmosphere total pressure. Total dissolved sulfur = 10^{-4} m. (Courtesy R. Natarajan and R. Garrels.)

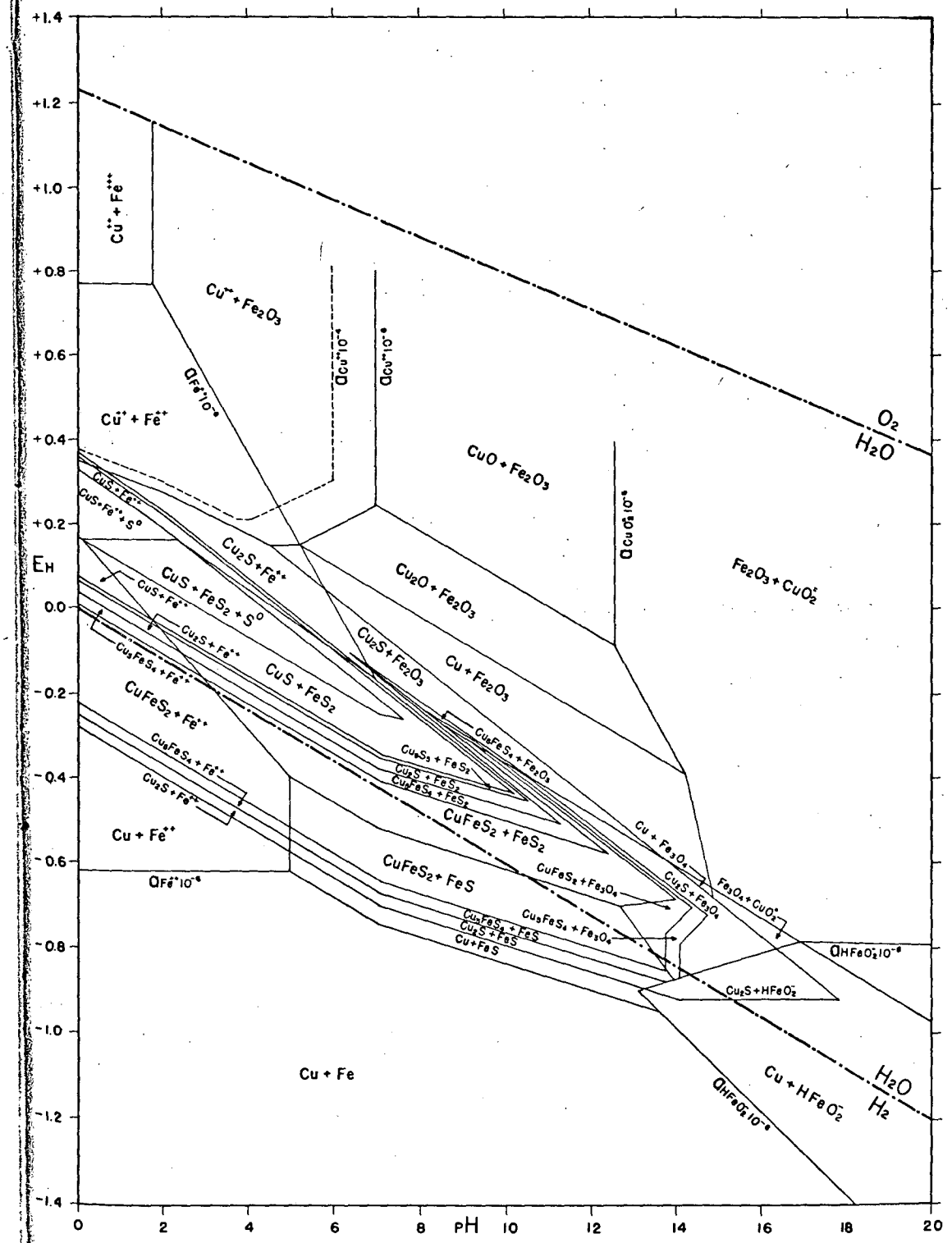


FIGURE 6.24 (~~opposite page~~). The system Cu–Fe–S–O–H (in part) at 25°C and 1 atmosphere total pressure. Total dissolved sulfur = 10^{-1} m. (Courtesy R. Natarajan and R. Garrels.)

Garrels

line for the boundary $Cu_5FeS_4 + FeS = CuFeS_2 + FeS$ (D) cannot be placed on the diagram unless it is within 1 or 2 log units of P_{S_2}. Furthermore, because the *slopes* of all equilibria are known, irrespective

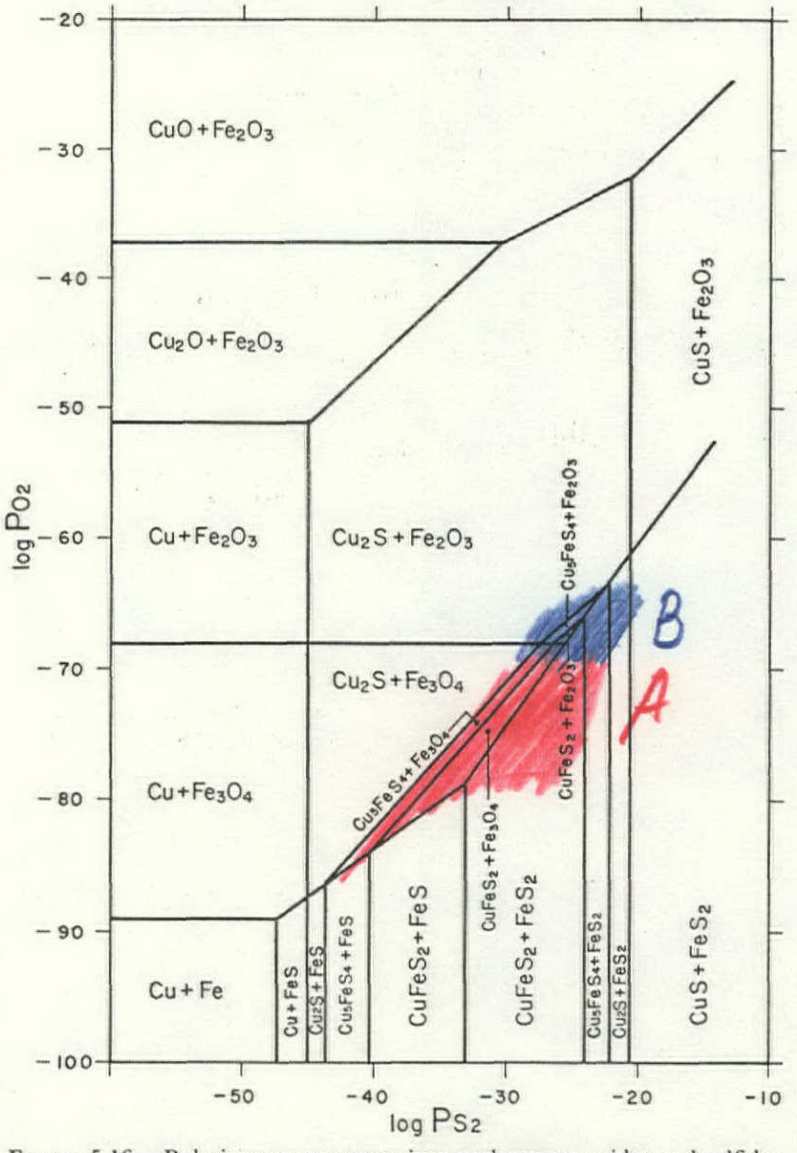


FIGURE 5.16. Relations among some iron and copper oxides and sulfides at 25°C and 1 atmosphere total pressure. Heavy lines are from superposition of iron and copper P_{O_2}–P_{S_2} diagrams; lighter lines are relations deduced from mineral associations. (R. Natarajan and R. Garrels, unpublished manuscript, 1958.)

5

Combination of equation 9 and 10 gives:

11. $2CuFeS_2 + 4O_2 = 2CuS + 2FeSO_4$

This is the reaction for in situ oxidation. Effectively the iron and sulfur in the chalcopyrite are being oxidized and the oxygen is being reduced, generating $FeSO_4$. Numerous intermediate reactions are possible. Also, reactions with other compounds, e.g. water, may alter the end products. In equation 11, the $CuSO_4$ of equation 9 may be generated but is less stable in solution than the $FeSO_4$ and therefore Cu is immediately precipitated out. The necessary redox conditions for in situ oxidation limit the degree of oxidation, transportation and leaching of Cu and Fe with development of mineral assembleges similar to, but distinct from, supergene enrichment mineral assembleges.

Bateman states that equation 9 and 10 are simplifications, in reality chalcopyrite and bornite are formed as transition products in the alteration of pyrite and chalcopyrite to chalcocite and covellite. Figure 6 shows the stability relations for some of the Cu-Fe sulfides, at 25°C, and details Bateman's statement. There is some question as to whether covellite or chalcocite forms first, with production of the other later. One should probably write schematically, (Bateman, p 277: Zies, p 407):

12. Pyrite + Cu^{+2} = chalcopyrite/bronite/chalcocite/covellite + ...
13. Chalcopyrite + Cu^{+2} = bornite + ...
14. Bornite + Cu^{+2} = chalcocite/covellite + ...
15. Djurleite and digenite may also be intermediate phases.
16. Idaite may also be an intermediate phase, especially in the oxidation in situ of bornite.

Sato (II) states, "The results indicate that the direct oxidation of a simple sulfide mineral is a process in which the metal atoms move into the surrounding solution to become aqueous cations accompanied by a step-wise decrease in the metal to sulfur ration of the remaining solid phase. When solid sulfur is finally left over, it undergoes a series of reactions to be oxidized to sulfate ion. In case of reduction, sulfur atoms move out into the solution to become aqueous sulfide ions with a step-wise increase in the metal to sulfur ration of the solid phase. To illustrate, the oxidation of chalcocite proceeds as:

17. $(Cu_2S) \rightarrow 1/5(Cu_9S_5 + Cu^{++})? \rightarrow (CuS + Cu^{++}) \rightarrow (2Cu^{++} + S)$.

The path of the step-wise oxidation is not necessarily the same as that of the step-wise reduction for a metal-sulfur system."

Sato (II) further states, "The probable oxidation steps for the metal-sulfur system involving the sulfide minerals studied are as follows:

18. Cu-S System: $(2Cu + S^{-2}) \rightarrow Cu_2S \rightarrow 1/5(Cu_9S_5 + Cu^{+2})? \rightarrow$
$(CuS + Cu^{+2}) \rightarrow (2\ Cu^{+2} + S)$.

In these reactions $(2Cu + S^{-2})$ represents chalcopyrite, bornite, etc. Schwarts (10) states that chalcocite, especially as a supergene mineral, commonly precedes cuprite in the oxidation cycle and therefore implies that chalcocite is the product of an oxidation of a previous phase.

Runnels (p 87) states, "The arrow" (Figure 7) "traces in a general way, the changes in bulk composition and mineralogy involved in the alteration of the bornite. Steps 1 and 2 along the arrow involve the loss of both iron and copper, probably through oxidation and leaching, whereas steps 3 and 4 probably reflect supergene enrichment." In other words chalcocite and covellite have been produced from oxidation and leaching as well as from supergene enrichment in the Ruby Creek deposit.

of compounds containing both iron and copper, such as chalcopyrite ($CuFeS_2$) and bornite (Cu_5FeS_4). R. Natarajan (unpublished manuscript, Department of Geology, Harvard University, 1958) has attempted to take into account such interactions, basing his interpretation of the position of the compounds on the diagrams of McKinstry and Kennedy [H. E. McKinstry and G. C. Kennedy, Some suggestions

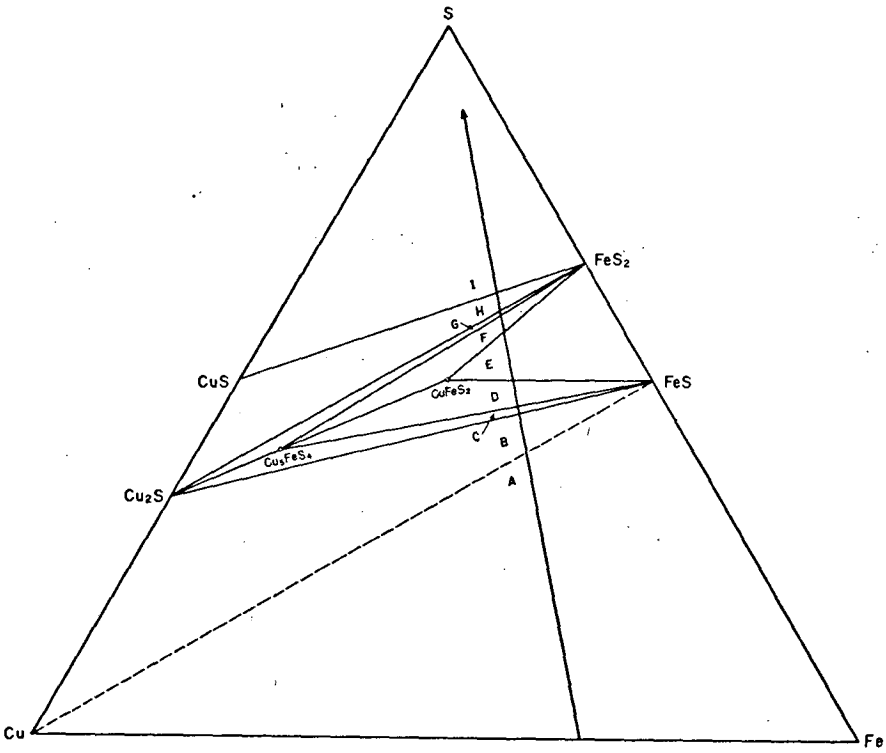


FIGURE 5.15. Stability relations in the system Cu-Fe-S at 25°C and 1 atmosphere total pressure, as deduced by R. Natarajan. There is some uncertainty concerning the composition of Cu_2S and FeS at this temperature.

concerning the sequence of certain ore minerals. *Econ. Geol.*, *52*, 379–390 (1957)], and McKinstry [H. E. McKinstry, Phase assemblages in sulfide ore deposits. *N.Y. Acad. Sci.*, Ser. II, *20*, 15–26 (1957)], who have assembled data on the natural associations in this system. For the 25°C relations, Natarajan studied the mineralogy of the zone of oxidation and secondary sulfide enrichment. The composition diagram for Cu-Fe-S_2 at 25°C, as deduced by Natarajan, is shown in Figure 5.15. The reactions involved in following the heavy arrow

Econ. Geol., v 64, p 75.

dolomitization in Mesozoic carbonates from the French Jura.

Malachite is fairly abundant, azurite less so, and cuprite is quite rare. Small amounts of supergene aragonite occur intimately mixed with sooty supergene sulfides and on the weathered surfaces of carbonate rocks.

Sulfides

Among the supergene sulfides, djurleite ($Cu_{1.96}S$) is the most abundant. The djurleite at Ruby Creek is most commonly present as a replacement of primary chalcopyrite.

Covellite is the second most abundant supergene sulfide. Supergene chalcocite and digenite are also present, but in the samples examined they are relatively rare. The distribution of the sooty chalcocite near the surface and along water courses distinguishes it from the coarse-grained, primary chalcocite.

Of particular interest is the supergene alteration of primary bornite. As illustrated in Figure 13, the first effect of the supergene alteration of bornite is to convert it to a fine-grained, reticulated mixture of covellite, chalcopyrite, and digenite. This early alteration begins along the boundaries between grains of the primary sulfides and gangue (Fig. 13). As the supergene alteration of bornite proceeds to more advanced stages, the digenite disappears and a two-phase mixture of supergene chalcopyrite and covellite appears. The chalcopyrite and covellite occur as a meshwork of tiny bladed grains. With continued alteration the meshwork of chalcopyrite and covellite is in turn veined and rimmed by pure covellite, and finally, rims of digenite surround the earlier products.

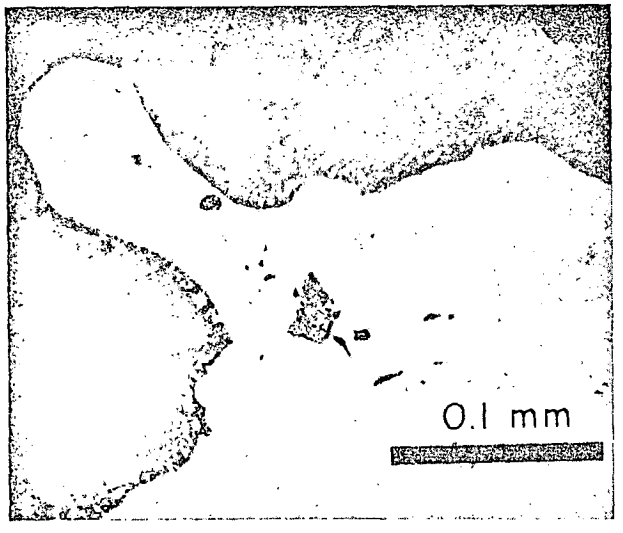


FIG. 13. Polished section of primary bornite (dark gray) and primary chalcopyrite, showing the first stages of supergene alteration. The contact between the primary grains is marked by a fine-grained mixture of supergene chalcopyrite, digenite, and covellite.

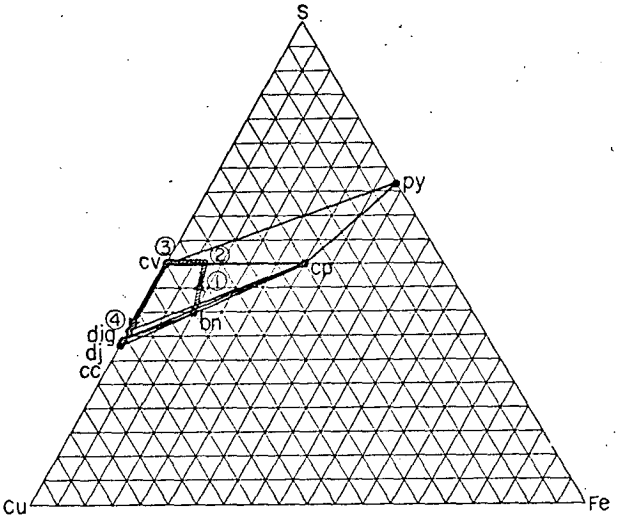


FIG. 14. Compositional triangle Cu-Fe-S showing phase relationships and changes in bulk composition during supergene alteration of primary bornite. Arrow traces path of alteration of bornite.

The changes involved in the supergene alteration of bornite are easily followed on a compositional triangle of the system Cu-Fe-S (Fig. 14). All of the tie-lines shown have been observed at Ruby Creek, with the single exception of dj-bn. McKinstry (1959) gives the tie-lines cv-bn and dig-bn instead of those shown in Figure 14 of dig-cp, dj-cp, and dj-bn. The assemblage dj-bn was not actually observed at Ruby Creek, but dj-cp and dig-cp-cv are definitely present. The arrow traces, in a general way, the changes in bulk composition and mineralogy involved in the alteration of the bornite. Steps 1 and 2 along the arrow involve the loss of both iron and copper, probably through oxidation and leaching, whereas steps 3 and 4 probably reflect supergene enrichment.

Other species

Small quantities of other minerals are also present in the zone of supergene alteration, including kaolinite, native copper, native silver, secondary uranium minerals (derived from traces of pitchblende), pseudomorphs of barite plus kaolinite after crystals of cymrite, and an unidentified weathering product of tennantite-tetrahedrite (Runnells, 1963).

Origin

Metals and Sulfur

With the sparse information which is now available, it is difficult to reach any firm conclusions concerning the origin of the deposit at Ruby Creek. There is no doubt that the metals were introduced into the host rocks from some outside source, but at this early stage in the development of the deposit

The sited references coupled with Figures 4, 5, 6, and 7 convencingly point out the fact that chalcopyrite, bornite and chalcocite are oxidized in the production of supergene chalcocite and/or covellite, regardless of whether the process is in situ oxidation or typical Southwestern supergene enrichment.

D. SUBSEQUENT OXIDATION HAS TAKEN PLACE

Sato (I) states, "In case both hematite and magnetite coexist in a rock, the Eh value of the solution long in contact with the rock would eventually approach the above value ($2Fe_3O_4 + H_2O = 3Fe_2O_3 + 2H^+ + 2e^-$, Eh = 0.221 - 0.0591 pH), the equilbrium with the two minerals being established." Therefore the straight line equation Eh = 0.221 - 0.0591 pH represents the lowest redox conditions of a sample containing hematite and points out a change in the direction of increased oxidation as compared to primary conditions. The trace of this equation is shown on Fig. 8 as the boundry between hematite and magnetite, plus H_2O. Analysis of the eight figures gives an idea of the redoz conditions involved in the Freida River sample as compared to primary conditions. The fact that the conditions have become more oxidizing is apparent. The figures suggests that the Freida River sample can be catagorized as a "depth environment" type having undergone distinct, but limited, oxidation.

That subsequent oxidation has taken place in the Freida River sample is also apparent from the magnetite-hematite relation. As Sato (I) points out, the development of hematite shows that conditions have become more oxidizing since deposition
(19) of the magnetite. Magnetite is stable in the reducing zone of an enrichment environment; however, as pointed out, 20% or more of the magnetite in the Freida River sample had been oxidized to hematite. This association indicates something other than a supergene enrichment environment for the Freida River sample.

The author believes that the above statements with the volumous material presented under heading "C", are enough to justify the "Subsequent oxidation ..." statement of the former (currently attached) memorandum. The environment of oxidation, whether hypogene or supergene, will be discussed under "E", and electron beam microprobe data verifies the analysis.

E. OXIDATION IN SITU DOMINATES

Below are three oxidation-in-situ geologic environments which could have produced the noted mineral association in the Freida River sample:

20. Hypogene: in which post-ore fluids became more oxidizing than were the ore fluids and the observed series of oxidation was produced.

21. "Depth environment" (Sato, I): in which hypogene conditions ceased to exist and "depth environment" conditions, which were not in equilibrium with the hypogene conditions, produced the observed assemblege.

22. Near surface, feebly oxidizing environment: in which the host rocks, containing hypogene ore mineralization, were exposed or semi-exposed to near surface weakly oxidizing supergene (downward percolating) conditions, which produced the observed association.

The last two in situ environments are not readily separable. Leaching of an individual component or components would be more likely in the latter case. This latter case may be correlated with the oxidizing zone of a sypergene enrichment, but, with markedly more feebly conditions. Marine or other possible cases will not be discussed.

Garrels

Figure 6.3 summarizes the stability relations of the iron oxides and water as functions of Eh and pH and will be used as the base for determining ionic activities.

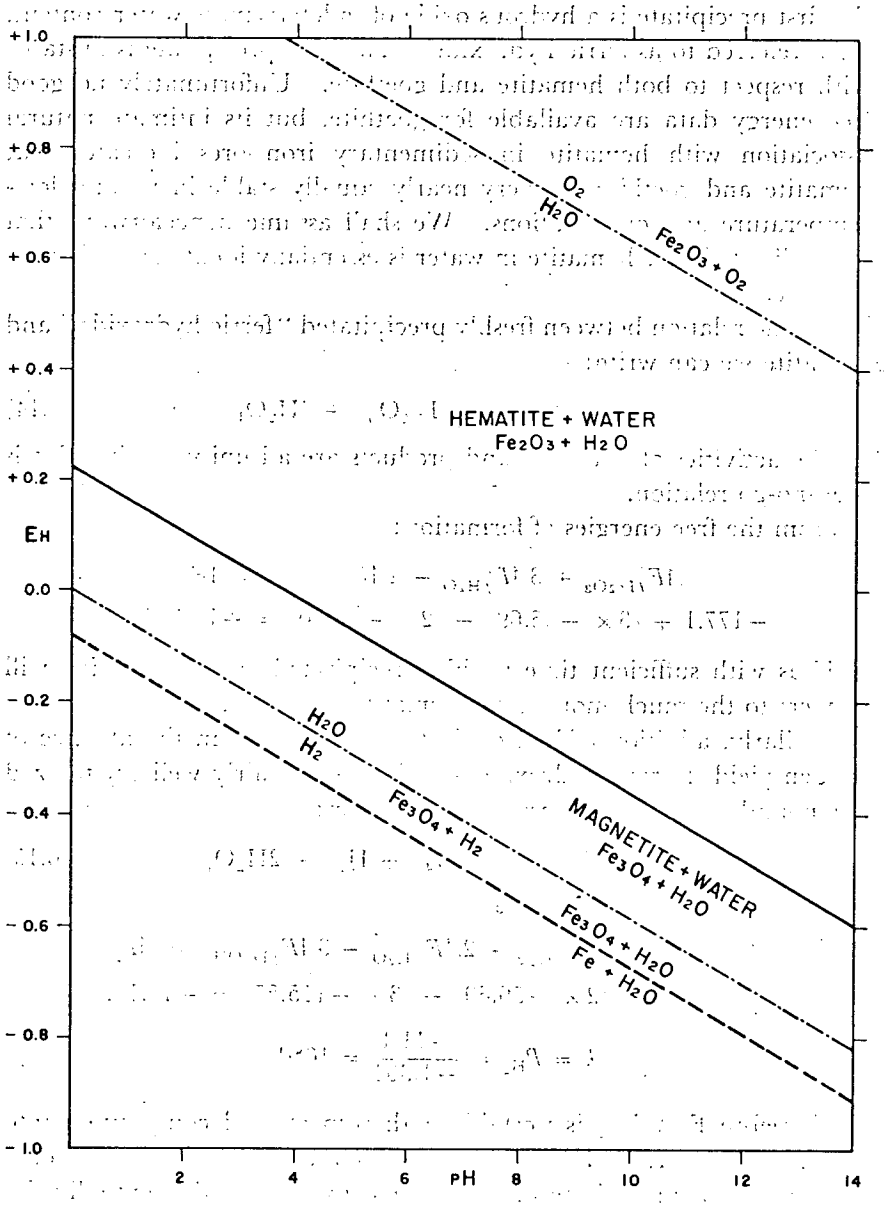


FIGURE 6.3. Stability fields of iron oxides as functions of Eh and pH at 25°C and 1 atmosphere total pressure. Dashed line indicates lower stability limit of magnetite in presence of metastable water.

In these three cases, the inequilibrium between the oxidizing solutions and primary metallic phases would be slight and the intermediate phases (chalcopyrite, bornite, chalcocite and digenite, as Bateman and others suggest) would have optimum opportunities to develope. Whereas, the change in the redox conditions is going from the odixizing zone to the reducing zone of a supergene enrichment environment
(23) would be marked; therefore, the opportunity for development of the intermediate phases would be substantially decreased and they probably would not significantly contribute to the apparent mineral assembleges.

Detailed examination of the literature has indicated that supergene bornite
(24) is more likely to form in a nuetral or slightly basic environment than in an acid environment (pH of 2 to 6). This indication is in agreement with Garrels redox diagrams (Figures 4a and 4b). The correlation is that one would more likely see supergene bornite in a feebly oxidizing environment with low contents of H^+, e.g. Freida River prospect, than in a typical Southwestern supergene enrichment zone in which the pH is notedly acidic, e.g. Silver Bell.

John M. Guilbert (University of Arizona, personal communications) stated
(25) that secondary bornite is "vanishingly rare" in a supergene enrichment zone. All bornite that the author has seen in polished sections of ore from the Silver Bell, Chino and Ely properties was considered to be primary, in agreement with Guilbert's statement and the paragraphs labeled 23 and 24.

When one compares the known minerology (with microprobe data) and the possible redox environments, the statement "Supergene enrichment is not indicated
(26) in the sample." does have significance. Use of the term supergene enrichment is as in the Southwestern porphyries, i.e., enrichment as a result of oxidation, mobilization and reprecipitation by the reduction of copper with one area being substantially leached and another area being enriched. The increase in copper content of one phase replacing an earlier phase with out migration (less than inches or centimeters) of copper, even when energized by supergene or downward percolating solutions, is not considered supergene enrichment, but rather, supergene alteration.

The above thinking draws a fairly well defined line between in situ oxidation phase enrichment and supergene enrichment and these are the connotations which were used in the earlier report. This use in the earlier report was not to deny any migration of copper in the Freida River prospect, but rather to point out that, (considering the mineral assembledge) in situ oxidation was preferred in that particular rock sample. As would be expected,portions of rock nearer the surface should show stronger oxidation, with production of more covellite and possibly Cu^0, Cu_2O and CuO with or without leaching.

Only three case studies of in situ oxidation with production of secondary sulfides were found in the literature:

27. Overbeck (1916, p 151): secondary chalcocite replaced chalcopyrite and bornite in dolomitic limestone, with minor malachite.

28. Schwartz (Ecom. Geol. v 23, p 765, 1928): "The most striking occurance is in the more or less spherical structures with chalcopyrite" (bornite replacing chalcopyrite). "Similar structures for the minerals have not been described for other deposits, so far as known." In this deposit bornite after chalcocite is characteristic and points out that retrograde oxidation (therefore reduction, Figure 4, area B to area A) has dominated the latest redox conditions.

29. Runnels (1969): The direct oxidation replacement of chalcopyrite by covellite and the supergene enrichment replacement of chalcopyrite by chalcocite both within the same ore deposit.

The Runnels paper is very recent; but, doesn't exhaustively describe the mineralogy. Nevertheless, that paper does describe oxidation replacement of chalcopyrite and provides levity for in situ oxidation versus supergene enrichment replacement.

ELECTRON BEAM MICROPROBE DATA

R. W. Wittkopp, at the University of California, Davis, California, ran several different types of analyses of the Freida River Prospect sample using the elecron beam microprobe. The microprobe is similar to an X-ray fluoresence instrument in that chemical analysis for most elements can be run. One of the main differences between the two instruments is that the microprobe can analyze a very small volume (0.5^3 microns). The microprobe effectively gives weight percentages within a fixed volume. This volume depends on the operating conditions of the instruments and the characteristics of a particular phase or sample; but, is held constant. This capability allows one to examine the boundries between mineral phases or other micro-areas of most minerals. The microprobe data convincingly demonstrates that leaching of the grain boundries has occurred in the Freida River sample and that the particular characteristics of the Freida River sample are distinctly different than those in samples from the Ely and Silver Bell deposits.

The microprobe work included: 1). Examination of oscilloscope displays of area scans for Fe, Cu and S in portions of each sample, 2). Comparisons of chemical analyses for Fe, Cu and S along line traverses across grain boundries in samples from the three locals, and 3). Single point chemical analyses for all elements.

The areal displays (Illustrations I and II) show rather clearly that the grain boundries in the Fredia River sample are much more gradational than in the Silver Bell or Ely samples. The line traverses (Illustrations III and IV) show this same feature; but more importantly, show that an intermolecular porosity exists in the Freida River sample and that it is distinct from the other two samples.

Illustration V shows a diagramatic comparison of line traverse chemical analyses given in Profile 1 (Illustration III, Freida River) and Profile 9 (Illustration IV, Tripp Veteran). As shown, there are distinct differences in the two sets of curves. Profile 12, sample from the Tripp Veteran pit, shows horizontal plateaus for the three elements in both mineral phases with a very steeply sloping curve portions between the two phases. This set of curves means that the two phases have rather consistant weight percentages of Fe, Cu and S across each phase and that the grain boundry is rather sharp or distinct between pahses. Profile 13 shows horizontal plateaus for Fe, Cu and S percentages in the chalcopyrite phase, gradually decreasing Fe, Cu and S contents in the chalcocite phase and not so steeply sloping curves between the two phases.

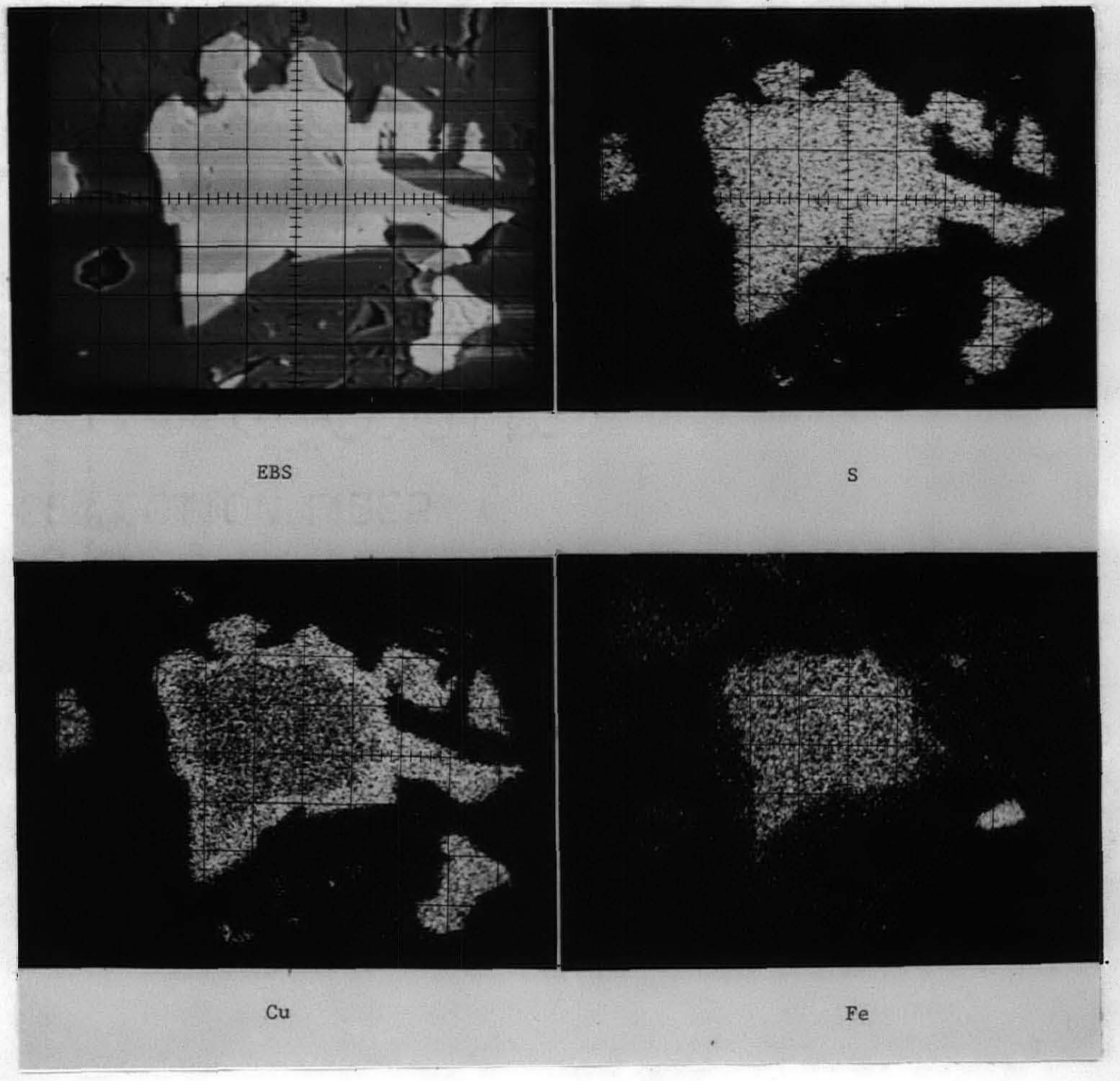


Illustration Ia, Photographs of Beam Scanning Oscilloscope Displays Showing Electron Back Scatter and Distribution of S, Cu and Fe in Metallic Grains from the Freida River Sample. Chalcocite (copper rich phase) Forms a Rim on Chalcopyrite with Gradational Grain Boundries. 3000X.

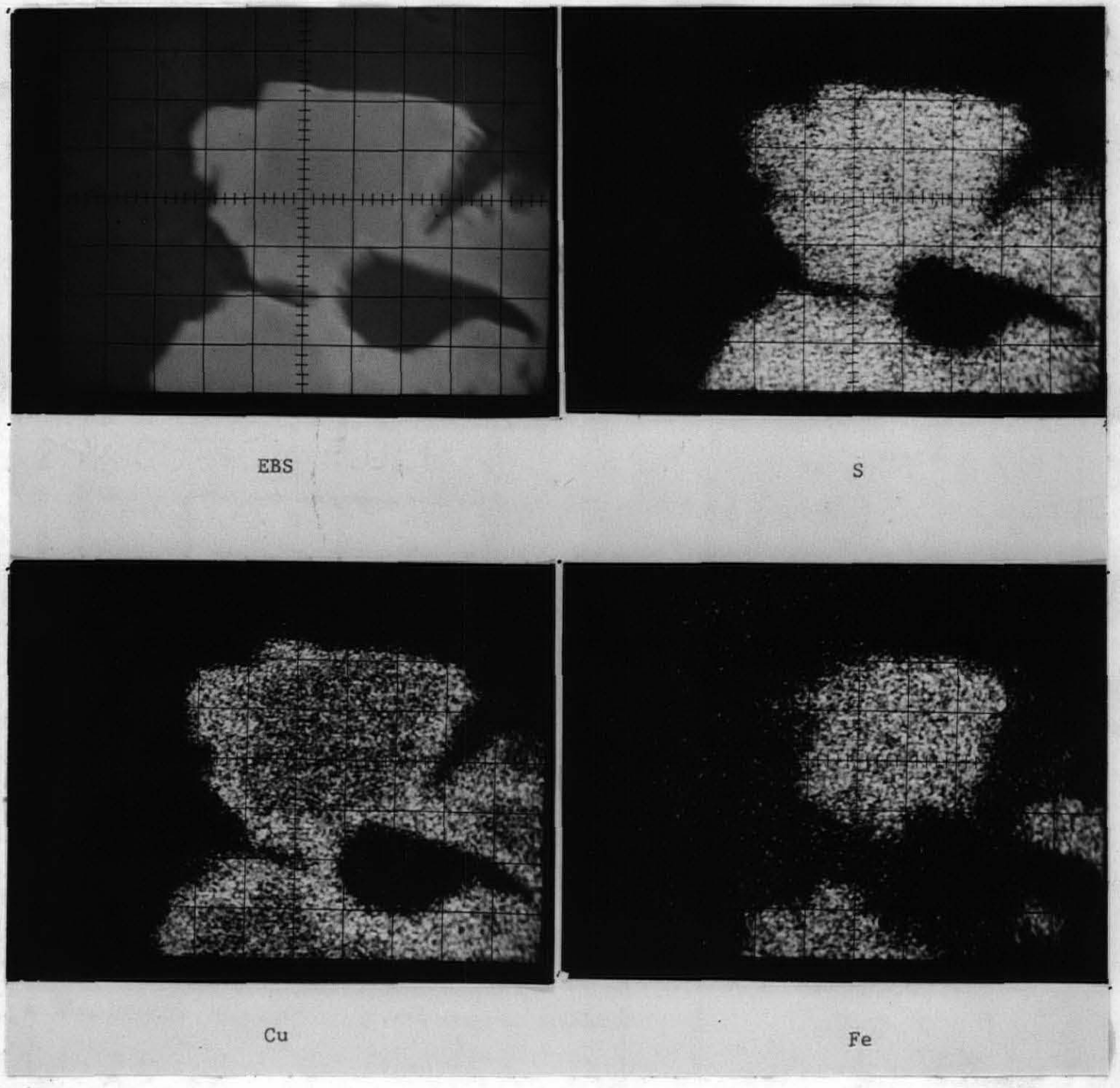


Illustration Ib, Same as Ia, Different Grain.
8100X.

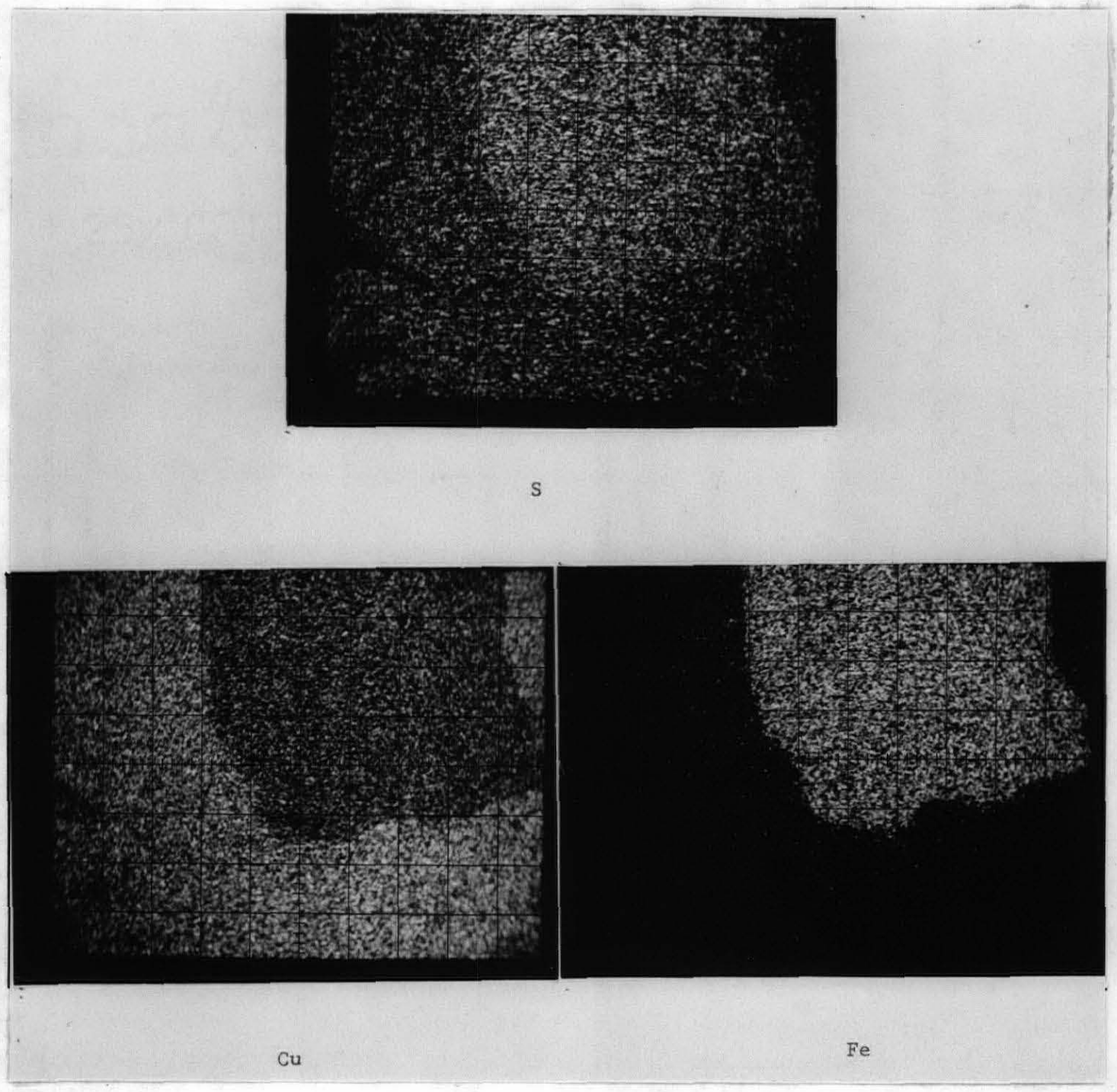


Illustration II, Photographs of Beam Scanning Oscilloscope Display Showing Distribution of S, Cu and Fe in Metallic Grains from the Ely Deposit. Chalcocite (copper rich phase) Rims Chalcopyrite with Sharp Grain Boundries. 3000X.

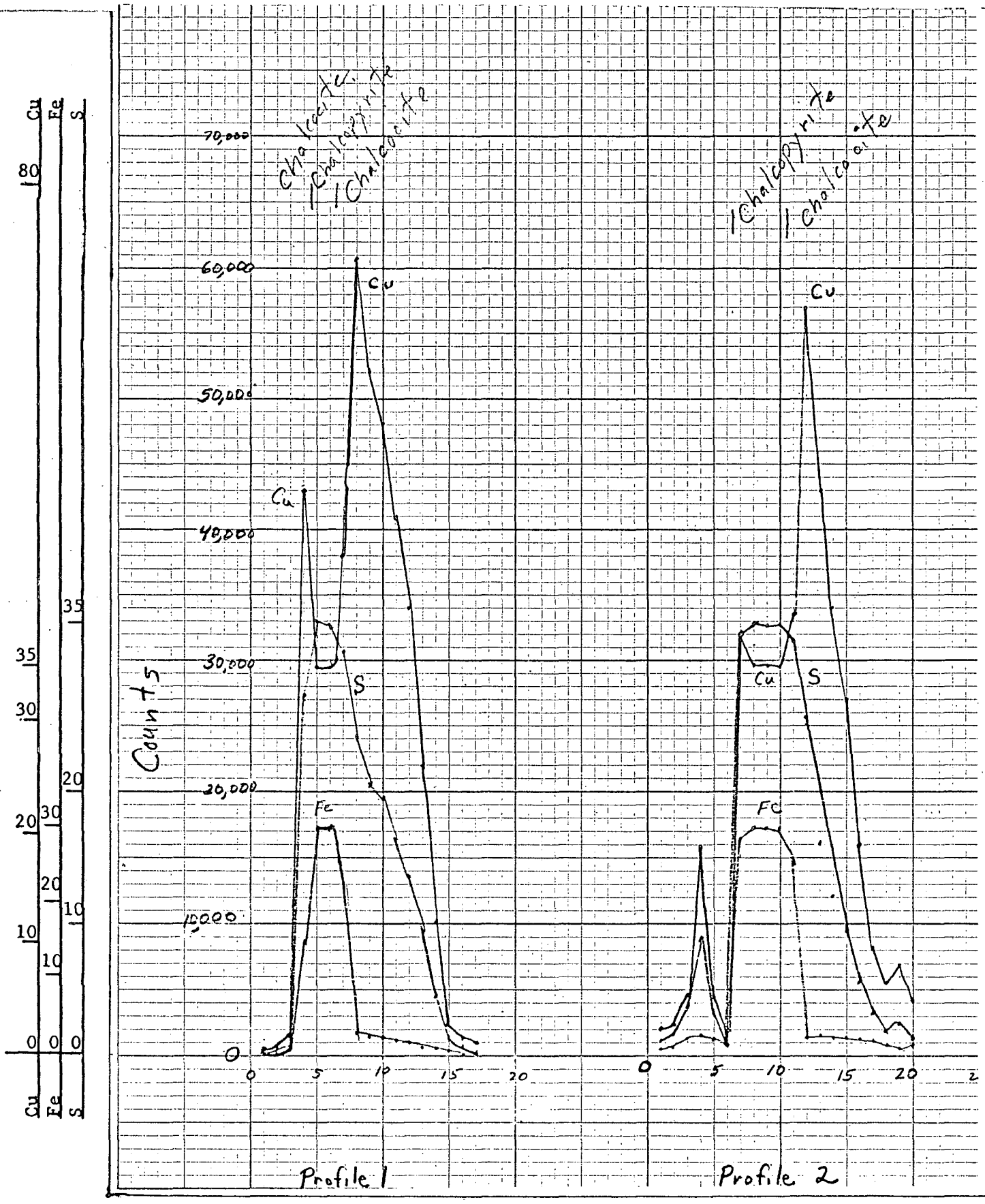
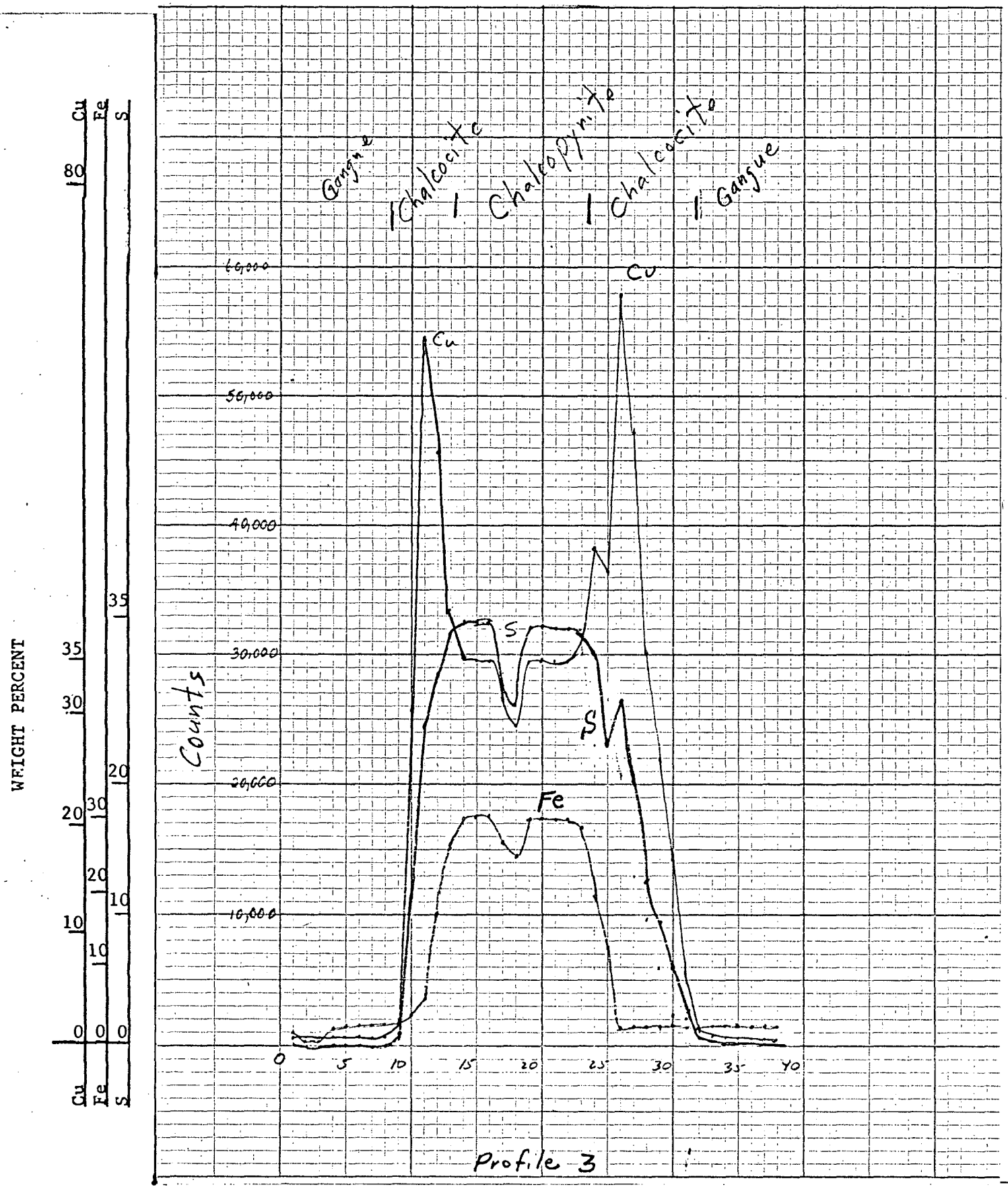
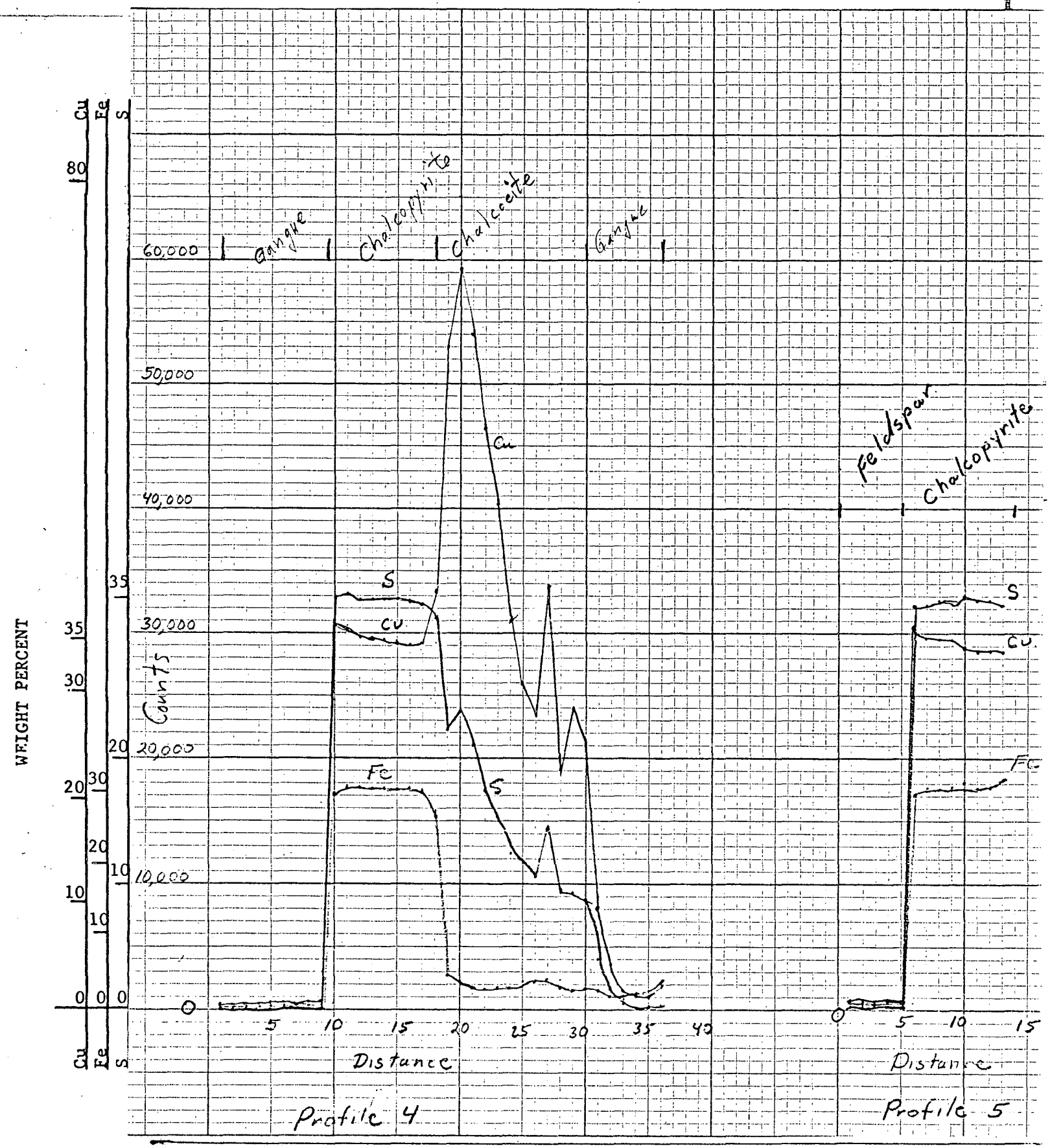


Illustration III - Chemical Analysis of Freida River Sample (Cu Fe + S, by electron beam microprobe).



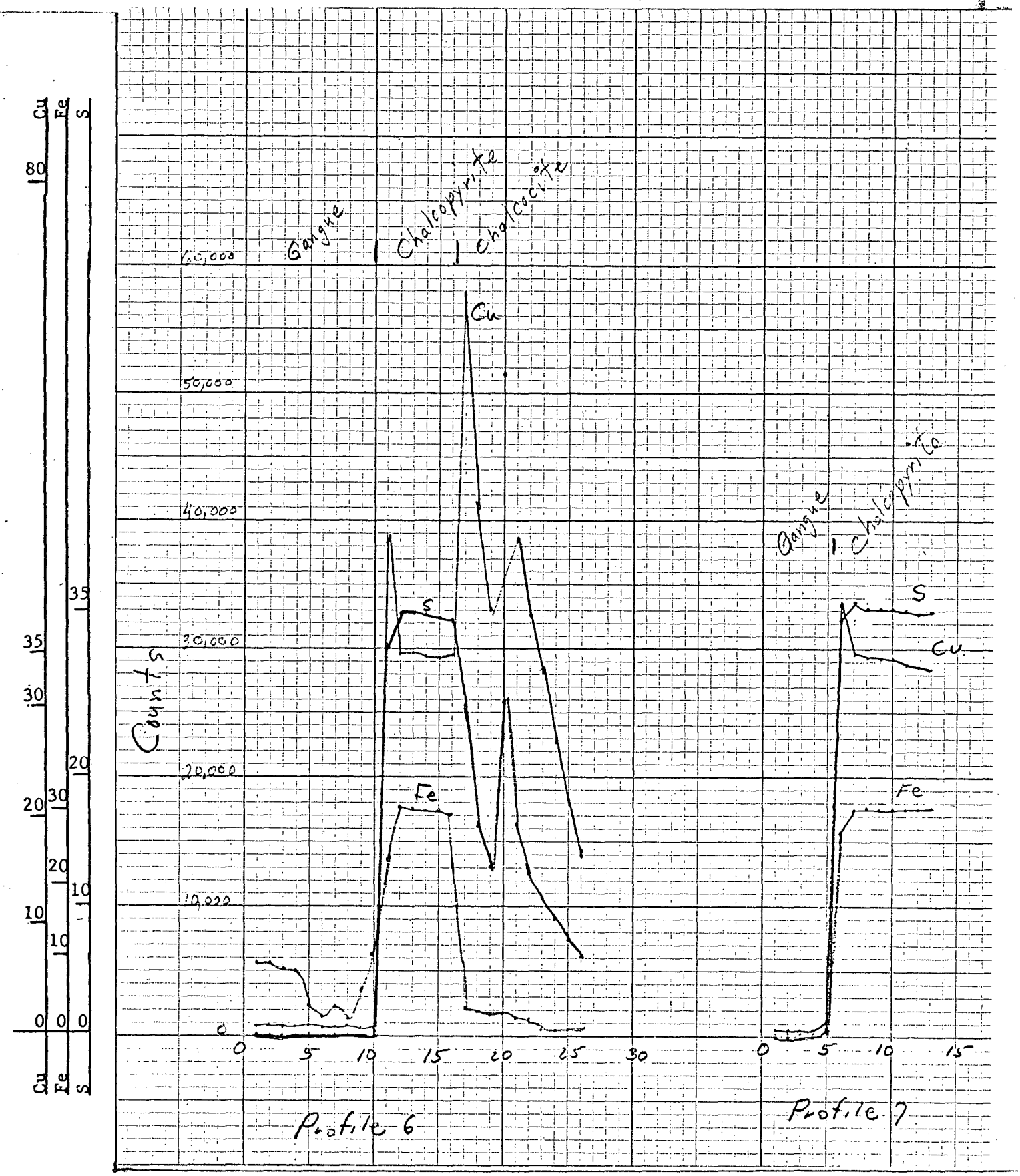
TRAVERSE DISTANCE (Microns)

Illustration III - Chemical Analysis of Freida River Sample (Cu Fe + S, by electron beam microprobe).



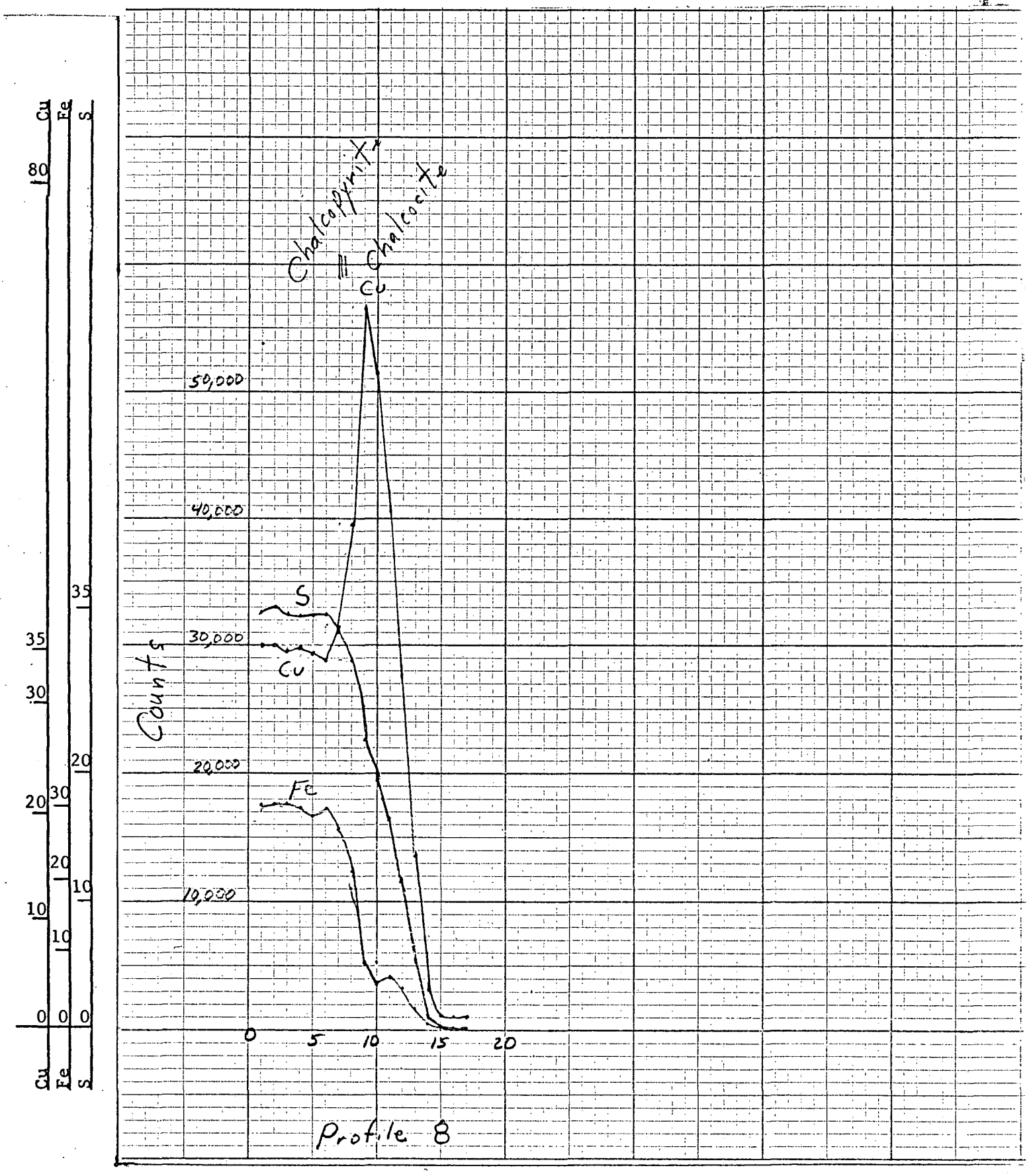
TRAVERSE DISTANCE (Microns)

Illustration III - Chemical Analysis of Freida River Sample (Cu Fe + S, by electron beam microprobe).



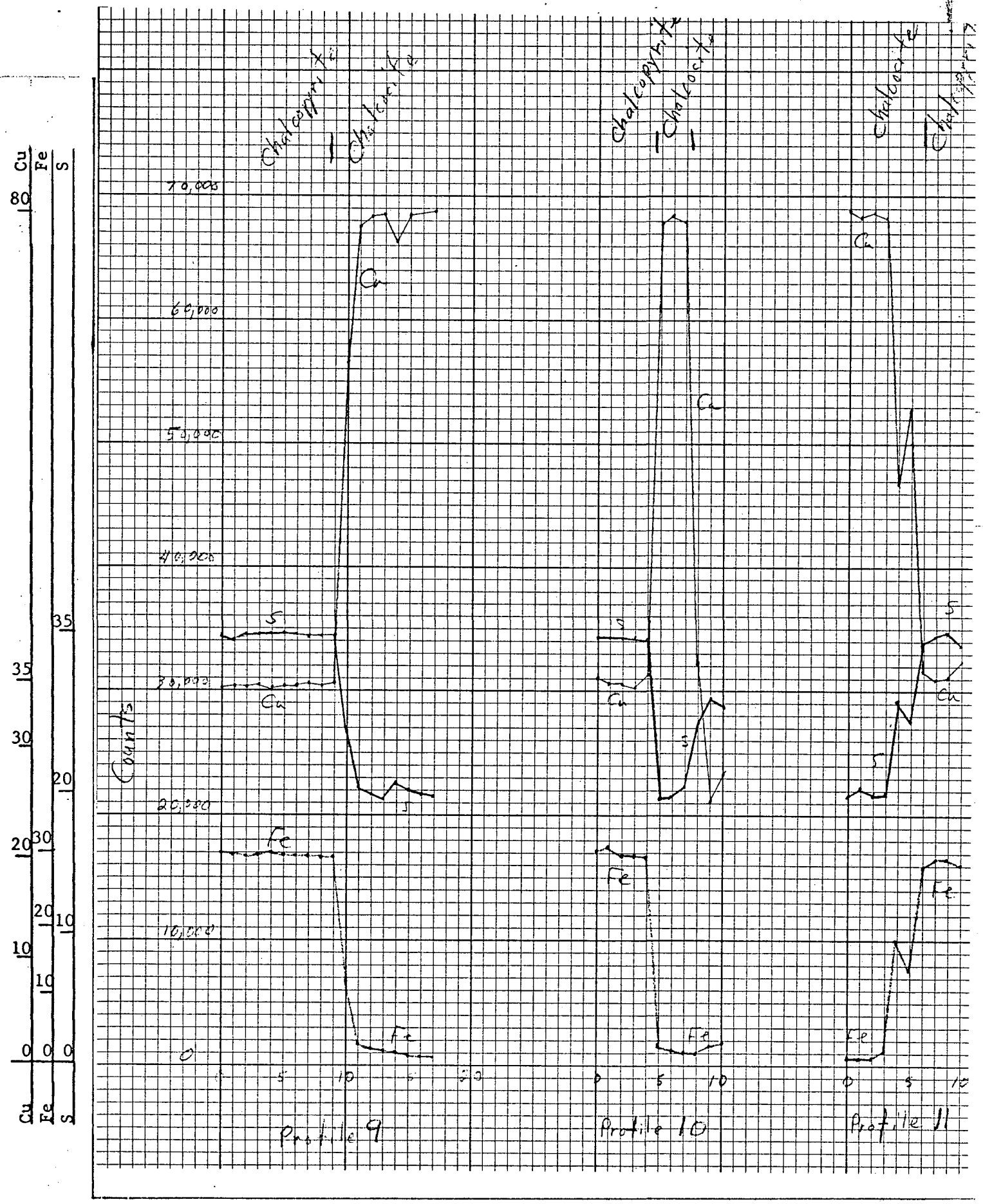
TRAVERSE DISTANCE (Microns)

Illustration III - Chemical Analysis of Freida River Sample (Cu Fe + S, by electron beam microprobe).



TRAVERSE DISTANCE (Microns)

Illustration III - Chemical Analysis of Freida River Sample (Cu Fe + S, by electron beam microprobe).



TRAVERSE DISTANCE (Microns)

Illustration IV - Chemical Analysis of Tripp Veteran Sample, (Cu, Fe + S).

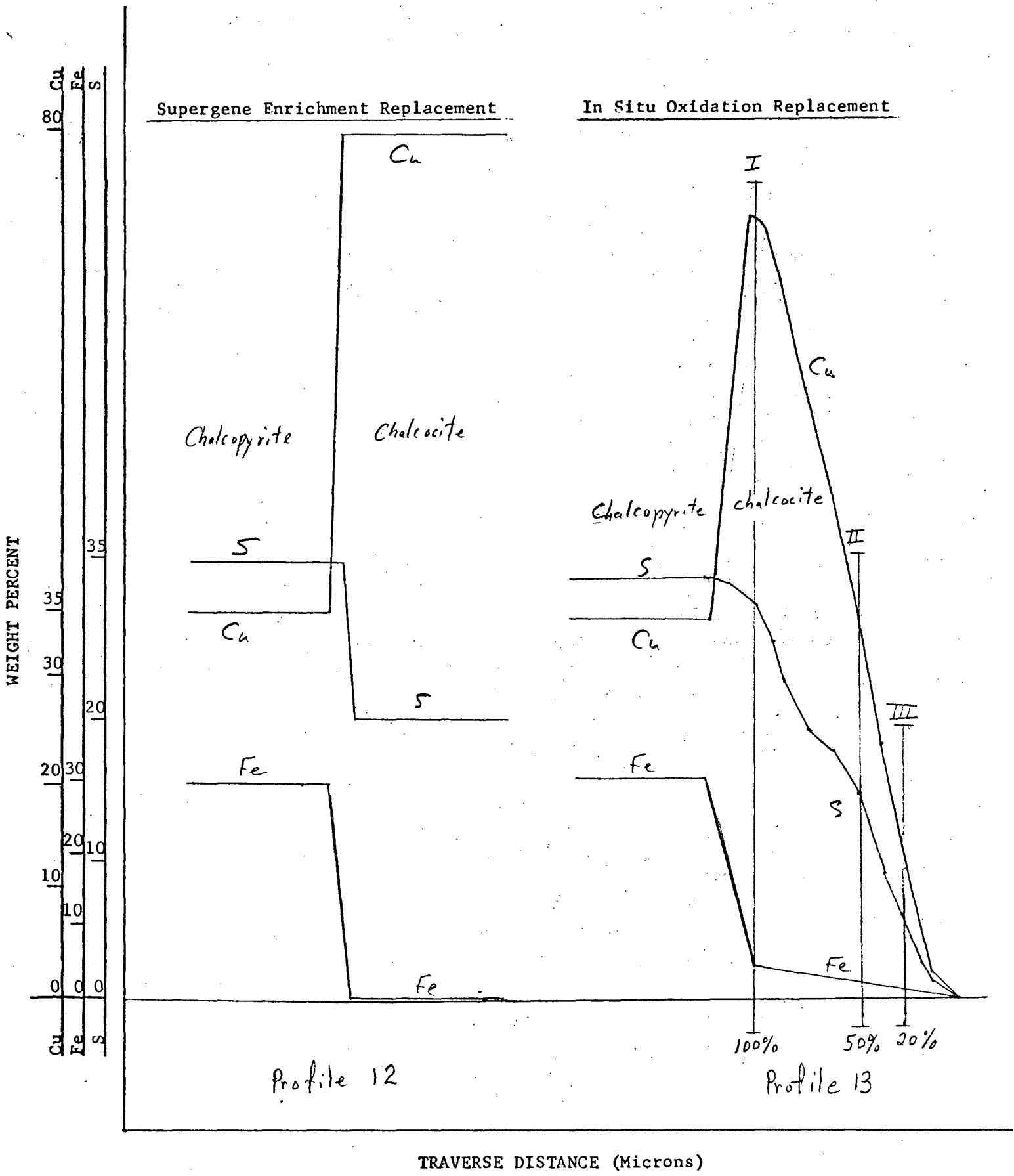


Illustration V - Diagramatic Comparison of Boundry and Leaching Characteristics of Tripp Veteran Sample (left) and Freida River Sample (right).

This latter set of curves shows that in the Freida River sample the grain boundries are gradational and that the chalcocite phase has been differentially leached of Fe, Cu and S. These two conditions are (as pointed out above) characteristic of in situ oxidation, with associated leaching. Therefore, one can say that the Freida River sample falls into case 22 (of this report) which is: an in situ, feebly oxidizing environment with associated leaching, attacking host rocks with primary mineralization.

Profile 13 (Ill. V) shows a diagramatic comparison of weighted percentages in the Freida River sample. The vertical lines marked I, II and III represent a summation of the Fe, Cu and S weight percentages, I=100%, II=50% and III=20%. The obvious question is, what comprises the remaining 50% or 80% of the given volumes? The microprobe, using a complete spectral scan for all elements on a fixed volume of the sample (e.g. III of Profile 13), showed that no other elements were present or that only space (or a porosity) exists to account for remaining volume percent. This is probably an intermolecular porosity.

THE ECONOMIC FACTORS ARE:

30. The magnetite should give an enormous magnetic response and therefore could provide a useful exploration tool.
31. If this is indeed oxidation in situ, then one can expect a very high grade protore and the target is not a two dimensional feature (blanket) but rather a three dimensional feature with much more attractive possibilities.

SUMMARY AND CONSLUSIONS

The combined results of the mineralogy and researched literature justify the statements in the previous report and prove with little room for doubt that the Freida River Prospect sample represents an in situ oxidation environment.

The mineralogical features that indicate this analysis (the features that were used in the original report) are low pyrite, replacement of magnetite by hematite, secondary bornite rims on the chalcopyrite, nuetral gangue minerals, and general supergene replacement textures (rims). The microprobe data only adds weight in favor of the original analysis.

J. W. Cameron

JON W. CAMERON
Geologist

JWC:jca

cc: WESaegart
GWBossard
RHall
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FREIDA RIVER PROSPECT MINERALOGY REPORT - 4/2/71

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AMERICAN SMELTING AND REFINING COMPANY
SILVER BELL UNIT
Silver Bell, Arizona

January 26, 1971

MEMORANDUM TO: Mr. J. H. Courtright, Chief Geologist

REF: EIGHTEEN GRAM SAMPLE FROM FRIEDA RIVER PROSPECT, NEW GUINEA.

Barry R. French submitted one sample for microscopic identification of opaque and non-opaque minerals. Two polished sections and one thin section were cut from the 18 gram sample. These sections were examined by reflected and transmitted light microscope techniques using magnifications up to 1000X.

Mineralization and alteration features observed in the polished sections were:

1. Magnetite: 6-7%, 400 to 1 micron size, 50% >30 microns, primary hydrothermal mineralization, possibly partially later than and replacing the biotite (see below), undergoing oxidation to hematite (Photograph #1), equidimensional, disseminated and occuring in discontinuous veinlets, probably chalcopyrite has replaced magnetite to some extent.

2. Chalcopyrite: 2%, hydrothermal replacement of magnetite, minor pyrites and possibly biotite, as inclusions in magnetite, is being oxidized to bornite, chalcocite and covellite, in that order.

3. Hematite: 2-1% occured as replacement rims and stringers after magnetite, which indicates oxidation (magnetite is stable in a supergene copper enviornment).

4. Chalcocite: 0.5%, oxidation replacement of secondary bornite and primary chalcopyrite (Photograph #2), occuring as replacement rims, stringers and inclusions.

5. Bornite: < 0.2%, primary (?) and secondary replacement of chalcopyrite, as rims and stringer, some bornite may have been primary, it showed a minor development of ex-solution textures with chalcopyrite.

6. Covellite: $<0.2\%$ replacement of chalcocite as rims, stringers and inclusion.

7. Pyrite: $<0.1\%$, occured along one vein, was strongly replaced and surrounded by primary chalcopyrite (Photograph #3), generally showed hydrothermal corrosion with little indication of alteration to hematite.

8. Sphalerite: A trace of sphalerite, associated with chalcopyrite, low iron content.

9. Rutile: 1-2%, a rather low-iron rutile was intimately associated with the magnetite as well as dispersed in the silicates.

Mineralization and alteration features observed in the thin section included:

1. Quartz: 60% grandular (20-200 microns) with inclusions of zircon and rutile, recrystalized.

2. Biotite: 20%, anhedral, tabular, with zircon inclusions.

3. Chlorite: 3-5% (prochlorite), not unfrequently along somewhat discontinuous veins.

4. Zircons: $<0.1\%$ 2-10 and $\pm$ 100 microns as inclusion in quartz and biotite, very rounded to slightly angular.

5. Plagioclose: Trace, <5 micron, twinned, unaltered.

6. Talc-Sericite: Possible trace amounts, stringers.

7. Opaques: 8-10%, anhedral, equidimensional.

Conclusions:

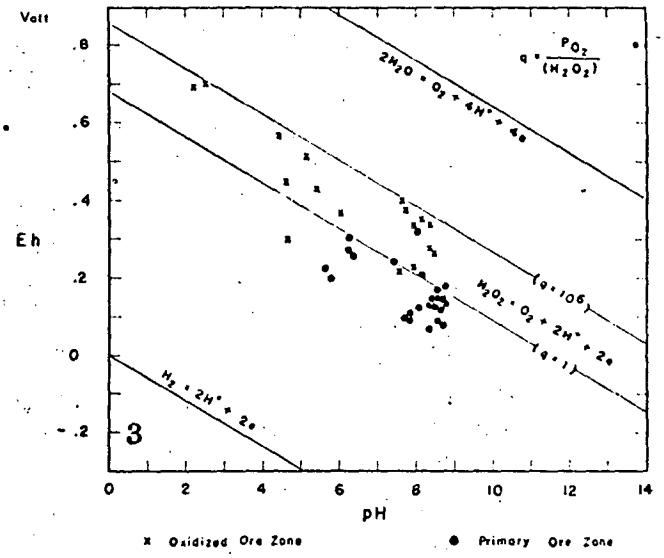
This rock fragment is from a hydrothermal deposit undergoing oxidation with or without supergene enrichment. The paragenetic sequence is (1) quartz-biotite-magnetite-rutile, (2) chlorite, (3) pyrite, (4) chalcopyrite-bornite-sphalerite, (5) hematite-bornite-chalcocite-covellite. Mineralization occured along veinlets and as disseminated grains. Subsequent oxidation has taken place. This oxidation is stronger along veinlets. Supergene enrichment is not indicated in this 18 gram sample. The host rock may have been an argillic siltstone which was silicified by hydrothermal processes. This rock fragment may be from within a broad quartz vein, and therefore not a siltstone.

J. W. Cameron

J. W. Cameron

JWC:jca

sediments of eleven lakes in Wisconsin, together with depth, temperature, and the amounts of dissolved oxygen, ferrous and ferric iron, and hydrogen sulfide. Their results are plotted on an Eh-pH diagram in Figure 4. Crosses



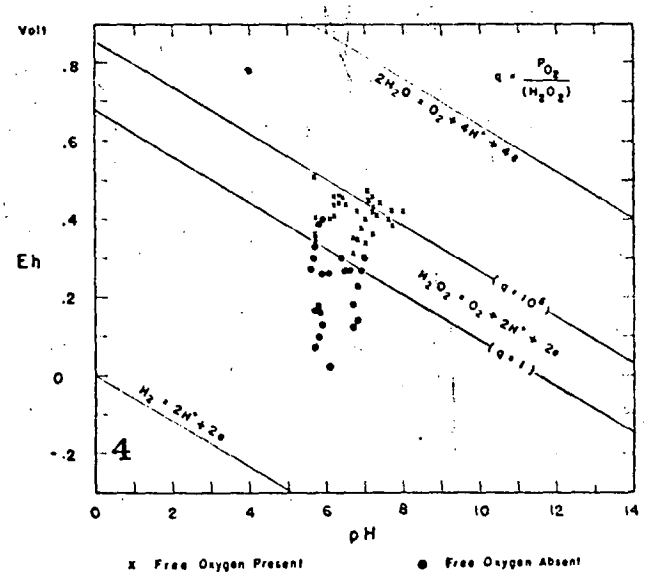


FIG. 3. Plots of Eh-pH values of natural waters in four mining districts. Circles represent measurements in the primary sulfide ore zone, and crosses those in the zone of oxidation.

FIG. 4. Plots of Eh-pH values of lake waters and of lake sediments in Wisconsin. Original data by Allgeier, Hafford, and Juday (1).

Magmatic Ore Deposits

that occur during emplacement and cooling of the ores.

Minerals and Phases of the Cu-Fe-Ni-S System

A large number of minerals whose compositions plot within the confines of the Cu-Fe-Ni-S system have been reported in the literature. A listing of these minerals with chemical formulae and the abbreviations used in this paper is found in Table 1. In addition, the compositional positions of the phases in the Cu-Fe-Ni-S system are shown in Figure 1.

Experimental Procedures and Reactants

The bulk of the experiments were performed using the silica-tube techniques described by Kullerud and Yoder (1959). These experiments were prepared on an analytical balance with an accuracy of ±0.02 mg. The experimental charges (100 to 200 mg) were prepared by direct weighing in silica tubes, eliminating loss of material due to handling and transfer.

The experimental charges were reacted in horizontal nichrome wound furnaces controlled within ±1° or ±5° C, depending on the accuracy required for the individual experiments. At the termination of each experiment the tubes were chilled in cold water and the products were examined at room temperature by means of reflected light microscopy and X-ray powder diffraction. In addition, a considerable amount of data were obtained by means of differ-

Table 1. Phases of the Cu-Fe-Ni-S System

Name	Abbreviation	Formula
Awaruite	aw	Ni_3Fe
Blue-remaining covellite*	bcv	CuS_{1-x}
Bornite	bn	Cu_5FeS_4
Bravoite	bv	$(Fe,Ni)S_2$
Chalcocite	cc	Cu_2S
Chalcopyrite	cp	$CuFeS_{2-x}$
Chalcopyrrhotite†	ccp	$\sim CuFe_2S_3$
Copper	--	Cu
Covellite	cv	CuS
Cubanite	cb	$CuFe_2S_3$
Digenite	dg	Cu_9S_5
Djurleite	dj	$Cu_{1.96}S$
Greigite	gr	Fe_3S_4
Heazlewoodite	hz	Ni_3S_2
Idaite	id	$Cu_{5.5}FeS_{6.5}$
Iron	--	Fe
Kamacite	ka	$\alpha(Fe,Ni)$
Mackinawite	mac	FeS
Marcasite‡	mc	$\sim FeS_2$
Millerite	ml	NiS
Monosulfide solid solution	Mss	$(Fe,Ni)_{1-x}S$
Nickel	--	Ni
Pentlandite	pn	$(Fe,Ni)_9S_8$
Polydymite	pm	Ni_3S_4
Pyrite	py	FeS_2
Pyrrhotite (several phases)	po	$Fe_{1-x}S$
Symthite	sm	Fe_3S_4
Sulfur	--	S
Taenite	ta	$\gamma(Fe,Ni)$
Troilite	tr	FeS
Vaesite	vs	NiS_2
Valleriite§	val	$\sim CuFeS_2$
Villamaninite	vill	$CuNi_2S_6$
Violarite	viol	$FeNi_2S_4$
X-bornite	x-bn	Cu_5FeS_{4+x}
Not known as mineral	--	Ni_7S_6

*Chemically the composition of "blue-remaining covellite" can be expressed as CuS_{1-x} (31); whether this formula is structurally correct is not known.

†Chalcopyrrhotite is considered by Yund and Kullerud (51) to represent tetragonal or cubic cubanite.

‡Marcasite is generally considered as a polymorph of pyrite (40). Kullerud (22) has recently suggested that it contains H-S bonds in its structure.

§Valleriite is considered by Evans et al. (11) to represent a low-temperature polymorph of chalcopyrite.

Symthite should read Smythite.

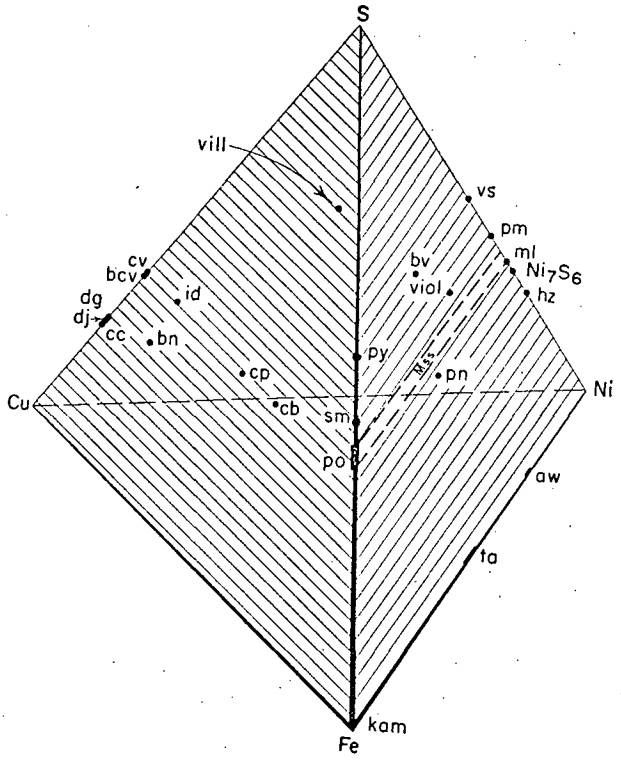


FIG. 1. Phases of the Cu-Fe-Ni-S system: pyrite (py), pyrrhotite of various types (po), smythite and greigite (sm, gr), chalcocite (cc), djurleite (dj), digenite (dg), "blue-remaining" covellite (bcv), covellite (cv), heazlewoodite (hz), Ni_7S_6, millerite (ml), polydymite (pm), vaesite (vs), pentlandite (pn), bravoite (bv), violarite (viol), chalcopyrite (cp), cubanite (cb), bornite (bn), x-bornite (x-bn), idaite (id), villamaninite (vill), copper (Cu), iron (kamacite) (Fe), nickel (Ni), sulfur (S), taenite (ta), and awaruite (aw). In addition the phases marcasite ($\sim FeS_2$), mackinawite ($\sim FeS$), chalcopyrrhotite ($\sim CuFe_2S_3$), and valleriite ($\sim CuFeS_2$) have been reported. Since the precise positions of these phases within the Cu-Fe-Ni-S system are unknown, they are not shown in the figure. Compositional limits of the Fe-Ni monosulfide solid solution (Mss, $[Fe,Ni]_{1-x}S$) at 600° C are shown dashed on the Fe-Ni-S face of the tetrahedron. The light lines on the Cu-Fe-S and Fe-Ni-S faces merely show perspective.