CONTACT INFORMATION
Mining Records Curator
Arizona Geological Survey
416 W. Congress St., Suite 100
Tucson, Arizona 85701
520-770-3500
http://www.azgs.az.gov
inquiries@azgs.az.gov

The following file is part of the

James Doyle Sell Mining Collection

ACCESS STATEMENT

These digitized collections are accessible for purposes of education and research. We have indicated what we know about copyright and rights of privacy, publicity, or trademark. Due to the nature of archival collections, we are not always able to identify this information. We are eager to hear from any rights owners, so that we may obtain accurate information. Upon request, we will remove material from public view while we address a rights issue.

CONSTRAINTS STATEMENT

The Arizona Geological Survey does not claim to control all rights for all materials in its collection. These rights include, but are not limited to: copyright, privacy rights, and cultural protection rights. The User hereby assumes all responsibility for obtaining any rights to use the material in excess of "fair use."

The Survey makes no intellectual property claims to the products created by individual authors in the manuscript collections, except when the author deeded those rights to the Survey or when those authors were employed by the State of Arizona and created intellectual products as a function of their official duties. The Survey does maintain property rights to the physical and digital representations of the works.

QUALITY STATEMENT

The Arizona Geological Survey is not responsible for the accuracy of the records, information, or opinions that may be contained in the files. The Survey collects, catalogs, and archives data on mineral properties regardless of its views of the veracity or accuracy of those data.

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

February 29, 1972

Memorandum:

TO: W. L. Kurtz

FROM: J. D. Sell

Acid Consuming Copper Deposits
Arizona (with short notes on
New Mexico, California and Nevada)

Compilation of acid consuming copper deposits and general areas has been made for Arizona with short notes on occurrences in New Mexico, California and Nevada. The work was based on company files, literature search, references submitted by the staff, and conversations with the U.S. Bureau of Mines. The references are included in the Tucson file.

Arizona has some twenty-five areas commercially producing leach copper and New Mexico has three. They are mainly located with or near a producing porphyry copper sulfide deposit. Tabulation of published reserves (both oxide and sulfide) of the major deposits in Arizona was submitted in a memo dated January 6, 1972.

The operating and potential leaching area tabulation is established on a radiating distance from the ASARCO acid producing plant at Hayden. All areas are potential acid sales areas. Large tonnage acid consumption areas in which ASARCO could participate directly by mining, joint venture, acid sales, include:

1) Miami district, especially Van Dyke (Occidental) and Cactus-Carlota (Miami-Homestake Prod.).
2) All of Florence district.
3) All of Casa Grande district.
4) All of Silver Bell district.
5) All of the Slate-Vekol district.
6) All of the Pima district.
7) All of the Johnson Camp district.
8) All of Kirkland district.
9) All of Cerbat district.

The most favorable areas are those in which ASARCO presently has a strong land position, namely in the Florence, Casa Grande, Silver Bell (and hence, Slate-Vekol), and Pima districts.

James D. Sell

JDS:sg
attach.

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Ray District					
Ray	KCC	X	X	36,726	Apparently large silicate reserves.
Copper Butte	KCC		X	119 (Year 1969)	15(?) million @ 0.4%
Chillito	KCC		X		tactite type
Christmas	ICC		X		
Rattler	ICC (?)		X		2.5 million tons @ 1-1.5%, mostly copper oxides, reported.

*Leach Operations Only

0 - Past or Anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Miami District					
Inspiration	ICC	X		10,038	Expansion
Miami*	Cities Service	X		7,483	Expansion potential
Ox Hide*	ICC	X		6,720	20-40(?) million @ ± 0.5%, all oxide
Blue Bird*	Ranchers	X		5,930	75 million @ 0.52%
Copper Cities	Cities Service	X		2,246	
Castle Dome	Cities Service	X		467	Operations ceased at end of 1970
Gibson Mine*	Private	X		3 (year 1969)	Oxidized sulfides in shear zone pods; probable limited
Barney*	ICC		X		Reported equivalent to Ox Hide
Miami East	Cities Service		X		100-140(?) million tons @ ± 1% Probably mixed oxide
Van Dyke	Occidental		X		80-100(?) million tons @ ± 1%; Half oxide (?)
Cactus*	Cities Service		X		40(?) million tons @ ±0.7%; Mostly oxide
Carlota*	Homestake Production		X		4-5 million tons @ ± 1.0%; All oxide
Copper Springs	Humble		X(?		
Diamond H	Cities Service	0	X		6 million @ 0.55%

*Leach Operations Only

0 - Past or Anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Florence District					
Poston Butte West	ASARCO		X		100-175 million tons @ ± 0.6%; Partially drilled
Poston Butte West	Conoco		X		75-100 million tons @ ± 0.5%; Partially drilled
Poston Butte East	Conoco		X		200-250 million tons @ + 0.4% Drilled oxide; Sulfides not included
Red Hill*	Private	X	X		Limited; Expansion drilling in progress
Mineral Butte	Duval		X		40 million tons @ 0.4%(?): Drilled

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
San Manuel District					
San Manuel	Newmont	X	X		Leachable 130 million tons @ 0.77% of which 0.47% is oxide
Old Reliable*	Ranchers	0	X		4 million tons @ 0.74% sulfide; Under development
Gold Hill	Little Hills Mines	X	X		Flux for San Manuel
North Star	Private		X		Apparently very limited
Control	Continental Mat.		X		Tactite in limestone; being drilled; 4 million tons @ 2.3% (open), sulfides.
Corncob	Essex		X		Being investigated
North Owl Heads	Conoco		X		Scattered occurrences in range
Ferguson Copper	Private		X		Reported 1 million tons @ 2.5%

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Casa Grande District					
Sacaton	ASARCO		X		6+ million tons with more probable. Drilled. Large potential @ ± 0.8% oxide. Partially drilled

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Silver Bell District					
Silver Bell Mines	ASARCO	X	X	2,807	16 million proven and probable, additional reserves possible.
Silver Bell West	Private		X		Minor development; good expansion possibilities.

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Slate-Vekol District					
Lakeshore	El Paso-Hecla	0	X		Plus 209 million tons @ 0.71% oxide. Drilled.
Vekol Hills	Newmont-Superior Oil		X		103 million tons @ 0.56% mixed. Drilled.
Copper Top	Private		X		Very limited tonnage in shear pods.

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Safford District					
Safford	KCC		X		+500 million tons @ 0.4% mixed
Sanchez	ICC		X		Being drilled. 40(?) million tons @0.4% (?)
Peacock*	Producers Chem.	X	X	10 (year 1969)	20(?) million tons @ 0.6%(?)

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Pima District					
Esperanza	Duval	X	X	2,214	
Pima	Pima	X	X		
Mission	ASARCO	X	X		
Eisenhower	Anaconda		X		
Twin Buttes	Anaconda	X	X		Oxide stock pile of 20 million tons @ 0.8%
San Xavier*	ASARCO	X	X	starting up	
Sierrita	Duval	X	X	starting up	
Helvitia	Anaconda		X		
Rosemont	Anaconda		X		

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Arivaca District					
Silver Ray	Private	0	X		Minor attempt
North San Luis	Private	0	X		Minor attempt in apparent small deposit
Arivaca Dist.	Private		X		Numerous small areas

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Morenci District					
Morenci	Phelps Dodge	X	X	8,450	Large

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Johnson Camp District					
Johnson C. East	Superior Oil		X		Tactites(?) Potentially large. Being drilled. Mainly sulfides.
Johnson C. North	Cont. Minerals		X		Tactites. Very spotty Partially drilled. Mainly sulfides.
Dragoon	Superior Oil		X		Tactites +80 million(?). Being drilled. Mainly sulfides.
Burro	Cyprus Mines	O	X	flux Douglas 196 (year 1968)	Lesse operations for silica flux.

*Leach Operations Only

O - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Courtland-Gleeson District					
Courtland-Gleeson	Mainly KCC		X		No major deposit known.
Courtland-Gleeson	Private	0	X		Minor production
Middle March	Private	0	X		Minor production
Aztec Mining*	Private	0	X	18 (year 1969)	Leaching old flooded workings; minor production

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Bisbee District					
Lavender Pit, etc.	Phelps Dodge	X	X	3,704	Preparing to close operations

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Ajo District					
Ajo	Phelps Dodge	0	X	Previous prod.	No known oxide left.? Under conglomerate?
Copper Giant	Private		X		Limited tonnage

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Kirkland District					
Zonia*	McAlester Fuel	X	X	1,743 (year 1969)	5(?) million tons @ 0.6%(?)
Copper Basin	PD & Others	O	X	74 (year 1969)	Partially drilled. Minor past production.
Little Cu Basin	Private		X		Partially drilled. 17 million (?) tons @ 0.37%, chalcocite
Porphyry Mtn.	Private		X		Est. 1 million tons @ 0.5%

*Leach Operations Only

O - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Jerome District					
Jerome	Verde Expl.		X	31 (year 1969)	Under exploration by Anaconda, 2 million tons @ 0.5%
Jerome Open Pit	Big Hole Mining	X	X	675 (year 1969)	

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Cerbat District					
Mineral Park	Duval	X	X	3,855	
Emerald Isle*	El Paso Nat. Gas	X	X	2,043 (year 1969)	

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Wickieup Dist.	Various		X		Various small potential producers

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Bagdad District					
Bagdad	Bagdad	X	X	6,220	Large potential.

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Planet-Swansea District					Potential in entire district and down river.
Planet	Private	0	X	1400 (year 1969)	Minor Operations
Swansea	Private	0	X		Minor Operations. 1.1(?) million tons @ 1.8%(?)
Moore (Solome)	Private	0	X		Minor Operations
Copper Hills (Bouse)	Private	0	X		Minor operations
Ohio Giant (Swansea?)	Private	0	X		Minor operations

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
White Mesa District					
White Mesa	Private	X	X	355 (year 1967)	0.5 million tons. Numerous attempts in sandstone coppers. Also several more in similar environment.

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
ARIZONA					
Patagonia District					
3 R Mine	Anaconda		X		
4 Metals	Norandex		X		

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
NEW MEXICO					
Santa Rita	KCC	X	X		425 million tons of leach material in dumps.
Tyrone	Phelps Dodge	0	X	Starting up	
Trigg	Private	X	X		250 thousand tons
Cuba	Earth Resources	X	X		Stockpiling oxides; sulfide reserves 10 million tons @ 0.7%
West Tyrone Area			X		Large area of scattered occurrences
Oro Grande			X		5-10(?) million tons @ 0.2%. Partially drilled.

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
CALIFORNIA					
Eureka District	Various		X		Picacho Mtns. (Calif.) and Trigo Mtns. (Ariz.) contain numerous showings of oxide copper.

*Leach Operations Only

0 - Past or anticipated leach production

State District Name	Owner	Operating	Potential	Production 1970 Tons of Leach Cu	Oxide Reserves Grade
NEVADA					
Yerington	Anaconda	X	X		28 million tons @ 0.57% oxide
Battle Mtn.	Duval	X	X		
Ely	KCC	X	X		
Big Mike	Ranchers	X	X		300,000 tons of oxide-sulfide in stockpile to be leached.

*Leach Operations Only

ASARCO

Southwestern Exploration Division

JDS

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.

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

H. G. Kreis

HGK:mek

cc: W. L. Kurtz

significant cost of doing business and, as of July 31, we had incurred $11.5 million in interest expense so far this year, compared the 70¢ we had anticipated in our budget plan, but somewhat more than the 65.96¢ we received during the same period in 1985.

Pinto Valley to study leaching of old Miami tailings

Pinto Valley has received approval for a $200,000 study of the feasibility of leaching tailings located at the old Miami open pit and underground mines.

Previous work has indicated a potential recovery of 115,000,000 pounds of copper over an eight year period.

Most of the tailings were deposited between 1911 and 1934 when mill recoveries were lower.

Some of the old tailings contain higher grades of copper than the present Pinto Valley mine.

Magma Update Sanderson, AZ
Vol. 2, No. 9, Aug. 1986 p 1

page 2

Pinto Valley has two SX-EW plants

Pinto Valley has two SX-EW plants. The older plant at Miami was built in 1976 to replace a copper precipitation plant which had been in operation since 1941.

This unit processes in-situ leach solutions from the caved area of the old Miami underground mine.

The newer plant near the Pinto Valley open pit went into operation in 1981 and processes copper from leach dumps.

The two plants have produced more than 105 million pounds of copper over the past decade.

One hundred million gallons of additional in-situ leaching solution have been injected in a new area this year to increase the flow rate to the SX-EW plant.

In-situ feed grade has increased 40% since May with a goal of another 10% increase which should substantially lower unit costs.

ASARCO

Southwestern Exploration Division

JAS

December 12, 1985

To: J. D. Sell

From: H. G. Kreis

Copper Leaching at
San Manuel Mine
Pinal County, AZ

As most everyone knows by now, Newmont has initiated a two phase program to profitably recover copper from the copper oxide mineralization at San Manuel. The first phase is an open pit-heap leach operation, and this was initiated in 1985 with the first production expected in July 1986. The second phase will be an in-place leach of copper oxide mineralization broken by block-cave mining of underlying sulfide ore. The block-cave in-place leach operation will start on a pilot scale basis within a year or two.

The open pit-heap leach operation is described in the attached text of a talk and is summarized in the following table. The speaker who gave this talk was asked to state the production cost of the open pit-heap leach operation. He referred the audience to an article in Pay Dirt (March 1985) which states "anticipated production costs of about 30¢ a pound." The speaker said he did not know the source of this information, but would like to know. The implication was that the 30¢ per pound cost figure was leaked from the company. This production cost figure seems reasonable, particularly if one assumes the production costs are probably half, or less, of the price of copper, say 65¢ per pound.

H. G. Kreis

H. G. Kreis

HGK:mek
Attachment

cc: W.L. Kurtz
J.R. Stringham

OPEN PIT OPERATION

$71 million investment
56 million tons oxide ore at 0.47% acid sol. Cu (0.72% total Cu)
0.2% acid soluble Cu cutoff grade
2 million tons sulfide ore at 1.27% Cu (0.16% acid sol. Cu)
2.2:1 stripping ratio (minimal vertical lift)
11 year mine life
≥85% Cu recovery of acid sol. Cu is predicted
Annual production of 20,000 tons of 99.9% pure Cu

BLOCK CAVE - IN-PLACE LEACH

175 mm tons at 0.43% acid soluble Cu
Apparently will involve doubling capacity of
SX-EW plant to 50,000 tons Cu per year

THE SAN MANUEL OXIDE OPEN PIT PROJECT

PRESENTED TO

ARIZONA CONFERENCE OF AIME

DECEMBER 9, 1985

BY

STEPHEN F. WALSH

THE SAN MANUEL OXIDE OPEN PIT PROJECT

Magma Copper Company, the largest underground copper mining company in the U.S., is joining the ranks of open pit copper producers. Magma has been producing copper since 1910 from underground mines at Superior and San Manuel, Arizona. In February 1985 Magma's parent company, Newmont Mining Corporation, granted full funding approval of $71 million to develop an open pit and construct a solvent extraction-electrowinning plant at the San Manuel mine site. The operation will utilize a carefully planned dump leaching program to recover copper from oxide copper ore by producing a pregnant leach solution suitable for processing in the SX-EW facility currently under construction. The San Manuel Oxide Open Pit Project will produce more than 450 million pounds of copper cathode from an estimated 56 million tons of oxide ore and approximately 50 million pounds of copper from sulfide ore over an eleven-year period. Full-scale mining is scheduled to begin January 1986 and cathode production will commence in July.

The oxide open pit is the first phase of a total program to recover copper from oxide ore. The oxide reserve has been divided into two categories: surface minable ore and in-situ leachable reserves. The second phase of the Oxide Project will utilize in-situ leaching to extract copper from proven reserves estimated at 175 million tons of oxide ore. The in-situ leachable reserve is deeper than the open pit minable portion of

the ore body and lies directly above depleted sulfide block caving panels. Underground openings in these areas are still intact and will serve as an excellent collection system for in-situ leach solutions. Block caving has extensively displaced and rubblized the oxide ore available to in-situ leaching, which should allow very good permeability, optimal solution-mineral contact, and maximum copper extraction. Large-scale testing of in-situ leaching is planned for 1987.

GEOLOGY AND MINERALIZATION

The first exploration drilling on the San Manuel property was conducted in 1916 and consisted of two churn drill holes drilled in the vicinity of an outcrop of oxidized copper ore. Exploration was halted when drilling indicated low grade oxide copper mineralization. The San Manuel group of claims was subsequently staked in 1925 by another party. In 1943, the U.S. Bureau of Mines began an exploratory drilling program that ultimately resulted in a total of 15,839 ft. drilled in 17 churn drill holes and indicated that sulfide copper mineralization was quite extensive. Magma Copper Company obtained options on the San Manuel claims and adjacent property in 1944. Extensive drilling of the deposit from both the surface and underground has been conducted by Magma, and has fully delineated the deposit and the surrounding area. The most recent exploration drilling was conducted in early 1984 and included a total of 16,240 ft. in 34

holes drilled with down-the-hole hammer reverse circulation techniques.

The geologic history of the deposit is similar to other porphyry deposits in Arizona. Precambrian quartz monzonite was intruded by monzonite porphyry of Laramide Age. Hydrothermal alteration and sulfide mineralization closely followed the intrusion of the monzonite porphyry. The ore body, which was then cylindrical in shape with a poorly mineralized monzonite porphyry core, was tilted to the northeast and buried beneath at least two sedimentary conglomerate units. The sedimentary units are locally referred to as Gila (?) Conglomerate and are believed to be of Oligocene or early Pliocene Age and vary in thickness from a feather edge to over 1,000 ft. above the ore body. The deposit was then bisected by the San Manuel Fault, a low dipping normal fault and a major structural feature in the area. The upper portion of the ore body was displaced approximately 8,000 ft. down dip to the southwest and is referred to as the Kalamazoo Ore Body. Oxidation of the Kalamazoo is insignificant. The portion of the deposit in the footwall of the fault zone and much nearer to the surface is the San Manuel Ore Body, which has been the target of sulfide mining for the past 28 years.

The San Manuel Ore Body is typical of most porphyry copper deposits in that the near surface portion of the sulfide ore body has been extensively oxidized. A period of erosion exposed the north limb of the deposit and oxidation of the sulfide ore

through normal weathering processes occurred as well as a comparatively minor amount of supergene enrichment. The ore body outcrop consists of approximately 80,000 sq.ft. of exposed oxide ore and has provided almost 20,000 tons of oxide ore for metallurgical testing. The mineral of major economic importance is chrysocolla, with cuprite probably contributing a minor amount to the soluble copper assay. Chrysocolla occurs principally as coatings and fillings of fracture surfaces and vugs. Chrysocolla also occurs with quartz stringers as small veins and disseminations distributed throughout the rock.

The original grade and mineral lensing patterns of the deposit have been displaced by a series of normal faults and radically disturbed by subsidence from block caving. The segment of the ore body that lies outside the vertical limits of block caving has retained recognizable grade lenses although displaced by varying amounts from the original position. The portion of the deposit within the vertical limits of block caving has been rubblized and the original grade and mineral patterns disturbed to the extent that a homogenous oxide ore zone now exists. These factors have also greatly diminished the competency of the mineralized zone which makes profitable exploitation of the deposit by leaching methods highly feasible.

The open pit minable reserve is estimated at over 56 million tons of oxide ore at 0.72% total copper and 0.47% acid soluble copper at a 0.20% soluble copper cutoff. Within the ultimate pit

limit as it is now planned are more than two million tons of sulfide ore with an average grade of 1.27% total copper and 0.16% soluble copper. The stripping ratio of the pit in tons of waste to total tons of copper ore is 2.20:1.

MINING

Mining of the oxide ore body using open pit, block caving, and in-situ leaching methods has been evaluated in the past but rejected basically because of interference with sulfide ore block caving. Underground mining has now advanced away from the oxide ore target, which allows for the timely recovery of copper from the oxide reserve utilizing open pit mining and in-situ leaching methods. The evaluation that led to the decision to develop the open pit was initiated in September 1982.

The oxide open pit as currently planned has been designed with the surface boundary constraints necessary to protect the existing facilities of the underground operation, minimize adverse effects of subsidence on the open pit walls, and limit pit encroachment on in-situ ore reserves. With these parameters limiting pit expansion, the dimensions of the ultimate pit will be 4,300 ft. long by 2,200 ft. wide and will reach 800 ft. below the oxide ore outcrop.

The pit design utilizes a bench height of 30 ft. and mining will incorporate a minimum mining bench width of 120 ft. An overall pit slope of 38° has been used in all mine plans to date and for initial mining. Several factors indicated that the slope angle selected is suitable for the operating conditions at San Manuel and could possibly be conservative. Underground mine pumping systems have created a depression cone in the water table which now intersects the underground mine on the deepest production level at an approximate elevation of 665 ft. The material within the pit boundary, therefore, has been effectively dewatered, which will enhance slope stability by the reduced pore pressure. Additionally, predominant geologic structures such as faults and bedding planes in the conglomerate, which would ordinarily adversely affect slope stability in a few locations, have been for the most part destroyed by subsidence. Also, the final pit slope on the north side of the pit will be located in competent, crystalline rock which has been affected only to a minor degree by subsidence-induced tension. Ultimately, pit slopes greater than the planned 38° may be utilized once practical experience is gained concerning mining in an active subsidence zone. A comprehensive slope monitoring program is being designed and will be implemented to insure safe and efficient mining throughout the project life.

The initial mining rate will average 11,700 stpd of ore and 24,000 stpd of waste for the first mining year. Mining rates in subsequent years will gradually increase due to slightly

declining ore grades, coupled with the requirement to maximize plant capacity over the life of the project. Operations will be scheduled on a 20-shift per week basis with the down-shift utilized for equipment maintenance.

Fragmentation of material will be accomplished by a well planned drilling and blasting program. The condition of the two types of waste material present within the pit boundaries varies from very competent virgin ground to highly fractured broken rock, depending on the proximity to block caving and the amount of displacement from the original position. Blastholes will be carefully logged and tension crack locations noted. The drilling and blasting foreman will adjust the pattern, spacing, and loading of the holes accordingly to insure that the material is adequately fragmented for loading and that the efficient use of explosives becomes routine. Blastholes will be loaded with ANFO and rounds will be delayed. Secondary blasting of waste material may be common, but hopefully not excessive.

The oxide ore exposed at the outcrop exists in a condition that resembles a shot bank. The method and type of mineralization, amount of faulting, and subsidence induced tension have combined to decrease the competency of the ore, both near the surface and at depth. Surface ore has been easily excavated directly from the bank or dozed with little effort. Tough, competent zones are anticipated, but should be the exception. Blasthole spacing in the ore will be determined by

ore control requirements, but blastholes will be lightly loaded and shot to enhance loading equipment productivity.

Multiple row shots of 9-7/8-in. blastholes will be used. Pattern type and spacing will be determined and modified as blasting experience is gained in the different material types and various ground conditions. Blastholes will be a nominal 35 ft. deep. Two truck-mounted Ingersoll-Rand T5BH rotary drills will be used for primary blasthole drilling and will also provide ore/waste boundary definition drilling depths up to 120 ft.

Initially, ore will be loaded by a Caterpillar 992C wheel loader with high lift arms and a 12-cu.yd. bucket. Waste loading will be handled by one P&H 1200 hydraulic mining excavator equipped with a 13-cu.yd. dipper. The hydraulic excavator, which was manufactured in Milwaukee, was selected for the high crowd and breakout forces, which allows excavation from a lightly shot or unshot face with a minimal reduction in productivity. The excavator is powered by two Cummins KT1150 diesel engines. The increased mobility provided by the elimination of trailing cables and associated towers, bridges, equipment, and problems was also of primary concern in excavator selection. A third loading unit is scheduled to be added to the fleet in mid 1986.

Truck haulage will be utilized at San Manuel. Material will be loaded into five Unit Rig M100 Lectra Hauls. The trucks purchased have been factory remanufactured with design

improvements installed by Unit Rig. Each haul unit is powered by a new Cummins KTA 2300-C engine with a rated 1,050 horsepower. A sixth unit will be added to the fleet in May 1986 with other additions scheduled in subsequent years.

Haul roads are 60 ft. wide with a maximum grade of 8%, with some temporary access ramps at 10%. A major factor contributing to the low cost of mining the oxide pit is the location of the waste dumps. Waste dumps are located in the large depression caused by subsidence. Waste hauls will be very short with minimal lift due to the daylighting of the southern benches, which allows favorable haul grades to be utilized in many instances. These combined factors will yield high haul unit productivity, which reduces the capital requirement for haulage trucks as well as minimizing operating costs. Oxide ore will be hauled to the leach dump located north of the pit. Sulfide ore will be hauled to a railroad siding near the mine production shafts where it will be loaded into 100-ton rail cars and hauled to the San Manuel mill and smelter for processing.

Additional mine equipment includes one Caterpillar 16G motor grader for road maintenance, one Cat 777 truck equipped with an 18,000 gallon tank for dust control, and a Clark 175C loader with a 5-cu.yd. bucket for rock chasing and other utility work. The dozer fleet includes one Cat 824C wheel dozer for loading area cleanup and minor dozing, one Cat D9L dozer for in-pit dozing, and two Cat D8L tractors equipped with U-dozers and radial

rippers for deep ripping on the leach dump. A D8L will also be assigned to waste dump dozing. The haulage trucks and the P&H 1200 excavator are equipped with automatic lubrication systems. Manually activated fire suppression systems have been installed on all the major mine equipment and all equipment used in the pit is radio equipped.

Equipment maintenance will play a major role in minimizing operating costs and providing optimum fleet availabilities to achieve operating goals. Magma will utilize maintenance programs developed by the experienced staff as well as those recommended by the original equipment manufacturers (OEMs). The programs are tailored specifically for the operation at San Manuel, which uses hydraulic systems extensively. Maintenance scheduling and history will be tracked by an information management computer program. In addition, the experience of the underground maintenance staff is available. Efficient preventive and repair maintenance will be performed in the new truck shop/office complex. The shop will have 7,500 sq.ft. of shop area under roof with three maintenance service bays, one grease bay, and one light truck bay. A parts warehouse is an integral part of the shop. The building will also include office space for the assistant superintendent of the operation, pit engineering, and operations staff.

The operation utilizes a simple line organization which permits excellent cooperation and coordination between pit

operations, engineering and maintenance. The Mine Manager directs both the underground operation at San Manuel as well as the open pit. The Mine Superintendent also has responsibilities in both the underground and open pit. Open pit operations, engineering, and maintenance are directed by the Assistant Mine Superintendent devoted exclusively to the open pit. The pit operating staff includes one general foreman, four shift foremen, two pit foremen, and a drilling and blasting foreman. The pit engineering department employs a chief engineer, two planning engineers, a geological engineer, one systems engineer, an ore control technician, and a draftsman. Two survey crews will be utilized. The pit maintenance staff consists of one general foreman and one craft foreman, and also relies on the direction of the General Mechanical Foreman, who has responsibilities in the surface, underground, and open pit.

At startup, approximately 35 hourly personnel will be employed by operations and are assigned to equipment as required due to a direct line of flow. The two job classifications in the pit are operators and truck drivers, with truck drivers to be utilized as operators when necessary and qualified. The maintenance department will have approximately 15 hourly employees assigned which may be utilized in underground or other surface assignments as well as the open pit. The flexibility in operations and maintenance will allow maximum utilization of the hourly work force.

DUMP LEACHING

The dump leaching process phase is a carefully planned program that will yield high copper recovery and is another major factor in achieving low cost production of copper cathode at San Manuel. Dump leaching will utilize a completely closed circuit. The flexibility of the dump leach environment will readily allow modifications to the leaching procedure when required and as operating experience is gained.

Leach testing of oxide ore has been conducted by both Newmont Research (beginning in 1970) and the Magma Metallurgical Department. Vat leaching of crushed ore and copper precipitation techniques were evaluated but were not considered to be cost effective. Other leach tests included bottle tests, 14-in. diameter by 5-ft. high column and 2-ft. diameter by 20-ft. high column tests, and large-scale dump tests of 2,000-ton and 8,000-ton samples. The testing provided information regarding dump construction techniques and dump lift height as well as establishing the metallurgical parameters of dump leaching. The leach testing program provided results that indicate recovery of at least 85% of the acid soluble copper in a 60-day dump leach cycle is achievable. Due to the friable condition of the ore, for reasons previously discussed, crushing prior to leaching is not required.

The area selected for the leach dump site covers approximately 83 acres. Preparation of the area to maximize solution recovery and protect the environment is necessary. The site has been completely grubbed and brought to grade to utilize the natural drainage pattern of the area for channeling of the pregnant leach solution (PLS) to the collection pond. A 60 mil high density polyethylene (HDPE) liner will be installed on the carefully prepared surface and will be an impermeable membrane which will confine leach solutions on the leach pad and channel the PLS to the collection pond. A cushion of 18 in. of select fill will protect the liner system from initial loading stresses.

The dump will be constructed in 20-ft. high lifts and individual increments of approximately 850,000 tons. This will provide about 750,000 sq.ft. of dump surface under various stages of leach at any given time. Each dump will be constructed by dumping ore from the haul trucks in the designated areas along an established haulage corridor in order to minimize compaction from the trucks on the dump. Ore will be dozed from the dumping site to the dump limit with a Caterpillar D8L tractor equipped with a universal dozer and radial attachment. After dozing, the entire surface of the dump, including the haulage corridor, will be extensively cross ripped by the D8L to a maximum depth of seven ft. to enhance permeability. The entire surface of each dump will be cross ripped again at the completion of the leach cycle to destroy impermeable formations or channels that may have formed during leaching. Eventually, subsequent 20-ft. lifts will

be constructed on previously leached areas which should provide additional copper recovery from secondary leaching of lower level lifts.

The typical 60-day leach cycle will consist of contacting the ore with a preconditioning solution of 100 to 150 gpl H_2SO_4 until the ore is saturated and allowing a seven-day curing period before applying the raffinate. Preconditioning the ore in this fashion will result in higher initial PLS heads and greater initial recovery during the leach cycle. Raffinate, with an input acid strength between 5 and 10 gpl H_2SO_4 will be distributed on top of the dump with a pipe and wobbler sprinkler system at a flow rate in the range of 1.0 to 1.5 gpm per 100 sq.ft. Maximum application rate will be 9,200 gpm. The flexibility of dump leaching allows modifying the application rate, acid concentration, or even ore production rate to compensate for age and grade variations in individual dumps or higher than projected recovery. Maximum tankhouse capacity can be utilized in this manner.

The PLS will be collected in a surge pond at the eastern toe of the dump with a surge capacity of 5,000,000 gallons. From the collection pond the PLS will be pumped to a plant feed pond with a capacity of 10,000,000 gallons located near the solvent extraction (SX) facility. Raffinate will be stored in a 10,000,000-gallon pond and pumped to the leach dump.

SOLVENT EXTRACTION-ELECTROWINNING

Solvent extraction-electrowinning is a two-stage process where the PLS from the leach dump is upgraded to a copper-loaded electrolyte and the copper ions won from the solution by applying an electric current. There are two phases of solvent extraction: extraction and stripping. In the extraction phase, the PLS is mixed with an organic media which consists of kerosene and a reagent that is highly selective of copper ions in weak acid. Copper ions are transferred to the organic and hydrogen ions to the aqueous. The copper carrying organic floats on top of the barren aqueous solution (raffinate) which allows separation of the two solutions. In the stripping stage the loaded organic is contacted by copper sulfate in a strong acid solution (electrolyte) and the copper ions are "stripped" from the organic to the electrolyte. The loaded electrolyte is pumped to the electrowinning cells where a DC circuit has been established. An electric current is conducted through the electrolyte, which causes positively charged copper ions to be plated on the surface of the negatively charged cathode. In all phases of SX-EW processing, the barren or stripped solutions are recycled in a closed circuit, resulting in zero discharge and maximum solution utilization.

Solvent extraction at San Manuel will utilize two conventional mixer-settler SX trains with two stages of

extraction and one stage of stripping in each train. The nominal rated flow through each train is 4,000 gpm of PLS and the maximum rating through the SX plant is 9,200 gpm.

The electrowinning tankhouse will have a rated capacity of 25,000 tons per year of high grade copper cathode. There are 94 cells in the tankhouse. Permanent anodes are of typical lead-calcium-tin composition. Copper cathodes are to be plated on reusable stainless steel mother blanks of Copper Refineries Ltd. patented design. Two cathodes weighing up to 100 lbs. each will be plated on one stainless steel blank every seven days and mechanically stripped. Use of the stainless steel blanks eliminates the need for copper starter sheets and will result in a substantial reduction in operating costs. Cathodes will be shipped from the San Manuel minesite to market or to the Magma Copper Company rod plant nine miles to the south in San Manuel.

Both the solvent extraction trains and electrowinning tankhouse have been designed and constructed to allow future expansion to produce 50,000 tons per year of 99.9% copper cathode.

SUMMARY

Magma Copper Company is developing an open pit mine and constructing a solvent extraction-electrowinning plant at the San Manuel minesite. The San Manuel Oxide Open Pit Project is the first phase of a total program to recover copper from oxide ore at San Manuel. Open pit reserves are estimated at 56 million tons of oxide ore at 0.72% total copper and 0.47% acid soluble copper grade and over two million tons of sulfide ore at an average grade of 1.27% total copper.

There are several factors that will contribute to the low cost production of copper over the life of the project. The ore has been extensively rubblized by subsidence from block caving, which will result in reduced drilling and blasting costs for ore fragmentation. Waste will be disposed of in the subsidence area, which will result in short hauls and the capability of using favorable haulage grades to reduce operating costs. The mineralogy and physical characteristics of the oxide ore are ideally suited to dump leaching. Crushing of the ore is not required and recovery of greater than 85% of the acid soluble copper is attainable in 60 days. Low cost solvent extraction-electrowinning will be used to produce LME grade one copper cathodes. The electrowinning process will utilize stainless steel cathode blanks to reduce operating costs.

The operation will produce more than 500 million pounds of copper over an eleven-year period, which will strengthen Newmont Mining Corporation's position as a major domestic copper producer and leader in the mining industry.

SFW:db710

REFERENCES

1. BAKER, D. A., "A General Description of the San Manuel Oxide Deposit." Paper presented at the 1984 Spring Session, Mining Geology Division, Arizona Conference of AIME.

2. DALE, V. B., "Mining, Milling and Smelting Methods, San Manuel Copper Corp., Pinal County, Ariz.", Bureau of Mines Information Circular 8104, 1962.

3. SCHWARTZ, G. M., "Oxidation and Enrichment in the San Manuel Copper Deposit, Arizona." Economic Geology, Vol. 44, No. 4, 1949, pp. 253 to 277.

4. THOMAS, L. A., "Subsidence and Related Caving Phenomena at the San Manuel Mine," Unpublished Magma Copper Co. report, 1971.

SFW:db853

ASARCO

Exploration Department

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.

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 28.24 lbs leach = 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

In-Place Leaching at Miami Mine, Miami, Arizona

by James B. Fletcher

Leaching of the block caved mine from 1941 through 1970 is described, and the improvements made are listed. Detailed is the increased knowledge of leaching gained through this operation, including methods used in forecasting production, prediction of the economical end point, rate of recovery, and ultimate recovery.

The Miami mine is located near the town of Miami, Gila County, Ariz. The Miami mine started mining in 1910 and finished in July 1959. The ore body was divided and mined as follows:

	Tons	
High Grade	24,400,000	Top slicing, sublevel caving, etc.
Low Grade	105,100,000	Block caving
Mixed ore	9,800,000	Block caving
Low Grade #2	13,100,000	Block caving
Total Mining	152,400,000	

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

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

In block caving there is always some dilution, and drawing of ore stops when the grade drops past an economical point. This results in a small amount of copper left in the stopes below the capping. There is some copper in the capping, probably below 0.03% or less than 0.6 lb per ton. Over the low grade No. 2 ore body and portions of the high grade, there was an oxide capping which was not mined. The crushed pillars are another source of copper. The leaching at Miami is an attempt to recover this last remaining copper of a worked-out mine.

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

Leaching

Leaching started on a small scale in December 1941. Some of the early problems that were met are as follows:

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

The next leaching was tried over the mixed ore body with water. The solution came through with very little copper. Water was tried over the sulfide stopes with no recovery of copper. Then acid was added to the water over the mixed ore. The first leaching was started with a 3% H_2SO_4 solution which was immediately dropped to a 0.6% solution. At the present time we try to control the H_2SO_4 feed by the amount of free acid in the pregnant solution. If the acid in the pregnant solution drops much below 0.5 lb H_2SO_4 per ton of solution, the pumps, pipe lines, sumps, etc., are clogged with iron salts.

The early thinking was that the reuse of the ferrous solution from the precipitation plant would clog the caved area and stop the leaching.

In order to leach sulfides it was realized that Fe''' would be necessary, and several attempts were made to manufacture a high Fe''' solution. It was discovered that by adding acid to the off-solution from the precipitating plant, Fe'' was converted to Fe''' as the solution went through the caved ground. The equation for this reaction is:

$$4FeSO_4 + 2H_2SO_4 + O_2 \xrightarrow{\text{Bacteria}} 2Fe_2(SO_4)_3 + 2H_2O.$$

The ferric sulfate reacts with the copper sulfide mineral, chalcocite, to form copper sulfate, ferrous sulfate, and elemental sulfur as shown in the following equation:

$$Cu_2S + 2Fe_2(SO_4)_3 \xrightarrow{\text{Bacteria}} 2CuSO_4 + 4FeSO_4 + S.$$

The ferrous sulfate is then reoxidized by the iron-oxidizing bacteria to form more ferric sulfate, and the cycle is repeated. The elemental sulfur set free in the dissolution of the copper mineral is oxidized by the sulfur-oxidizing bacteria, Thiobacillus Concretivoues, in the presence of oxygen and water, to form sulfuric acid as shown in the following equation:

$$2S + 3O_2 + 2H_2O \xrightarrow{\text{Bacteria}} 2H_2SO_4.$$

The iron-oxidizing bacteria, Thiobacillus Ferrooxidan, and the sulfur-oxidizing bacteria, Thiobacillus Concretivoues, are present in the Miami solutions.

Miami reports assay values in terms of pounds per ton of solution. Practically, it is a simple term, because all pumps are rated in gallons per minute and we talk in gallons per minute and days. 1 gal = 8-1/3 lb; therefore 1 gpm for 24 hr = (8-1/3 × 1440/2000 = 6 tons.

Requirements: The Miami plant was designed to treat 2000 gpm and to produce 1,500,000 lb Cu per month or 50,000 lb Cu per day. Therefore

2000 gpm × 6 = 12,000 tons of solution per day

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

+0.04 Tail

4.21

+0.07 Smelter Loss

4.28 Required Assay

J. B. FLETCHER, Member AIME, is Leaching Superintendent, Miami Copper Operations, Cities Service Co., Miami, Ariz. SME Preprint 71AS40, AIME Centennial Annual Meeting, New York, March 1971. Manuscript, Nov. 6, 1970. Discussion of this paper, submitted in duplicate prior to Mar. 15, 1972, will appear in SME Transactions, June 1972, and in AIME Transactions, 1972, Vol. 252.

Leaching can best be thought of in four dimensions. The first two dimensions are the surface to be covered. We have approximately 5,000,000 sq ft to cover. One-half of the area was covered in the first 20 years of leaching. The entire area had been covered in the next four years. Since 1963, the entire area has been under leach.

Tailing water from the precipitating plant constitutes the most important source of leach solution. Water losses occur principally from evaporation and at Miami they approximate 10%. Fresh water is used as wash water in the precipitating plant and usually supplies the makeup water to maintain the 2000 gpm from underground. Acid is immediately added to the off-solution at the sump to bring the solution to 0.5 lb H_2SO_4 per ton of the solution. This keeps the iron from precipitating out, which gives us a clean sump, pump, and pipelines to the caved ground. This strength acid does not deteriorate the cement asbestos pipeline. When the solution reaches the caved ground it is measured in weir boxes and additional acid is added to bring the strength up to the acid required at each area. From this point on, the solution is carried in polyethylene plastic pipe to the point of application. We have used ponds, sprays and drill holes to introduce the solution to the caved ground. Due to the uneven surface slope encountered in the caved area, sprays are the most satisfactory. Sprays also allow the solution to be introduced slower than by ponding. Drill holes were used in an attempt to introduce the solution below the conglomerate cover east of the Miami fault.

The solution descends vertically from the point of application except where sealed off by the heavy gouge of the Miami fault. Holes that were drilled through this gouge leached an area approximately 50 ft in diam directly below the hole. To leach this area would required drilling on 50-ft spacing.

The third dimension is the amount of ore and waste to be penetrated. This averages about 600 ft. Of this 600 ft, the bottom 150 ft contains the mixture of ore and waste we are leaching. We try to maintain the underground collecting system so that we have a free flow of water in the main haulage ways and we have a few key sampling points.

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

The next point is how long can the spray remain at one place before the grade drops. This depends on the amount of ore in the area being leached. It is practically impossible to correlate the mining extraction records with the material to be leached. So when a new area is to be leached, the spray is turned on and there is a waiting period of 3 to 4 weeks before a sample can be taken underground.

The sprays are left in one place until the grade from that area drops below 2 lb Cu per ton. The spray is then turned off and the area allowed to lie idle for a period of time. The rest and leach periods are alternated over and over. The second time over an area we can remain about one-half the time used originally and recover about one-half the amount of copper. The third time over it is one-half the second, the fourth is one-half the third, etc.

There are several explanations for this. During the rest period oxidation of the sulfides may take place, but since we are leaching with an acid ferric sulfate solution and from observed action on the surface it appears that a reverse capillary action is the best explanation. A copper-bearing boulder can be put in a flow of a leaching solution, then the solution removed, and as the boulder begins to dry, a copper-bearing film forms on its surface. When a copper-bearing rock is broken, it usually cracks on the mineralized seams and shows that the solution is actually penetrating the rock.

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

$$(1)\ Cu_2S + Fe_2(SO_4)_3 = CuSO_4 + 2FeSO_4 + CuS$$

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

The first reaction takes place fairly rapidly. The second reaction is much slower. Chalcopyrite, a minor ore mineral at Miami, does leach but at a much slower rate.

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

To Jan. 1, 1970, the Miami mine has produced by leaching 261,000,000 lb of copper. The iron to copper ratio is 1.3 and the acid to copper ratio is 2.4, both figures based on net smelter returns. It is apparent that the largest factors affecting the cost per pound of copper are the iron and acid consumption. If the acid in the pregnant solution becomes too high, the iron consumption per pound of copper will go up. (The ferric iron in the pregnant solution also affects the iron consumption.) If the acid strength is too high, the gangue consumption of acid will go up. The acid strength is kept below 15 lb even in the high grade oxide areas unless the ore is very close to the surface. The time lag between the solution entering the caved ground and the same solution appearing underground where it can be sampled and assayed makes this problem of control difficult. The best way we have found is to keep graphs of the assays of the underground sample points. From these graphs we can check the

MIAMI COPPER COMPANY
DIVISION TENNESSEE CORPORATION
CERTIFICATE OF ASSAY

LEACHING PLANT — TYPICAL — 1962

Beaker Nos	MARKS, ETC.	SAMPLE GMS	CU	FE	FE^{++}	FE^{+++}	H_2SO_4	PH
			- POUNDS PER TON -					
Feed	ABC		4.28	[illegible]	2.3	[illegible]	[illegible]	[illegible]
Tail	A		0.04	9.4	9.0	0.4	[illegible]	[illegible]
	B		0.05	9.2	[illegible]	0.1	[illegible]	[illegible]
	C		0.03	9.7	9.5	0.2	[illegible]	[illegible]
C A Feed	ABC		0.03		8.5		[illegible]	[illegible]
Sump	A		0.04	9.3	8.5	0.8	[illegible]	[illegible]
	B		0.03	9.2	8.4	[illegible]	[illegible]	[illegible]
	C		0.06	9.6	8.8	[illegible]	[illegible]	[illegible]

Assayer

MIAMI COPPER COMPANY
DIVISION TENNESSEE CORPORATION
CERTIFICATE OF ASSAY

LEACHING PLANT — TYPICAL — 1968

Beaker Nos	MARKS, ETC.	SAMPLE GMS	CU	FE	FE^{++}	FE^{+++}	H_2SO_4	PH
			- POUNDS PER TON -					
Feed	ABC		3.10	2.90	1.90	1.00	0.5	2.5
Tail	A		0.03	6.10	6.00	0.10	Tr	4.0
	B		0.04	5.90	5.90	-	Tr	4.0
	C		0.02	5.90	5.90	-	Tr	4.0
C A Feed	ABC		0.03	-	5.50	-	9.0	4.5
Sump	A		0.03	5.90	5.20	0.70	0.6	2.7
	B		0.01	5.80	5.20	0.60	0.5	2.7
	C		0.01	5.80	5.20	0.60	0.6	2.7

Assayer

Fig. 1—Typical assay reports for 1962 and 1968.

OF MATERIAL MINED BY OPEN PIT AND DUMPED INTO CAVED AREA

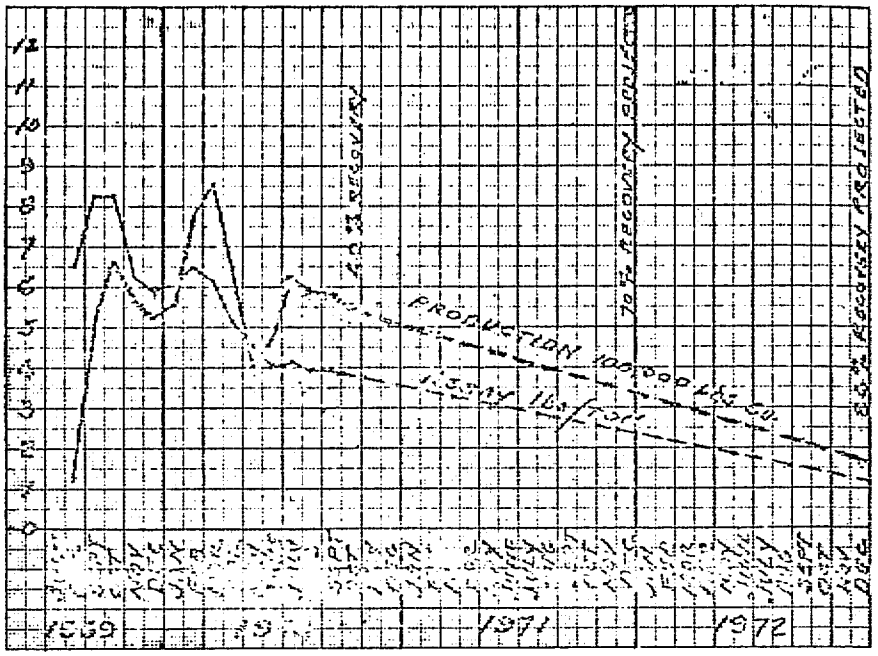


Fig. 2—Assay and copper production by months from start (1969) to November 1970 with a projection to 1972, Miami mine.

trend and anticipate a drop in grade or a decrease in acid consumption.

The operation since 1963, the time when no new areas to leach were added, shows a gradual drop in grade of the pregnant solution from 4.03 lb per ton in 1963 to 3.10 lb per ton in 1968.

Iron salts do precipitate as the solution goes through the caved material. From the assay report, Fig. 1, it can be seen that iron loss from the cave area feed to the precipitation plant feed is 4.2 lb Fe per ton of solution to 1962. In 1968 the loss is 2.6 lb Fe per ton of solution. It has been our experience that there is no plugging action of the cave ground from these basic iron salts. It is not known where the iron precipitates, but is probably high in the column.

Since the tail solution from the iron launders is to be recirculated the argument always occurs; why make a good recovery? It has often been proposed to recirculate low grade solution back through the same area (with addition of acid) in order to upgrade the solution to the iron launders. A strike was called at Miami in August 1954. The pregnant solution was pumped from the mine and stored in a pond on the mill tailings dam. In September this solution, which assayed 5 lb Cu per ton, was pumped back to the leaching areas, more acid was added, and sprayed back on to areas. No pickup in the pregnant solution was noted in the following months. The pregnant solution from the mine continued to assay 5 lb per ton as it had prior to the strike. Miami was again on strike in 1967. Only the minimum amount of washing of the iron launders was allowed. The assay results in Table 1 show that recirculation did not upgrade the solution.

Acid was added to bring the acid strength of the recirculated solution to the normal spray solution.

Miami Copper has two other properties, Castle Dome and Copper Cities, where dump leaching is carried on. In both operations the dumps contain enough pyrite so that acid is not required for leaching. No. 14 dump at Copper Cities mine was producing a low grade solution. In December 1964 it was decided to try recirculating No. 1 dam pregnant solution over this dump with the hope that the Fe^{+++} and H_2SO_4 in the solution would help the leaching. On Dec. 21, 1964, 500 gpm of No. 1 dam solution was started on the dump. The assays for the month of January 1964 showed the results given in Table 2.

From Table 2 it is apparent that no copper is being leached from the dump to date. The copper assay shows such a small drop that it could be in assaying. The only positive reaction is the loss of iron in the dump. The assay from this dump the last time it was run (October 1964) with tail solution is given in Table 3.

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. This resulted in 200,000 sq ft of level area ideal for leaching, containing 1,348,172 tons assaying 0.78% total Cu, 0.59% oxide Cu, and 0.19% sulfide Cu or 21,000,000 lb Cu. The solution came through to the 1000 level and mixed with the solution from other parts of the mine. However, we were able to weir and assay this solution. From the results obtained from this area a close approximation of the overall recover can be made. The graph (Fig. 2) shows assay and copper production by

Table 1. Assay Results in 1967

Year 1967	Cu Assay, Lb per Ton Feed to Plant	Cu Assay, Lb per Ton Recirculated Tail Solution
July	2.73	0.05
August	2.69	1.04
September	2.29	1.37
October	2.40	1.07
November	2.30	1.08
December	2.40	0.75

Table 2. Copper Cities Dam No. 1 Assay Report, January 1964

	Gpm	Tons	Assay	Lb Cu	Fe	H_2SO_4
To Dump	477	88,722	1.59	141,067	1.09	0.35
From Dump	396	73,656	1.58	116,376	0.42	0.24
Difference	81	15,066	0.01	24,691	0.67	0.11

Table 3. Copper Cities Dam No. 1 Assay Report, October 1964

	Cu Assay	Fe	H_2SO_4
To Dump	0.01	3.03	0.00
From Dump	1.10	0.20	0.10
Difference	1.09	2.83	0.10

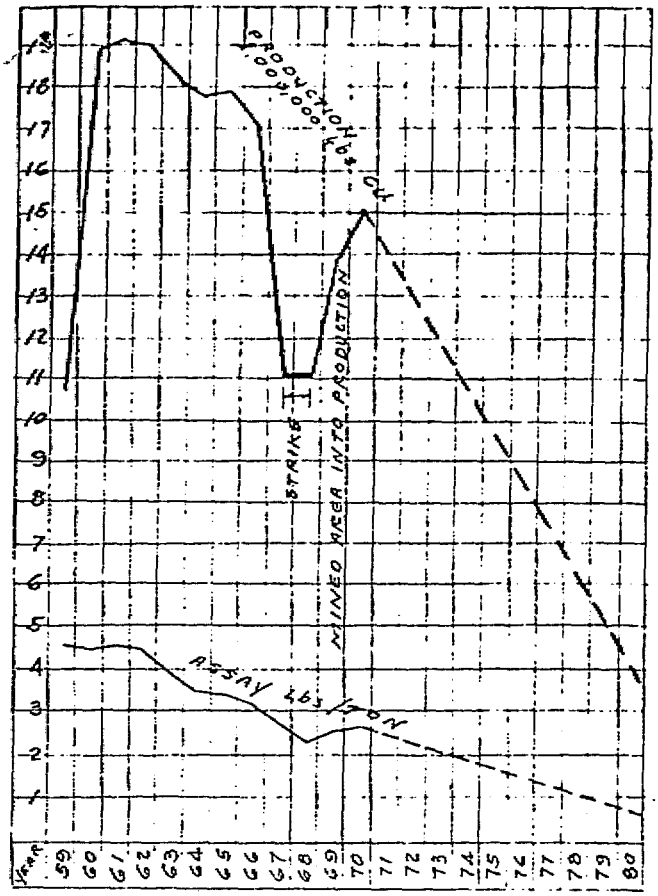


Fig. 3—Projected assays and production, with flow of 2600 gpm, Miami mine.

months from the start to November 1970 with a projection to 1972. The maximum flow that can be sprayed on this area without excessive pooling is 900 gpm. This gives a flow rate of 0.45 gpm per 100 sq ft with a loss of 10% by evaporation and gives a recovery of 800 gpm from this area.

Although the plant was designed for a flow of 2,000 gpm, as the grade dropped the flow was increased in order to keep the production as high as possible. At the present time 2600 gpm is the maximum flow without major changes in the pumping system and precipitation plant. With the 5,000,000 sq ft of area available, if the entire area were covered, it would give a flow rate of 0.052 gpm per 100 sq ft. Using 2600 gpm, a graph of the projected assays and production was made (Fig. 3).

Assuming the following cost and the projected production of 4,000,000 lb for 1980, the economics would be as follows:

Assumed Cost:	Acid	\$25.00 ton, 1.25¢ per lb
	Iron	\$64.00 ton, 3.2¢ per lb

Iron to Cu ratio: 2 (as the grade of the solution decreases, the iron consumption increases due to Fe^{+++})
Acid to Cu ratio: 2.3
Gives:

$$1.25¢ \times 2.3 = 2.875¢ \text{ per lb Cu}$$
$$3.2¢ \times 2 = 6.400¢ \text{ per lb Cu}$$

Freight, Smelting, and Refining = 7.00¢ per lb Cu (Assumed)
Total Fixed Cost 16.275¢

As the production drops, the manpower could probably be cut. But for this study, assume an operation cost of \$3000 per day which would include labor, power, taxes, overhead, etc. Using the projected copper production of 4,000,000 lb for the year 1980 gives:

$$\frac{4{,}000{,}000}{365} = 10{,}959 \text{ lb Cu per day}$$

$$\frac{\$3{,}000}{10{,}959} = 27.375¢ \text{ per lb Cu}$$

16.275¢ + 27.375¢ = 43.65¢ total cost per lb Cu for the year 1980.

Therefore, the ultimate recovery will depend on the price of copper.

The 1,348,172 tons mined in 1969 were dumped into an area of 200,000 sq ft with an average depth of 135 ft. From this dump it is projected to have a recovery of 80% in 3 years. If the area could have been doubled, 1800 gpm of solution could have been sprayed on the dump with a quicker recovery. This dump was mostly oxide which gives a comparably rapid recovery. The ore broke fairly fine with very few rocks over 12 in. in diam.

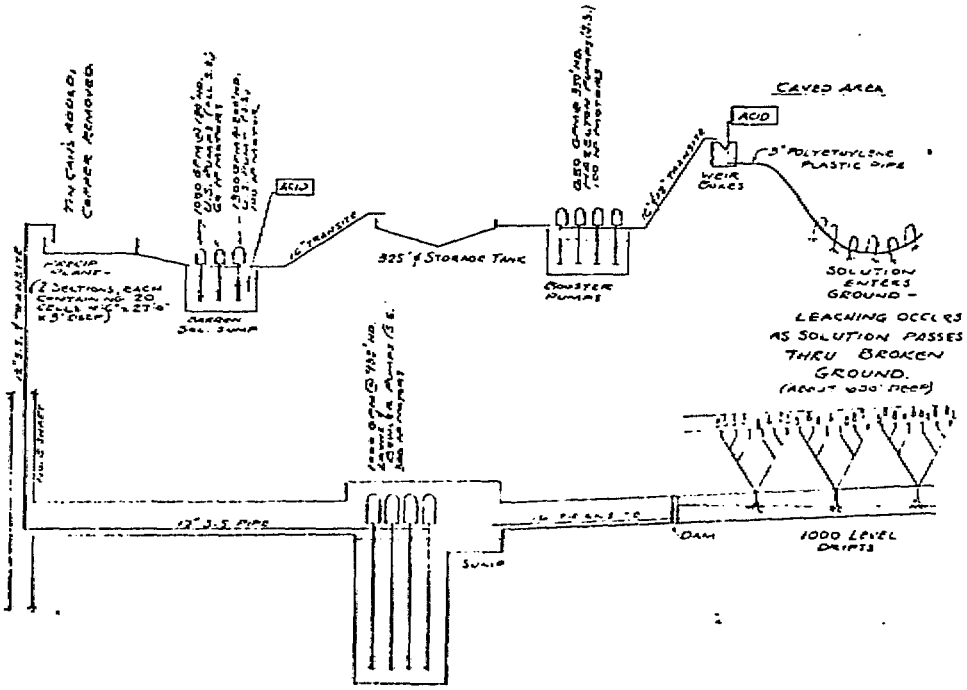


Fig. 4—Schematic of leaching at Miami Copper unit. All pumps are type 316 SS.

Table 4. Operating Crews for Miami Leach Operation

Precipitating Plant, 7-day operation	1 Operator or pumpman	3 shifts
	1 Equipment operator	Day shift
	1 Cave area pumpman	Day shift
	3 Cell washers	Day shift
	1 Foreman	Day shift
Underground, 5-day operation	3 Timber crews, 6 men	Day shift
	2 Hoist engineers	Day shift
	1 Foreman	Day shift
Maintenance crews from Copper Cities, 5-day operation	1 Pipeman	Day shift
	1 Machinist	Day shift
	1 Mechanic	Day shift

At the Copper Cities mine, as the dump material came from lower in the mine, where chalcopyrite was the main mineral, the rate of recovery dropped rapidly. At Castle Dome mine, at the toe of the dumps, which are composed of large boulders, leaching is very slow.

The precipitating plant at Miami is similar to others in the southwest. The only difference is the method of cell loading. Detinned shredded cans are received by rail and unloaded by a magnet. The magnet drops the cans into a hopper which feeds a belt conveyor. The belt travels over the center of the cells and by means of a traveling tripper the cells are evenly charged with iron. The precipitated copper is washed from the cans through wooden screens by high pressure water. The precipitates discharge onto a decant slab. Washing is done on the day shift only. Solutions are drained from the precipitated copper and pumped back to the cells on the afternoon shift. The copper is moved from the decant slab to a drying slab early the next morning before washing begins. The copper is moved with a front-end loader. The same loader loads the dry copper into the railroad car. The copper is shipped to the smelter at Douglas.

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

The operating crew of the plant is listed in Table 4.

The overall recovery by leaching at Miami will be impossible to determine as we do not know exactly how much copper we had at the start. A percent recovery figure for leaching is a misleading figure. A 50% recovery of a 1% ore body would be poor.

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

To give a complete picture of the leaching at Miami, a flowsheet (Fig. 4) and two typical assay reports (Fig. 1) are included.

Summary and Conclusions

1) Leaching in place of a block caved mine recovers the remaining copper.

2) Oxide minerals leached with H_2SO_4 solution.

3) Ferric sulfate solution is necessary for leaching sulfide minerals and can be obtained by utilizing the action of bacteria on the ferrous solution from the iron launders.

4) There is no plugging of the ground by disposition of iron salts from the reclaimed solution.

5) Sprays were found to be the most economical and simplest way to introduce solution to the caved area.

6) Recirculation does not upgrade the solution; therefore, a good stripping of the solution in the iron launders is good practice.

7) A rest period is utilized to induce reverse capillary action.

8) Rate of leaching is controlled by the size of the material, type of mineralization, and the area available.

9) The economical end point and recovery can be closely predicted by graphs of the leaching.

TECHNICAL NOTE

Size Distribution General Law of Fragments Resulting from Rock Blasting

by C. Dinis da Gama

In mining, all unit operations are closely interrelated, and the results of blasting are probably the key factor within that sequence.

The drilling program prior to blasting is just as dependent upon the desired rock fragmentation as are the loading, hauling, and mechanical crushing operations which are economically and technically influenced by the results of blasting, namely, the size and scatter of the product.

What is known today as optimum blasting[1] is the process of designing a blast in order to obtain a certain degree of fragmentation so that overall mining costs are minimized. Applying this criterion often results in a different program than when an individual blasting pattern is chosen only on the basis of the lowest operational cost for that pattern, disregarding what comes before or after the blast. Fig. 1 shows the characteristic variation of mining unit costs with the degree of fragmentation, or the maximum size of blocks obtained from blasting.[2] From the figure, it can be seen that optimum breakage must correspond to the lowest total costs, but reducing drilling-blasting costs only does not achieve such an optimum solution.

Methods of designing surface or underground blasting operations should comply with the optimization criterion, but this compliance is feasible only if a detailed mining system analysis is available. Therefore, predictions of costs of loading, transporting, and mechanical breakage of ore as a function of its size will define a narrow range of variation for drilling and blasting parameters in order to get the minimum total costs.

C. D. DA GAMA, Member AIME, is Assistant Professor, University of Luanda, Angola, Portugal. TN 71AM203. Manuscript, Jan. 19, 1971.

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 Lbs/Ton	Consumed 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

Milton H. Ward

Ranchers Exploration and Development Corporation
Albuquerque, New Mexico

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-

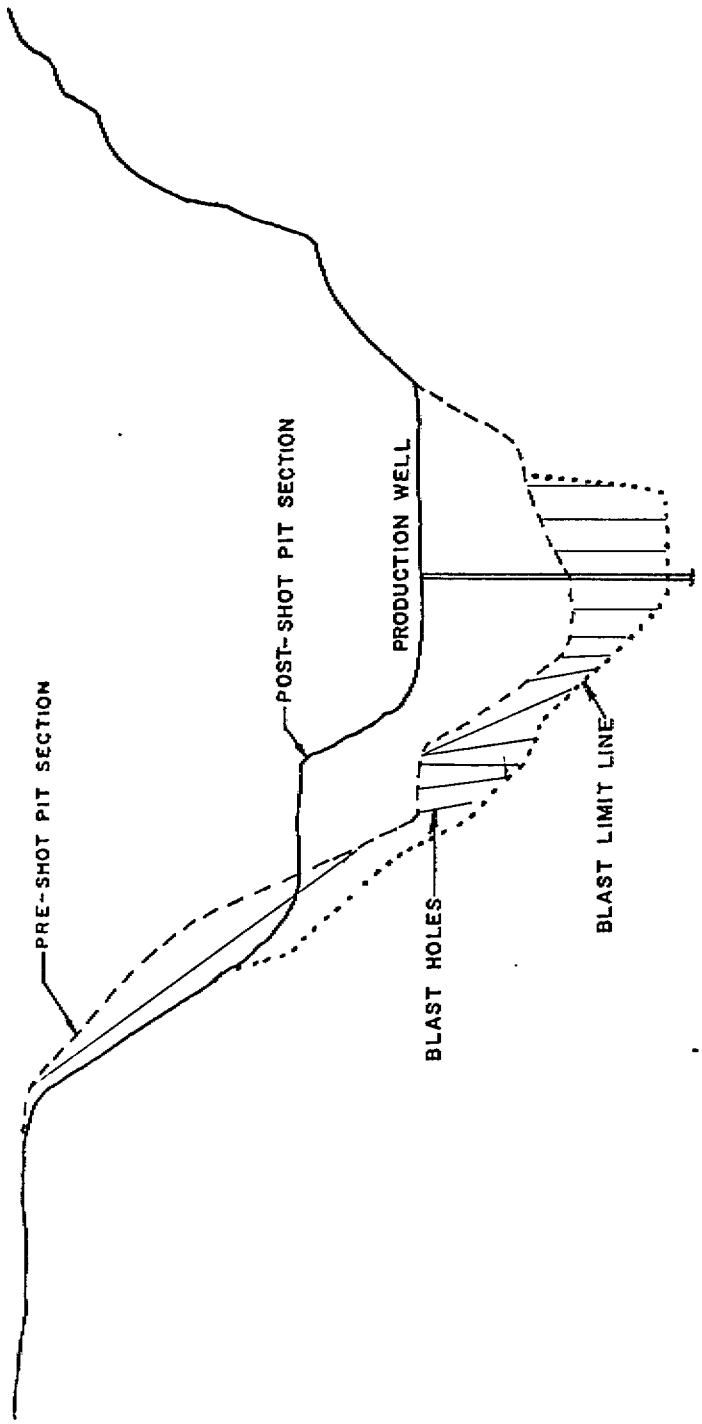


FIGURE I- PIT CROSS-SECTION BEFORE & AFTER BLAST

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 hac 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

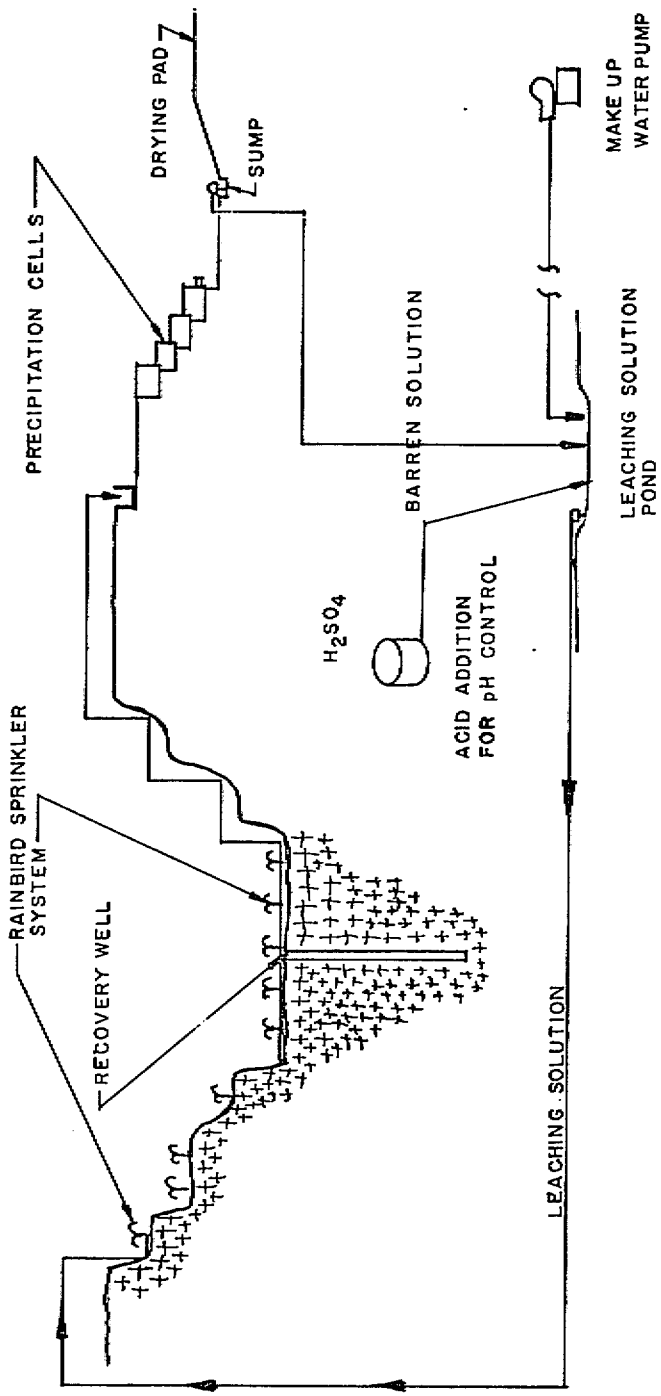


FIGURE 2 – FLOWSHEET–PIT LEACH OPERATION

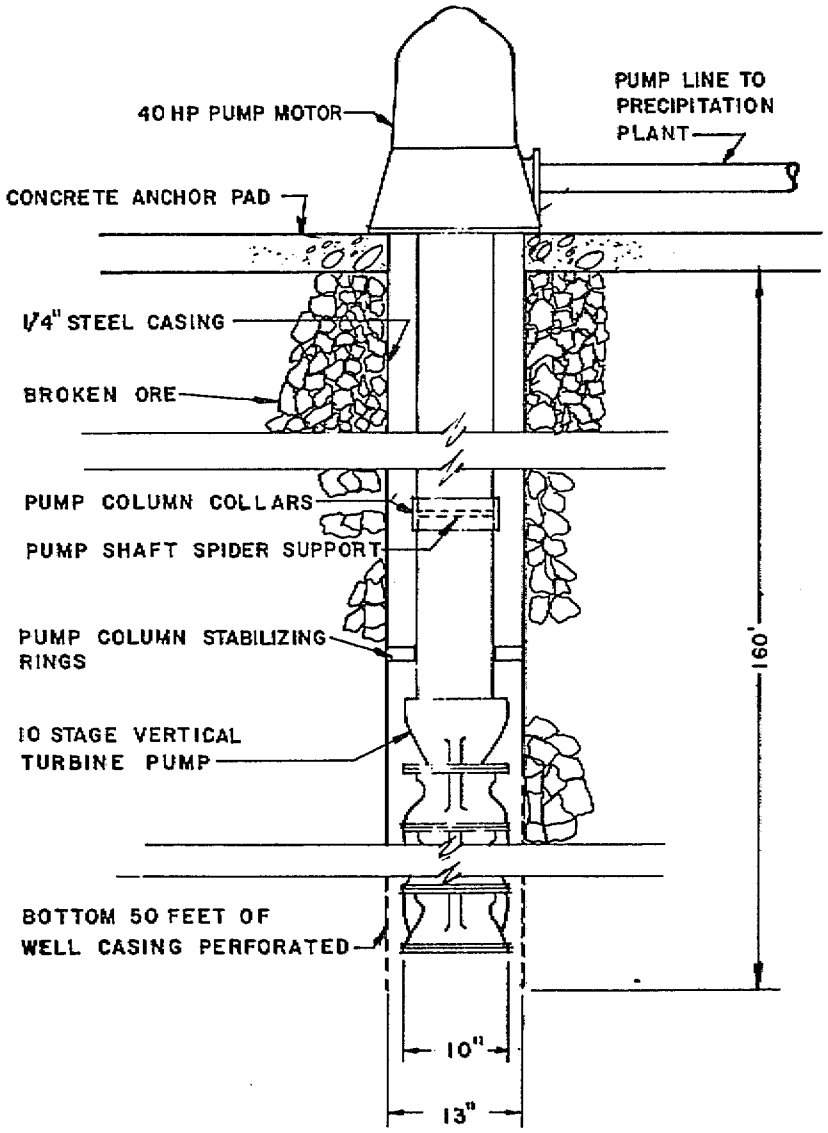


FIGURE 3- RECOVERY WELL- PUMP COLUMN & CASING

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

by Dennis V. D'Andrea[1] and Sidney M. Runke[2]

[1] Geophysicist, Twin Cities Mining Research Center, Bureau of Mines, U.S. Department of the Interior, Twin Cities, Minn.

[2] Metallurgist, El Paso Mining and Milling Company, El Paso, Tex.

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.

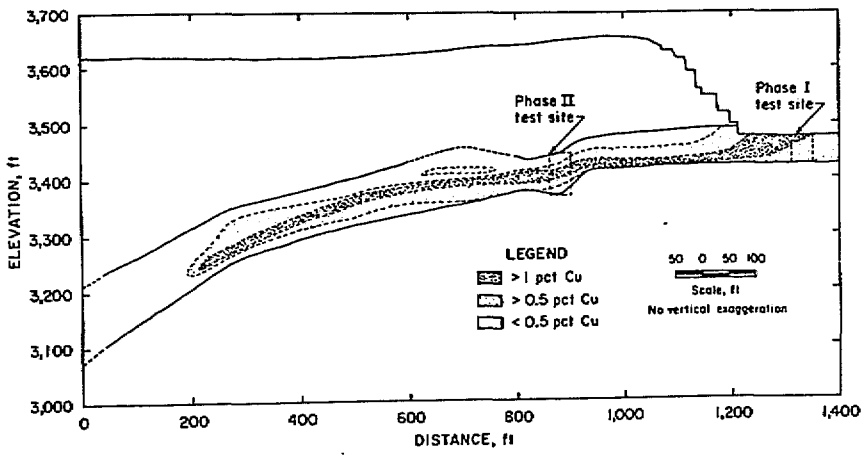


Figure 1. - Cross section through Emerald Isle deposit

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.

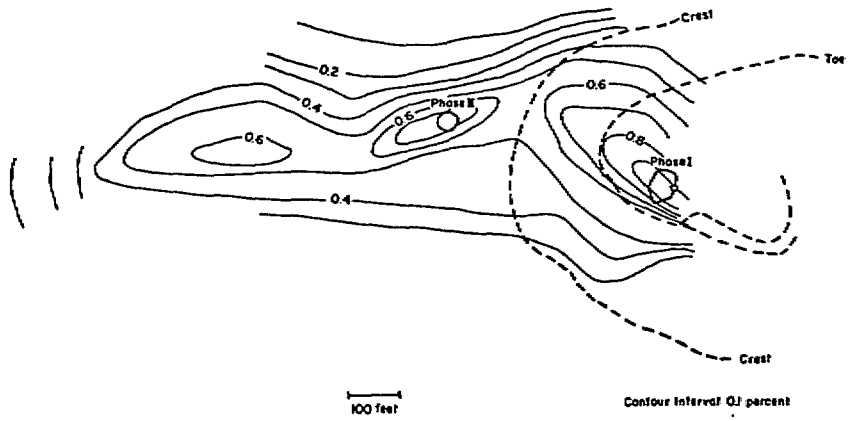


Figure 2. - Contour map of copper grade

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) = $1814 \frac{Pe}{Pr} \left(\frac{D^2}{S^2}\right)$) for ore in powder column zone only

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

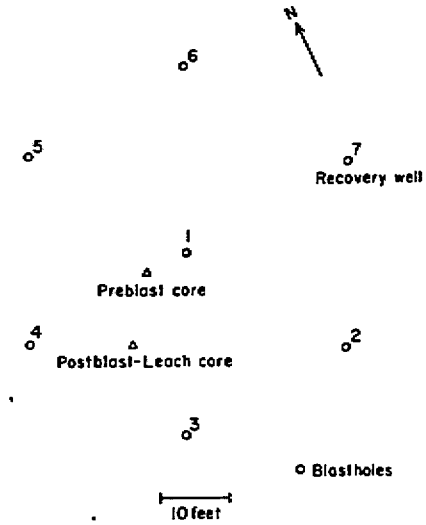


Figure 3. - Phase I test blast design

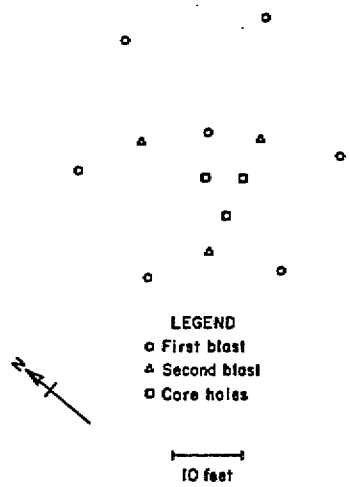


Figure 4. - Phase II test blast design

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

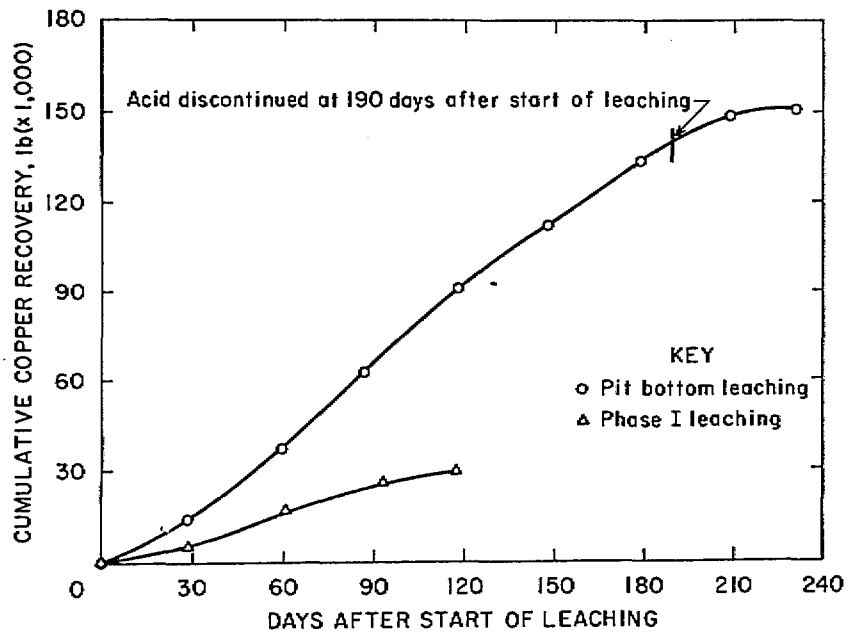


Figure 5. - Cummulative copper recovery - Phase I test and pit bottom leaching

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

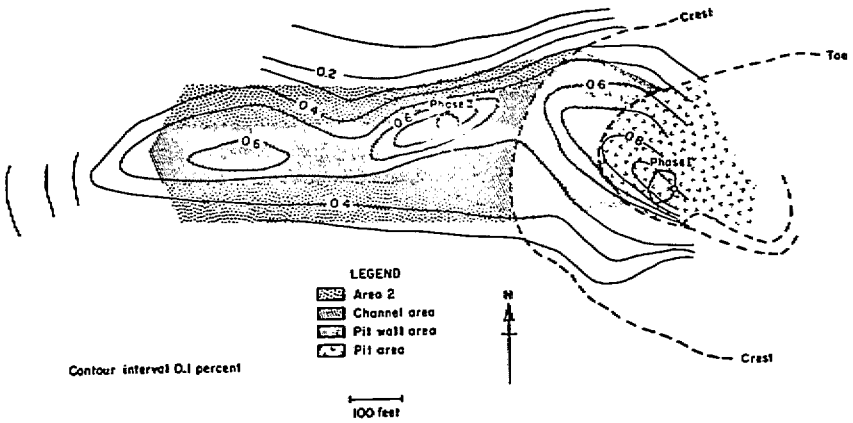


Figure 6. - Blast design for full-scale operation

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

JDS

November 17, 1987

To: J.D. Sell

From: H.G. Kreis

In Situ Conference
Santa Barbara, CA

During the week of October 25th, I attended an Engineering Foundation conference on In Situ Recovery of Minerals. A list of those who attended the conference and a copy of the program are attached. As you can see from the participant list, about 50 to 55 people attended the conference, and they represented a wide variety of interests. Most of those who attended the conference were either employees or consultants for the mining industry. About 15-20% of the attendees were employees of the U.S. Bureau of Mines or a contractor or sub-contractor to the U.S. Bureau of Mines. In addition there were several university professors who are involved in in situ research projects.

Nearly all the talks were directly or indirectly related to copper and uranium in situ leaching. In situ leaching of gold mineralization was rarely mentioned, but it was obvious that several of the attendees were there to evaluate the feasibility of in situ leaching of gold.

Most all of the data in the various talks was at least 5 to 10 years old with the exception of the recent U.S. Bureau of Mines work on Santa Cruz and Lakeshore. Aside from the recent U.S. Bureau of Mines work, the talks represented the state-of-the-art of in situ mining when the severely depressed uranium and copper prices put an end to in situ research and development.

The conference was in part a forum for the U.S. Bureau of Mines to present its work on Santa Cruz and Lakeshore, and to take a leadership role in the field of developing in situ leaching.

Of all the technical information presented at the conference the only information of direct consequence to the Santa Cruz Project was that information provided by the U.S. Bureau of Mines and its sub-contractors Don Davidson, Ray Huff, and John Edwards. Of interest to the Santa Cruz Project were several comments made by Tom Belk of the E.P.A. Mr. Belk stated that in situ leaching involves Class V type wells, and during the last three years the E.P.A. had each state evaluate the effect of Class V wells on the environment within the boundaries of their state. From the information provided by the states, the E.P.A. now has 30 categories of Class V wells and is in the process of developing appropriate regulations based on these various categories of wells. It will be another two to four years before the E.P.A. has a final set of regulations, and it is

expected that regulations for Class V wells will vary from very simple to very detailed depending on the category of the Class V well.

Ray Huff gave a paper on the results of Occidental Minerals in situ leach test work on the Van Dyke Copper Deposit located in Miami, Arizona. I have permission from Ray Huff to use that information along with additional information he gave me over the phone. I will be reporting all this information in the near future.

A number of the talks from the in situ conference are going to be published by the Engineering Foundation in the spring of 1988.

H. G. Kreis

H. G. Kreis

HGK:mek
Att.

cc: R.L. Brown/F.T. Graybeal
W.L. Kurtz

ENGINEERING FOUNDATION CONFERENCES

345 East 47th Street, New York, New York 10017 • Telex: 126022 • Telephone: (212) 705-7835

IN SITU RECOVERY OF MINERALS CONFERENCE
OCTOBER 25-30, 1987

P R O G R A M

SUNDAY, OCTOBER 25, 1987

3:00 PM - 9:00 PM	REGISTRATION AND CHECK-IN
6:00 PM	DINNER
9:00 PM - 11:00 PM	SOCIAL HOUR

MONDAY, OCTOBER 26, 1987

7:30 AM	BREAKFAST
9:00 AM	OPENING AND WELCOMING Conference Chairman: Ken R. Coyne, Bechtel, Inc.
9:15 AM - 12:00 Noon	KEYNOTE ADDRESSES Chairman: J. Brent Hiskey, University of Arizona "The Leaching of Dilute Solid Phases" Milton E. Wadsworth, University of Utah "Future Direction for the Domestic Mining Industry" David S. Brown, USBM "Industry Perspective of In Situ Leaching" Richard L. Brown, ASARCO Panel Discussion Panelists: Keynote Speakers Fred W. DeVries W. C. (Bill) Larson Ray V. Huff
12:00 Noon	LUNCH
2:00 PM - 5:00 PM	AD HOC SESSION
5:00 PM	SOCIAL HOUR
6:00 PM	DINNER

MONDAY, OCTOBER 26, 1987 (Continued)

7:00 PM - 10:00 PM	ENGINEERING DESIGN, EQUIPMENT AND ECONOMICS Chairman: Ray V. Huff, Ray V. Huff & Associates, Inc. "Wellfield and Well Design" Steven G. Axen, Ray V. Huff & Associates, Inc. "Metal Recovery Processes for In Situ Fluids" John Edwards, Davy McKee Corp. "Economics of In Situ Leaching" Donald Davidson, Science Applications International Corporation
10:00 PM - 11:00 PM	SOCIAL HOUR

TUESDAY, OCTOBER 27, 1987

7:30 AM	BREAKFAST
9:00 AM - 12:00 Noon	CHARACTERIZATION, PERMEABILITY MEASUREMENTS AND FLOW ANALYSIS IN FRACTURED HETEROGENEOUS OREBODIES Chairman: W. C. (Bill) Larson, USBM "Measurement of Absolute and Relative Fracture Permeability" Chet R. McKee, In Situ, Inc. "Effect of Sweep Efficiency on Mineral Recovery in a Heterogeneous Orebody" Larry Lake, University of Texas-Austin "Analysis of Discrete Fracture Flow in a Phreatic Boundary Problem Using the Analytic Element Method" Robert D. Schmidt, USBM "Geologic Characterization of Oxide Copper Ores for In Situ Mining" Sterling S. Cook, USBM "Role of Rock Mechanics and Geologic Discontinuities in Hydraulic Fracture Design and Growth" Carol Tosaya, Bechtel Civil, Inc. Panel Discussion Panelists: Session Chairman and Speakers

TUESDAY, OCTOBER 26, 1987 (Continued)

12:00 Noon	LUNCH
2:00 PM - 5:00 PM	AD HOC SESSION
5:00 PM	SOCIAL HOUR
6:00 PM	DINNER
7:00 PM - 10:00 PM	MINERALOGICAL AND CHEMICAL CHARACTERISTICS Chairman: Donald Davidson, Science Applications International Corp. "Mineralogy and Petrography" Won Park, Dowell Schlumberger Inc. "Application of Alternative Lixiviants in the Leaching of Atacamite" J. Brent Hiskey and R. K. Bazaj, University of Arizona "Core Leaching Experiments to Determine Solution-Rock Interactions During In Situ Mining of Oxide Copper Ores" Steven E. Paulson, USBM Panel Discussion Panelists: Session Chairman and Speakers
10:00 PM - 11:00 PM	SOCIAL HOUR

WEDNESDAY, OCTOBER 28, 1987

7:30 AM	BREAKFAST
9:00 AM - 12:00 Noon	DRILL HOLE AND BORE HOLE SLURRY RECOVERY TECHNIQUES Chairman: D'Arcy A. Shock, Consultant "Basic Well Completion Methods" Rudy H. Jacobson, Consultant "Propellant Stimulation of Shallow Injection Wells" Steve Trost, Hi Tech Natural Resources, Inc. "Introduction to USBM Computer Programs" John Ahlness/Robert Schmidt, USBM Panel Discussion Panelists: Session Chairman and Speakers

WEDNESDAY, OCTOBER 28, 1987 (Continued)

12:00 Noon	LUNCH
2:00 PM - 5:00 PM	AD HOC SESSION
5:00 PM	SOCIAL HOUR
6:00 PM	DINNER
7:00 PM - 10:00 PM	CASE HISTORIES OF DRILL HOLE AND BORE HOLE SLURRY TECHNIQUES Chairman: Rudy H. Jacobson, Consultant "Case History: Van Dyke ISL Copper Project" Ray V. Huff, Ray V. Huff Associates, Inc. "Borehole Slurry Extraction of Phosphate" M. F. (Mike) Dibble, Consultant "Uranium Solution Mining - IEC Corporations Operating Experience" Velu Annamalai, IEC Corporation "Uranium Solution Mining with Ammonia" Jerry T. Laman, In-Situ Inc. Panel Discussion Panelists: Session Chairman and Speakers
10:00 PM	SOCIAL HOUR

THURSDAY, OCTOBER 29, 1987

7:30 AM	BREAKFAST
9:00 AM - 12:00 Noon	RUBBLIZATION TECHNIQUES Chairman: Peter G. Chamberlain, USBM "Rubblization of Shallow Deposits Using Vertical Blast Holes" Richard Dick, USBM "Permeability After Confined Blasts In Flooded Shale At Two Levels Of Tectonic Stress" D. Linn Coursen, E.I. Dupont de Nemours & Co.

THURSDAY, OCTOBER 29, 1987 (Continued)

"In Situ Leaching: A Blasting Challenge"
Darrell D. Porter, Science Applications International Corp.

"Large-Scale Underground Expansion Blasting for In Situ Recovery of Minerals
Dr. Thomas E. Ricketts, Science Applications International Corp.
} Occidental's Oil Shale Project

"Energy Considerations in Fragmenting Rock for In Situ Leaching"
Peter G. Chamberlain, USBM

12:00 Noon — LUNCH

2:00 PM - 5:00 PM — AD HOC SESSION

5:00 PM — SOCIAL HOUR

6:00 PM — DINNER

7:00 PM - 10:00 PM — CASE HISTORIES OF RUBBLIZATION AND OTHER TECHNICQUES
Chairman: Ed E. Malouf, Consultant

"In Place Solution Mining of Remnant High Grade Ore Deposits Inaccessible to Conventional Mining"
Geoffrey G. Hunkin, Hunkin Engineers, Inc.

"Percolation Leaching of Rubblized Materials"
Robert W. Bartlett, University of Idaho

"Case History of Old Reliable, Zonia and Big Mike Rubblization Projects for the Leaching of Copper"
Ed E. Malouf, Consultant

"Solution Mining of Water Soluble Salts"
D'Arcy A. Shock, Consultant

Panel Discussion
Panelists: Session Chairman and Speakers

10:00 PM — SOCIAL HOUR

FRIDAY, OCTOBER 30, 1987

7:30 AM	BREAKFAST
9:00 AM - 12:00 Noon	ENVIRONMENTAL ISSUES Chairman: Rick W. Lawrence, Coastech Research Inc. "In-Situ Leaching Control" Tom Belk, U.S. Environmental Protection Agency "Environmental Management of In-Situ Mining Sites" Corale L. Brierley, Advanced Mineral Technologies, Inc. "Natural and Man-Aided Attenuation of Contaminants at In-Situ and Hazardous Waste Sites" Jim V. Rouse, Geochemical Engineering, Inc. Panel Discussion Panelists: Session Speakers Rudy H. Jacobsen Clement K. Chase Fred W. DeVries
12:00 Noon	LUNCH
	ADJOURNMENT

Engineering Foundation Conferences

IN SITU RECOVERY OF MINERALS
Sheraton Hotel and Spa
Santa Barbara, California
October 25-30, 1987

PARTICIPANTS LIST

Ahlness, Jon K., Mining Engineer, U.S. Bureau of Mines
5629 Minnehaha Ave., S., Minneapolis, MN 55417

Annamalai, Velu, Prod. & Restoration Mgr, IEC Corporation
Route 1, Box 662, Three Rivers, TX 78071

Axen, Steven, Vice President, Ray V. Huff & Associates
P.O. Box 1310, Golden, CO 80402

Belk, Tom, U.S. EPA
Underground Injection, 401 M Street, S.W., Washington, DC 20460

Brierley, Corale L., Vice President, Advanced Mineral
Technologies, Inc., 5920 McIntyre Street, Golden, CO 80403

Brown, David S., Deputy Director, U.S. Bureau of Mines
2401 E. Street N.W., Washington, DC 20241

Chamberlain, Peter G., Staff Engineer, U.S. Bureau of Mines
2401 E Street, Mail Stop 6020, Washington, DC 20241

Chamberlin, Paul D., Chamberlin & Associates
7463 West Otero Place, Littleton, CO 80123

Chase, Clement K., Metallurgical Consultant, KD Engineering Co., Inc.
1844 W. Grant Rd.,RM 106, Tucson, AZ 85745

Cook, Sterling S., Geologist, U.S. Bureau of Mines
Twin Cities Research Ctr., 5629 Minnehaah Av S., Minneapolis, MN 55417

Coursen, D. Linn, Research Fellow, E.I. Dupont de Nemours & Co
Explosive Products Div., Brandywine Bldg., Rm B1222, Wilmington, DE 19898

Coyne, Kenneth R., Vice President, Bechtel, Inc.
P.O. Box 3965, San Francisco, CA 94119

Dandrea, Dennis V., Research Supervisor, U.S. Bureau of Mines
5629 Minnehaha, Ave. S., Minneapolis, MN 55417

Davidson, Donald H., Mgr., Tech. Dev., Science Application
International Corp., 8400 W. Park Drive, McLean, VA 22102

THIS LIST IS THE PROPERTY OF THE ENGINEERING FOUNDATION AND MAY NOT BE COPIED, DUPLICATED, OR USED FOR DISTRIBUTION OF ANY MATERIAL TO THE PERSONS WHOSE NAMES ARE CONTAINED HEREIN WITHOUT WRITTEN PERMISSION OF THE FOUNDATION.

De Vries, Frederick W., Senior Consultant, DuPont Co.
C4P Chesnut Run 709, Wilmington, DE 19898

Dibble, Merton F., Consulting Engineer,
5908 Oakmont Lane, Lakeland, FL 33813

Dick, Richard A., Senior Staff Engrg, U.S. Bureau of Mines
5629 Minnehaha Ave., S., Twin Cities Research Ctr, Minneapolis, MN 55417

Disbrow, David Alan, Mine Engineer, FMC Gold
1801 California Rm 2720, Denver, CO 80202

Edwards, John F., Senior Process Engineer, Davy McKee Corp.
2302 Camino Ramon, P.O. Box 5500, San Ramon, CA 94598

Fisher, Gordon, Engineering Foundation
, NY 10017

Ghassemi, Ahmad, Research Assistant, University of Minnesota
Civil &Mineral Engrg Dept, 501 University Ave SE #2, Minneapolis, MN 55414

Hickson, Robin J., Vice President, Freeport Mining Company
P.O. Box 61520, New Orleans, LA 70161

Hildebrand, Stephen B., Project Engineer, Magma Copper Company
P.O. Box M, San Manuel, AZ 85631

Hiskey, J. Brent, University of Arizona
Metallurgical Engrg. Dept, Tucson, AZ 85721

Huff, Ray V., President, Ray V. Huff & Associates
P.O. Box 1310, Golden, CO 80402

Hunkin, Geoffrey G., President, Hunkin Engineers, Inc.
4061 South Eliot St, Englewood, CO 80110

Jacobson, Rudy, Consultant,
2502 Overland Road, Laramie, WY 82070

Johnson, Allan M., Michigan Technological Univ
Benedict Laboratory, P.O. Box 114, Houghton, MI 49931-0114

Johnson, Daniel L., Michigan Tech. Univ.
P.O. Box 114, Houghton, MI 49931-0114

Johnson, Guy A., Staff Engineer, U.S. Bureau of Mines
Denver Research Center, Bldg. 20, Den Fed. Ctr, Denver, CO 80225

THIS LIST IS THE PROPERTY OF THE ENGINEERING FOUNDATION AND MAY NOT BE COPIED, DUPLICATED, OR USED FOR DISTRIBUTION OF ANY MATERIAL TO THE PERSONS WHOSE NAMES ARE CONTAINED HEREIN WITHOUT WRITTEN PERMISSION OF THE FOUNDATION.

Kline, John T., Plant Superintendent, Cyprus Cash Grande Corp.
P.O. Box C-6, Casa Grande, AZ 85222

Kreis, Henry G., Exploration Geologist, ASARCO SWUS
P.O. Box 5747, Tucson, AZ 85703

Lake, Larry, Professor, University of Texas
Petroleum Engrg. Dept., Austin, TX 78712

Larson, William C., Geologist, U.S. Bureau of Mines
Twin Cities Research Ctr., 5629 Minnehaha Ave., S., Minneapolis, MN 55417

Lorenzen, Harold, Billiton Research Arnhem
P.O. Box 40, Westervoortsedyk 67d, 6800 AA Arnhem, NETHERLANDS

Mc Ardle, Donna Marie, Engineering Foundation
345 East 47th Street, New York, NY 10017

Niklason, Michael L., Sr. Region Engineer, Union Cardibe Linde Div
2420 Camino Ramon, San Ramon, CA 94583

Park, Won, Dowell Schlumberger Corp.
P.O. Box 2710, Tulsa, OK 74102

Paul, Raj, Senior Mine Engr Spec, Esso Resources Cananada
234 4th Ave. S.W., Rm 1157, Calgary, Alberta T2P 0H6, CANADA

Paulson, Steven E., Geologist/Geochemist, U.S. Bureau of Mines
Twin Cities Research Ctr, 5629 Minnehaha Ave., S., Minneapolis, MN 55417

Porter, Darrell D., Sr. Scientist, SAI
5438 S. Iola Way, Englewood, CO 80111

Press, David C., Senior Mining Engrg, Rio Algom Ltd.
120 Adelaide St. W., Toronto, Ontario M5H 1W5, CANADA

Ricketts, Thomas E., Senior Scientist, Science Applications Int'l
3351 S. Highland Drive, Las Vegas, NV 89109

Rouse, Jim V., Aware Incorporated
P.O. Box 26572, Lakewood, CO 80226

Schmidt, Robert D., Hydrologist, U.S. Bureau of Mines
5629 Minnehaha Ave S., RM 207, Minneapolis, MN 55417

Sharples, Eric Paul, Project Manager, Cencor General Mining
Union Corp. Ltd, PO Box 61820,74-79 Marshall, Marshalltown 2107, SOUTH AFRICA

Shock, D'Arcy A., Consultant,
233 Virginia, Ponca City, OK 74601

THIS LIST IS THE PROPERTY OF THE ENGINEERING FOUNDATION AND MAY NOT BE COPIED, DUPLICATED, OR USED FOR DISTRIBUTION OF ANY MATERIAL TO THE PERSONS WHOSE NAMES ARE CONTAINED HEREIN WITHOUT WRITTEN PERMISSION OF THE FOUNDATION.

Simons, Stephanie, Metallurgical Engineer, BHP-Utah International
550 California St., San Francisco, CA 94104

Tosaya, Carol A., Director, Laboratory, Bechtel, Inc.
P.O. Box 3965, 45/31/C-1, San Francisco, CA 94119

Trost, Steven A., High Tech Natural Resources
7138 S. Highland Drive, Salt Lake City, UT 84121

Tweeton, Daryl R., Research Physicist, U.S. Bureau of Mines
5629 Minnehaha Ave., S., Twin Cities Research Center, Minneapolis, MN 55417

Van Aswegen, Pieter C., Consulting Metallurgist, General Mining Union Corp.
P.O. Box 61820, Marshalltown 2107, SOUTH AFRICA

Wadsworth, Milton E., Dean, University of Utah
College of Mines, 209 Browning Bldg, Salt Lake City, UT 84112-1183

Wahl, Shirley J., New Mont Mining Corp.
200 W. Desert Sky Rd, Tucson, AZ 85704

Woodward, John R., Director Env. Compl, NERCO Minerals Company
8100 NE Parkway Dr., Vancouver, WA 98662

Yamaguchi, Takao, Consultant Geologist, Nihon Erdart, Company
901-62, Ohkawa, Higashi-Izu-Machi, Kamo-gun, Shizuoka-ken 413-03, JAPAN

GUESTS
Mrs. Mary S. De Vries
Mrs. Marie A. Dibble
Mrs. Margot A. E. Hunkin
Mrs. Mary M. Wadsworth

THIS LIST IS THE PROPERTY OF THE ENGINEERING FOUNDATION AND MAY NOT BE COPIED, DUPLICATED, OR USED FOR DISTRIBUTION OF ANY MATERIAL TO THE PERSONS WHOSE NAMES ARE CONTAINED HEREIN WITHOUT WRITTEN PERMISSION OF THE FOUNDATION.

ASARCO

Southwestern Exploration Division

JDS

November 17, 1987

To: J.D. Sell

From: H.G. Kreis

In Situ Leaching
Van Dyke Deposit
Gila County, AZ

In late October Ray Huff of Ray Huff Associates Inc. gave a talk on Occidental Minerals in situ leach test work on the Van Dyke Deposit located at Miami, Arizona. A copy of the talk is attached.

Using injection wells and recovery wells Occidental Minerals was able to recover 100,000 lbs. of copper in solution from the Van Dyke Deposit. A diluted solution of H_2SO_4 (2-4%) was used to dissolve the copper and the head grades averaged 2½ grams per liter of copper, and they ranged from 2-3 grams per liter. Approximately 4-5 lbs. of sulphuric acid were consumed for each pound of copper recovered. Towards the end of the test work the acid consumption went up to 8 pounds of sulphuric acid per pound of copper recovered. Fifty per cent of the injected fluids were captured by the recovery wells using an air lift. It was felt that the recovery of the solutions could have been increased by having a greater draw-down in the recovery wells and more recovery wells. The permeability of the Van Dyke Deposit in the test area was determined to be two millidarcys before the ground was hydrofractured.

Occidental's in situ leach test on the Van Dyke Deposit involved using a process of hydrofracting the bedrock. Both the recovery and injection wells were each hydrofracted with one horizontal fracture and the fractures were sanded to hold them open. The injected solutions were thought to move horizontally along the hydrofracture and then vertically through the rock to a point where the horizontal fracture from the recovery well was able to capture the solutions. It was felt that the hydrofracting process was a success.

H. G. Kreis

H. G. Kreis

HGK:mek
Att.

cc: R.L. Brown/F.T. Graybeal
W.L. Kurtz

CASE HISTORY: VAN DYKE ISL COPPER PROJECT (1)

ISL

Abstract: An in-situ leaching technology was developed for copper oxide minerals at the Van Dyke deposit at Miami, Arizona. This paper discusses the process developed and the activities that occurred. Specifically, the activities of well installation, deposit data acquisition, leach testing and equipment development are reviewed. Installation and operation of the test facility are described.

11/13/87

Hank:

Here is a copy of the paper you requested.

Regards,

Ray *

RECEIVED
NOV 18 1987
EXPLORATION DEPARTMENT

* Ray Huff

(1) Ray V. Huff, Steven G. Axen and David Baughman
Ray V. Huff & Associates, Inc., Golden CO

Introduction

In January, 1976, a mining company embarked on the development of an ISL process for copper oxides at the Van Dyke deposit. In Phase I, the technical feasibility of the process was established. Phase II of the development was initiated in October, 1977, and established the potential economic viability of the process. Phase III, which would have been operated in a fashion similar to a commercial operation, was designed to demonstrate the commercial application of the process.

Process Review

An ISL facility is composed of five segments. They are:

1. Solution preparation plant
2. Injection equipment
3. Wellfield
4. Production equipment
5. Copper extraction plant

Solution preparation facilities and the copper extraction plant with a few exceptions are little different than that used for conventional leaching operations. Copper extraction can be accomplished with SX-EW. However, certain minerals that can precipitated from the leaching fluid may be deleterious to the process. These can be controlled by addition of chemicals at the solution preparation plant and special circuits in the extraction plant.

The heart of the process is associated with the injection of the solvent into the deposit, dissolution of the desired metal values, and production of the metal-laden liquor to the surface. With suitable information about the geological properties of the deposit, the wellfield and the solvent can be designed and selected, respectively.

Because copper is soluble in either acidic or ammoniacal aqueous fluids, a choice between the two solvents may be made. This choice is dictated by the consumption of the active agents in the solvent by gangue. If the gangue is a high acid consumer, an ammonia solution may be chosen. On the other hand, ion exchange minerals may tie up the ammonium or cupric ammonium ion and render the ammoniacal fluid less suitable.

The process utilized on the Van Dyke deposit can be outlined as follows:

* A dilute (2-4%) H_2SO_4 aqueous lixiviant was prepared.

* Solvent was pumped at high pressure into injection wells.

* Solvent exited injection wells through induced horizontal hydraulic fractures.

* Solvent moved vertically (up or down) through the ore, dissolving oxide copper and entered the induced fracture of the production wells.

* Leachate (copper laden solvent) flowed into the production wells and was lifted to the surface with submersible pumps.

* Leachate was delivered across the fence to a neighboring mine.

In a commercial operation the leachate would be handled as follows:

* Copper would be extracted at the SX-EW plant.

* Raffinate from the SX plant would be reconstituted in the solvent preparation plant and the cycle repeated.

At Miami, Arizona, the company completed exploration drilling on an orebody lying directly under the Town of Miami. The mineralization is one hundred million tons of 0.5% total copper[1]. The copper occurs as chrysocolla, malachite and azurite in fractures in schist and granite porphyry, some cuprite, chalcocite and native copper occur but the majority of the mineralization is fully oxidized or "acid soluble".

The deposit dips from the west to the east, with the top of the orebody 1000 feet (305m) below surface. The host rocks are well fractured, imparting some porosity and permeability to the rock mass. It is thought that the orebody was formed by migrating groundwaters carrying the metal values down-dip from a large copper deposit to the west. The Van Dyke is now covered with a minimum of 600 feet (183m) of Gila Conglomerate, a moderately cemented alluvial fill.

The pilot plant site was located near the northwestern edge of the orebody, to avoid the potential problems of an industrial operation within the town limits.

Surface Operations

At the test site, surface facilities were built to mix and inject dilute sulfuric acid and to recover pregnant liquor from the wellfield. Backup systems and equipment were provided wherever possible to avoid losing data due to routine maintenance and unexpected breakdowns. In operation, the pilot plant operated 24 hours a day for several months.

The nearest available water for the operation was an aquifer in the Bloody Tanks Wash, approximately 1500 feet (458m) from the test site. Two wells were maintained in the wash, providing a backup well in case one should fail. The wells delivered up to 500 gpm (1892 l/min) to a central tank. From this point, a horizontal centrifugal pump transferred the water 1500 feet (458 m) to a tank above the site. Next, water flowed by gravity to the acid mixing facility. The produced well water contained approximately 50 ppm of dissolved copper, and was quite corrosive. Because of this, all components of the water system were corrosion resistant. Fiberglass reinforced plastic (FRP) was used for tanks, stainless steel for pumps, and HDPE (polyethylene) for piping. The water system operated automatically, but could be run manually. Further, each well could be operated independently.

Concentrated sulfuric acid was received by truck and was stored above the operations site in carbon steel tanks. Storage capacity was divided into two tanks of 10,000 gallons ($37.9m^3$) or 60 tons (54,430 kg) each. With both tanks piped and valved independently, maintenance and repairs could be performed as necessary on one tank while the other was in service. A berm surrounded both tanks to prevent potential spills. Desiccant dryers were provided, to lessen the corrosion rate of the steel. The tanks were filled by gravity from trucks, and emptied by gravity through steel piping.

Acid and water were delivered to a weirbox, where they were mixed and the flow rate adjusted. Acid concentration (2 to 4%) was manually controlled. The diluted acid was stored in a 40,000 gallon ($151m^3$) tank for delivery to the pumps. The mixed acid was delivered by gravity through an 8" (20 cm) pipeline and distributed to the pumps through an intake manifold.

Well Installation

Because the wells were to be operated in pseudo-radial flow a screened completion was envisioned. Wells were drilled by rotary rigs using a polymer based mud. An intermediate steel string was set through a clay formation immediately beneath the Gila conglomerate. It was suspected

that this clay formation would swell or heave and prevent subsequent installation of a fiberglass reinforced plastic (FRP) casing or perhaps later cause damage to the FRP casing. (The last hole drilled and completed through the clay formation was cased only with the FRP casing and no problems were experienced with swelling or heaving of the clay.)

Injection wells were equipped using an FRP injection tubing string with a packer attached to the bottom of the tubing. The packer engaged the casing above the injection interval and isolated the annulus between the casing and injection tubing from the lixiviant. The production wells were originally equipped with a production tubing string with an electrical submersible pump attached at the bottom. Because of corrosion problems with pumps available at that time, the production wells were subsequently equipped with air-lift equipment. (Pumps today can be fabricated from suitably corrosion-resistant materials.)

Injection and production wells were stimulated with hydraulic fractures and the fractures were installed in the ore zone through holes made in the casing. These perforations had previously been installed using shaped charges. Suitable shaped charges generate a hole that extends through the casing, the cement behind the casing and into the ore zone.

Fluid Injection and Production

The fluid was pumped into the injection holes with plunger pumps. These pumps were similar to those used for water flooding in the oil industry. The pumps were equipped with relief valves, and each was capable of a maximum of 150 gallons per minute (568 l/min), with a maximum pressure of 1600 psi (11,000 kpa). Several pumps were used to provide excess capacity in case one pump was taken out of service. By changing plunger sizes, or drive sheaves, and by the use of more than one pump, almost any flow rate up to 450 gpm (1703 l/min) could be achieved. Solvent from each pump was delivered to a stainless steel manifold, where it was distributed to individual wells. A bypass was provided to return liquid to the dilute acid storage tank. This bypass piping provided another method for controlling the injection flowrate. From the injection manifold, fluid traveled through meters and then through flexible rubber hoses to the individual wells. These flexible hoses provided a convenient connection to the individual wells.

Two methods had been used for fluid recovery from production holes, those being submersible pumps and gas lift. The submersible pumps had not been designed to adequately handle the pregnant liquor. Pumps lasted, at best, only a few weeks. In addition, the available pumps would only achieve 700 feet (213m) of drawdown on the wells.

During pilot plant operations, because pump downtime was severe, an airlift system was designed to provide a more reliable system. This system consisted of a surface compressor capable of 500 psi (3,400 kpa), surface air piping and control valving on each wellhead. The air flow rate to each hole was controlled by valves and was measured with rotameters. Air and pregnant liquor were separated with the liquid flow measured in weirboxes. Pregnant liquor flowed by gravity to a central sump, and was pumped to FRP holding tanks. In the transfer piping, a flow meter recorded the total production from the wellfield. Pregnant liquor was pumped to a neighboring company.

Throughout the injection and production system, different corrosion problems were studied. Many different metals, plastics, ceramics and elastomers were tested and evaluated. For this system corrosion problems were solved.

Tracer tests were run using chloride to determine residence time and rock volume swept by the fluids. In operation, the pilot plant ran around the clock. Plant operators maintained the equipment, and collected samples and data. After most operating problems had been eliminated, injection and recovery continued with only a few percent downtime, for a period of months.

These are some of the significant observations:

1. Produced 100,000 lb.(45,360 kg) copper

2. Grade of solutions averaged 2.5 gpl

3. Grade of pregnant liquor declined very slowly

For economic forecasts, in order to make an estimate of copper production versus time, pilot plant results were coupled with laboratory core leaching data. Using these data, and heap leaching experience in the district, a copper recovery curve was built, showing copper extraction versus time. This curve was used to project copper production over the life of the orebody with a computer model. The program divided the orebody into leach blocks and assigned copper grades to the blocks. The computer sequentially "mined" the blocks in any given order, producing copper at the desired yearly rate. With the use of this program, economic evaluations for project feasibility could be quickly completed. The computer model was tailored for the Van Dyke orebody, but could be readily adapted to other properties.

Flow Geometry

Radial and axial fluid flow[2,3] can be used in wellfield design. Radial flow is typically used in uranium in-situ leaching operations, and was originally used in design of the Van Dyke site wellfield; it is also used in most oil-

field operations, and is suitable for operations in high permeability rock.

Radial flow occurs when fluid flows into or out of a well along paths following radial lines, originating at the wellbore and extending outward in a horizontal plane. Radial flow in a five-spot well pattern can be described by the variation of Darcy's law, as follows:

$$q = \frac{1.035 \times 10^{-4}\ k\ h\ \Delta p}{u\ [\ln (d/r_w) - 0.629]}$$

Where:

1.035×10^{-4} = Conversion factor
q = Flow rate, gpm
h = Exposed ore interval, (ft.)
Δp = Differential pressure, psi
u = Fluid viscosity, cp
d = Radial distance between injector & producer, (ft.)
r_w = Wellbore radius, (ft.)
k = Permeability, (md)

Axial flow occurs along parallel flow lines, such as flow between two planar surfaces. This type of flow may be effected by creating horizontal hydrofractures at different elevations (h_1 & h_2) in two wells, and applying a pressure difference to cause fluid flow vertically (Figure 1).

This causes axial flow between the fractures as shown. After ore is leached within the interval shown, the recovery well is fractured an equal distance above the injection fracture, the old fractured interval is plugged, and the next vertical interval is leached. This sequence continues until the ore column between the wells has been leached.

Axial fluid flow can be described by the following variation of Darcy's law:

$$q = \frac{3.287 \times 10^{-5}\ k\ A\ \Delta p}{u\ L}$$

Where:

3.287×10^{-5} = Conversion factor
k = Permeability (md.)
A = Area of fracture

overlap (ft^2)

Δp = Pressure differential, (psi)
u = Viscosity (cp)
L = Vertical distance between fractures, ft.

This system was tested at Van Dyke, and appears superior in low permeability rock. Well spacing is determined by hydrofracture radius, which must be optimized during wellfield design, along with the vertical fracture spacing.

It should be realized that while axial flow predominates in this configuration, an element of radial flow simultaneously exists. Radial and axial flow equations are used to determine the most economical operating configuration, indicating required well spacings and/or fracture sizes and spacings, for a given recovery flowrate.

Once flow type and optimum well spacing is determined, the number of injection and production wells can be calculated, indicating the minimum economic wellfield size. Knowing the ore deposit geometry and number of operating wells required, a well pattern can be chosen, and operating plans made.

Pattern Design

Several well patterns are in common use in oilfield water-floods, and uranium leaching operations. The most common of these is the 5-spot pattern, which has 4 injection wells located on the corners of a square and a recovery well in the center. Another common pattern is the inverted 5-spot, which has 4 recovery wells surrounding one injector in the same square geometry. The inverted pattern allows better fluid control in an unconfined system.

A brief description of Van Dyke site wellfield design and operations is included here to illustrate development of the axial flow concept.

The Van Dyke site wellfield was designed as an inverted 5-spot pattern, elongated in one direction (Figure 2). The elongation allowed two different injector-producer spacings to be tested, and was oriented according to suspected anisotropic permeability thought to be related to structural features observed in drill core.

Originally, the Van Dyke wellfield was designed for and operated in a radial flow configuration. Early operations suffered frequent surface and downhole equipment failures, resulting in excessive downtime, and significantly extending or precluding the operating time necessary to obtain useable data.

Analysis of data collected during these early radial flow operations indicated irregular recovery rates and excessive fluid residence time.

To test the effect of well stimulation on radial flow operations, and to evaluate axial flow operation on a limited scale, one recovery and one injection well were hydrofractured, at different elevations. This allowed radial flow between the fractured injector and several surrounding recovery wells, as well as axial flow between the overlapping hydrofractures of the injection well and the stimulated recovery well.

Comparison of these test results with radial flow test results favored axial flow operations, particularly in terms of shorter fluid residence times, higher initial copper loadings and more constant fluid recovery rates.

The Van Dyke test site wells were next recompleted for a full scale axial flow test. This was accomplished by installing hydraulic fractures in the remaining unfractured recovery wells, at an elevation above the existing injection well fracture.

The full scale axial flow test was run for approximately 3 months, during which time equipment reliability and operating time was over 90%. Van Dyke field operations, including the axial flow test, ceased prematurely, as a result of permitting problems. Offsetting the relatively short test run was the higher percentage of operating time, which allowed a good comparison with previous test results; comparison verified that higher initial copper loadings and more constant recovery rates resulted from the axial flow configuration under existing conditions.

Well Testing

Equipment, procedures, and deposit characteristics were all subjects of various tests. This discussion will cover only tests performed to define fluid flow characteristics within the deposit. Injectivity, drawdown and tracer tests, as well as wireline logging, fall into this category.

Injectivity and drawdown tests consist of injecting fluid into or withdrawing fluid from a well at a constant rate, while recording bottomhole pressure and time. These data can be analyzed and used as an indicator of fluid flow capacity near the wellbore, and rock permeability. If two wells are available, rock porosity can be determined by interference (multiple well) testing.

Tracer tests consist of injecting an inert, detectable fluid into a well, and recording its arrival and concentration with time. This can be used to determine plan view flow geometry, residence time, and dilution of injected liquid by deposit liquid.

Wireline temperature, spinner and tracer logs can be used in a similar manner to define fluid flow in profile view,

and find fluid entrance and exit points within the well.

Equipment failures plagued many early Van Dyke well tests, making test results difficult to evaluate. Subsequent tests indicated rock permeability between 1 and 3 md, and injectivity increased as a result of hydraulic fracture stimulation.

Downhole Equipment

Downhole operations utilizing high pressure corrosive fluids over extended periods of time are uncommon, and little specialized equipment or operating experience exists in this area. As a result, many untried techniques and types of equipment were used during early operations and many equipment failures occurred.

Failures of submersible pumps, cables, and instruments at first, then wellheads, casing and other downhole tools later, caused numerous expensive delays in early operations as suitable equipment and techniques were developed.

Failure of FRP well casing and tubing, which was operated well within the factory specifications, caused the most serious delays and required considerable expense, time, and effort on the part of both the company and the manufacturers to resolve.

Several types of corrosion resistant coatings were evaluated on submersible pumps and other downhole equipment. Wellheads, instruments, and many other types of downhole equipment and compatible operating techniques were designed, obtained and evaluated before achieving satisfactory operating reliability. A new type of packer was designed, and is currently being marketed by the manufacturer to the leaching industry as a result of development on the Van Dyke project.

Geological Considerations

The classical methods of geological and assay data gathering for a conventional mining system of a copper deposit are basically the same as when considering an in-situ leaching method. However, there are significant differences in mineralogical detail and deposit characteristics required. The following are some of the characteristics that need to be determined and which are directly related to geology and mineralogy.

The following factors should be considered when evaluating a deposit for ISL.

1) The deposit should not be more amenable to conventional mining methods than to ISL.

2) Geologic structures should not be such that they have adverse effect on fluid flow.

3) Preferably the ore zones should be below the water table. This condition gives fluid flow control.

4) Permeability should be at least 2 millidarcies at 1000 ft. (305m) depth or 0.5 md at 4000 ft. (1220 m).

5) For copper, the deposit should contain at least 100 million tons (91 million metric tons) at 0.5% soluble copper with 50-60% recoverable values.

6) Metal values should be soluble in known solvents.

7) Host rock and gangue minerals should be relatively non-reactive to solvent.

8) Clay content should not exceed 10% depending on type and occurrence.

9) The solvent must have access to the ore minerals.

If an oxide deposit exists, it suggests that sufficient permeability may exist for in-situ leaching. Without permeability, there could not be secondary deposition of minerals. Of course post ore changes to the permeability can exist.

The determination of soluble versus non-soluble copper minerals is important. This requires a choice of assay procedures and assurance that these procedures are conducted properly. Since economic predictions are quite sensitive to recovery rates, the assay data must be sound.

Leaching Model

A computer leach model utilized assay data and a unique equation for copper recovery to estimate copper production and acid consumption over the life of the mine. In addition, the model allowed for the elimination of uneconomic blocks of ore from the mine plan.

With this predictive model, one can design a wellfield which will provide sufficient copper production to reach the required annual rate. To do this calculation once by hand takes an engineer four to ten days. The computer leaching model was built to do these computations. With some modification, the model is suitable for use with any deposit.

Summary

The Van Dyke project was designed, installed and operated in an effort to evaluate ISL of copper oxides. Many problems were encountered in the operation of the facility because of insufficient knowledge about materials of construction, flow systems, and operating procedures. Little prior experience existed with injection of highly corrosive fluids through wells into orebodies. In addition lift systems were inadequate for producing corrosive, copper-bearing solutions from wells 1000ft (305m) deep. Most problems were solved in this project. However a demonstration project would have been required to evaluate the economics for ISL of the Van Dyke deposit.

References

1. _____ : Oxymin Details Plans for In Situ Leach Project in Arizona. Mining Engineering, November 1977.

2. Axen, S.G., Ray V. Huff, and Jerry R. Bergeson, Well Repair for In Situ Leaching. U.S. Patent 4,386,806. June 7, 1983

3. Baughman, D.R., and Jerry R. Bergeson, Method of Repair of Short Circuits for In-Situ Leaching U.S. Patent 4,438,976, March 27, 1984.

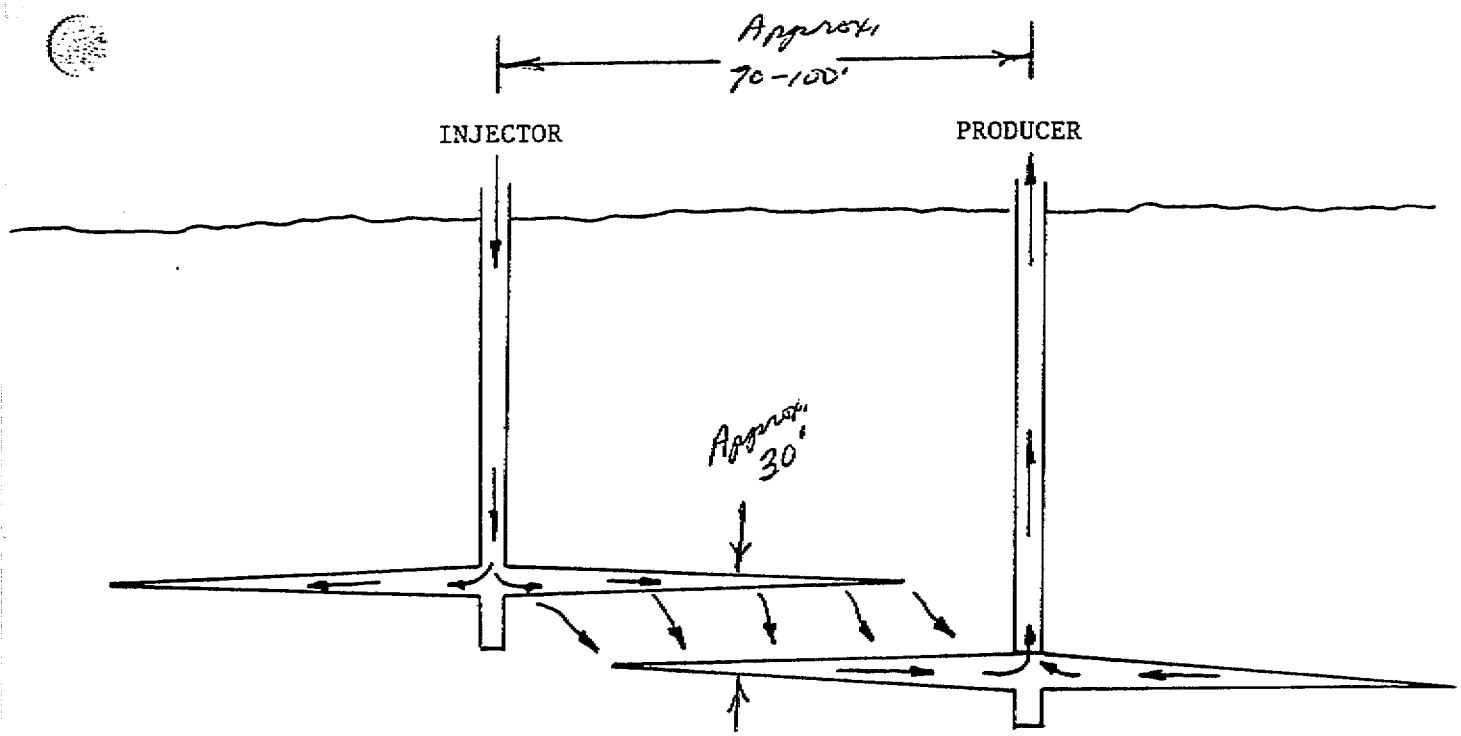


FIGURE 1

AXIAL FLOW BETWEEN FRACTURES

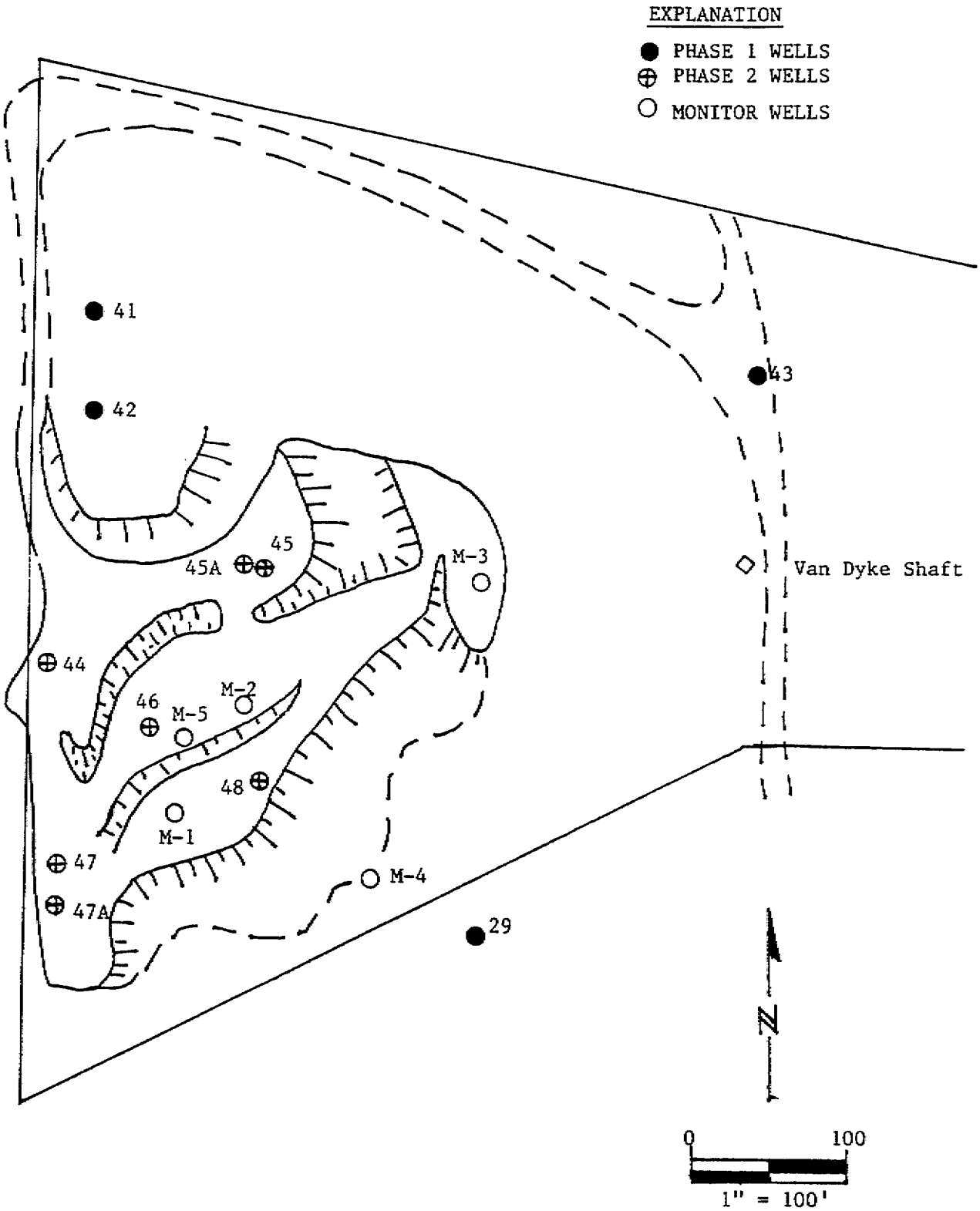


FIGURE 2

VAN DYKE TEST SITE

ASARCO

Southwestern Exploration Division

JDS

November 19, 1987

To: J.D. Sell

From: H.G. Kreis

Occidental Petroleum's
in Situ Technology
Van Dyke Deposit
Gila County, Arizona

This morning I talked with Ray Huff of Ray Huff and Associates about Occidental Petroleum's rights to the information it generated in situ leach testing the Van Dyke Deposit located in Miami, Arizona. Mr. Huff said Occidental sold Codelco a non-exclusive license to use the Occidental technology to develop certain deposits in Chile.

Occidental is said to have applied for, but was not granted, a patent on the horizontal hydrofracturing technique used on the Van Dyke Deposit. When the patent was not granted for this process, Occidental planned to keep the information proprietary. However, Occidental described the horizontal fracturing technique in another patent application, but the applied for patent was not granted. Because the patent was not granted, the information in the patent became public information; and because it included the horizontal hydrofracturing technique, this information is now regarded as public information. Thus, it is no longer proprietary to Occidental. According to Mr. Huff, Occidental still regards all its reports on the Van Dyke in situ leach test as being proprietary information; and, therefore, is not willing to make them public. Occidental has, however, given Ray Huff permission to talk about the results of leach test work on the Van Dyke Deposit.

There may come a time when Asarco wishes to consider using some of the leaching technology that was generated from the Occidental Van Dyke in situ leach test work. At that time it would be appropriate for Asarco to determine what rights, if any, Occidental Petroleum has to that technology.

H. G. Kreis

H. G. Kreis

HGK:mek

cc: R.L. Brown/F.T. Graybeal
W.L. Kurtz

ASARCO

Southwestern Exploration Division

November 17, 1987

To: Files

From: H.G. Kreis

In Situ Leaching
Van Dyke Deposit
Gila County, AZ

Yesterday and today (November 16-17) I talked with Ray Huff about Occidental's in situ leach testing of the Van Dyke Deposit. Ray said that Occidental had given him permission to present the results of Occidental's test work at the October 1987 Engineering Conference on in situ leaching (Santa Barbara, California), and also at the February 1988 AIME meeting.

Ray said he is sending me a copy of the talk he presented in Santa Barbara along with the slides for the talk. He said I could use the contents of the report and the slides for an in-house Asarco exploration talk in December 1987. Furthermore, he said it was alright for me to make duplicate copies of these slides as long as I did not show them publicly before the February 1988 AIME meeting.

H.G. Kreis

HGK:mek

H. G. Kreis

cc: R.L. Brown/F.T. Graybeal
W.L. Kurtz
J.D. Sell

ASARCO

Southwestern Exploration Division

JDS

November 17, 1987

To: J.D. Sell

From: H.G. Kreis

U.S.B.M. Permeability
Test Work, Santa Cruz Project
Pinal County, Arizona

The U.S.B.M. has completed its field test work on the permeability testing of the Santa Cruz Deposit. Permeability tests were run in a wedged-out portion of SC-46, a portion of the old hole SC-19, and a wedged-out portion of SC-19. The field results are currently being evaluated and it will be at least a week or two before some firm conclusions can be made as to the permeability of the Santa Cruz Deposit. Jon Ahlness of the U.S.B.M. has promised to send Asarco the results of the permeability test work as soon as it becomes available to him.

Cursory observations of the field work by those involved with the permeability testing suggest that the recently wedged-out portions of the drill holes have low permeability because of a severe skin effect caused by the drilling mud. The old portion of drill hole SC-19 gave relatively high permeability results of five to twenty millidarcys depending on how one interprets the data. The permeability in the old portion of SC-19 is thought to be high because the drilling muds have had sufficient time to break down.

The U.S.B.M. appears to be encouraged by the permeability test results in SC-46 and SC-19, and they are going ahead with plans for the next permeability test program in the area of SC-46.

H. G. Kreis

HGK:mek

H. G. Kreis

cc: R.L. Brown/F.T. Graybeal
W.L. Kurtz

ASARCO

Southwestern Exploration Division

November 17, 1987

To: S.A. Anzalone

From: H.G. Kreis

Copper Mountain and
Copper Leach Copper Deposits
Burro Mountain District
Grant County, N.M.

In regard to our discussions on open pittable-leachable reserves, the Copper Mountain and Copper Leach deposits, collectively known as Copper Mountain Mines, are worthy of investigation.

Fifteen years ago the property was controlled by USNR, and they were looking for either a joint venture partner or someone to take over the property. At that time USNR had mined out and put on leach dumps a total of 2 million tons at 0.53% copper, and left in the ground were 24 million tons at 0.42% copper. As best I can recall, the reserves remaining in the ground were open pittable with about a 1.5 waste to 1 ore stripping ratio.

It would be worthwhile to see who presently owns the property, to determine how much ore was mined from the ground, and to find out how much copper was actually extracted from the leach dumps.

If I can be of any further help to you in evaluating this property, please let me know.

H. G. Kreis

H. G. Kreis

HGK:mek

cc: W.L. Kurtz
J.D. Sell

ASARCO

Southwestern Exploration Division

November 17, 1987

To: S.A. Anzalone

From: H.G. Kreis

Cerrillos Copper Deposit
Santa Fe County, N.M.

In response to our discussions on small copper oxide deposits I suggest examining the Cerrillos Copper Deposit. It is located in Sections 5 and 8, T14N, R8E, about 30 miles south of Santa Fe, New Mexico. As best I can recall, the Cerrillos Copper Deposit contains 27 million tons of leachable copper mineralization having an average grade of 0.40% copper. About one-third of this tonnage runs greater than 0.40% copper, and it averages 0.6% copper. The copper oxide mineralization is present from the surface to a depth of 180 feet, and it occurs in a zone that is 3000 feet long and 500 feet wide.

In the early to mid 1970's Occidental Minerals Corporation evaluated the Cerrillos Copper Deposit with the intent to blast it and leach it in place. Occidental did a lot of work evaluating these reserves, and it is probable that the land owners have retained this information.

If I can be of any further help to you, please let me know.

H. G. Kreis

H. G. Kreis

HGK:mek

cc: W.L. Kurtz
J.D. Sell

ASARCO

Southwestern Exploration Division

JDS

December 15, 1988

J. D. Sell

Copper Butte Deposit
Ray Mines Area
Pinal County, Arizona

On a recent tour to the Ray Mine, Niel Gambell and Dick Banghart spoke about the Copper Butt Deposit west of Ray Mine. The exotic copper mineralization at Copper Butte consists of two deposits: the Copper Butte Deposit and the Buckeye Deposit which lies about two miles to the west of the Copper Butte Deposit. Both deposits are strata-bound with the copper oxide mineralization occurring in a limestone-rich unit of the White Tail conglomerate. This limestone-rich unit contains about 20% limestone. The copper oxide mineralization is in the form of chrysocolla and neotocite, and published information suggests that a significant amount of the copper may be tied up in the limonite minerals. The Copper Butte Deposit was said to consist of 20 million tons of 1% copper, and the Buckeye Deposit contains a similar tonnage and grade.

The geology of the Copper Butte mineralization is such that the ores will be treated differently than the Ray Mine leach ores. Leach ore from Copper Butte will be crushed to -4" and screened. The limestone goes out in the screen oversize. The undersize will contain most, if not all, of the copper oxide mineralization because it occurs in the matrix of the conglomerate.

The screen undersize material will be agglomerated with sulfuric acid and a binder. The agglomeration will be tried on a conveyor belt, and they may go to a drum agglomeration if necessary. The agglomerated ore will be heap leached similar to the rest of the Ray Mine leach operation.

H. G. Kreis

HGK:mek

H. G. Kreis

cc: F.T. Graybeal
W.L. Kurtz
A.R. Raihl

JDS

ASARCO

Southwestern Exploration Division

December 15, 1988

To: A.R. Raihl

From: H.G. Kreis

Copper Leaching Magma Copper's San Manuel Mine

On October 13, 1988 Steve Bywater (Manager of Mining Research MIM), Kit Clifford (Chief Mining Engineer MIM), Peter Forrestal (Chief Geologist MIM), Ed Malouf (Consultant for MIM) and I visited the leaching operations at Magma Copper's San Manuel Mine. Michael Eamon, SX-EW General Foreman, gave us a tour of the SX-EW plant, the open pit heap leach operation, and the surface of the block cave-in place leach operation.

Block Cave-In Place Leach Operation

The solution distribution system on the top of the block caved ground at San Manuel is very similar to that used at the Miami block cave-in place leach operation. The applied leach solution is mine water acidified with 15 gpl sulfuric acid. The acidified leach solution is applied to the block caved ground through drill holes spaced on 50 ft. centers. The drill holes are estimated to be 600 to 700 ft. deep to get through the base of the Gila conglomerate to the top of the copper oxide mineralization. The holes are drilled by a private contractor at a cost of $10,000 per hole, and they are cased with plastic pipe. Currently there are 180 drill holes actively being used, and there will be more as the area under leach expands. Some of the cased drill holes are being lost because of ground subsidence caused by the adjoining active block cave operation. Currently Magma is putting 100 gpm of leach solution into many of the drill holes, but they plan to operate at 20 to 30 gpm per drill hole. The rate of solution application is presently 0.8 gpm per 100 sq. ft., and they plan to operate at 0.4 gpm per 100 sq. ft. As previously mentioned, the leach solution contains 15 gpl sulfuric acid; but initially they put on 30 gpl sulfuric acid. They felt 30 gpl was too acidic and was damaging the leach column by dissolving minerals at the top of the column and precipitating minerals at the base.

They are presently applying 2,240 gpm of leach solution to the block cave ground, and they are getting a return of 700 gpm. The rate of solution return is increasing by 40 to 50 gallons per month. Magma feels that they have not saturated the ground yet, and it will be some time before the ground is fully saturated. They predict, that once the ground is saturated, they will have an operating loss of 10% of the leach solution. Magma now has a full time hydrologist to look after the movement of leach solutions in the block cave ground.

Published information describes the leach column as being 800 to 1000 ft. thick. A total of 1.5 billion pounds of acid soluble copper are present in 175 million tons grading 0.43% acid soluble copper. Virtually all the copper in the leach column is in the form of copper oxide mineralization with the exception of some sulfide remnants.

The pregnant leach solutions are collected in the underground mine workings and are pumped 2300 ft. back to the surface for treatment in the SX-EW plant. Head grades from the block cave-in place leach operation are presently averaging 1.2 gpl copper. Head grades to the SX-EW plant are maintained by blending the lower grade solutions from the block cave-in place leach with the higher grade solutions from the heap leach operation. Magma predicts the block cave-in place leach will produce head grades of 1.5 to 1.6 gpl once the block cave operation is in a normal operating mode. The pregnant leach solution from the block cave area is running 20 ppm suspended solids which is very high compared to the Miami block cave-in place leach operation. The reason is that the San Manuel operation is an active block cave area, and the activity of the block caving operation produces an abundance of suspended solids.

The block cave-in place leach operation at San Manuel is operating in a hydrologic cone of depression. The cone of depression is caused by pumping 3,000-4,000 gpm of groundwater out of the mine.

Open Pit - Heap Leach Operations

Magma's San Manuel open pit - heap leach operation is described in detail in the paper "San Manuel Oxide Open Pit Project," by Stephen F. Walsh (text of a talk given to the Arizona Conference of AIME December 9, 1985; copy attached to H.G. Kreis memo to J.D. Sell of December 12, 1985). The following information on the open pit - heap leach operation will help to update that paper.

The total of 66,000 ppd are being mined from the open pit. This includes 24,000 tons of ore ranging from 0.5 to 0.6% copper and 0.4% readily soluble copper. The remaining 42,000 tons is waste. The oxide ore is dumped on the leach dumps without crushing or any type of agglomeration. Ed Malouf felt that Magma could significantly increase their copper recovery if they agglomerated their ore before dumping it.

The leach dumps sit on plastic liners having a gravel cover layer on top of the plastic. The dumps are constructed in 15 ft. lifts, and each lift is leached a minimum of 49 days and a maximum of 80 days. Presently there are 70 acres of dump now under leach and this area will expand to a total of 160 acres under leach.

Once the 15 ft. lift has been completed, the ore receives a very effective form of pretreatment. The ore is saturated with a 250 gpl sulfuric acid solution (this accounts for 15 lbs. acid consumed per ton ore), and there is no outflow of leach solution at this time. The ore, saturated with the acid solution, is allowed to rest for 4 days; and then it is saturated again with 250 gpl sulfuric acid (this accounts for 6 lbs. of acid consumed per ton or ore). The resaturated ore is allowed to rest another 3 days. At the end of the 3 day rest period the pretreatment of the ore is considered complete--the rock has been neutralized and much of the copper is in solution. The copper is then leached from the pretreated rock with a solution of 10 gpl sulfuric acid applied at the rate of 0.3 to 0.5 gpm per 100 sq. ft. At this time very little of the acid in solution is consumed by the dump material. The pregnant leach solution coming from the dump runs 8-10 gpl sulfuric acid. Nearly all of the acid consumed by the dump is consumed during the pretreatment of the ore. The total acid consumption is 21 lbs. of acid per ton of rock, and this calculates to 2.3 lbs of acid per pound of copper recovered. As previously mentioned, the ore is leached for 49 to 80 days following the 7 day pretreatment period.

Currently, 190,000 lbs. of copper per day is obtained in solution from the heaps. This is slightly more than 100% of the acid soluble copper being added to the dumps daily. The copper head grades from the dump vary from 1.0 to 2.4 gpl depending upon the gallons per minute applied. The iron content of these solutions varies from 1.3 to 1.4 grams per liter.

The total operating cost of the SX-EW plant was said to be 14¢ per lb. of copper recovered. This includes 6¢ per lb. of copper for reagents and other supplies and 6¢ per lb. for electric power. Open pit mining costs were said to be 25-30¢ per lb. of copper, and their open pit is engineered to cost no more than a total of 55¢ per lb. of copper recovered.

H. G. Kreis

HGK:mek

H. G. Kreis

cc: W.L. Kurtz/J.D. Sell
S.A. Anzalone
D.E. Crowell
F.T. Graybeal

ASARCO

DEC 10 '91

RECEIVED Mining Department
Tucson, Arizona
JAN 2 1992
EXPLORATION DEPT

MINING DEPT.
RJK
RSP
TGC
RLP
FTB

December 5, 1991

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

R. J. K.
DEC 12 19

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

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

OXIDE LEACHING

The leaching of siliceous low-grade copper-bearing ores and waste rock that cannot be processed economically by milling and concentrating is an important low-cost process for recovering copper. It presents a growing factor in the U.S. production of primary copper, amounting to 8% of the total U.S. production in 1963; 12%, in 1968 (175,000 tpy); and an estimated 20% by 1975. This leaching practice may be classified into the following four categories:

1. Vat leaching, which is the percolation of dilute sulfuric acid through crushed and sized oxidized ore that is bedded in rectangular tanks with a filter bottom. Slimes made in crushing may be agglomerated and leached with the coarse material or leached separately by agitation. The leach cycle is one to two weeks in the vats.

2. Heap leaching, which uses dilute sulfuric acid solution for extracting copper from uncrushed porous, oxidized ore that has been piled on prepared drainage pads. The leach cycle duration is months.

3. Dump leaching, which is used to extract copper from low tenor, uncrushed strip waste material that must be moved during mining. Both oxide and sulfide minerals in the waste rock may dissolve during years of exposure to oxidizing solutions of dilute sulfuric acid.

4. In-place (in situ) leaching, which extracts copper from broken low-grade rock in and overlying a now abandoned copper mine or which has been fractured by an underground explosion, or from underground porous strata. As with dump leaching, the cycle duration is in years, and there is thus time for the oxidation and dissolution of sulfide minerals.

The choice of the leaching method thus depends upon the chemical and physical characteristics of the specific material to be treated. The grade of the ore, the solubility of the copper minerals, the amount of acid-consuming associated gangue material, the size of the operation, and the mode of occurrence of the copper-bearing minerals are some of the important factors to be considered.

The principal copper oxide minerals dissolved by leaching are azurite, chrysocolla, and malachite, although varying amounts of cuprite, tenorite, and native copper are also known to be contained in some ores and are simultaneously dissolved. The copper sulfide minerals prevalent in many sulfide ores and dissolved during oxidation leaching are chalcocite, chalcopyrite, and covellite. Pyrite is an important mineral found in many sulfide ores and is oxidized during leaching to form ferrous sulfate and sulfuric acid.

Both dump leaching and in place leaching result in leach effluents whose copper content varies between 0.75 and 2.25 g/L and the pH of which varies between 1.9 and 3.3. While some deposits require no acid addition because of the generation of sulfuric acid as the result of pyrite oxidation, for example, other deposits consume modest quantities of sulfuric acid. ASARCO has a dump leaching operation at Silver Bell.

Because of the higher grade of the heaps, the stronger acid solution which is generally employed, and the ready solubility of oxidized copper minerals, the pregnant solutions tend to be considerably higher than those obtained from dump leaching. The copper content of the pregnant liquor at Ranchers Exploration and Development Corporation varies from 3 to 7 g/L.

Vat leaching is employed to extract copper from oxide or mixed oxide-sulfide ores containing more than 0.5% acid-soluble copper. This method is preferred to heap leaching if the ore is not porous and crushing is necessary to permit adequate contact between leach solution and copper minerals. Higher copper recovery is realized in shorter periods, pregnant liquor losses are negligible, and the copper content of the pregnant liquor is much higher. Thus, for example, ASARCO's San Xavier leach plant will treat about 4000 tpd of ore averaging about 1% Cu to produce 12,000 tons of copper per year while consuming 50,000 tons of sulfuric acid from ASARCO's Hayden plant. The level of copper in the leach solution is on the order of 15 g/L.

Copper Recovery

Copper is recovered from the various leach liquors principally by means of galvanic precipitation by metallic iron. In the case of San Xavier vat leach solution, 15,000 tons of shredded detinned cans are used annually to accomplish this precipitation, while the Hecla Lakeshore flowsheet indicates that sponge iron will serve this purpose.

The most common practice is typified by the use of gravity launders such as those shown in Figure 3. Copper-bearing liquors are introduced into the upper end of gravity launder plants and allowed to trickle downwards through the shredded cans. Launder arrangement is such that solution flow is in series. Copper is washed three times a week from the first few launders, which normally precipitate more than 60% of the total copper. The copper precipitated in the remaining launders is removed usually once a week to once a month.

The amount of iron consumed per pound of copper precipitated is always higher than the theoretical 0.88 pound of iron and, in fact, ranges between 1.2 and 2.5 pounds of iron per pound of copper precipitated. The copper precipitate from this process is typified by the product of the San Xavier Plant and contains 80-82% Cu. It is dried and usually returned to a copper smelter (El Paso in the case of the San Xavier precipitates).

While a number of variations exist in the equipment employed for copper cementation on iron, only the cone-type precipitator developed by the Kennecott Copper Corporation will be mentioned here. A cutaway diagram of the precipitation cone is shown in Figure 4. Copper-bearing solution is injected through nozzles into the bottom of the cone and travels upward through shredded, detinned iron scrap, which is semicontinuously loaded into the inner cone above a heavy-gage stainless steel screen. Copper is rapidly precipitated and quickly removed from the iron surfaces by the turbulent action in the lower part of the cone so that fresh, clean surfaces of iron are created for continued copper precipitation. Cement copper settles through the stainless steel screen which forms the upper cone, settles to the bottom of the outer shell, and is removed intermittently.

It is claimed that iron consumption in the cone-type precipitator is lowered by 0.5-1.0 pound per pound of copper and that the copper content of the precipitate is improved to 90-92%.

In recent years, solvent extraction of copper from low-content pregnant leach solutions, followed by stripping of the copper from the solvent into a sulfuric acid solution which would then be suitable for copper electrowinning, has been adopted commercially both by Bagdad Copper at Bagdad, Arizona and at Ranchers Development and Exploration's Bluebird operation near Miami, Arizona.

The solvent should have the following properties:

1. It should be selective for copper at the natural low pH of the leach solution.

2. It should have a low water solubility.

3. It should be compatible with a cheap diluent, such as kerosine.

4. It should be nontoxic to bacteria beneficial to the leaching process.

5. It should be stable between 0° and 80°C.

6. It should not be detrimental to the electrolytic process.

Such a family of reagents has been developed by General Mills, Inc., first for copper extraction from ammoniacal solutions (LIX-63) as applied, for example, in the Arbiter process, and subsequently for use with low copper, low pH leach solutions (LIX-64) such as those encountered typically in dump or in-place leaching. Recently, a new reagent (LIX-70) has been developed which, it is claimed, will be able to treat more highly concentrated acidic solutions of copper, perhaps as high as 60 g/L Cu.

Since solvent extraction is an equilibrium chemical process whereby, in the case of most reagents, the copper ions in the leach solution are exchanged for hydrogen ions in the solvent, the pH of the leach solution decreases during extraction and eventually limits the process. This is the reason for the continuing search for reagents which are able to handle higher copper concentrations in the pregnant leach solutions, and, consequently, higher acid concentrations as the solutions are depleted in copper. Moreover, since solvent extraction is an equilibrium operation, several stages of extracting and stripping are usually required to achieve the desired degrees of extraction and stripping. These operations serve the two-fold purpose of transferring copper from an impure leach solution to the final aqueous phase which is sufficiently pure for copper electrowinning, and to concentrate the copper from relatively dilute leach solutions to levels suitable for electrowinning.

The various combinations of oxide leaching with either copper cementation or copper electrowinning serve the very important purpose of recovering copper from low-grade ores and waste

rock which could not be economically processed by milling, concentrating, and subsequent extractive metallurgy. As the name indicates, this type of leaching effectively recovers oxide copper, although some oxidative attack may also occur on sulfide mineral constituents when leaching occurs over extended periods. Within this limited scope of applicability, oxide leaching has been applied to ASARCO ore deposits at Silver Bell and San Xavier, but they are not suitable for sulfide ores or concentrates which constitute ASARCO's major source of copper.

The quality of the recovered copper is such, in the case of cementation, that it must be treated for further purification, usually in a copper smelter. As in all other electrowinning processes, there is some doubt that the lead content of the copper obtained in this fashion will equal that of the electrorefined product.

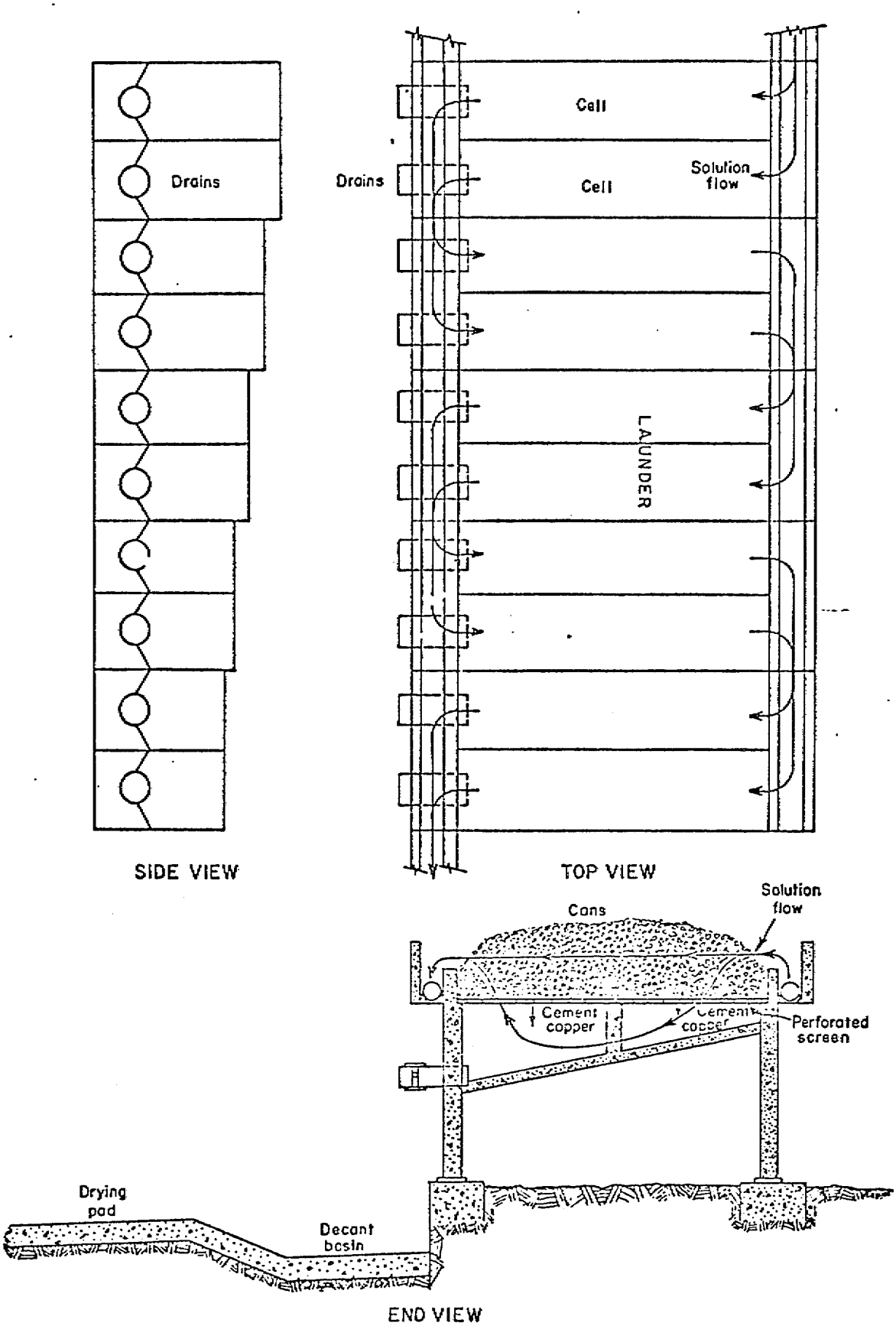


FIGURE 3 Typical Design of Gravity Launder-Precipitation Plant.

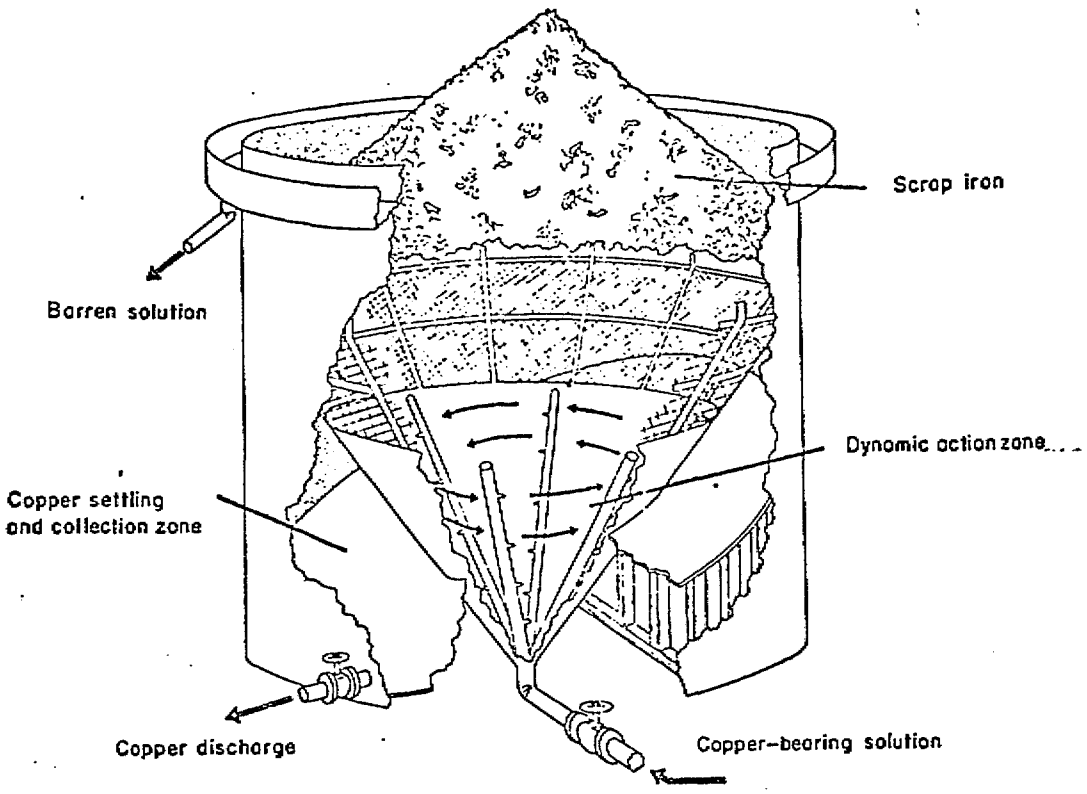


FIGURE 4 Cutaway Diagram of Cone Precipitator Designed by Kennecott Copper Corp. (Neal Bishop, Kennecott Research Center.)

AMERICAN SMELTING AND REFINING COMPANY
Central Research Laboratories
South Plainfield, New Jersey

~~D.E.G.~~ File Copper Meeting
OCT 1973
OCT 12 1973

September 7, 1973

Dr. W. P. Roe
B U I L D I N G

J. H. C.
OCT 17 1973

ANNUAL RESEARCH MEETING

OCTOBER 4, 1973

AGENDA

9:30 A.M.	WELCOME AND INTRODUCTION	W. P. Roe
	Copper Smelting Alternatives	V. Kudryk
9:45 A.M.	NEW SOURCE STANDARDS	J. M. Henderson
	EPA Proposed SO_2 Standards	
10:00 A.M.	SULFUR DIOXIDE ABATEMENT	W. H. Wetherill Bill
	Scrubbing Systems and H_2SO_4 Neutralization	
10:20 A.M.	SULFUR DIOXIDE RECOVERY	A. Bogeatzes
	ASARCO, Allied Chemical, Bureau of Mines, DMA and other processes	
11:35 ~~10:45~~ A.M.	COPPER PYROMETALLURGICAL DEVELOPMENTS	H. R. Larson
	Roast-blast Furnace, Brixlegg, Amax	
12 ~~11:10~~ A.M.	CONTINUOUS COPPER SMELTING	J. R. Wettlaufer
	Noranda, Mitsubishi, Worcra	
1 ~~11:30~~ A.M.	ESTABLISHED PYROMETALLURGICAL SMELTING	J. M. Henderson
	Reverberatory, Electric and Flash Smelting	
12:30 P.M.	LUNCH	
2 ~~1:30~~ P.M.	HYDROMETALLURGICAL DEVELOPMENTS	W. H. Wetherill
	Cymet, DuVal, Treadwell, DuPont, Sherritt-Gordon, Bureau of Mines	
2:00 P.M.	PROPOSED COMMERCIAL PROCESSES	M. L. Hollander
	Anaconda, Hecla, Oxide Leach	
2:15 ~~2:20~~ P.M.	BACTERIA LEACHING	E. Martinez
2:45 ~~2:30~~ P.M.	COMPARISON OF ENERGY REQUIREMENTS	H. R. Larson
~~2:50~~ P.M.	COST EVALUATION	J. E. Fay
~~3:30~~ P.M.	SUMMARY AND DISCUSSIONS	

V. Kudryk

V. Kudryk

VK/lk

MINERAL
OCT 17 1973
BENEFICIATION DEPT

ASARCO

Exploration Department

July 30, 1976

To: W.L. Kurtz, F.T. Graybeal
D.P. Cadwell, J.C. Balla,
J.H.G. Fuchter, D.M. Fletcher

EXOTIC COPPER DEPOSITS
(LITERATURE SEARCH)

Enclosed is Mr. Giesecke's brief comment on an article, "Copper Silicate Deposits of Northern Chile" with a copy of the summary attached. The author concludes that such deposits are restricted to arid or semiarid regions.

La Exotica, containing 150 million tons of 1.6% Cu, is perhaps the largest known occurrence of this type in South America. It is also unique in that two-thirds of the deposit is in bedrock beneath the gravels. Apparently, copper-bearing meteoric waters (derived from erosion-oxidation-leaching of the chalcocite enriched Chuqui ore body) moved down slope (2 kms.) into a basin where copper precipitated as silicate in pervious bedrock as well as in overlying gravels.

The source of the Sagasca (85 kms. east of Iquique) copper is not so readily evident. The nearest known porphyry copper occurrence, Cerro Colorado, is 25 kms. NE ---- and in another drainage system. The Sagasca deposit (12 million tons of 2.5% Cu) occurs in flat-lying conglomerates of probable Late Tertiary age, which have been exposed by erosion. In contrast, La Exotica was totally concealed. Reportedly it was discovered during the course of drilling for water.

Several exotic copper deposits are known in Arizona, such as Emerald Isle (near Ithaca Peak), Copper Butte (near Ray) and Superior East (west of Miami). All were derived from the weathering of porphyry copper deposits and no doubt there are others that remain to be discovered under covered areas. But, the search for them will involve rather high risk exploration in most cases.

Fred Graybeal has estimated (memo of 1-2-75) that the El Tiro-North Silver Bell zones have had some 4 billion tons of .20% Cu removed by erosion and that most of this copper, transported in solution and/or mechanically, could be contained in the north-striking graben lying (down slope) to the west.

Although to date the exotic deposits have not proven profitable producers (except for a few very small operations), they may become so in the future with their advantage over sulphide ores environmentally in conventional treatment. And, being in general relatively permeable, they may be more amenable to in-situ leaching. La Exotica (open pit) encountered metallurgical problems and is currently inoperative.

J. H. Courtright
J.H. Courtright

JHC:vh
cc: TCOsborne w/attach.
RBSprague
JJMerz

ASARCO

Madrid Office - Exploration Department

J. J. Merz
Director of Exploration

A. Giesecke
Chief Geologist

June 30, 1976

Literature search

Title: "Silikatische Kupferlagerstaetten in Nordchile."

Author: Gustav Roethe

Publication: Geologische Rundschau, 1975, Vol.64, N°2. p.421-456

Above article on Copper silicate deposits in northern Chile is of special interest to anybody dealing with secondary Copper deposits (Exotica type).

Three individual prospects are described:

1. Sagasca, Tarapacá Prov.
2. La Exotica, Antofagasta Prov.
3. Venado, Atacama Prov.

Geology, mineralogy (Chrysocolla, Copperpitch), and genesis are discussed. Microfotos, X-ray studies, and chemical composition are enclosed. There are geological maps and sections to show the general situation. Some data on grade and tonnage are given.

The article is in german but carries an english summary. Explorationwise, it is of definite importance to anybody assessing the mineral potential of "post-mineral" cover rocks.

A. Giesecke

x-c: R.B. Sprague
R.L. Brown
J.H. Courtright ✓

Abstract

The formation of isolated ~~siliceous~~ silicate copper deposits is controlled by two main conditions:

1. oversaturation of groundwaters in silica
2. lateral inflow of copper bearing and slightly acid solutions originating from the oxidation of primary sulfide deposits.

The basic copper silicate chrysocolla is formed by chemisorption of copper ions by disilicic acid with a net-like structure. Under favourable conditions copper silicate deposits are formed which contain up to several million tons of copper as for example the La Exotica deposit in Northern Chile.

The copper values of these deposits sometimes exceed those of the primary sulfide deposits. Thus, the oxidation of sulfide deposits, the migration of copper bearing solutions and the chemisorption of copper ions by the disilicic acid can result in workable new copper concentrations originating from sulfide protores.

High concentrations of silica in groundwaters and thus the chemisorption of copper ions by the disilicic acid as an ore forming process is restricted to arid or semiarid regions. Therefore it is proposed to call such deposits arid silicateous infiltration deposits.

Infiltration deposits are related to weathering processes. "They comprise products of weathering by which valuable substances were leached from the host rocks, migrated in groundwaters and were redeposited in other neighbouring rocks" (Smirnov, 1970, p. 324).

BACTERIAL LEACHING OF COPPER CONCENTRATES

A number of years ago investigators discovered the presence of bacteria in copper mine waters that were capable of oxidizing sulfide minerals. Pyrite was oxidizing to ferric sulfate and sulfuric acid and copper sulfide minerals to copper sulfate due to action of bacteria.

The strains of bacteria were identified as Thiobacillus ferrooxidan and Thiobacillus thio-oxidans. Both strains derive the energy for their life cycle from oxidation of iron and sulfur phases. The oxidizing bacteria are capable of living on inorganic substance such as dissolved salts, nitrogenous material, carbon dioxide and oxygen.

The activity of the bacteria is optimum at about 35°C and a pH of 2.0 to 3.5. The rate of growth drops, as well as activity, as the temperature and pH go on either side of the given range. The bacteria are destroyed at >50°C.

Thiobacillus ferrooxidan is a robust organism. Although it is inhibited by low metal ion concentrations, it can be adapted over a period of time to tolerate concentrations as high as 30,000 ppm of Cu, Zn and U. It can also adapt to living in an environment of pH 0.5. Mercury, however, is highly toxic and it was found that the bacteria could not adapt to its presence. Direct sunlight inhibits the activity of bacteria.

Oxidation Reactions

The bacteria aid in the oxidation reactions via a catalytic action which speeds up reactions that are thermodynamically possible but kinetically too slow. There are two forms of reactions - direct and indirect.

In direct oxidation the bacteria must be kept in physical contact with the mineral:

$$CuFeS_2 + 4O_2 \rightarrow CuSO_4 + FeSO_4$$
(chalcopyrite)

The bacteria will then oxidize the $FeSO_4$, accelerating the reaction which would be very slow in their absence:

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

The ferric sulfate formed will leach other copper sulfide minerals as well as chalcopyrite:

$$CuFeS_2 + 2Fe_2(SO_4)_3 \rightarrow CuSO_4 + 5FeSO_4 + 2S$$

$$CuS + Fe_2(SO_4)_3 \rightarrow CuSO_4 + 2FeSO_4 + S$$

The latter reactions are referred to as indirect since the bacteria are not attacking the mineral, but providing the chemical that will. The sulfur will react with water and oxygen to form sulfuric acid in the presence of the bacteria.

Sulfuric acid and ferrous sulfate can be generated by using bacterial action to oxidize pyrite. The ferrous iron in solution is converted to ferric by bacterial oxidation. The acid obtained in the oxidation of pyrite holds the ferric sulfate in solution until it comes into contact with the copper sulfide minerals.

B. C. Research Council Proposed Leaching Process

A preliminary feasibility study on the recovery of metal values from sulfide concentrates by a microbiological leaching procedure was presented by B. C. Research. The proposed procedure was the adaptation of microbiological leaching to a continuous process.

The system selected was a chalcopyrite concentrate containing 28% Cu. As noted above, bacteria will accelerate the direct oxidation of chalcopyrite, as well as oxidize $FeSO_4$ to $Fe_2(SO_4)_3$ which will also attack the mineral. A schematic representation of the leaching process for treating 100 tons per day of concentrate is shown in Figure 1. A material balance is given in Table 1.

The leaching equipment involved a series of 20-ft. square tanks with a 16-ft. pulp depth. The tanks are charged with a -325 or -400 mesh chalcopyrite concentrate, bacteria, H_2SO_4, air, CO_2, and nutrients for the bacteria. Suitable means for aerating and agitation are required.

Leaching proceeds at 35°C (maintained) at a pH of 2.0 to 2.8. The proposed pulp density is 10%. The residence time is based on a leaching rate of 300-350 mg/l. hr. for copper. Based on these conditions, leaching of 100 tpd of concentrate would require 18 tanks (20' x 20' x 16' deep).

The leaching tank effluent goes to a 2-stage gravity separation unit. In the first stage the unreacted mineral products are separated and returned to the leaching tanks (after regrinding if necessary). In the second a sludge consisting of basic ferric sulfate and clays are removed. The pregnant solution, which is the overflow from the second clarifier, would be allowed to build up until it contains 30 gpl of Cu prior to electrowinning.

The advantages of the process are given as:

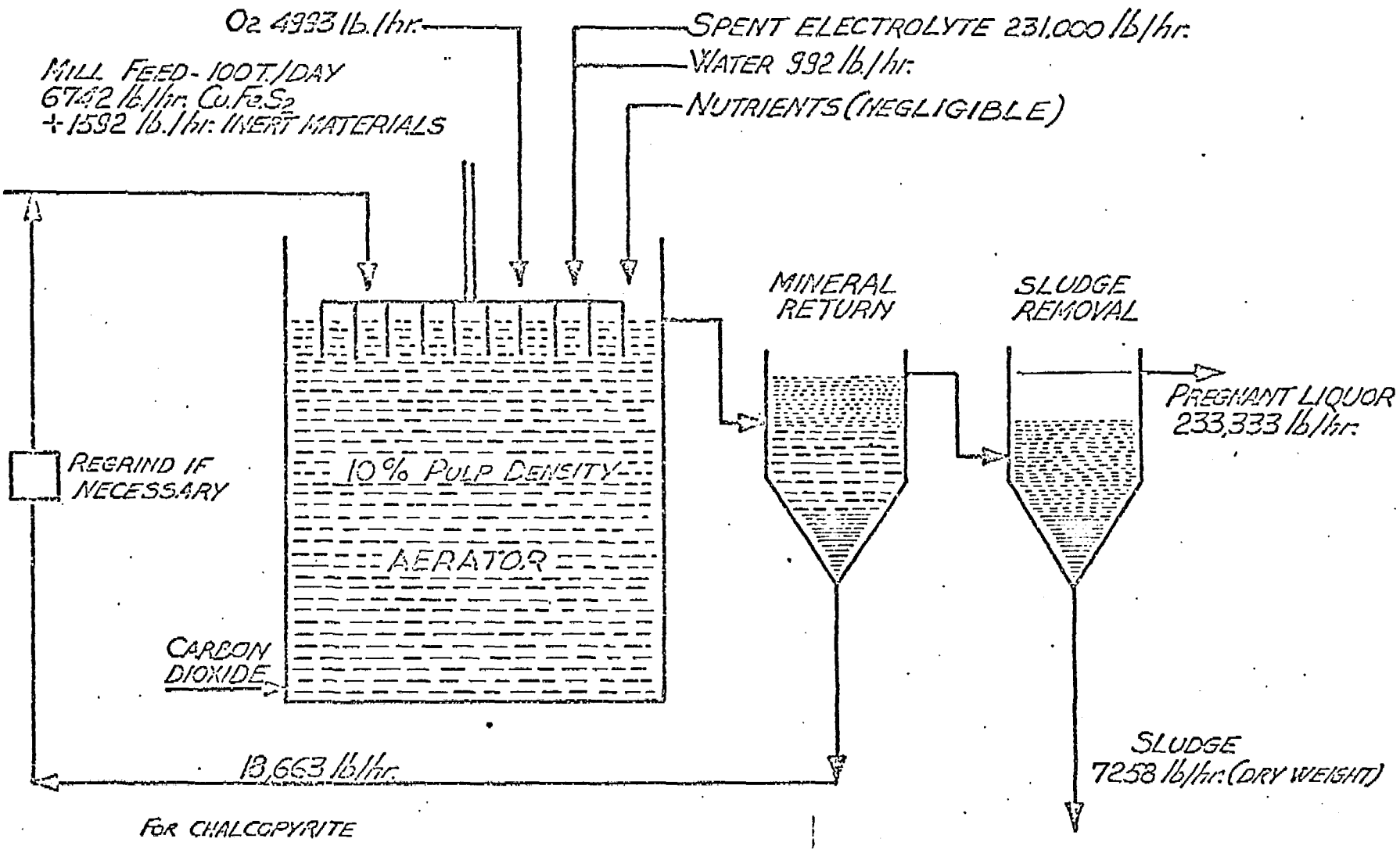
1. Product of mill is upgraded at mill site.
2. Relatively low capital cost compared to a conventional smelter.
3. Process equipment can be designed in modules.
4. Output of small or large mines can be handled.
5. Eliminates SO_2 air pollution.
6. Au, Ag, & Pb may be recovered from sludge.

The disadvantages are:

1. Sulfur by-products not recovered.
2. Formation of dilute H_2SO_4 constitutes a potential water pollution problem.
3. Higher capitalization required than in procedure of selling concentrates to existing processes.

Other difficulties not mentioned in B. C. proposal:

1. Temperature control of tanks.
2. Grinding of concentrates to -400 mesh or finer.
3. Possible build up of deleterious impurities in solutions such as zinc, magnesium, or H_2SO_4 requiring a bleed-off.
4. Separation of sludge (clays and basic ferric sulfate) from unleached mineralized concentrate.
5. Treatment of copper concentrates containing a mixture of chalcopyrite and chalcocite. Chalcocite requires a two stage process.



SCHEMATIC REPRESENTATION OF LEACHING SYSTEM

TABLE 1

MATERIAL BALANCE

Material In					
Mill Feed (28% Cu or 81% chalcopyrite)					8,333.3 lb/hr
$CuFeS_2$			6,741.7		
Inert gangue			1,591.6		
Oxygen	$\frac{25.5 \times 32}{6 \times 183.5}$	x	6,741.7	=	4,993.3 lb/hr
H_2O	$\frac{9 \times 18}{6 \times 183.5}$	x	6,741.7	=	991.9 lb/hr
			TOTAL IN		14,318.5 lb/hr
Material Out					
Copper Sulfate	$\frac{6 \times 159.5}{6 \times 183.5}$	x	6,741.7	=	5,859.9 lb/hr
Sulfuric Acid	$\frac{2 \times 98}{6 \times 183.5}$	x	6,741.7	=	1,204.5 lb/hr
			Soluble out		7,064.4 lb/hr
Basic Ferric Sulfate	$\frac{2 \times 462.7}{6 \times 183.5}$	x	6,741.7	=	5,666.5 lb/hr
Inert Material					1,591.6 lb/hr
			Solids out		7,258.1 lb/hr
			TOTAL OUT		14,322.5 lb/hr

Material balance is based on the following overall reaction proposed by B. C. Research:

$$6CuFeS_2 + 25\ 1/2\ O_2 + 9H_2O \rightarrow 6CuSO_4 + 2HFe_3(SO_4)_2(OH)_6 + 2H_2SO_4$$

AMERICAN SMELTING AND REFINING COMPANY
Central Research Laboratories
South Plainfield, New Jersey

~~DEG~~ / Copper Mills
OCT 12 1973

September 7, 1973

Dr. W. P. Roe
B U I L D I N G

J. H. C.
OCT 17 1973

ANNUAL RESEARCH MEETING

OCTOBER 4, 1973

AGENDA

9:30 A.M.	WELCOME AND INTRODUCTION	W. P. Roe
	Copper Smelting Alternatives	V. Kudryk
9:45 A.M.	NEW SOURCE STANDARDS	J. M. Henderson
	EPA Proposed SO_2 Standards	
10:00 A.M.	SULFUR DIOXIDE ABATEMENT	W. H. Wetherill Bill
	Scrubbing Systems and H_2SO_4 Neutralization	
10:20 A.M.	SULFUR DIOXIDE RECOVERY	A. Bogeatzes
	ASARCO, Allied Chemical, Bureau of Mines, DMA and other processes	
~~10:45~~ 11:35 A.M.	COPPER PYROMETALLURGICAL DEVELOPMENTS	H. R. Larson
	Roast-blast Furnace, Brixlegg, Amax	
~~11:10~~ 12 A.M.	CONTINUOUS COPPER SMELTING	J. R. Wettlaufer
	Noranda, Mitsubishi, Worcra	
~~11:30~~ 1 A.M.	ESTABLISHED PYROMETALLURGICAL SMELTING	J. M. Henderson
	Reverberatory, Electric and Flash Smelting	
12:30 P.M.	LUNCH	
~~1:30~~ 2 P.M.	HYDROMETALLURGICAL DEVELOPMENTS	W. H. Wetherill
	Cymet, DuVal, Treadwell, DuPont, Sherritt-Gordon, Bureau of Mines	
2:00 P.M.	PROPOSED COMMERCIAL PROCESSES	M. L. Hollander
	Anaconda, Hecla, Oxide Leach	
~~2:20~~ 2:15 P.M.	BACTERIA LEACHING	E. Martinez
~~2:30~~ 2:45 P.M.	COMPARISON OF ENERGY REQUIREMENTS	H. R. Larson
~~2:50~~ P.M.	COST EVALUATION	J. E. Fay
~~3:30~~ P.M.	SUMMARY AND DISCUSSIONS	

V. Kudryk

V. Kudryk

VK/lk

MINERAL
OCT 17 1973

file : copper

THE FORMATION OF OXIDIZED COPPER MINERALS IN DISSEMINATED COPPER DEPOSITS

J. H. C.

DEC 13 1965

J. J. Durek

Chemical Environment During Weathering

Present Evidence. An oxidizing ore deposit appears to be one of the simplest of natural chemical systems, where reactions occur near the surface at normal temperatures in water solutions. But the inherent variabilities of superimposed igneous, hydrothermal, and supergene processes, and the relative openness of the system in terms of air and water have forced an acceptance of qualitative generalizations.

The copper and iron sulfate systems have been determined for elevated temperatures, and the combined system has been partially determined (Posnjak,1929). Much of the theoretical support for relationships determined empirically during early capping studies was developed from these system studies. They indicate that, at 50°C, cupric oxide and the basic sulfates may form in solutions containing greater than 97 percent water when the CuO concentration is larger than SO_3, and that chalcanthite forms in solutions containing less than 80 percent water when CuO is equal to or less than the SO_3 in the system. Only goethite, of the common stable iron minerals, is present in the aqueous iron sulfate system, and no field occurs for hematite or, of course, jarosite. By extrapolation, only goethite is shown to form in solutions containing from 96 percent water to almost infinite dilution when the ratio of SO_3 to Fe_2O_3 is less than approximately 3 to 1; no precipitate forms when the SO_3 exceeds this ratio.

Within these limits it is necessary to speculate about the probable limitations in its range. However, an environmental restriction based on modern climate is not warranted since marked climatic changes have occurred in the Southwest during historical time and major changes may be inferred to have occurred during the formation of the enriched orebodies.

It is certain that no single homogeneous solution prevailed, and two kinds of solutions probably existed. More concentrated acidic solutions were associated with the decomposing sulfides, and relatively mild solutions formed chrysocolla and the acid soluble tenorite or copper carbonates. Except for the common confinement of native copper and cuprite to the lower part of the capping, there are no major transitions within the capping and in its basic features, especially the iron minerals, probably persisted as formed.

Acidity. The leaching solutions began as rain water, which is about pH 6 when in equilibrium with atmospheric carbon dioxide at the present time. The acidity was only slightly changed during percolation through the relatively stable capping, but when the oxygenated solutions encountered sulfides they become acidic through the formation of sulfuric acid and by cation hydrolysis. Ferric iron readily hydrolyzes and a conspicuous precipitate forms when the pH is greater than 2.5; ferrous iron remains in solution in similar concentrations to pH 5. Of course, both of these ions

will remain in solution at a higher pH in concentrations equal to the solubility of their oxides or hydrated oxides, and a large amount of iron could be eventually transported in weakly acidic solutions within the geologic time involved. A pH of 2-2½ is readily achieved in moderately concentrated iron solutions such as mine water where the pH is depressed by hydrolysis of iron as rapidly as ferrous iron oxidizes until an equilibrium is reached or the iron is depleted.

Copper also hydrolyzes in aqueous solutions, and the precipitation of tenorite lowers the pH until the solution is in equilibrium with the hydroxide. The amount of copper in solution above pH 6.0 is negligible, and this is often cited as the highest pH at which copper will remain in solution. In the absence of strong complexing agents cuprous ions are not stable in aqueous solutions.

Jarosite is very abundant in capping and is relatively insoluble in acid. It is synthesized at elevated temperatures in very acidic solutions of about pH 1, but its association with hematite makes it doubtful that this acidity was achieved during weathering.

Silicate reactions offer few quantitative clues, although a large amount of kaolinite is formed from feldspar by the acid solutions. The often observed alteration of biotite or chlorite to illite is achieved in dilute sulfuric acid of pH 0.5, but lack of knowledge of reaction rates, as with the synthesis of jarosite, makes an application doubtful.

The highest probable basicity is controlled by the calcite equilibrium and may be achieved locally. When neutralization occurs in solutions containing calcium, it proceeds only to pH 8, where precipitated calcite is in equilibrium with dissolved CO_2. In more basic solutions the precipitation of calcite causes ionization of carbonic acid, and the acidity is increased by the release of hydrogen ions until equilibrium is again achieved. In sulfate solutions, gypsum may precipitate before equilibrium is achieved, and the equilibrium pH is increased.

The range in acidity of leaching solutions is probably pH 2-8. Greater acidity may occur in pyritic veins where solution flow is restricted or in film reactions during evaporation where the solution may not even be aqueous in nature. The synthesis of jarosite or the alteration of chlorite require solutions having a pH of 1 or less to cause reactions observable within hours or weeks, but no evidence exists that this condition is present in nature.

<u>Copper Concentration</u>. During the decomposition of a sulfide ore, the copper in solution is restricted only by the precipitation of basic or normal sulfates. In sulfide leaching experiments, the copper retained in solution is between 10^{-3} and 10^{-7} mole per liter. Solutions passed through pyrite-chalcocite samples are in equilibrium with brochantite, but solutions from pyrite-chalcopyrite samples are more acidic and chalcanthite forms when evaporation occurs.

When copper sulfate solutions are evaporated, the metal concentration may become very large before chalcanthite is formed. Because of the free sulfuric acid and iron sulfates in the solution, the maximum copper concentration should be less than 1.4 moles per liter, which is that derived from the solubility of chalcanthite. During industrial leaching of sulfide dumps and in mine sumps, the copper in solution is generally less than 0.1 mole per liter, but these solutions are not saturated. When chalcanthite is redissolved, solution concentrations equal to its solubility may occur, especially where circulation is impeded.

Sulfate Concentration. The dissolved sulfur readily forms sulfate in the presence of oxygen, and bivalent sulfur is rare in mine waters. Mine sump water may contain 0.7 mole per liter sulfate, and concentrations of 2 moles per liter probably occur in evaporating solutions where chalcanthite is forming. The range of sulfate in ground water draining several copper deposits has been reported to be 10^{-1} to 10^{-3} mole per liter (Emmons, 1917), but many of the samples were from non-oxidizing zones and the larger concentrations are more characteristic of solutions associated with decomposing sulfides. In sulfide leaching experiments, the largest sulfate concentrations are 10^{-2} to 10^{-4} mole per liter.

Carbon Dioxide Concentration. The carbon dioxide present in ground water is derived from the atmosphere or by solution of carbonate minerals. A concentration greater than predicted by gas equilibrium has been attributed to organic action or decay, selective adsorption on soil particles, reduced ventilation, or loss by diffusion in water saturated sediments (Adams,1937).

The maximum CO_2 dissolved in water at 25°C in equilibrium with the atmosphere is 10^{-5} mole per liter (Harned,1943). This is reduced approximately 3.4 percent per 1,000-foot rise in elevation and 1.7 percent per degree rise in temperature. Solubility is also reduced in the presence of dissolved salts, and the absorption coefficient is nearly linear to the salt concentration but varies little from one salt to another (Johnson,1915).

When acid solutions react with limestone near the surface, the effective partial pressure of CO_2 approaches one atmosphere and the equilibrium solubility of CO_2 will increase to $10^{-1.5}$ mole per liter.

In surface waters, carbonate is usually not present in appreciable quantities but bicarbonate occurs in concentrations of 200-400 ppm in limestone areas to less than 50 ppm in waters that drain areas of relatively insoluble igneous rocks. In waters from 42 mines in non-calcaleous rocks, carbonate was present in concentrations of 13-240 ppm, but in a single sample exceeded 1,500 ppm (Emmons,1917).

It must be noted that variations in the CO_2 content of the atmosphere have probably occurred during the oxidation of these copper deposits. An increase probably occurred during the period of Tertiary volcanism, and a decrease occurred during the Pleistocene epoch when solubility increased in the cooler oceans. The present contributions from domestic and industrial fuels is

also creating an imbalance.

Chemical Stability of Oxidized Copper Minerals

Introduction. During the past decade a major effort has been made to apply thermodynamic and chemical data to theoretical studies of geologic processes. Because most data are determined for a temperature of 25°C and atmospheric pressure, it is chiefly in the study of surficial processes, such as weathering or sedimentation, that these data may be applied even semi-quantitatively.

A calculation of mineral stabilities was made by Garrels (1954) for the formation of copper oxides and sulfides in solutions derived from the oxidation and leaching of a sulfide deposit. The only assumption necessary was the sulfur concentration of the solution. For the model calculation 0.1 mole per liter was used, and the activity of the various ions could then be defined for any selected oxidation potential and acidity.

Mineral stabilities are defined by oxidation potential only when valence changes are relevant. Reduced phases are negligible in the near-surface zone of oxidized capping, and only cuprite may be of local importance. It is possible, therefore, to discuss capping minerals simply in terms of relative solubilities.

Effect of Temperature. The standard temperature at which thermochemical measurements are made (25° C or 77° F) is higher than the mean annual temperature of much of the world. This discrepancy is commonly discounted but is significant if the exclusion of a mineral stability field is expected.

At lower temperatures the brochantite field expands into the normal tenorite field and the chalcanthite field expands to replace the antlerite field. It is approximated that the antlerite field is absent below 18°C (66°F), and the chalcanthite and brochantite fields are greatly expanded.

At standard temperature and pressure neither malachite nor azurite will form in solutions in equilibrium with the atmosphere, and azurite will not form even in solutions saturated with CO_2. Qualitative experiments suggest that azurite may form by dehydration, and temperature changes are probably a major factor.

The solubility of malachite and azurite decrease with increasing temperature, in contrast with the other oxidized copper minerals, and their formation is therefore enhanced in warmer environments. It is probable that azurite forms only at temperatures above 25°C or where CO_2 is present in the atmosphere in greater than normal abundance.

Occurrence of the Oxidized Copper Minerals

Calculated Stability Fields. Calculated mineral stability diagrams

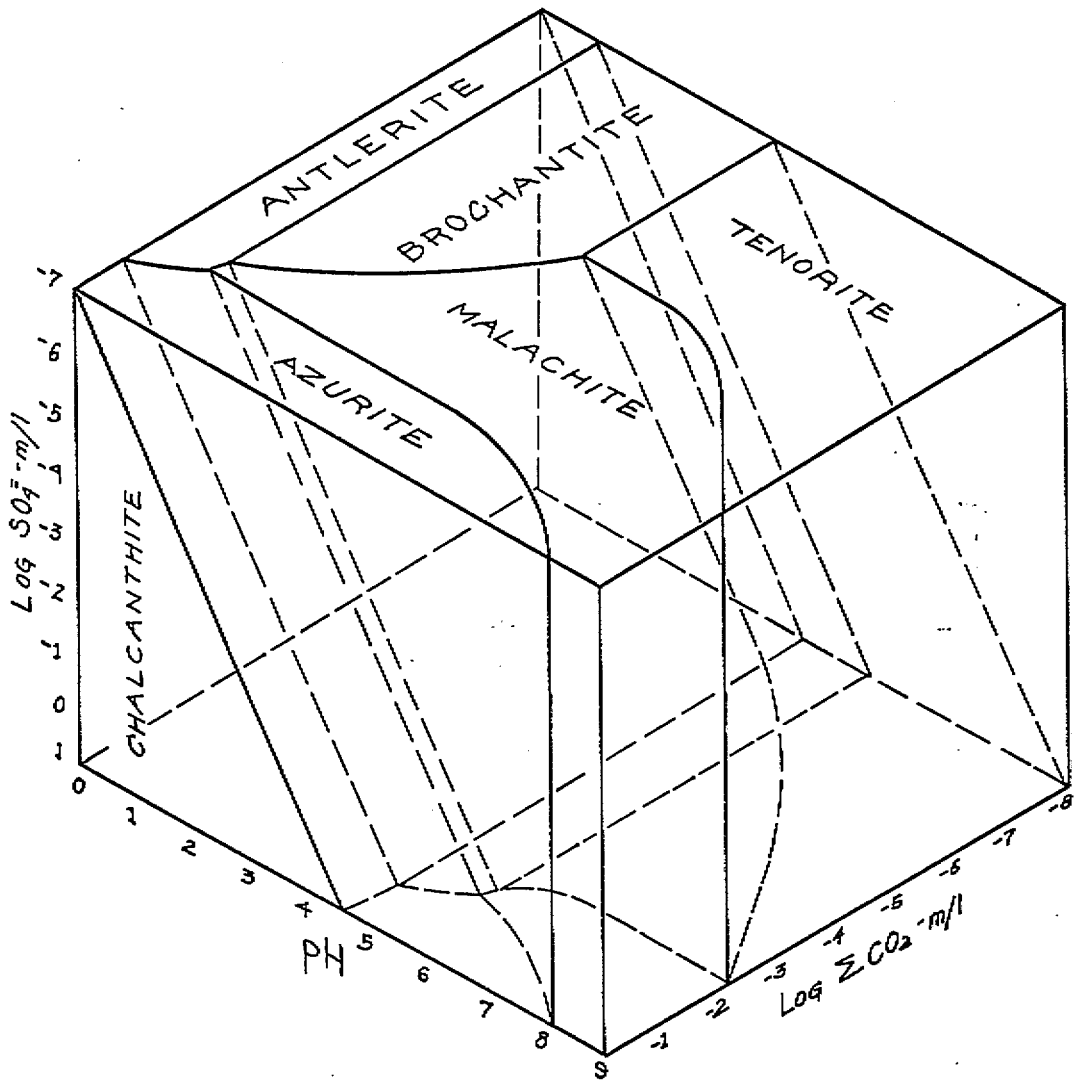


Fig. 28. COMBINED
STABILITY FIELDS OF OXIDIZED COPPER MINERALS
at 25°C.

FIG. 25

STABILITY FIELDS OF OXIDIZED COPPER MINERALS at 25°C

Contours indicate Cu^{++} activity

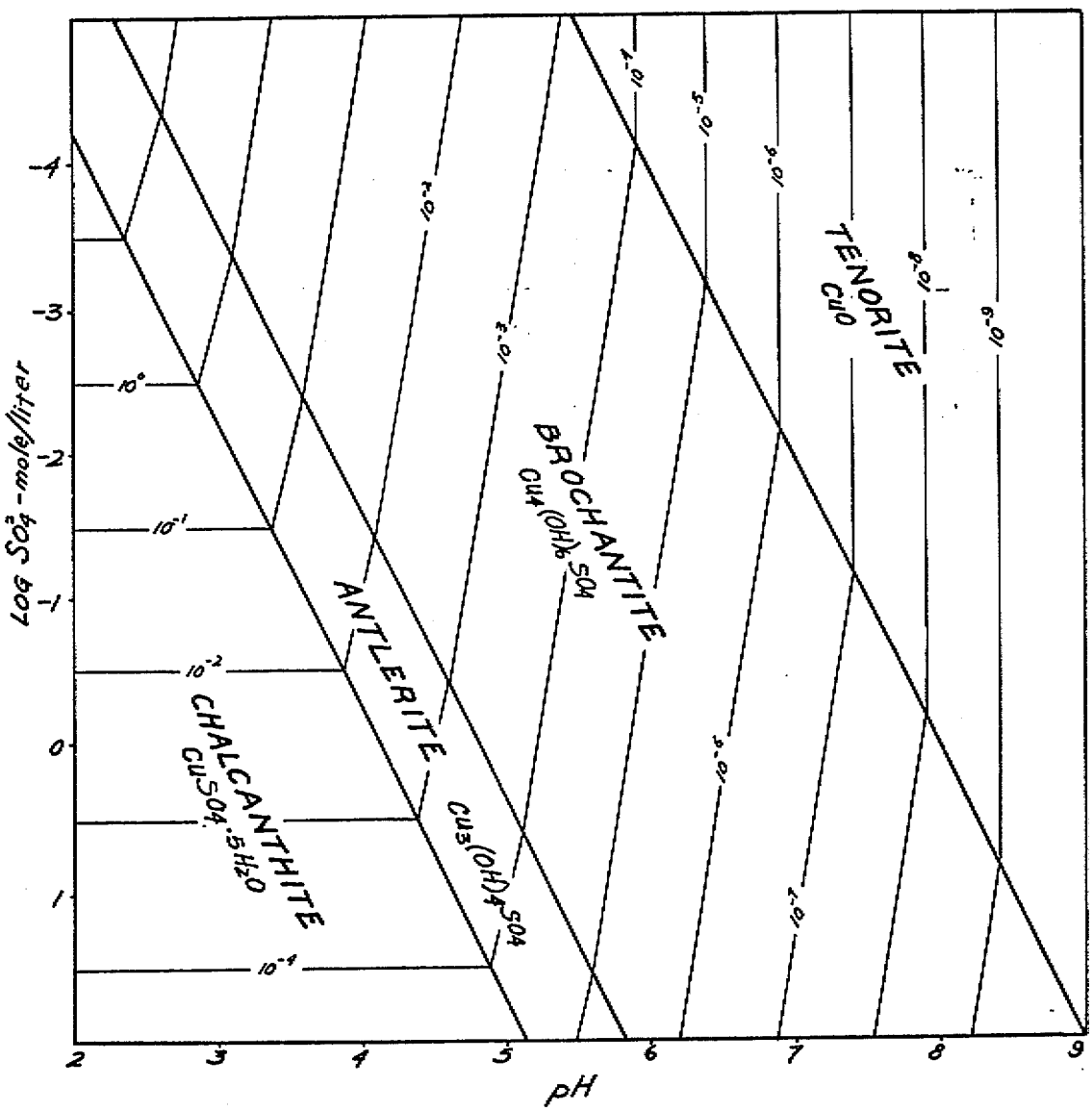


FIG. 26

APPROXIMATE STABILITY FIELDS OF OXIDIZED COPPER MINERALS at 18°C

ANTLERITE IS UNSTABLE BELOW THIS TEMPERATURE

Contours indicate Cu^{++} activity

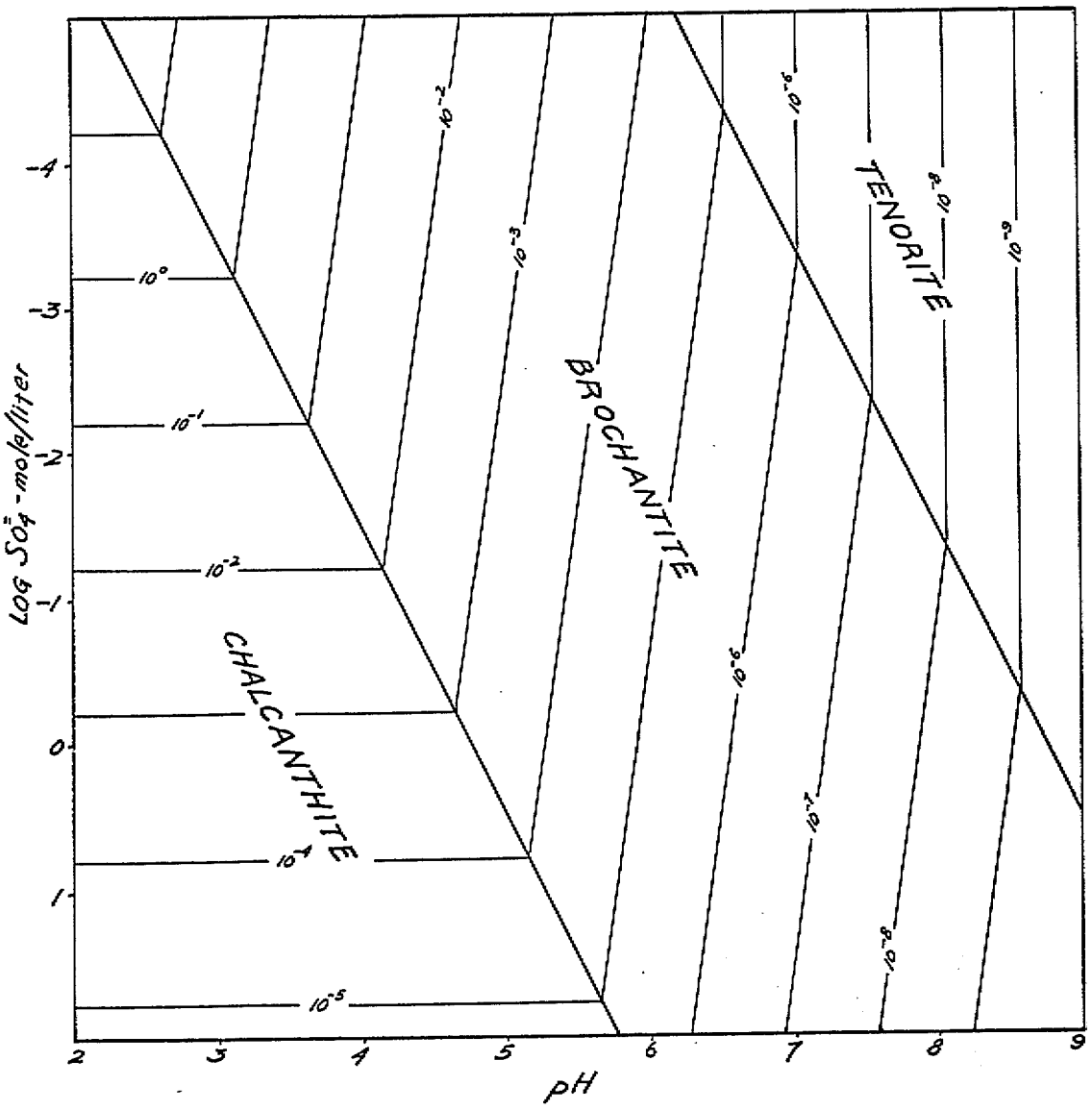


FIG. 27. STABILITY FIELDS OF THE COPPER CARBONATES IN NON-SULFATE SOLUTIONS AT 25°C

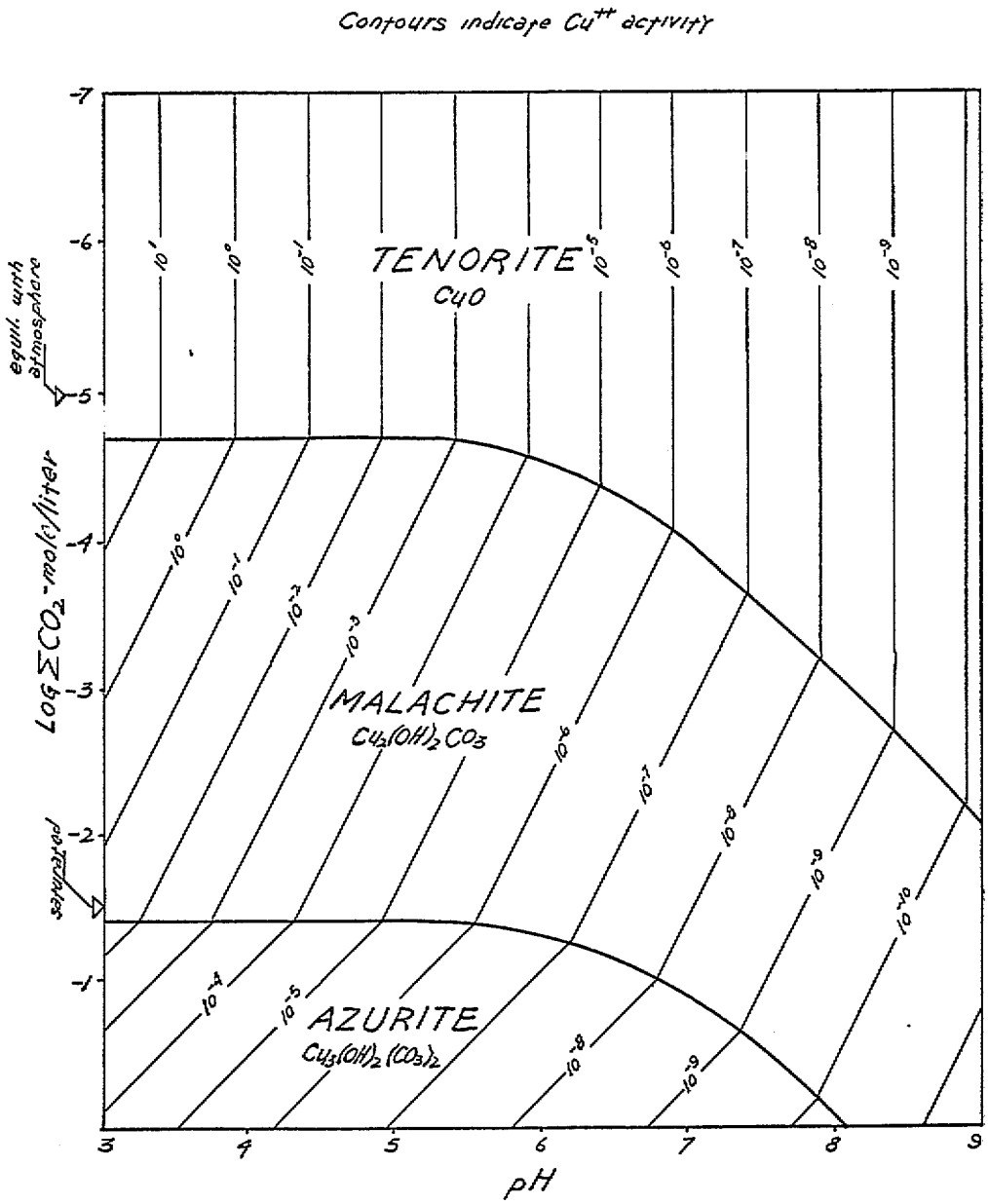


FIG. 33.
STABILITY FIELD OF ATACAMITE VS
ANTLERITE AND BROCHANTITE
at 25°C

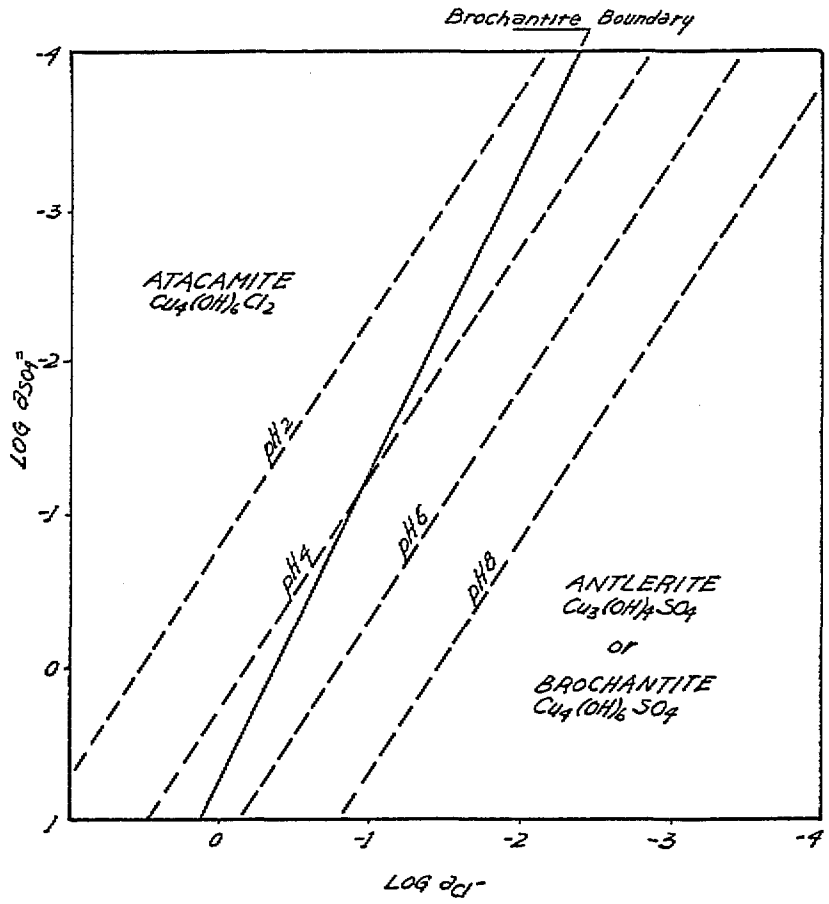
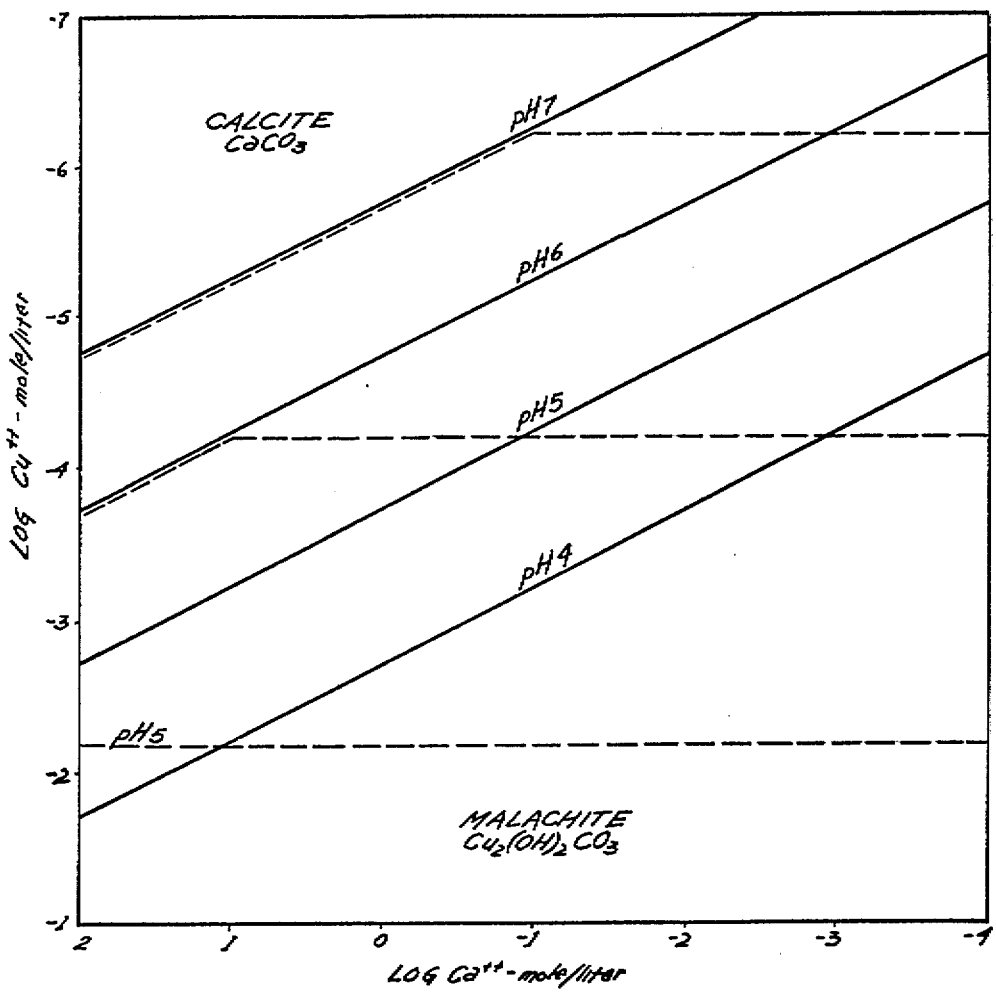


Fig. 31.
STABILITY FIELDS OF CALCITE vs MALACHITE

Dashed lines indicate limiting case where $\Sigma CO_2 = 10^{-4.7}$ and tenorite is in equilibrium with malachite.



represent only models from which possible mineral occurrences or associations can be predicted or explained. They may differ from the natural system through a failure to evaluate kinetic factors, metastability and unknown complexes, or transitional species.

The calculations are based on the premises that the activity of a metal ion is defined by the equilibrium constant if the activity of the anion is specified, the activity of the hydroxyl ion is defined when the pH is specified, and that the solid phase will consist of the least soluble mineral where equilibrium is achieved.

Chalcanthite. Chalcanthite does not generally persist at outcrop because of its large solubility or because of dehydration. It is often abundant at shallow depth, however, and readily forms as incrustations or efflorescences on mine walls or on the floor and walls of open pits. It clearly forms from a very concentrated solution during evaporation and is readily redissolved.

The pentahydrate forms in the evaporating solutions and is stable below 108°C. It dehydrates in a dry atmostphere to a powdery whitish monohydrate that is probably not always recognized in capping.

Chalcanthite generally occurs free but may be in proximity to other water soluble sulfates, such as yellowish brown coquimbite, translucent greenish melanterite, and white fiberous alums. Underground it forms as stalactitic masses, platy fracture fillings, and clusters on exposed faces. When exposed in open pits, it is generally admixed with clay and presents a dramatic appearance far greater than its quantitative significance. It has no direct role in the interpretation of capping, but by isolating copper in a transient phase it influences the nature of the more stable phases that are subsequently formed.

Antlerite. Antlerite does not occur commonly or is not recognized often. It was a major ore mineral at Chuquicamata, Chile, where for many years it was called brochantite. Both antlerite and brochantite are commonly identified as "malachite", and when qualitative chemical tests are made the more common name "brochantite" is frequently assigned. However, even when adequate optical or chemical tests are made, antlerite is rarely detected and appears to be of minor significance in domestic copper deposits.

Because of the almost unique occurrence of large tonnages of antlerite in the former oxidized ore at Chuquicamata, its formation has been attributed to a very arid environment. Bandy (1938) has stressed the factor of high solution concentration caused by slight wetting of the voluminous soluble sulfates, but at Potrerillos, a similar deposit on the Atacama Desert, the predominant oxidized mineral is brochantite. Jarrell (1939) has speculated that this occurred because the acidity of the leaching solution was lower at Potrerillos where there is less pyrite.

The instability of antlerite at temperatures comparable to those

existing widely at the present time and probably during the oxidation of domestic copper deposits explains its relative absence. The presence of antlerite does not provide clear evidence of a past warmer climate, however, and the rate of oxidation or solution complexity may be important factors.

Brochantite. Brochantite appears to be present in all the porphyry copper deposits, and when proper tests are made is often found to be relatively abundant. Extensive tests made at Morenci, Arizona indicated that it is the principal green oxidized copper mineral associated with porphyry ore, although malachite is abundant in the peripheral sedimentary replacement deposits and may have been more abundant near the pre-mining surface.

Brochantite also forms, with chalcanthite, as an incipient oxidation product in chalcocite ore exposed by mining in open pits. It occurs as powdery films or minute spherulites on chalcocite and may constitutemore than ten percent of the total copper contained in a normal sulfide ore. It may not be detected by superficial examination but was present in all enriched ore examined in the Southwest. It is generally conspicuous and may be the only oxidized mineral present in enriched sulfide veins. The brochantite is also common as the "verdigris" on bronze statues and plaques or copper coins.

Therefore, brochantite appears to be the first copper mineral formed during the oxidation of enriched chalcocite orebodies in the Southwest. When oxidation progresses to the stage where pyrite decomposes, the brochantite is probably destroyed. It will certainly not persist in an environment in which ferric iron is being transported, but reprecipitates in coarsely fiberous masses or layers in fractures or voids where the evidence of solution migration is clear.

Atacamite. Atacamite, the basic cupric chloride, has been reported to occur in some mines in the southwestern United States, and it is probably more abundant than now recognized.

The stability of atacamite relative to brochantite is dependent only on the anion concentrations. In an environment where antlerite normally forms, atacamite is more stable when pH is high but the chloride concentrations must be very large. It is probable that atacamite forms by replacement of earlier formed minerals in chloride solutions derived from saline ground water, hot springs, or playa deposits.

Tenorite. The black cupric oxide forms in a weakly acid or basic environment, probably from relatively dilute solutions. Its formation is analogous to hematite in that it is considered to form from a gelatinous precipitate of a hydroxide or hydrated oxide.

The solutions, after migrating from the decomposing sulfides, may be neutralized by reaction with silicate minerals and locally becomes as basic as pH 8. Iron is generally absent from the solutions before tenorite precipitates, although manganese is almost always coprecipitated with the copper.

The tenorite generally shows evidence of solution migration even in a limestone environment. It forms in layers or nodules, rarely in crystals, which concurs with the evidence that gelatinous precipitates are formed when tenorite is synthetized by neutralization of copper solutions. It may be derived from cuprite or native copper and occurs as coatings on the surface of these minerals, but this relationship is rarely seen in the capping of the porphyry orebody.

Forming relatively late, it succeeds the basic sulfates and is closely associated with malachite, azurite, and chrysocolla. Of this group, all of which formed late from transported solutions, tenorite is most often the first to precipitate, and the copper carbonates often appear to have been derived from it. Some trace of tenorite is present in most specimens of copper carbonate, and they clearly form in a similar environment.

Black coatings or nodules consisting of copper, iron, or manganese oxides frequently occur in capping. They may be pulverulent to pitch-like, thinly layered, or minutely botyroidal. The metals occur in various ratios characteristic of mixtures and silica is commonly present. Materials of this type have been called "copper pitch", or melaconite, and other names have often been applied to local varieties.

The copper mineral contained in these mixtures is probably either tenorite or an amorphous cupric oxide, but some specimens contained delafossite, a copper-iron oxide. The vitrous varieties are often transitional from normal blue chrysocolla, and zoned or banded black and blue specimens occur. The black specimens differ chemically from chrysocolla only in a high manganese content, but a black color can be produced in normal chrysocolla by heating (Schwartz,1934).

When cupric oxide is precipitated the initial color is blue. When allowed to age the gelatinous precipitate dehydrates and changes in color to green, brown, and black. These are the colors of natural chrysocolla or chrysocolla-like minerals. When manganese is precipitated with the copper, it does not completely mask the blue copper precipitate but appears to accelerate the transformation to a black color. The color change is retarded when gelatinous silica is precipitated with copper.

The change in color has been reported to be influenced by factors such as temperature, alkalies and certain salts, or the colloidal concentration (Weiser,1926). These factors are erratic in their effect and, as characteristic of other gels, slight changes in the method of preparation or aging may influence the nature of the product.

Cuprite. Cuprite is often abundant in vein or replacement deposits and near the bottom of leached capping. It is very commonly associated with native copper, and alters to tenorite or malachite. Because it is stable in a less oxidizing environment than normally encountered near the surface, it is significant chiefly as an intermediate phase that may influence the nature of subsequently

formed cupric minerals. However, in reactive environments or where pyrite is not abundant it may persist in the capping.

Malachite and Azurite. The ubiquitous basic carbonates are associated with copper deposits of all types and were formerly a major ore, especially in veins and replacement deposits in limestone. In common with most carbonates, they are very soluble in strong acid and therefore form only in weakly acid or basic environments. They precipitate late, or perhaps most commonly are derived from earlier formed minerals, and probably only chrysocolla is immune to this alteration.

The solubility of these carbonates is dependent only on the total carbon dioxide in the solution and the acidity, and no stability field exists in a system in which the carbon dioxide content is low. When the total carbon dioxide in the solution reaches or exceeds $10^{-4.7}$ moles per liter (at 25°C), tenorite is no longer stable except in basic solutions and malachite preferentially forms. At slightly larger concentration of carbon dioxide the malachite field also replaces part of the normal brochantite field when the sulfate concentration is less than 10^{-5} mole per liter. If the carbon dioxide in solution is increased, the brochantite field is further reduced and, when a concentration of CO_2 of $10^{-1.4}$ moles per liter is reached, azurite becomes the stable phase and completely supplants the former malachite field.

The stability of malachite and azurite therefore depends on the solution and dissociation of carbon dioxide, and rainwater, or any solution in equilibrium with the atmosphere, is therefore not capable of precipitating the copper carbonates except at higher temperatures in which mineral solubility is reduced. It is evident that malachite should be the principal carbonate formed, even in a limestone environment, and the relative absence of azurite, forming only in saturated or near saturated solutions of CO_2, is also understandable.

The transformation to the most stable carbonate appears often to proceed relatively rapidly, and many mislabeled museum specimens are attributed to this. Both minerals commonly coexist, however, with no fixed sequence of formation, and the long-standing ambiguity about "first formed" carbonate is unavoidable. Present consensus favors azurite as most commonly being the later mineral to form.

Chrysocolla. The most common copper mineral in porphyry capping is chrysocolla. It is associated with, but often later than, the late-forming minerals tenorite, malachite, and azurite, and appears to have formed in solutions removed from the acid environment of the decomposing sulfides. No quantitative evidence relating to the chemical aspects of its formation is available, and considerable doubt exists about its character.

The common "chrysocolla", which varies in composition, and other copper silicates such as diptase, bisbeeite, ajoite, and shattuckite, some of which are found in a single district, are believed to have formed in a similar environment. Because specific evidence is lacking, they are assumed to be of equal significance in capping.

Conclusions

The formation of supergene enriched ores and oxidized capping probably occurred during the Tertiary period, and the sulfide ore is being modified or destroyed at the present time. The changes that occur within the capping are minor and are confined to the further leaching of some copper and iron near the surface and the alteration of some earlier-formed copper oxides and basic sulfates to copper carbonates or chrysocolla.

During the incipient oxidation of copper sulfides, chalcocite and chalcopyrite are leached at comparable rates. Brochantite forms on chalcocite but the copper derived from chalcopyrite migrates because of the lower copper-sulfate ratio and the greater acidity; chalcanthite forms as a transition phase if evaporation occurs. Cuprite may form early, especially in limestone or other reactive environments, but it is not generally abundant in porphyry ores. Antlerite is unstable at lower temperatures, probably below 18°C, and is rare in ore deposits, but brochantite is more common than generally believed. The early precipitates are modified after the enriched chalcocite coating the pyrite is denuded or the rapidly decomposing chalcopyrite is consumed, and the environment becomes solely pyritic. All of the stable copper minerals present in the upper part of the capping show evidence of being formed from migrating solutions or of being derived from other oxidized copper minerals.

Tenorite, malachite, azurite, and chrysocolla form late from relatively basic solutions removed from the zone of decomposing sulfides. Malachite will not form at 25°C in solutions in equilibrium with the average partial pressure of CO_2 in the present atmosphere and azurite will form only in buffered solutions in which CO_2 is supersaturated. Malachite forms readily, often with tenorite, especially in or near a limestone environment, but azurite is relatively rare. Both of the carbonates are more stable at higher temperatures, and some azurite may form by dehydration.

Although oxidized copper minerals are not generally conspicuous in capping, some are always evident locally. However, geochemical anomalies most commonly reflect unseen copper that is intimately, although not uniformly, associated with iron minerals.

References

1. Adams, C.S., and Swinnerton, A.C., 1937, Solubility of limestone: AGU Trans., vol. 18, p. 504-508.
2. Bandy, M.C., 1938, Mineralogy of the three sulfate deposits of nothern Chile: Am. Min., vol. 23, p. 713-760.
3. Emmons, W.H. 1917. The enrichment of ore deposits: U.S. Geol. Survey Bull. 625, 530 pp.
4. Garrels, R.M., 1954, Mineral species as functions of pH and oxidation-reduction potentials, with special reference to the zone of oxidation and secondary enrichment of sulfide ore deposits: Geochim. Cosmochim. Acta, vol. 5, p. 153-168.

5. _____, 1960, Mineral equilibria at low temperature and pressure, Harper Brothers, New York, 254 pp.
6. Harned, H.C., and Davis, Jr., Raymond, 1943, The oxidation constants of carbonic acid in water and the solubility of carbon dioxide in water and aqueous salt solutions from 0° to 50°; Jour. Am. Chen. Soc., vol. 65, p. 2030-2037.
7. Jarrell, O. W.; in Palache, Charles, 1939, Antlerite: Am. Min., vol. 24, p. 293-302.
8. Johnson, John, 1915, The solubility product constant of calcium and magnesium carbonates: Jour. Am. Chem. Soc., vol. 37, p. 2001-2020.
9. Posnjak, Eugen, and Tunell, George, 1929, The system cupric oxide-sulphur trioxide-water: Am. Jour. Sci., 5th. ser., vol. 18, p. 1-34.
10. Schwartz, G.M., 1934, Paragenesis of the oxidized ores of copper: Econ. Geol., vol. 29, p. 55-75.
11. Weiser, H.B., 1926, The hydrous oxides, McGraw-Hill, New York, 452 pp.

6-18-54

SEDIMENTARY COPPER ORES IN THE

"RED BEDS"

S. T. B.
JUN 28 1954

F. V. R.
JUN 26 1954

Conclusion

In the literature of sedimentary copper deposits (exclusive of those on the Colorado Plateau), most writers believe that the deposits are syngenetic, and their source was fissure veins in pre-Cambrian rocks or formations older than the sediments in which the copper horizons are found.

The ore horizons, confined to the sandstones, shales, and grits of late Carboniferous to Jurassic formations, often contain abundant amounts of carbonaceous material. Descending solutions leached copper from overlying formations and concentrated it in favorable localities.

No uranium minerals were directly associated with sedimentary copper ores, although vanadium oxide was reported in coal analysis of the red beds of Fremont County, Colorado. Since yellow and orange secondary uranium minerals are easily recognized, it seems improbable that examining geologists failed to note their association with the sedimentary copper, although the dark-colored uranium mineral pitchblende-uraninite could have been overlooked or improperly identified as one of the secondary copper sulphides.

The most logical conclusion is that no secondary uranium mineralization was found in the copper-ore horizons.

On the Colorado Plateau, sedimentary copper deposits are found in many places adjacent to uranium properties, while at several uranium-producing mines, secondary copper sulphides and oxides occur in significant amounts to make their recovery economic. The proximity of uranium to copper deposits might be partially explained by the difference in solubility of the two minerals.

Assuming that the minerals had been originally deposited at the same time from the same solution, then copper sulphides, oxides, carbonates and silicates once deposited would tend to remain fixed, while the more soluble uranium could be taken into solution by ground water and redeposited in other favorable localities.

Willis Johns

W. R. L.
JUN 23 1954

READ AND RETURN

FVR	✓	
SIB	✓	
DWN	✓	
RJL		

SEDIMENTARY COPPER ORES
COLORADO PLATEAU

Reference: U.S.G.S. Bulletin 315, pp. 102-105

Locality: Hartville uplift, 14 miles north of Fort Laramie, Wyoming.

Ore Horizon: Carboniferous Guernsey sandstone.

Occurrence: Lenticular or tabular masses in sandstone.

Origin: Source of copper was from upper Guernsey fm., or some other formation above it that has been removed by erosion. This presupposes the original source as being a pre-Carboniferous copper deposit probably of the fissure-type in pre-Cambrian rocks.

Copper was included as a chemical or mechanical impurity in the Guernsey fm., or one overlying it, and was deposited in its present state by descending and laterally moving solutions which concentrated the copper in pockets, sink holes or irregularities of the underlying pre-Cambrian surface. Clays may have been responsible for precipitating the copper from the solution.

The ore bodies are not connected with fractures, lie on an impervious clay stratum, and are associated with detrital iron ore in some places.

Relation of copper to uranium: No uranium mineralization was mentioned in the report; apparently no primary uranium minerals were associated with the primary copper in the pre-Cambrian land mass.

Reference: U.S.G.S. Bulletin 340, pp 170-174.

Locality: "Red Beds" of Fremont County, Colorado.

Ore Horizon: Shales, conglomerates, and limestones of 550' Pennsylvanian(?) "Red Beds." Ore is found at several horizons.

Occurrence: In conglomerate ore is associated with coaly material, fossil wood(?) and in shales ore is found in 1" - 2" coal seams.

Origin: Pre-Cambrian rocks containing copper deposits furnished material for "Red Beds." Circulating surface waters were instrumental in forming strong copper sulphate solutions. The coal seams of the "Red Beds" were replaced as follows:

$$4\ CuSO_4 + 5C + 2H_2O \text{ -- } 2\ Cu_2S + 2H_2SO_4 + 5CO_2$$

Relation of Copper to Uranium: Coal analysis showed a vanadium oxide content of 0.114%. No mention was made of any diagnostic secondary uranium mineralization. Apparently little or no uranium was associated with the primary copper, or it was removed entirely by solution activity.

SEDIMENTARY COPPER ORES
NEW MEXICO, ARIZONA

Reference: N.M.B.M. Bulletin 15, pp. 37,42,58,62,67, and 94.

Locality: Union, Mora, Harding, San Miguel, Guadalupe and Quay Counties, New Mex.

Ore Horizon: Triassic Santa Rosa sandstone. Sedimentary copper deposits are also found in the following formations: Pennsylvanian Magdalena ls., Permian Glorieta ss., Triassic Dockum sh. and ss.

Occurrence: Bedded deposits in a depression with impervious clay underlying the ore and frequently overlying the ore. The Santa Rosa sandstone appears to be laid down under conditions of shallow stream flow. A structural trap (syncline) formed from gentle flexing of the beds, and it is surmised that this same movement metamorphosed the underlying and overlying sediments to clay.

Origin: Copper may have originated from pre-Cambrian masses and was possibly removed in solution or as disintegrated particles of the mineral. Surface waters migrating through the Santa Rosa ss. and overlying formations carried the copper to its present location where plant remains and other organic matter acted as a reducing agent precipitating the copper in the ancient stream beds.

Relation of copper to uranium: No uranium mineralization was mentioned in this report, and it appears safe to assume that none was associated with the primary copper.

Reference: U.S.G.S. Bulletin 540, pp. 159-163.

Locality: White Mesa district, Arizona, 100 miles northeast of Flagstaff, Ariz.

Ore horizon: Jurassic LaPlata sandstone(?).

Occurrence: In strongly cross-bedded sandstone which has numerous joint planes that may influence ore deposition. Ore occurs in small bunches.

Origin: Copper solutions have replaced cementing clay of the sandstone.

Relation of Copper to Uranium: No uranium minerals present.

SEDIMENTARY COPPER ORES
IDAHO, UTAH

Reference: U.S.G.S. Bulletin 430, pp. 113-121.

Locality: Montpelier, Bear Lake County, Idaho.

Ore Horizon: Triassic Ankareh shale and sandstone.

Occurrence: Ore is confined to same stratigraphic horizon and is found in joints along the bedding. Also in a 5' clayey, sandy rock between massive, even-bedded red sandstones.

Origin: This "Red Bed" type of deposit belongs to a well-recognized type. Carbonized plant fragments acted as precipitating agents for deposition of copper from solutions. This copper was widely distributed in disseminated form in the overlying sedimentary rocks as they were laid down.

Association of copper and uranium: No uranium minerals mentioned in this report.

Reference: U.S.G.S. pp. 111, p. 614.

Locality: LaSal Mountains, "Big Indian Valley".

Ore horizon: ~~Middle~~ Upper Cret ~~Triassic~~, at or near horizon of ~~Shinarump~~ Dakota conglomerate.

Occurrence: In permeable gray, medium-grained sandstones with lenses of fine cong.

Origin: Mineralization appears to have taken place along a general strike of fissuring in sandstone. In some places no relation between fissuring and sedimentary copper ore is apparent. Chalcocite occurs as a replacement of a sulphide, probably chalcopyrite, and it cements the sand grains together, and in places partially replaces them.

Association of copper and uranium: Uranium mineralization has been found in numerous localities around the LaSal Mts., but no uranium was mentioned in connection with the copper deposits, although it appears probable that uranium does occur in minor amounts.

Reference: U.S.G.S. pp. 111, p. 615

Locality: Lisbon Valley, near Colorado-Utah line, 14 miles SE of LaSal post office.

Ore horizon: Interbedded sandstones and shales and Carbonaceous shales near the base of the Triassic?(contact of Shinarump and Moenkopi)?

Occurrence: Occurs as cement of the sandstone and masses lining openings in rocks. No fissuring noted in connection with deposits. Porous beds appear most favorable.

Origin: Most of copper is present as a sulphate and has possibly leached downward from the sandstone beds above. Abundant plant remains are present in main ore horizon.

T-10.6.0

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

June 14, 1954

RECEIVED
JUN 16 1954
WESTERN MINING DEPT.

Mr. W. R. Landwehr, Chief Geologist
Western Mining Department
Salt Lake City Office

"Red Bed" Copper Deposits
Origin of

Dear Sir:

Excerpts from "Copper Resources of the World", are attached. These are of interest in connection with uranium occurrences in sedimentary beds, as well as those of copper. Proponents of the hypogene theory of origin have stressed the presence of sulphides as evidence; Finch believes they were formed through the reaction of copper and other metals with hydrogen sulphide derived from plant remains under a reducing environment (well below the surface).

The comparatively recent age (Pliocene) of the Boleo deposits of Baja California is a rather strong point against a hypogene origin.

Very truly yours,

J. H. Courtright

J. H. Courtright

JHC:ar
enc. as noted above
cc: KRichard

W. R. L.
JUN 16 1954

EXCERPTS FROM COPPER RESOURCES OF THE WORLD, VOLUME 1

Sedimentary Copper Deposits of the Western States

by J. W. Finch

P 375 —

Characteristics and origin of the copper deposits. Although the copper deposits occur in the Red Beds, they are rarely found in red strata. Possibly the beds that contain them were bleached by the copper-depositing reactions. These deposits differ from hydrothermal deposits in the absence of any related quartz or other gangue. Silicification of woody fossils and the presence of calcite and gypsum are independent of copper deposition. The deposits show no geomorphic, structural, or geographic relation to Jurassic or Tertiary igneous regions — in fact, in areas near the Red Beds deposits the Tertiary magmatic solutions lacked copper. Pre-Cambrian ore deposits in and around the Red Beds region were characteristically copper-bearing. The Red Beds deposits contain copper sulphides, but not with hydrothermal mineral associations, and they were not restricted to special epochs, like hydrothermal ores, but were formed through several long geologic periods. They are so widely distributed and so uniform in character that all must have had a common origin. They were apparently dependent upon local sources of copper that were also local sources of the sediments and were deposited only under special conditions of sedimentation.

P 377 —

The Boleo Copper Area, Baja California

Evolution of the copper deposits. Three stages may be assumed in the evolution of the copper deposits — (1) the original deposition of copper with the beds, (2) a long period of deep burial that placed the copper deposits in a reducing environment, (3) a period of reoxidation as erosion exposed them. The deposits were syngenetic in stage 1, epigenetic thereafter.

Stage 1. The general method of deposition in solution and as fragments (15,21,22,23) has already been suggested. The sediments were derived from extensive highlands generally devoid of copper. There was, however, a localization of copper along slopes heading in areas containing copper outcrops, so the deposits in stage 1 were controlled by position of sources, routes of run-off, and torrential temporary streams. Woody material was protected from normal decay by dessication. Its tendency was to become silicified if it remained long near the surface, but if rapidly and deeply buried it was lignitized. At ordinary temperatures the organic matter in the beds would not reduce sulphates to sulphides; native copper would be more likely to be precipitated (24), but this is rare in the deposits. Some carbonate might have been converted to chalcocite in ground water, as has been shown by experiment (25), if moderate organic decay evolved some hydrogen sulphide.

Stage 2. In late Jurassic time the uplands had become peneplains. The late Jurassic, Cretaceous, and later seas deeply buried the 1,500 to 8,000 feet of Red Beds with 5,000 to 10,000 feet of overlying beds. Plant material was converted to coal, and covellite or chalcocite replaced the coal as fast as hydrogen sulphide was produced (3, p.79). There was some segregation of plant remains during deposition (15, p.172), and hence some localization of copper in stage 2. Covellite

and chalcocite were precipitated, also by the hydrogen sulphide reaction, in seams and lenses of porous rocks. Small amounts of pyrite and chalcopyrite were apparently deposited contemporaneously with the chalcocite (3, p. 79). Some replacement of these sulphides by chalcocite (8, pp. 615, 622) was possibly effected in stage 3. The copper ores of the Boleo mine (26), in Baja California, may have been formed in stage 2 by segregation of copper from beds, partly volcanic, that contained copper when they were laid down.

Stage 3. In stage 3, which has extended to the present time, copper has accumulated at and near the surface in local bodies of commercial value. Although there is no complete record, the total production from them in six States may have been about 10,000,000 pounds of copper. Remnants of chalcocite are found. Sulphates have migrated from scattered sulphides undergoing oxidation to places of discharge, where evaporation and reaction with lime have generally precipitated shallow bodies of copper carbonates. In some localities (27, 28) plant remains appear to have been lacking, and sulphate solutions entrapped in the sands were never precipitated as sulphides but were finally deposited as silicates and oxides in stage 3.

P-412

Ore bodies.

Although the erosion before and during the deposition of the cover has not been measured, it is possible that the top copper bed was never buried deeper than 100 meters.

Touwaide (2) concluded that the copper was originally present in the volcanic flows and tuffs (probably in the pyroxenes) to the amount of 0.2 percent. It was extracted thence, as he believed, by connate waters and penetrated the clay beds by diffusion, where it was precipitated as chalcocite, probably by H_2S or some other derivative of the organic matter in the clay. However, any such copper content as 0.2 percent in the fresh rocks is unproved.

ASARCO

Exploration Department
Western USA

August 31, 1981

R. L. Brown
New York Office

Van Dyke Copper Oxide Deposit
Arizona

Attached is Mr. Kreis' report on the Van Dyke Copper Oxide Deposit and his cover letter comparing the Van Dyke with the oxide copper zones at Santa Cruz and Poston Butte.

Mr. Kreis states that Occidental conducted an in-situ test but elected not to continue the next $16 million phase recommended by the project geologist. Mr. Kreis has verbally reported two additional items concerning the test:

1) Occidental now knows how to put together the correct hardware to conduct a test -- they learned this the hard and very expensive way.

2) Occidental did deliver pregnant solution, apparently of relatively high grade, to one of the local SX plants.

Mr. D.H.Davidson, in a report dated 21 Dec. 79 to Mr. Graybeal, enumerates some of the parameters necessary for a successful in-situ copper operation, It would appear, based on the fact that Occi did deliver pregnant solution, that the Van Dyke deposit will meet these. The ore reserve is not much above the minimum indicated as being feasible by Davidson.

The property owners would accept $7 million over 3 years for a complete buyout -- this figure is negotiable. Also, they would consider a five-year option for $50,000 cash plus requirement that we initiate a 500 gpm plant and expand it to 3,000 gpm by the end of the five-year period -- not an attractive option.

The $7 million figure appears reasonable if Asarco were to acquire the Cities Service's properties which include two SX plants that could process the Van Dyke pregnant solution. This assumes that over a period of time the Town of Miami problem can be resolved.

W. L. Kurtz

W. L. Kurtz

WLK:lb

cc: WDPayne
HGKreis

Aa-0.0.15B

ASARCO

Southwestern Exploration Division

October 19, 1978

TO: F. T. Graybeal

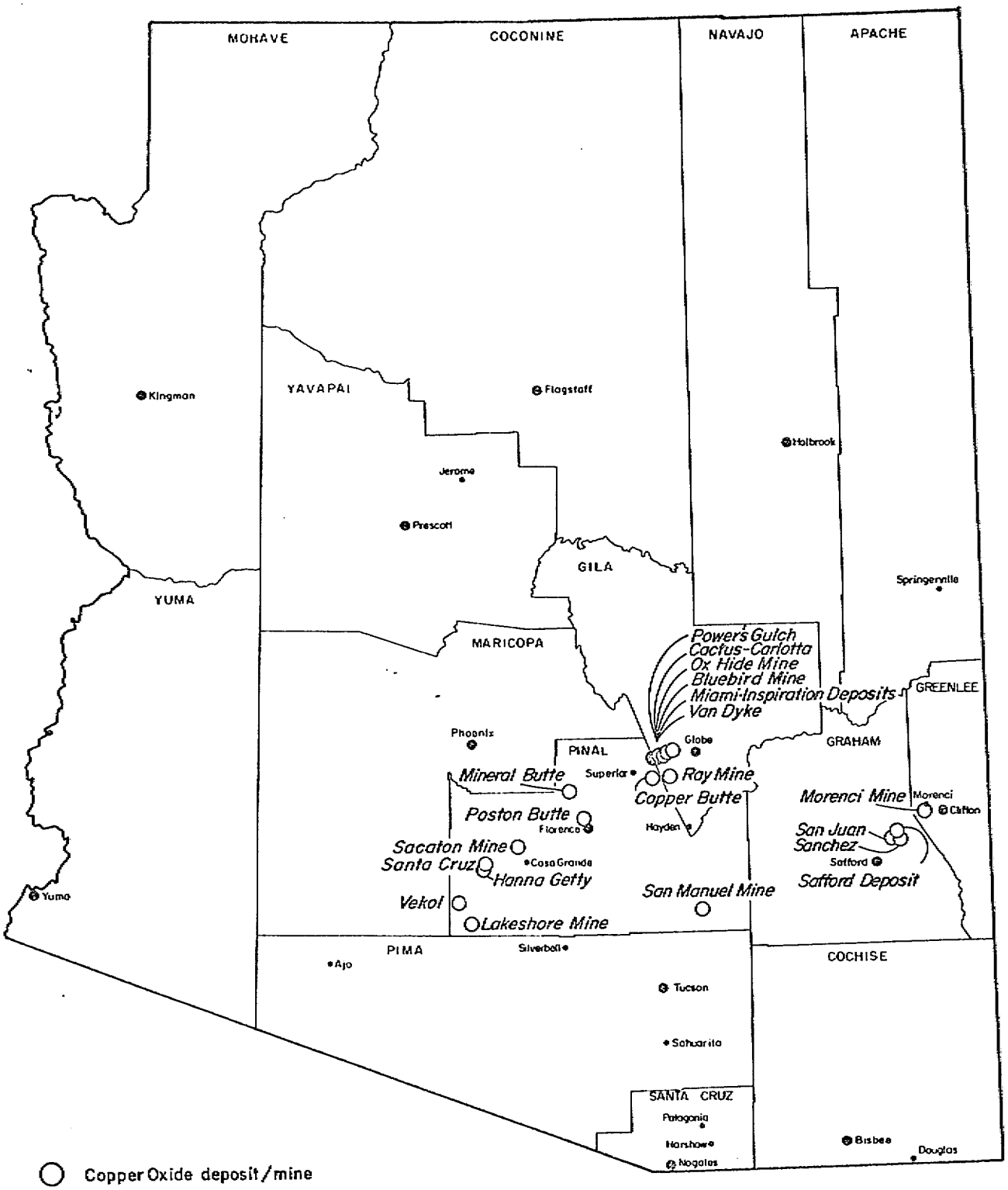
FROM: H. G. Kreis

Exploration-Exploitation of Copper Oxide Reserves

The history of copper production by leaching has been one of increased production, advancements in technology, and increased capital investments. Since the advent of open pit copper mining most open pit copper mines have incorporated heap leaching of low-grade "waste." In general these leach operations required very small capital investments and nearly all the mining costs were charged to ore reserves processed in a mill. In the last twenty years, however, leachable copper reserves have justified larger capital investments. Vat leaching operations, for example, have required considerable capital costs, and many copper mines have built or are seriously considering the use of solvent extraction-electrowin plants. Mines such as the Bluebird Mine and the OxHide Mine in Arizona have been brought into production solely as heap leaching operations.

The most successful variation from traditional heap-vat leaching practices is Cities Service's leaching of block caved ground at Miami, Arizona. Down-the-drill hole in-situ leaching has yet to be developed, but multi-million-dollar expenditures to pilot test this leaching method have been made in the last few years. Kennecott initiated a pilot test to leach primary copper sulfides at its Safford deposit, but abandoned the project earlier this year for reasons of corporate business rather than technological difficulties (rumored information). Occidental Minerals Corporation is attempting down-the-drill hole in-situ leaching of copper oxide reserves at Miami, Arizona. Occidental reported initial success and is now attempting a much larger pilot test (five drill holes). If Occidental is successful with their larger pilot test they will have accomplished a major technological achievement in the field of leaching.

There are major copper oxide reserves in Arizona, New Mexico, and Nevada. Large reserves of copper oxide mineralization occur in the Casa Grande-Miami-Morenci area of Arizona and include those reserves shown on the attached plan map and table. Those reserves having tonnage and grade figures in the attached table amount to 2 billion tons at 0.54% copper. About half this tonnage averages 0.6% copper and either is being mined or is thought to be reasonably committed to being mined under more favorable economic conditions. The other half of the tonnage appears much less favorable to mining in the near future, and of this amount 800 million tons at 0.4-0.5% copper are located in four separate deposits, all more than 1000' deep to the top of copper mineralization. These four deep copper oxide deposits are Santa Cruz, Poston Butte (West), Van Dyke, and Safford. Asarco's position in these four deposits amounts to ownership of a possible reserve amounting to 175 million tons at 0.46% copper at Poston Butte and a 50% undivided ownership of a probable-possible reserve of 200 million tons at 0.51% copper at Santa Cruz.



○ Copper Oxide deposit/mine

LOCATION MAP FOR MOST MAJOR
COPPER OXIDE RESERVES
IN
CASA GRANDE- MIAMI- MORENCI
AREA OF ARIZONA

H. Kreis Oct., 1978

scale 1" = 60 miles

ma 5037 dam

The economic value of these four deep copper oxide deposits lies in the development of suitable in-situ mining technique(s) and a reasonable copper price. In the period from 1971 to 1974 Occidental made comprehensive feasibility studies on its Van Dyke deposit (100-150 million tons at 0.4-0.5% copper). These studies were offered to many mining companies, including Asarco, for purposes of joint-venture discussions. These studies concluded that limited block cave mining (enough to generate a broken leach column) and subsequent leaching of the block caved ground could yield a favorable rate of return on a relatively small capital investment; furthermore, the studies concluded with a recommendation for an underground pilot leach test utilizing such a leach method. Apparently the unfavorable copper market has rendered this proposed mining technique uneconomic. Consequently Occidental has initiated a down-the-drill hole in-situ pilot leach test, as previously mentioned. If the technique is developed successfully, Occidental plans to sink a shaft and do the down-the-drill hole leaching from drifts below the Miami townsite. Asarco is currently evaluating the economics of down-the-drill hole in-situ leaching of the Santa Cruz deposit and the Poston Butte (West) deposit.

Most of the copper oxide reserves listed in the attached table are hosted by Precambrian Oracle granite with Cretaceous intrusive rocks or Precambrian Pinal schist with Cretaceous intrusive rocks. Copper oxide reserves at Lakeshore, Safford, and Sanchez are hosted by Cretaceous andesitic volcanics and Cretaceous intrusive rocks. Copper oxide reserves in altered Precambrian-Paleozoic sediments and altered diabase occur at Ray and Vekol. Almost all the copper oxide reserves listed in the table are thought to have resulted from the oxidation of either chalcopyrite or chalcocite depending on the particular deposit. Copper Butte, with copper oxide mineralization in Tertiary(?) conglomerate, is the only exotic copper oxide deposit. Slightly less than half of the listed copper oxide reserves exhibited explorationally significant copper oxide mineralization in outcrop, and virtually all the reserves are associated with evidence of alteration-mineralization in outcrop.

The listed copper oxide reserves were discovered either in the process of drilling for sulfide copper deposits or by following a few million tons of outcropping copper oxide mineralization. Future exploration for non-exotic copper oxide mineralization will be done similarly. However, when copper oxide mineralization is encountered it will be explored more aggressively than in the past. Companies that produce excess acid from their smelters have an added incentive to find copper oxide reserves.

There appears to be sufficient leachable copper reserves within the mining industry to justify the research and development of in-situ mining techniques. The advantage of in-situ leaching would be smaller capital investments, lower operating costs, and larger profit margins. Low copper prices and the risk of developing a mine on unproven mining techniques are obvious disadvantages. If profits are to be made by in-situ leaching in the future, those companies that will profit the most are those that presently hold the copper oxide reserves and have kept abreast of the advancement of in-situ leaching technology. For those companies holding major copper oxide reserves the risk capital involved in the research and development of in-situ leaching

might be comparable to risk capital in exploration for porphyry copper deposits: high risk, high total cost, numerous "failures," and an eventual discovery that will pay back the exploration-development costs and yield profits commensurate with the risk.

H. G. Kreis

H. G. Kreis

HGK:lb
Atts.

MOST MAJOR COPPER OXIDE RESERVES IN CASA GRANDE-MIAMI-MORENCI AREA OF ARIZONA

I think this/these needs revision

Name	Ownership	Tons (millions)	Grade (% Cu)	Reserve Classification	Dominant Cu Mineralogy	Major Host Rock Type	Depth (Ft.)	Thickness (Ft.)	Partial Reference
Vekol	Newmont	103	0.56	1978, probable		diabase, pЄ and Paleoz. sed.	<1000		14
Lakeshore Mine	Hecla-El Paso	207	0.71	1969, probable	chry.	and., porphyry	0-1000; 500 av.	550 av.	10
Santa Cruz	Asarco-Freeport	200	0.51	1978, probable and possible	chry., atacam., brochan.	Oracle granite	1200-2200	700 av.	7
Hanna-Getty Discovery	Hanna-Getty	No information available.							
Sacaton Mine	Asarco	20	1.0	1978, probable	brochan., cc.	Oracle granite	1000-1800		2
Mineral Butte	Gila River Ind. Res. (formerly Duval)	40	0.4	1978, rumored	chry., neotocite	Oracle granite	crops out		14
Poston Butte (East)	Conoco	200(+?)	0.4	1978, probable	chry.	Oracle granite	400-700	300	
Poston Butte (West)	Conoco	85	0.49	1978, probable	chry.	Oracle gr., granodio.	1200-2200	up to 800	8
Poston Butte (West)	Asarco	175	0.46	1978, possible	chry.	Oracle gr., granodio.	1200-2000	700 av.	8
Copper Butte	Kennecott	10-60		1978, rumored	CuOx	Tertiary(?) conglom.	crops out		
Pay Mine	Kennecott	156	0.9	1977, proven	chry.	schist, db., qtzite	crops out	areas >300	4
Power's Gulch	Inspiration	100	0.4	1978, speculative	CuOx	schist			17
Cactus-Carlotta	Cities Service-Private	40	0.7	1978, probable	chry., cc.	schist	200-700	100-500	6
Oxide Mine	Inspiration	35-55	0.4	1972, probable	chry.	schist, granite	crops out		16
Bluebird Mine	Rancher's	115	0.5	1974, probable	chry.	schist, granite	0-1000	500 av.	9
Van Dyke	Occidental Minerals	100	0.5	1978, probable	chry., az., mala.	schist	1000-2000	400 av.	12
Miami-Inspiration Deposits	Cities Service; Inspiration	substantial		pre-mining					18 11
Morenci Mine	Phelps Dodge	substantial		pre-mining					
San Manuel	Newmont	130	0.70	1969, probable	mixed, 0.47% as CuOx, chry.	Oracle gr., porphyry	300-1500	up to 1000	15
San Juan Mine	Producer's Chemical	5-15	0.8±	1978, possible	chry.	qtz. latite	crops out		3
Safford Deposit	Kennecott	250(+)	0.4	1978, probable	chry., brochan.	andesite	1000-2000 to top of CuOx	400 (av.?)	5
Sanchez	Inspiration	79	0.36	1978, proven	chry.	andesite, monz. porph.	0-1100 to top of CuOx	≈800	1 13

I think 130 M

H. Kreis
October, 1978

REFERENCES FOR ATTACHED TABLE

1) ASARCO files, circa 1965, United Nuclear(?) cross sections; ASARCO file Aa-7A.12.0.

2) Anzalone, S. A., 1973, personal communication.

3) Davis, S. R., 1978, personal communication.

4) Gambell, N. A., 1977, Geology and Mineralization of the Ray Silicate Orebody; AIME Mining Geology Division Meeting at Ray Mine, April 16, 1977.

5) Kennecott, U.S.A.E.C., U.S. Bureau of Mines, and Lawrence Radiation Laboratory, 1967, Sloop; a public report, 44 pages.

6) Kreis, H. G., circa 1973, private report.

7) _____, 1978, Possible Copper Reserve for Bulk Mining Consideration, The Lands, Santa Cruz Project; ASARCO report.

8) Kurtz, W. L., 1971, Evaluation of the 1970 Poston Butte Prospect, Pinal County, Arizona; ASARCO memo.

9) _____, 1974, Bluebird Mine; ASARCO memo.

10) Pay Dirt, 1969, Stockholders to Consider Development of Lakeshore Properties; Pay Dirt, Apr. 28, 1969, p. 6-8.

11) Peterson, N. P., 1962, Geology and Ore Deposits of the Globe-Miami District, Arizona; USGS Prof. Paper 342.

12) Potter, S., 1972, The Geology and Ore Deposit at the Bluebird Mine; Rancher's Exploration and Development Company application for mineral and millsite patents.

13) Seagart, W. E., 1971, Sanchez Copper Deposit; ASARCO memo.

14) Sell, J. D., 1972, Acid Consuming Copper Deposits, Arizona; ASARCO memo.

15) _____, 1972, Reserve Figures — Major Copper Deposits; ASARCO memo.

16) _____, 1972, Tour Notes Miami District, Gila County, Arizona; ASARCO memo.

17) _____, 1978, personal communication.

18) Zache, Bob, 1977, The Oxymin Project at Miami, Arizona; A summary of a presentation made to Miami Town Council, 9 pages.

ASARCO

~~Circulate to staff~~
General Files - Oxide Copper

Exploration Department

F. T. G.

AUG 1 1976

July 30, 1976

To: W.L. Kurtz, F.T. Graybeal
D.P. Cadwell, J.C. Balla,
J.H.G. Fuchter, D.M. Fletcher

EXOTIC COPPER DEPOSITS
(LITERATURE SEARCH)

Enclosed is Mr. Giesecke's brief comment on an article, "Copper Silicate Deposits of Northern Chile" with a copy of the summary attached. The author concludes that such deposits are restricted to arid or semiarid regions.

La Exotica, containing 150 million tons of 1.6% Cu, is perhaps the largest known occurrence of this type in South America. It is also unique in that two-thirds of the deposit is in bedrock beneath the gravels. Apparently, copper-bearing meteoric waters (derived from erosion-oxidation-leaching of the chalcocite enriched Chuqui ore body) moved down slope (2 kms.) into a basin where copper precipitated as silicate in pervious bedrock as well as in overlying gravels.

The source of the Sagasca (85 kms. east of Iquique) copper is not so readily evident. The nearest known porphyry copper occurrence, Cerro Colorado, is 25 kms. NE ---- and in another drainage system. The Sagasca deposit (12 million tons of 2.5% Cu) occurs in flat-lying conglomerates of probable Late Tertiary age, which have been exposed by erosion. In contrast, La Exotica was totally concealed. Reportedly it was discovered during the course of drilling for water.

Several exotic copper deposits are known in Arizona, such as Emerald Isle (near Ithaca Peak), Copper Butte (near Ray) and Superior East (west of Miami). All were derived from the weathering of porphyry copper deposits and no doubt there are others that remain to be discovered under covered areas. But, the search for them will involve rather high risk exploration in most cases.

Fred Graybeal has estimated (memo of 1-2-75) that the El Tiro-North Silver Bell zones have had some 4 billion tons of .20% Cu removed by erosion and that most of this copper, transported in solution and/or mechanically, could be contained in the north-striking graben lying (down slope) to the west.

Although to date the exotic deposits have not proven profitable producers (except for a few very small operations), they may become so in the future with their advantage over sulphide ores environmentally in conventional treatment. And, being in general relatively permeable, they may be more amenable to in-situ leaching. La Exotica (open pit) encountered metallurgical problems and is currently inoperative.

J. H. Courtright
J.H. Courtright

JHC:vh
cc: TCOsborne w/attach.
RBSprague
JJMerz

ASARCO

Madrid Office - Exploration Department

J. J. Merz
Director of Exploration

A. Giesecke
Chief Geologist

June 30, 1976

Literature search

Title: "Silikatische Kupferlagerstaetten in Nordchile."

Author: Gustav Roethe

Publication: Geologische Rundschau, 1975, Vol.64, N°2. p.421-456

Above article on Copper silicate deposits in northern Chile is of special interest to anybody dealing with secondary Copper deposits (Exotica type).

Three individual prospects are described:

1. Sagasca, Tarapacá Prov.
2. La Exotica, Antofagasta Prov.
3. Venado, Atacama Prov.

Geology, mineralogy (Chrysocolla, Copperpitch), and genesis are discussed. Microfotos, X-ray studies, and chemical composition are enclosed. There are geological maps and sections to show the general situation. Some data on grade and tonnage are given.

The article is in german but carries an english summary. Explorationwise, it is of definite importance to anybody assessing the mineral potential of "post-mineral" cover rocks.

A. Giesecke

x-c: R.B. Sprague
R.L. Brown
J.H. Courtright ✓

Abstract

The formation of isolated ~~silicaceous~~ silicate copper deposits is controlled by two main conditions:

1. oversaturation of groundwaters in silica
2. lateral inflow of copper bearing and slightly acid solutions originating from the oxidation of primary sulfide deposits.

The basic copper silicate chrysocolla is formed by chemisorption of copper ions by disilicic acid with a net-like structure. Under favourable conditions copper silicate deposits are formed which contain up to several million tons of copper as for example the La Exotica deposit in Northern Chile.

The copper values of these deposits sometimes exceed those of the primary sulfide deposits. Thus, the oxidation of sulfide deposits, the migration of copper bearing solutions and the chemisorption of copper ions by the disilicic acid can result in workable new copper concentrations originating from sulfide protores.

High concentrations of silica in groundwaters and thus the chemisorption of copper ions by the disilicic acid as an ore forming process is restricted to arid or semiarid regions. Therefore it is proposed to call such deposits a r i d s i l i c a t e o u s i n f i l t r a t i o n d e p o s i t s.

Infiltration deposits are related to weathering processes. "They comprise products of weathering by which valuable substances were leached from the host rocks, migrated in groundwaters and were redeposited in other neighbouring rocks" (Smirnov, 1970, p. 324).

Gen Ex

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

September 14, 1971

TO: W. L. Kurtz

FROM: J. C. Balla

OXIDE COPPER DEPOSITS

There are a number of oxide copper prospects in Arizona; some of which have been examined in detail, while others are still raw prospects. At the present time, I do not know the ownership situation on very many of them. Our belated interest in these types of deposits is unfortunate since several other companies have already surveyed them (Humble Oil & Refining Co. in 1969, Du Pont in 1971, Essex International in 1970-1971, to name a few).

The following list is in no special order since the potential and/or ownership is somewhat in doubt.

1. Copper Springs, south of Miami. This area was drilled by Kerr-McGee and subsequently Humble Oil (1969). Good potential but ownership uncertain.

2. Copper Butte, west of Ray. Mentioned in my memo of March 12, 1971, this deposit has been purchased by Kennecott. Reserves are uncertain, either 16,000,000 or 60,000,000 tons of leachable copper.

3. Southwest of Granite Peak mountain, Whetstone mountain. Granite Peak may represent the roots of a porphyry copper deposit which has been eroded away. Adjacent to it are a sequence of Cretaceous(?) sandstones which contain azurite and malachite, exposed in a dozer cut. The amount of copper in the prospect is unknown. The prospect has been rumored to be controlled by Humble Oil.

4.

4. Red Hills-Price prospect, east of Florence. This property is currently in production by Red Hills Mining Corp. We have looked at it a number of times, but have never adequately examined it and the apparent amount of oxide copper ore appears to continue to increase.

   It may well be the upper plate of a flat fault and the source of the mineralization has still not been found.

5. Red bed copper deposits of the Colorado Plateau. Thw White Mesa prospect is the best known but numerous other occurrences are known but have not yet been tested for size and grade.

6. South side Silver Reef Mountains, Papago reservation. An attempt to leach oxide copper occuring inaa fault zzone in post mineral volcanics was tried several years ago. The prospect may be more of interest as an indicator of a nearby porphyry copper deposit than as a leach caddidate.

7. Dragoon, Cochise County. Phelps Dodge drilled a series of altered limestone thrust slivers immediately west of the town of Dragoon. The limestones contain oxide copper. Superior Oil may currently own this prospect.

8. Courtland-Gleeson, Cochise County. There are several oxide copper deposits in limestone in these districts. Land ownership is unknown.

9. Planet-Swansea districts, Yuma County. These districts in northern Yuma County, along the Bill Williams River, contain copper in limestone. Ownership is unknown.

I think the best way to examine these prospects is to have a geologist assigned specifically to this program as opposed to having various people cover various parts of the state. By having one person doing it, that person can become more familiar with the geology and economics of these deposits than several geologists who only examine a couple or so deposits.

John C. Balla

JCB:sg

General Exploration

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

May 17, 1971

TO: C. K. Moss

FROM: R. Van Blaricom

Re: DECAY CURVE MEASUREMENTS FOR THE
SOUTHWEST EXPLORATION DEPARTMENT
GILA RIVER
MINERAL MOUNTAIN
WEST WATERMAN
CASCABEL PROJECT AREA C
CASCABEL PROJECT AREA B
POSTON BUTTE

INTRODUCTION

This report covers the decay curve measurements that were made for the Tucson office over several IP anomalies. Mr. Jon Kingman and Frank Blackham of the Salt Lake office took the actual field data.

SUMMARY AND RECOMMENDATIONS

Equipment

I would strongly recommend that this particular set of equipment be preserved for decay curve measurements. I would also recommend taking decay curve measurements as a routine evaluation of all IP anomalies.

Recommendations

I would recommend that further work be done on the Cascabel Area C and the Gila River area. These two areas have the characteristic curve of sulfides. The West Waterman, although its curve is quite high, it is much higher than that of Silver Bell. I would say the response is from syngenetic pyrite and carbonaceous material and is not of interest.

RESULTS

The equipment was calibrated at Silver Bell and Ryan Field. Comparing this data with previous surveys it was decided to adjust the readings so as to make them compatible with previous readings. To do this, take the NDC value and multiply it by 1.14.

Silver Bell

The areas tested gave an IP responce of 25 MV/V. There was little noise and the normalized decay curve (NDC) values were 24.6%. This will be

assumed to represent a valid value for the NDC of sulfide ore deposits.

Ryan Field

This value was also normalized to previous results and gave a NDC of 18.6%. This represents a clay response above the water table.

Mineral Mountain (Plate 1)

The NDC values as follows: 1000'a N=1, 19.9%, 100'a N=2, 18.66%, 2000'a N=1, 18.9%. This is definitely a clay response.

Cascabel Project Area B (Plate 2)

The NDC values are; 500'a N=1, 16.08%, 1000'a N=1, 18.18% and 2000'a N =1, 17.65%. This is also a clay response.

Cascabel Project Area C (Plate 3)

The NDC values are; 500'a N=1, 23.19%, 1000'a N=1, 23.65% and 2000'a N =1, 21.17%. This response is much higher than a normal clay response and is probably due to sulfides.

West Waterman (Plate 4)

The NDC values are 250'a N=1, 27.07%, 500'a N=1, 27.94% and 1000'a N =1, 26.86%. This is an extremely high response. From my knowledge of the area I would say the response is from syngenetic pyrite and carbonaceous material. The extremely high resistivity of the rock may have some effect on the response to make it so high. The response could also possibly be from sulfide mineralization.

Gila River (Plate 5)

The NDC values are 250'a N=1, 20.54%, 500'a N-1, 23.88% and 1000'a N=1, 23.31%. This response is probably from sulfides. The resistivity of the area is somewhat higher than would be expected from gravels. I would recommend further work be done on this anomaly.

Poston Butte (Plate 6)

The NDC values are 100'a N=1, 16.34%, 500'a N=1, 20.75% and 1000'a N=1, 22.01%. This is a good case history area showing the transition from a

characteristic clay curve at shallow depths to a curve that is more sulfide-like at depth.

Plate 7

This shows all of the NDC values. There is a transition zone from 20.5% to 21.5%; any value above 21.5% should be viewed as possibly representing a sulfide response.

R. Van Blarlcom

RVB:sh

cc: WESaegart
WGFarley

Gen. Expl

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

April 6, 1971

TO: W. L. Kurtz

FROM: S. J. Bellock

Re: OXIDE COPPER DEPOSITS

In reply to your March 4th memo concerning the consumption of sulfuric acid in oxide copper prospects, I briefly examined the Silver Hill deposit on the south side of the Waterman Mountains on the Papago Indian Reservation adjacent to the Silver Bell District.

The Silver Hill deposit is a shallow (maximum depth of mine workings approximately 400 feet) oxide deposit consisting of veins and stringers of copper carbonates and sulfates in limestones. Gangue is almost exclusively calcareous in nature, however, limited and occasional zones of skarn mineral assemblages do exist.

The mine workings consist of two main tunnels with associated adits and rooms connected by a 200 foot verticle shaft, that reaches bottom at the 400 foot level. In plan the workings cover an area approximately 150-200 yards by 100-150 yards.

The average grade of the deposit is low but certainly more than .25% (personal observation).

This deposit, not withstanding the problems involved in its location, may warrant further consideration as a source of economic disposal of sulfuric acid.

Other deposits in the area that may also deserve eventual consideration include the Arizona-Indiana, and Squabble deposits.

S. J. Bellock

SJB:sh

Gen. Expl.

AMERICAN SMELTING & REFINING COMPANY
Tucson Arizona

March 12, 1971

MEMORANDUM

TO: W. E. Saegart

FROM: John C. Balla

Oxide Copper Deposits
Worthy of Review

I agree with Kurtz that we should review all of the known oxide copper deposits to determine their attractiveness in light of the acid disposal problem.

Specific area of interest that I know of are the red bed copper deposits of the Colorado Plateau; the Copper Springs deposit south of Miami; a copper in sandstone deposit south of Granite Peak in the Whetstone's, and Copper Butte (along the Gila River east of Florence).

I suggest that a geologist be assigned specifically to this review program, as opposed to having various people cover various portions of the state.

John C. Balla

JCB:van

Gen Expl

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

March 8, 1971

TO: W. E. Saegart

FROM: J. D. Sell

Re: Oxide Copper Deposits

In response to your request on oxide copper deposits as suggested in the memo by W. L. Kurtz, three areas are known in the vicinity of the Superior East Project.

1. The Carlota area presently being drilled by Homestake Production Company. Reported figures are given as 8.6 million tons at + 1% nonsulfide copper. Estimated reserves may amount to 15 million tons (See Carlota Drilling memo of January 18, 1971). ASARCO has initiated contact with Homestake for logging and evaluating their new and previous drilling but to date the work has not been started.

2. The Powers Gulch area presently controlled by Inspiration has an unknown amount of nonsulfide copper which extends under the dacite to the west (Doug Friel will report on this in his memo). ASARCO will continue to test the westward extension and potential source area during the drilling program projected for Superior East.

3. Bellevue area, Section 21, T1S, R14E, Pinal Ranch Quadrangle, has a small heap leaching workings coming from several small areas. No further information is known at this time.

James D. Sell

JDS:mw
cc: W. L. Kurtz

2a - O.O. 19F

August 31, 1965

Mr. E. P. Cadwell
Treadwell Corporation
277 Park Avenue
New York, N. Y. 10017

Dear Pete:

I have been out of the country since the end of June so your letter of August 3 only came to my attention today.

Over the past years I have seen many of the copper sandstone deposits throughout the Southwest but don't have a thing in the form of samples.

I had heard that you left Occidental but did not know just where you moved. Dawn and I were in New York recently; I came up from Central America on business and she flew east to spend a couple of weeks. We are both leaving shortly for another Central American trip and I'll be gone a couple of months but Dawn will probably return somewhat earlier.

I have just noticed that the AIME Phoenix meeting will be in October so neither Dawn nor I will be in the States, and, of course, the same for the Mining Congress. So far this year I have been away over five months but Dawn did spend a few weeks in Dominican Republic earlier this year.

Sorry that I can't help on the oxide copper business.

Best wishes to all from both of us!

Sincerely,

L. P. Entwistle

LPE:bam

Aa-0.0.19F

TREADWELL CORPORATION
277 PARK AVENUE/NEW YORK, N. Y. 10017/826-2929

L. P. F.

AUG 31 1965

August 3, 1965

Mr. Lawson Entwhistle
American Smelting & Refining
Valley National Bank Building
Tucson, Arizona

Dear Lawson:

As you will note from the letterhead, I have changed jobs again and am now back in the New York area. The Occidental job went bad when they discovered that even though you drilled thirty or forty holes, none of them could be connected to a main and they could start producing. This was one oil company whose temperment was not suitable to the mining game. Unfortunately, I didn't find this out until after I had been with them about a year. I am now the Manager of Mining and Metallurgy for Treadwell Corporation and this is a real live-wire group--interested in manufacturing, engineering, research and development. We have a very nice laboratory and plenty of capable people with good ideas and background.

I have talked with Lyman Hart about the Red Beds of Arizona and New Mexico. He suggested that I should write to you and perhaps you would have a sample I could get for test purposes. We are not interested in entering the mining game as such, but in our company we have some new approaches to both leaching and precipitation. Our thought is that we would like to attack some of the ores from the famous Red Beds and see if this would lend itself to our processes. The Sagasca ore from Northern Chile lends itself very well, that is, the Chrisacolla dissolves and we are able to make a chemical precipitation and a subsequent copper product which compares favorably with electro-winning. If you have a sample of from five

TREADWELL CORPORATION

to ten pounds of this material which you could afford to let us have for a preliminary examination, we will be happy to accept it (Express, collect) and talk over the results with you upon completion. Incidentally, are you planning to attend the American Mining Congress in Las Vegas? If so, I'll plan on seeing you there. However, the thought occurs to me that you will, in all probability, attend the Northwestern Mineral Conference in Phoenix the week prior to the Mining Congress and I plan on being there also.

In the meantime, the best regards from the Cadwells to both you and Dawn. I remain,

Yours very truly,

TREADWELL CORPORATION

E. P. Cadwell, Manager
Mining and Metallurgy

EPC/alf

0.0.19F

T A.S

JUL 31 1958

July 31, 1958

A. C. H.

AUG 1 1958

Mr. H. P. Harman
226 S. W. 44th Street
Oklahoma City 9, Oklahoma

SANDSTONE COPPER PROSPECT

Dear Sir:

This will acknowledge your letter of July 15 to Mr. Welch. Your letter answers questions which Mr. Welch had asked you, at my instigation, in his letter of June 27.

We are very glad to have this information; however, I am sorry to advise you that this is not the type of mineral deposit which this Company would be interested in developing. We thank you kindly for bringing this to our attention.

Yours very truly,

Original Signed By
K. Richard

KENYON RICHARD

KR/ds
cc: RFWelch

226 S.W. 44th st
Okla. City Okla.
July-15-58

American Smelting & Refining co
810 Valley National Bldg.
Tucson arizona
Mr Reed F. Welch,

Richard this is for your reply.
Pete

Dear Mr Welch:

In reply to your letter dated June 27, 1958 regarding the mineralization of some oil specimen sent you. The area from which these samples was taken is about 200 or 300 ft wide about 1600 or 1800 long, and 6 ft thick.

The assays report which was sent to Mr Richard W. Ray of 1309 Hillcrest Road, Pauls Valley Okla. are as follows

From, American Smelting & Ref. co, 810 Valley National Bldg. Tucson arizona (date. 12-27-56)

	Silver oz-Ton	Copper oz
First Bot sample		
sample no 2	not run	2.16
" 3	" "	3.78
" 4	" "	1.54
" 5	" "	14.35
Second Bot		
No 2	0.4	1.66
" 3	.3	0.57
" 4	.4	2.75
" 5	.2	.96
" 9	.3	1.27

Yours Truly
H. P. Harman
226 S.W. 44 st
Okla City
Okla

Lucon Uranium Co
Sandstone Cr Prospect
Paul's Valley
Garvin County, Okla

RECEIVED
JUL 18 1958
REED F. WELCH

Tucson, Arizona

June 27, 1958

Mr. H. P. Harman
226 S.W. 44th St.
Oklahoma City 9, Okla.

Dear Mr. Harman:

This is in reply to your letter of June 23rd advising that you and Mr. James Pratt hold a lease on seventy acres of land showing mineralization similar to the small specimen samples you sent me.

Before assaying these samples it would be of interest to us to know what each represents in the way of thickness of mineralization and the area over which the samples were taken.

None of these samples represents material that could be shipped to the smelter. As for our possible interest in the property for copper recovery we would need to know more about the extent of the mineralization.

Yours very truly,
ORIGINAL SIGNED BY
REED F. WELCH

REED F. WELCH

bc:KERichard
RMMcGeorge

A2-C-C.19F

T. A. S.
APR 16 1958

J. H. C.
APR 14 1958

K. R.
APR 15 1958

L. S. H.
APR 17 1958

Denver, Colorado
April 10, 1958

MR. TAS ~~ACH~~ ~~RFW~~
READ AND RETURN
PREP ANSWER HANDLE
FILE

Mr. W. P. Hewitt
Western Mining Department
American Smelting and Refining Co.
600 Crandall Building
Salt Lake City, Utah

A. C. H.
APR 16 1958

SANDSTONE-COPPER DEPOSITS

Dear Sir:

Herewith is my report of an investigation of sandstone-copper deposits in New Mexico, Arizona, Utah, and Colorado.

The work was started in the winter of 1956-57, during our search for uranium deposits, with instructions to consider copper as well as uranium. The completion of the field work and the report were delayed by the inaccessibility of some areas during the winter months and by other assignments.

Although some areas known to contain sandstone-copper deposits have not yet been investigated, the more important ones were visited. No deposits that would be of interest to the Company were found.

Yours very truly,

R. K. Kirkpatrick

R. K. KIRKPATRICK

RKK:fb
CC: 1 extra with report to W.P.H.
K. Richard ✓ With report

REPORT ON

SANDSTONE-COPPER DEPOSITS

IN

NEW MEXICO, ARIZONA, UTAH, and COLORADO

R. K. Kirkpatrick
December, 1957.

INTRODUCTION

By a letter dated September 6, 1956, Mr. L. H. Hart assigned to me "a broad study of sandstone-copper deposits (considering both copper and uranium possibilities)". The work was to be undertaken during the winter months when field work in much of Colorado is generally not possible. However, some sandstone-copper deposits in Colorado and in northern New Mexico occur at higher altitudes and are inaccessible during winter months due to snow. Field work in New Mexico and Arizona was done in the winter of 1956-57; during 1957 several areas in Colorado that were known to contain copper in sandstone were visited. Work on the project was interrupted by other assignments during the spring and summer of 1957.

A review of the literature of sandstone-copper deposits was first made and from this study it became apparent that copper occurrences in sandstone, and allied sediments, are widespread in several states. Also apparent was the fact that many if not most of the deposits are unimportant and insignificant except from a scientific standpoint. In consultation Mr. Hart suggested that the examination should be confined to the larger deposits or more promising areas.

DISTRIBUTION

Sandstone or "Red Bed" copper deposits are widespread in the southwestern United States; deposits occur in New Mexico, Arizona, Colorado, Utah, Texas, Oklahoma, Wyoming, and Idaho. The most important deposits are found in the first four states and at the localities shown on the accompanying index maps.

PRODUCTION

The production of copper from deposits in sandstone in the southwestern United States has been relatively small. Although reasonably complete statistics are available for New Mexico, only a rough estimate of total production from all states can be made. From the states of New Mexico, Arizona, Utah and Colorado, I estimate that about 16,000 tons of copper has probably been produced from the sandstone-copper deposits.

Of the total, about 11,000 tons has come from New Mexico. I estimate that about 500,000 tons of crude ore has been produced from deposits in the four states.

Only two deposits, the Stauber mine near Santa Rosa, New Mexico, and the Big Indian mine near Moab, Utah, are believe to have produced more than 100,000 tons each. These two deposits appear to have yielded about half of the copper produced from the sandstone-copper type of deposits. Probably only five or six other mines in addition to the two mentioned above have produced more than 5,000 tons each.

URANIUM IN THE COPPER DEPOSITS

One of the purposes of the investigation was to look for uranium in association with copper. None of any consequence was found in areas or deposits that were examined during this investigation.

When the Happy Jack mine in White Canyon, Utah, which was first prospected as a sandstone-copper deposit, was found to contain rich uranium ore, the attention of uranium prospectors was naturally directed to other deposits of copper in sandstone. No doubt, in the late'40s and early '50s known deposits of sandstone-copper were carefully scrutinized for uranium more than once. Many of the deposits in sandstone that have been mined for uranium, of course, contain copper. However, most of the sandstone-copper deposits do not contain detectable uranium.

Because of the common association of uranium and copper in several of the commercial uranium deposits of the Colorado Plateau, a reconnaissance of copper disseminated in sandstone was made by the U. S. Geological Survey on behalf of the Atomic Energy Commission in 1951. The results of this investigation were reported in Geological Survey Circular 219.

GEOLOGY

Sandstone-copper, or "Red-Bed" Copper deposits as they have often been called, in the western

United States occur in beds that range in age from Upper Carboniferous to Cretaceous. The majority of the deposits are in formation of Permian or Triassic age that are commonly called "Red Beds". Although the general color of the rocks within which the copper deposits are found is reddish, the actual rock containing the copper is usually light-colored or buff and, consequently, the term "red-bed" is something of a misnomer.

The ore occurs in sandstone, arkose, conglomerate, and shale, and is commonly associated with carbonaceous trash or plant remains in the sediments. The beds are generally of the type that are considered to be continental origin and to have accumulated rapidly as shallow water or subaerial deposits. Sedimentation is generally irregular and crossbedding a common feature.

The deposits are typically tabular in shape and essentially parallel to the strata in which they occur; in detail they are usually irregular and spotty. In areas where more than one deposit occurs, the several deposits are generally in the same stratigraphic horizon. The deposits range in size from a few cubic feet of sandstone or shale containing a small amount of copper to the few, such as the Stauber deposit, that contained over 100,000 tons of ore.

The most abundant copper minerals in the deposits are malachite, chrysocolla, azurite and chalcocite. The most typical ore consists of disseminated specks of the oxidized copper minerals in sandstone. Undoubtedly, chalcocite was the primary copper mineral in practically all of the deposits. It occurs as discrete grains in the sandstone, as nodules, typically about one-half inch in diameter, and as a replacement of carbonized vegetal remains. Lead occurs in at least two sandstone-type deposits in appreciable quantities, the Warnock mine in the Sacramento district, New Mexico, and the Copper King mine, near McCoy, Colorado. The Warnock mine was mined principally for lead; the ore of the Copper King mine contains about 1.5 percent lead, which is about the same as the copper content. Lead may be more prevalent in sandstone-copper deposits than it is generally thought to be. The near surface ore is usually so oxidized that the lead-bearing mineral is not readily recognized

and probably little assaying of the ore has been done for lead.

The grade of the average sandstone-copper deposit is generally low when the material is considered in minable quantities, and is estimated to be well below one percent. The Stauber mine, which has probably been the most productive sandstone-copper, is estimated to have yielded about 200,000 tons of 3.5 percent ore. Selective mining or hand-sorting generally would be necessary to obtain a grade of three percent or better from most of the deposits that were examined.

SUMMARY AND CONCLUSIONS

An investigation of copper deposits in sandstone revealed no significant uranium mineralization in any of the deposits, other than in those already being mined or explored for uranium. After the discovery of associated copper and uranium in sandstone in the Happy Jack mine, the type of occurrence has been the subject of scrutiny for several years past.

No sandstone-copper deposits of a size and grade that would be of interest for operation or exploration by the Company were found. The average deposit is low-grade and insignificant from the standpoint of tonnage.

Many of the sandstone-copper ores are desirable as a siliceous flux at the copper smelters and it is for this use that the deposits have been of value in the past. However, most deposits can be worked economically only at times of high copper prices.

No important potential sources of fluxing ore were found during the investigation. The Stauber mine, in the past an important source of this ore for the El Paso smelter, is now depleted and no deposit to take the place of this source appears to be in sight. Deposits near Cuba, New Mexico, such as the Conglomerate and San Miguel mines, may be a source of ore again at high copper prices. The Copper King mine in Colorado may likewise furnish siliceous ore to Garfield at high copper prices.

The outlook for siliceous copper ore from the sandstone-copper deposits at normal copper prices is believed to be poor.

DESCRIPTIONS

Descriptions of some of the more important and relatively larger deposits follow.

R. K. Kirkpatrick

ARIZONA, COCONINO COUNTY
NAVAJO INDIAN RESERVATION
WHITE MESA DISTRICT

LOCATION

The White Mesa copper district, on the Navajo Indian Reservation in northern Arizona, is approximately 60 miles north of Cameron. A new highway from the Gap on U. S. Highway 89 to the Glen Canyon Dam passes through the district, which is about 25 miles north of the Gap; the Glen Canyon dam is estimated to be 20 to 25 north of the district.

Copper mineralization extends over an area which is about four miles long in an east-west direction and two miles wide in a north-south direction. The principal deposits are in the western part of the area.

GEOLOGY

The bedrock in the district is the Navajo sandstone of Jurassic age. The sandstone, which is eolean in origin, is several hundred feet thick, light-colored, strongly cross-bedded, and essentially horizontal. The topography is rolling with a few low hills, and a large part of the area is covered with a thick veneer of windblown sand, although knolls and some hillsides are bare rock.

A system of faults and sheeted zones that strike about N. 20 degrees W. in the western part of the district and a northeasterly striking joint system are the principal structural features. The northwesterly trending fractures are characterized by silicified breccia and silicified zones that commonly stand out as low ridges.

The most abundant copper minerals are malachite, chrysocolla (?), and chalcocite. Most of the copper appears to be in a chrysocolla-like material which replaces the cement of the sand grains, and may be a mixture of copper carbonates and silicates. Chalcocite, which is not common, generally occurs as residual nodules and specks. The

two most characteristic forms of mineralization are disseminated discrete grains of the oxidized copper minerals in the sandstone and a flooding of the cementing material of the sand with a bluish-green copper-bearing material. In detail the mineralization is usually spotty and erratic.

The ore of the district is generally associated with fractures but occurs in the adjoining and nearby sandstone rather than in the actual fractures. As a rule, the ore is not controlled by bedding like the ore of typical sandstone-copper deposits. No fissure-type ore was seen in the district. Most individual deposits are small and limited to an area of a few hundred square feet. The Copper World and Little Dick workings, the two largest in the district, are shown on the accompanying map, and illustrate the extent of two of the better deposits. Spots of fairly intense copper mineralization are widely distributed in the district but most of the individual deposits contain only an insignificant tonnage.

PRODUCTION AND EXPLORATION

Thomas V. Keams and companions prospected the region in 1882 and located 40 claims. Numerous examinations have been made of the district since then and several attempts made to produce ore or concentrates. The lack of water, low grade of the ore, and long haul (110 miles) to the railroad at Flagstaff have prevented a profitable mining operation.

The Navajo Copper Company in 1917-18 made an attempt to develop high-grade copper ore; the Coconino Copper and Chemical Company attempted to operate a leaching plant in 1939-40; and the Mardun Company attempted a dry concentration process in 1941. The Stark Lathing Company of Prescott, Arizona, was the last company to try to exploit the deposits. This company's rather elaborate dry concentration plant, erected in 1955-56, is still on the property. No record is available of the total copper produced from these various operations.

Shattuck-Denn Corporation, the Mardun Company, and Coronado Copper Zinc Company are among the companies that have conducted exploration and drilling on the properties.

ORE RESERVE

Cut samples and drill holes indicate that the average grade of material in the mineralized zones is in the neighborhood of 0.75 percent copper. Estimates of the assured and indicated tonnage range up to a maximum of 2.5 million tons for the district as a whole. Obviously, without the benefit of specific drill hole information, it is impossible to estimate the available tonnage. On the basis of exposures of ore in the various open-cuts and prospect pits, an estimate of 100,000 tons of indicated ore with a grade of less than one percent would seem to be a liberal estimate.

The ore of the district would be a desirable siliceous fluxing ore for the copper smelters, and by selective mining a grade in the range of 2 to 3 percent copper could probably be produced. However, the deposits are not presently economic for this purpose due to the distance to smelters.

COLORADO, CHAFFEE COUNTY
FUTURITY PROSPECT

LOCATION

The Futurity prospect is about 15 airline miles north of Salida, and is estimated to be in Section 13, T. 15 S., R. 77 W. Access is via a Forest Service road leading south from Highway 285 from a point about four miles west of Trout Creek Pass.

GEOLOGY

Copper mineralization occurs in sandstone and shale beds of the Pennsylvanian Weber formation that strike northwesterly and dip about 40 degrees to the northeast. Exposures in surface cuts show copper staining in locallized areas in the sediments, commonly in a 2 or 3-foot band; in material on a dump bornite is closely associated with fissuring in the sandstone and shale.

Although the Futurity prospect occurs in sediments and on casual inspection appears to be a typical sandstone-copper-type of deposit, there is clear evidence that copper sulfide was deposited in fissures that in places are closely parallel to the bedding.

PRODUCTION

An inaccessible, inclined shaft, the dump of which indicates at least 200 or 300 feet of work is the principal working. No record of production is known, but it is possible that a small tonnage of sorted or selected ore may have been shipped in the past.

The small amount of copper showing in surface exposures and dumps indicates that no future production can be expected. A few tons of 1 to 2% copper could be sorted from the dump.

COLORADO, EAGLE COUNTY
McCOY AREA
COPPER KING MINE

LOCATION

The Copper King mine is near the north boundary of Eagle County, about two miles east of the town of McCoy, and is estimated to be in Section 33, T. 1 S., R. 83 W. A spur of the Denver and Rio Grande Railroad is only two or three miles from the mine, a factor which has been of some aid in shipping the low-grade ore of the mine.

GEOLOGY

The mine is in rolling hills formed on a thick section of Pennsylvanian "Red Beds", consisting principally of arkosic sandstone and conglomerate with a moderate southeasterly dip. The ore deposit is in a bedded, light-colored, medium-grained sandstone about 10 feet thick; individual beds vary from a few inches to a few feet thick.

About equal amounts of lead and copper (1.4 and 1.3%) were contained in the last ore shipped from the mine. However, no lead minerals were recognized in megascopic examination of the ore and it is likely that the lead is present as oxidized minerals that are more or less masked by the general oxidation of the deposit.

Oxidized copper minerals occur in streaks and disseminated in the sandstone. The principal mineralization is concentrated in a zone 2 to 4 feet thick but copper minerals have flooded into adjacent sandstone for a greater, but variable, thickness. The deposit, as presently developed by stoping, is a narrow manto extending from the outcrop down the dip for about 400 feet. Mineralization is present on the edges of most of the workings but presumably the best part of the ore body was followed with the stoping.

PRODUCTION

Full production records are not available, but the total ore shipped from the mine is estimated to have been less than 6000 tons.

In 1956, the following tonnage of ore was shipped to the Garfield smelter:

Tons	Oz. Au	Oz. Ag	% Pb	% Cu	% SiO2
4146	.005	0.32	1.4	1.3	88.5-89.0

ORE POTENTIAL

The ore at the Copper King mine is not exhausted and future production under high copper prices can be expected. The ore could be obtained by extending the stopes laterally, probably with a diminishing grade and thickness, and by deeper exploration of the ore-shoot. A tonnage of ore comparable to that already produced might be a reasonable expectation for the mine.

COLORADO, FREMONT COUNTY
DEPOSITS NEAR HOWARD

LOCATION

Copper deposits in sediments occur in western Fremont County about four miles southeast of the small town of Howard. The deposits are north of the Rio Grande River and in the western part of T. 48 N., R. 11 E.

GEOLOGY

A section of Permian red-beds, consisting of shale, sandstone, arkose and conglomerate, possibly as much as several thousand feet thick exists in the area. In the vicinity of the deposits the beds strike northeasterly and dip steeply to the northwest. The copper is in a fine-grained silty arkosic bed three to five feet thick which is enclosed in coarser arkosic sediments. Chalcocite and malachite are the principal copper-bearing minerals. Mineralization on the outcrop is spotty and discontinuous for a length of at least 1500 feet along the strike of the bed. Prospecting has been by shallow open-cuts and short shafts. The grade of the 2-4 foot bed, where mineralization is most intense is about 1.5 percent copper.

ORE POTENTIAL

The deposits are too small and low grade to be profitably mined except by close hand-sorting, and, consequently, the area is not considered to have a reserve of any importance.

COLORADO, FREMONT COUNTY
RED GULCH AREA

LOCATION

The Red Gulch area is about nine miles due north of the small town of Cotopaxi on U. S. Highway No. 5. The deposits are in Section 13, T. 49 N., R. 11 E. Although the area is mentioned in the literature on red-bed copper deposits, the mineralization evident at the present time is insignificant.

GEOLOGY

Along a major north-south fault of the region the block on the west side has been dropped relative to the block on east side of the fault. In the vicinity of Red Gulch upper Paleozoic red beds are faulted against lower Paleozoic limestones - as shown on the accompanying sketch. A thick section of Permian red-beds, consisting of reddish conglomerate, arkose and shale dips easterly toward the fault.

A bed of conglomerate four to five feet thick that outcrops about 2000 feet west of the fault contains, at the outcrop, scanty patches of oxidized copper minerals. The copper content is very low in all of the exposures and careful sorting would be necessary to produce even a few tons of ore-grade material.

The relationships in the area (see map) suggest that the mineralizing agencies are probably closely related to the fault.

PRODUCTION

Several carloads of ore are reported to have been produced from the shafts near the fault, but no substantiating records are available.

No future production can be expected from the area on the basis of the evidence exposed at the surface.

NEW MEXICO, GUADALUPE COUNTY
STAUBER MINE

LOCATION

The Stauber mine in northeastern New Mexico, in Section 6, T. 7 N., R. 20 E., is about 12 miles southwest of Santa Rosa. The mine is 2 miles from a loading dock on the Southern Pacific Railroad, a factor of prime importance in enabling operators to ship relatively low-grade ore to the El Paso smelter.

GEOLOGY

The ore at the Stauber mine is in the generally flat-lying Santa Rosa sandstone member of the Dockum formation of Triassic age. The sandstone is fine to medium grained, crossbedded, and relatively clean. The mineralized part of the sandstone that has been mined is reported to have ranged from 2 to 17 feet thick. The ore bearing sandstone is underlain by a mudstone, and is overlain in part by sandstone and in part by mudstone. Practically all of the ore deposit has now been strip-mined and waste stacked in the area formerly occupied by the deposit; consequently, the detailed geology of the deposit is missing or obscured. Various reports and publications on the mine are available.

PRODUCTION

The Stauber mine has had the largest production of any sandstone-copper in New Mexico. Appreciable production started in 1925 and has continued sporadically until the present time. The mine has been an important source of fluxing ore for the El Paso smelter.

The deposit was first mined by underground methods, but during the last year the 5 to 40 feet of overburden has been stripped from the deposit to recover the pillars and unmined remnants of ore. The underground workings extended over an area about 350 by 850 feet in size.

The grade of ore that has been shipped from the mine has ranged from one to five percent. During 1956, about 24,000 tons of 2.6% copper ore was mined in the stripping operation. I estimate that the total production has

been about 13 million pounds of copper from 150,000 to 200,000 tons of ore.

ORE RESERVE

The ore deposit of the Stauber mine is largely depleted. The current operator, Q. M. Drunzer, estimates only a few thousand tons of ore remain to be mined by stripping. Drilling in the past to find an extension or other ore bodies in the area has not been successful. Windblown sand covers the surrounding surface and the remains of surrounding drill holes. It is reported that the area immediately surrounding the deposit has been rather extensively drilled.

Note: Previous reports on the mine are in the files at the Tucson office.

NEW MEXICO, RIO ARRIBA COUNTY
CONGLOMERATE MINE

LOCATION

The conglomerate mine is about 6 miles due east of the town of Cuba in northern New Mexico. It is in Section 32, T. 21 N., R. 1 E., about one mile north of State road No. 126.

GEOLOGY

The Poleo sandstone of Triassic age, estimated to be about 75 feet thick, forms a rim at the top of a canyon in which the mine occurs. The dip is about 10 degrees westerly in the vicinity of the mine. The Poleo sandstone is generally light-colored, medium-grained, and crossbedded. At the mine, a conglomeratic sandstone lens with pebbles up to 1 inch in diameter forms the lower part of the Poleo formation. The conglomerate has a maximum thickness of 15 feet and a length of 300 to 400 feet at the outcrop of the sandstone bed. Most of the copper mineralization is in the lower part of the conglomerate but copper also occurs in non-conglomeratic parts of the sandstone. Chalcocite and copper carbonates are the ore minerals.

At this mine the conglomerate appears to have been important in locallizing the copper mineralization. Carbonaceous trash in the sandstone also appears to have had some bearing in locallizing the ore. A few pieces of high grade chalcocite that replace woody material have been found in the deposit.

PRODUCTION

The Conglomerate deposit was opened by an adit some years ago but the principal production from the property has been since R. D. Morton started mining at the beginning of 1956. In the first eleven months of that year, 9,860 tons that averaged 2.3% copper and about 83% silica were shipped to the El Paso smelter.

ORE RESERVE

I estimate that about 8,000 to 10,000 tons of ore, comparable in grade to that produced, is indicated by the present mine workings. There is a good possibility that

additional ore will be found by an extension of the ore shoot into unexplored ground.

NEW MEXICO, OTERO COUNTY
SACRAMENTO DISTRICT

LOCATION

The deposits are in a north-south belt on the west side of the Sacramento Mountains, about 7 miles east of Alamagordo. They are reached by proceeding easterly from Alamagordo on State Highway 83 for 9 miles to High Rolls, and thence south from High Rolls on a Forest Service road. Deposits are from 1.5 to 10 miles south of High Rolls. They are in the western part of T. 16 N., R. 11 E.

GEOLOGY

The Abo formation, dipping east at about 10 degrees, outcrops as a belt on the west side of the Sacramento Mountains. The Forest Service road, mentioned above, more or less follows the base of the formation in which the sandstone copper prospects are located.

The copper mineralization occurs in arkose, and sandy shale as nodules of chalcocite that have been oxidized to copper carbonates near the surface.

WORKINGS

Workings consist of numerous small pits, generally with insignificant showing of copper mineralization, and several larger workings shown on the accompanying map.

The Adycopt mine, 10 miles south of High Rolls by road, has an incline, not entered due to bad ground condition, reported to be 1200 feet long. The dump from the incline is nearly barren.

The best known mine of the district is the Warnock (or Holmes) mine containing sandstone-lead ore. This property was examined for ASARCO by B. N. Webber in 1948. He reported in detail on the mine after 33 days of field work.

ORE POTENTIAL

Small shipments of sandstone-copper

ore will probably be made from the district from time to time but production will probably be insignificant. Sorting will be necessary to produce even a low grade of ore. The arkose in which mineralization occurs does not make the ore desirable as a flux.

NEW MEXICO, OTERO COUNTY
TULAROSA AREA

LOCATION

Deposits are approximately 5 miles northeast of Tularosa in the southern part of T. 13 S., R. 10 E., in the drainage of Coyote Creek. They are reached by going north on U. S. Highway 54 about 1.5 miles from Tularosa, thence turning on a dirt road just before the railroad overpass, and traveling northeast for 4 miles. Topography is rolling hills.

GEOLOGY

The Abo formation dips 10 degrees to the northeast. Several basic dikes, 1 to 10 feet wide, vertical, and trending northeasterly traverse the area. The dikes have indurated the sediments very slightly at their contacts but have otherwise produced no alteration. No evident copper mineralization is in or associated with the dikes. However, the presence of copper in sediments at various stratigraphic intervals in the general vicinity of the dikes suggests the possibility of a genetic relationship. Copper showings in the area consist of small quantities of malachite and azurite in shaly sandstone.

WORKINGS

In a distance of 1.5 miles along Coyote Creek are two caved adits and seven or eight shallow open pits. Most of the prospects are practically barren. One pit of recent origin has an exposure of 2 feet of black gypsiferous shale with malachite staining that might carry as much as 1% copper.

The deposits of the district are insignificant and unimportant. No production should be expected from the area.

NEW MEXICO, OTERO COUNTY
DEPOSIT NEAR BENT
(TULAROSA COPPER COMPANY)

LOCATION

The mine is about one mile southeast of Bent, a post office on Highway 70 about 10 miles east of Tularosa, in Section 26, T. 13 S., R. 10 E. It can be reached by crossing the river at Bent and following a dirt road for about one mile up the south bank of the river.

GEOLOGY

The copper deposit near Bent is principally in an intrusive diorite porphyry although some copper mineralization is present in overlying shaly sandstone. A thick mass of diorite porphyry, probably a sill, has intruded sediments of the Chupadera formation. The base of the diorite is not exposed in the vicinity of the mine. Steep, northeasterly-striking stringers of barite-dolomite containing chalcocite occur in the diorite but do not continue into the overlying sediments. The stringers are up to one inch wide and are bordered by weak alteration. A shaly sandstone overlying the diorite has been chloritized. Sparse oxidized copper minerals occur in the overlying sediments. The deposit is unquestionably hydrothermal in origin.

WORKING

The principal accessible working is an open pit about 60 feet in diameter and 30 feet deep. Underground workings which are not safely accessible extend from the pit. A large dump of diorite indicates that possibly a 1000 feet underground workings may be present. Apparently production from the mine was obtained by handsorting the stringers from the barren rock.

ORE POTENTIAL

The chalcocite stringers are too narrow and widely spaced to make ore of the general mass of diorite. Although handsorting might produce some copper ore, the potential of the deposit is considered practically nil.

NEW MEXICO, SOCORRO COUNTY
RAYO DISTRICT

LOCATION

The deposits of the Rayo district, about 8 miles southwest of the town of Scholle on U. S. Highway 60, are in the western part of T. 1 N., R. 4 E. Two areas of prospects are about two miles apart in a north-south direction.

GEOLOGY

The copper occurs in a light-colored, cross-bedded, medium-grained sandstone near the top of the Abo formation of Permian age. The beds in the area are flat-lying and the surface generally covered with windblown sand.

The copper in the sandstone occurs in chalcocite and malachite. At the north end of the district only sparsely disseminated malachite is present in prospect pits; at the south end of the district in recently mined shallow pits of the Meader Corporation, chalcocite and malachite both are present. Most of the chalcocite occurs as nodules up to a half inch in size but in one of the pits a layer of nearly solid chalcocite about one inch thick is present over an area of a few square feet. Mineralization generally extends through a thickness of 3 or 4 feet of sandstone but is spotty and locallized as shown on the accompanying map.

WORKINGS

At the north end of the area are the Parker-Doyle prospects where work consists of a few shallow pits scattered over a wide area. The largest pit is about 75 by 15 feet and six feet deep. No copper mineralization that would approach one percent in grade was seen in any of the pits. The maximum copper content in 29 wagon drill holes of the Bureau of Mines was reported to be 0.37 percent.

The Meader Corporation pits at the south end of the district from which ore was mined and trucked to the Meader plant near Scholle are shown on the attached map. Possibly 300 or 400 tons of ore that might average between 1 and 2 percent copper was mined from the pits in the latter part of 1956.

ORE POTENTIAL

The ore potential of the Rayo district is believed to be very low. The ore would be a desirable siliceous flux but the deposits are small, relatively low grade, and too far from copper smelters to be profitably shipped. By sorting, a few tens of tons of ore that would probably support the cost of freight, could be produced from the Meader pits.

NEW MEXICO, VALENCIA COUNTY
SCHOLLE DISTRICT

LOCATION

The Scholle district is in southeastern Valencio County about 60 miles southeast of Albuquerque. Prospects of the area are accessible by dirt roads leading from the town of Scholle on U. S. Highway 60. Most of the prospects are north of the highway.

GEOLOGY

The deposits occur in the lower part of the Abo formation not far above the Magdalena limestone. The beds which contain copper are shaly sandstone, shale, and arkose. The original mineralized bed appears to have been a shaly sandstone a few inches thick in which disseminated chalcocite occurs in irregular patches and lenses. Individual deposits are small and range from a few feet to a few tens of feet in lateral dimensions. The copper in most of the workings is now present in oxidized minerals.

WORKINGS

Numerous small workings and pits are present along the outcrop of the mineralized bed for a distance of about 5 miles north of Scholle. The workings show that the copper mineralization is spotty and lacks continuity.

Meader Corporation. Along the highway one-half mile west of Scholle the Meader Corporation (of Houston, Texas) has built a plant to treat copper ores of the area. Equipment consists of a portable crusher, and two flat concrete tanks. An attempt is being made to produce copper by leaching the ore with acid and precipitating the copper on tin cans. Several hundred tons of ore from deposits in the Scholle and Rayo districts had been stockpiled at the mill site in December, 1956, but no copper precipitate profitably produced. The plant is reported to have closed shortly thereafter.

UTAH, SAN JUAN COUNTY
BIG INDIAN COPPER MINE

LOCATION

The Big Indian Copper mine, in Section 27, T. 29 S., R. 24 E., northern San Juan County, is about 35 miles southeast of Moab and on the east side of Big Indian Wash. The district is, of course, now famous for the large deposits of uranium that have been found in recent years.

GEOLOGY

In the Big Indian district the sequence of formations ranging from the Pennsylvanian to the Cretaceous that is common to most of the Colorado Plateau is present. The beds are folded into the Lisbon Valley anticline with a northwest trending axis and faulted by the Lisbon Valley fault which is roughly along the axis of the anticline. The beds northeast of the fault have been dropped about 2000 feet in relation to those southwest of the fault in the vicinity of the mine where the Dakota and Morrison formations northeast of the fault are against the Cutler formation southwest of the fault.

The copper ore of the Big Indian mine is in a flat-lying sandstone bed about 15 to 20 feet thick and lies a few hundred feet northeast of the fault. The ore zone has been removed by open-pit mining over an area about 400 feet in diameter (see map); only sparsely disseminated malachite and azurite are present in the sandstone at the edge open-cut. All of the sandstone now exposed in the pit is very low grade and will probably not average over one or two tenths of a percent copper. Presumably the principal part of the deposit which was treated in a mill contained several percent copper.

PRODUCTION

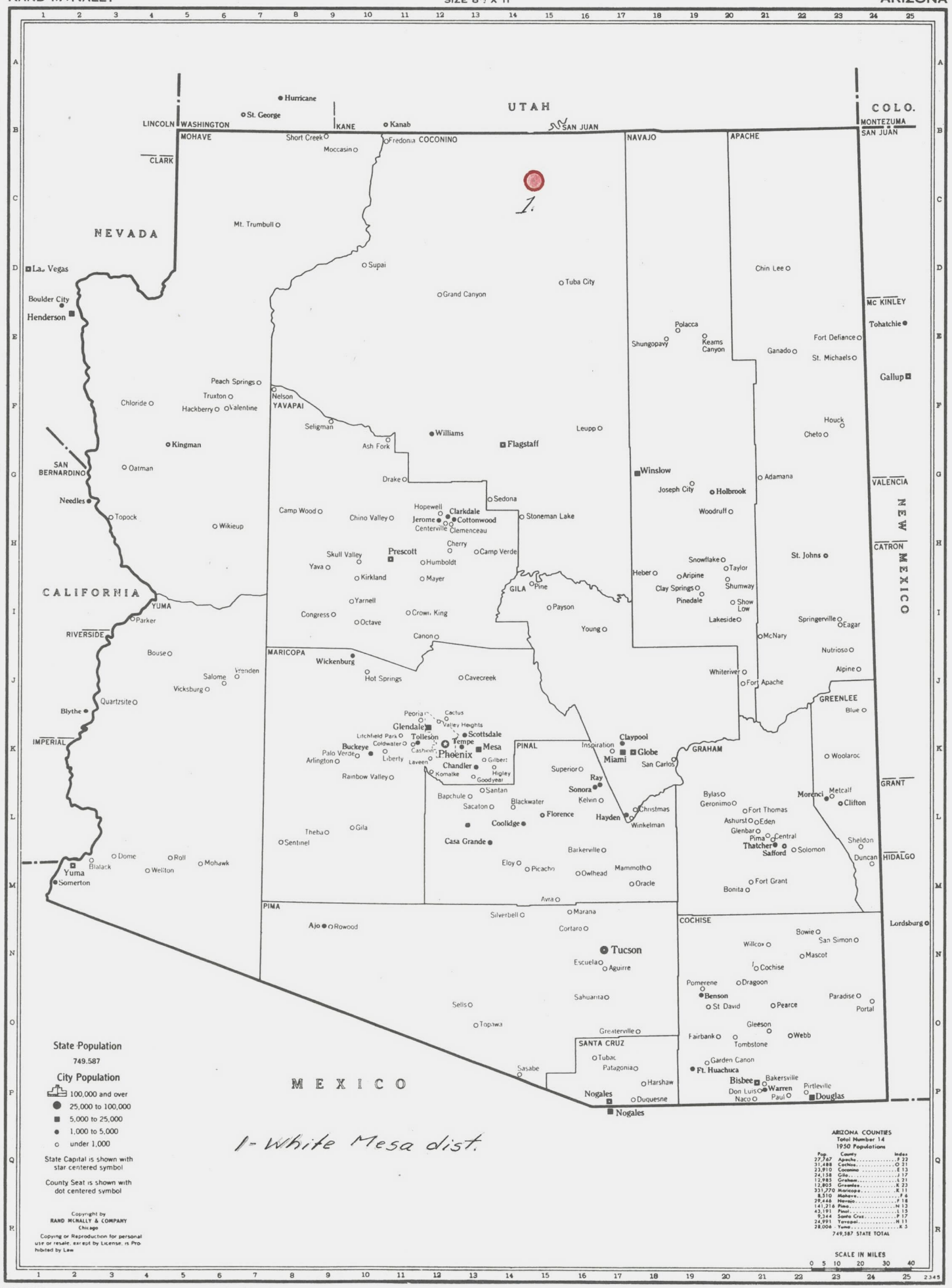
No record of the production of the mine is presently available to me. A production in the range of 100,000 to 150,000 tons is estimated from the size of the open pit.

ORE POTENTIAL

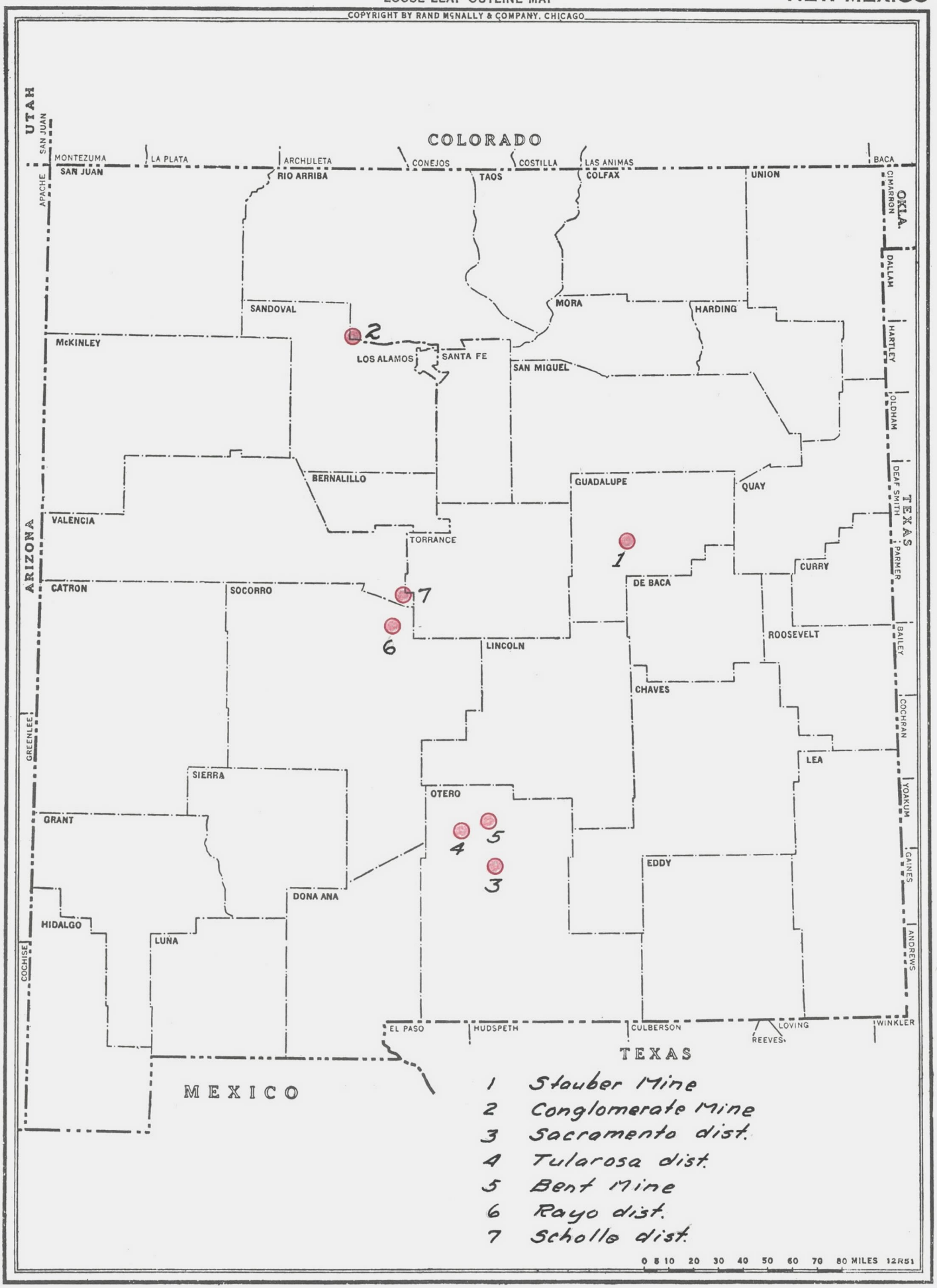
Drilling in recent years has failed to find any ore of commercial grade on the property. However, I understand that some copper mineralization in the range of a few tenths of a percent copper has been found in drill holes.

INDEX

	Page	Figure
Introduction	1	
Distribution	1	
Production	1	
Uranium in the Copper Deposits	2	
Geology	2	
Summary and Conclusion	4	
Descriptions of Individual Deposts or Areas		
White Mesa district, Arizona	6	1, 2
Futurity prospect, Colorado	9	3
Copper King mine, Colorado	10	4
Deposits near Howard, Colorado	12	5
Red Gulch area, Colorado	13	5A
Stauber mine, New Mexico	14	6
Conglomerate Mine, New Mexico	16	7, 8
Sacramento district, New Mexico	18	9
Tularosa area, New Mexico	20	-
Bent deposit, New Mexico	21	10
Rayo district, New Mexico	22	11
Scholle district, New Mexico	24	-
Big Indian Copper mine, Utah	25	12
State Index Maps, following	26	

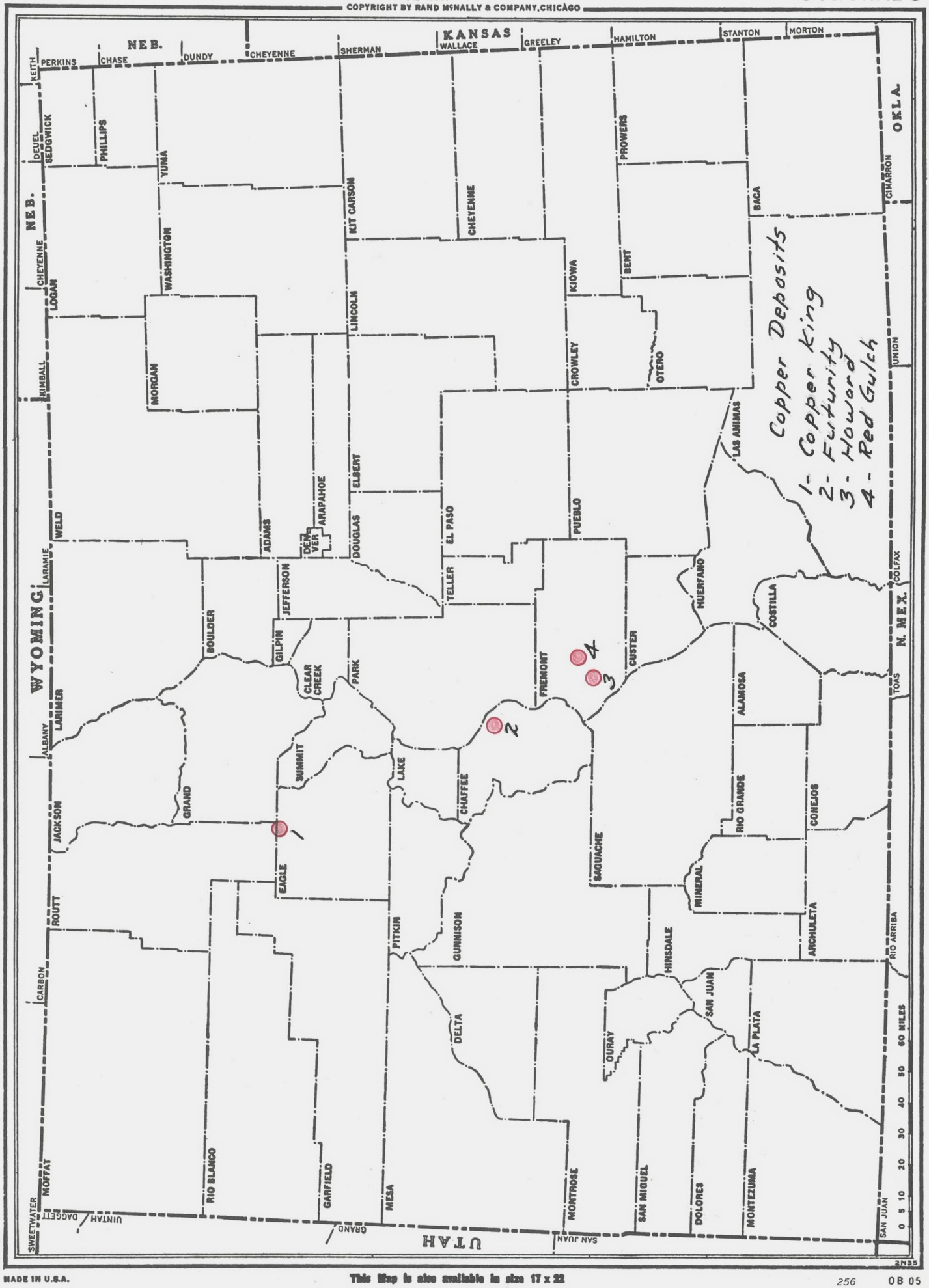

UTAH
COLO.
NEVADA
CALIFORNIA
NEW MEXICO
MEXICO
LINCOLN
WASHINGTON
KANE
SAN JUAN
MONTEZUMA
MOHAVE
COCONINO
NAVAJO
APACHE
CLARK
MC KINLEY
YAVAPAI
SAN BERNARDINO
VALENCIA
CATRON
YUMA
RIVERSIDE
MARICOPA
GREENLEE
IMPERIAL
PINAL
GRAHAM
GRANT
HIDALGO
PIMA
COCHISE
SANTA CRUZ
Hurricane
St. George
Kanab
Short Creek
Moccasin
Fredonia
Mt. Trumbull
La. Vegas
Boulder City
Henderson
Supai
Grand Canyon
Tuba City
Chin Lee
Polacca
Shungopavy
Keams Canyon
Ganado
Fort Defiance
St. Michaels
Tohatchie
Gallup
Peach Springs
Truxton
Hackberry
Valentine
Chloride
Nelson
Seligman
Williams
Flagstaff
Leupp
Houck
Cheto
Kingman
Oatman
Ash Fork
Drake
Winslow
Joseph City
Holbrook
Adamana
Needles
Topock
Wikieup
Camp Wood
Chino Valley
Hopewell
Jerome
Clarkdale
Cottonwood
Centerville
Clemenceau
Sedona
Stoneman Lake
Woodruff
Cherry
Camp Verde
Skull Valley
Prescott
Humboldt
Yava
Kirkland
Mayer
Snowflake
Taylor
Heber
Aripine
Clay Springs
Shumway
Pinedale
Show Low
St. Johns
Pine
GILA
Payson
Yarnell
Congress
Crown King
Octave
Lakeside
Springerville
Eagar
Parker
Canon
Young
McNary
Nutrioso
Bouse
Wickenburg
Hot Springs
Cavecreek
Whiteriver
Alpine
Salome
Vicksburg
Vrenden
Fort Apache
Quartzsite
Blythe
Peoria
Cactus
Glendale
Valley Heights
Litchfield Park
Tolleson
Scottsdale
Coldwater
Buckeye
Tempe
Mesa
Palo Verde
Cashion
Phoenix
Arlington
Liberty
Laveen
Chandler
Gilbert
Komatke
Higley
Goodyear
Rainbow Valley
Santan
Bapchule
Sacaton
Inspiration
Claypool
Globe
Miami
San Carlos
Superior
Ray
Sonora
Kelvin
Blackwater
Florence
Hayden
Christmas
Winkelman
Coolidge
Casa Grande
Theba
Gila
Sentinel
Barkerville
Eloy
Picacho
Owlhead
Mammoth
Oracle
Avra
Woolaroc
Bylas
Geronimo
Fort Thomas
Ashurst
Eden
Glenbar
Pima
Central
Thatcher
Safford
Solomon
Morenci
Metcalf
Clifton
Sheldon
Duncan
Fort Grant
Bonita
Dome
Roll
Mohawk
Blaiack
Weliton
Yuma
Somerton
Silverbell
Marana
Cortaro
Ajo
Rowood
Tucson
Escuela
Aguirre
Sahuarita
Sells
Topawa
Greaterville
Tubac
Patagonia
Sasabe
Harshaw
Nogales
Duquesne
Bowie
San Simon
Willcox
Mascot
Cochise
Pomerene
Dragoon
Benson
St. David
Pearce
Paradise
Portal
Gleeson
Fairbank
Webb
Tombstone
Garden Canon
Ft. Huachuca
Bisbee
Bakersville
Warren
Pirtleville
Don Luis
Naco
Paul
Douglas
Lordsburg
1.
1- White Mesa dist.
State Population
749,587
City Population
100,000 and over
25,000 to 100,000
5,000 to 25,000
1,000 to 5,000
under 1,000
State Capital is shown with star centered symbol
County Seat is shown with dot centered symbol
Copyright by RAND McNALLY & COMPANY Chicago
Copying or Reproduction for personal use or resale, except by License, is Prohibited by Law
ARIZONA COUNTIES
Total Number 14
1950 Populations
Pop. County Index
27,767 Apache F 22
31,488 Cochise O 21
23,910 Coconino E 13
24,158 Gila J 17
12,985 Graham L 21
12,803 Greenlee K 23
331,770 Maricopa K 11
8,510 Mohave F 6
29,446 Navajo F 18
141,216 Pima N 13
43,191 Pinal L 15
9,344 Santa Cruz P 17
24,991 Yavapai H 11
28,006 Yuma K 5
749,587 STATE TOTAL
SCALE IN MILES
0 5 10 20 30 40

COPYRIGHT BY RAND McNALLY & COMPANY, CHICAGO

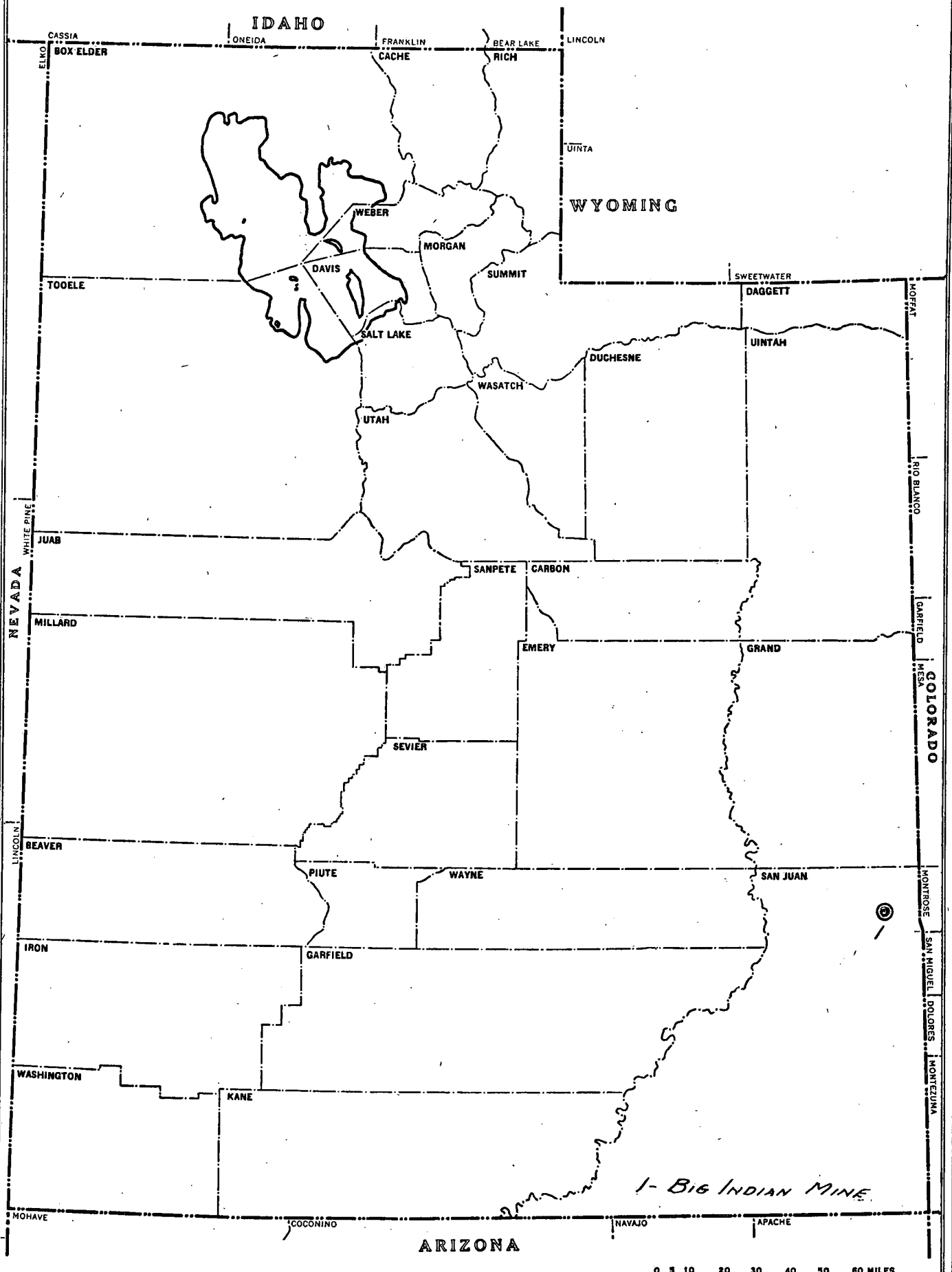


1 Stauber Mine
2 Conglomerate Mine
3 Sacramento dist.
4 Tularosa dist.
5 Bent Mine
6 Rayo dist.
7 Scholle dist.

COPYRIGHT BY RAND McNALLY & COMPANY, CHICAGO



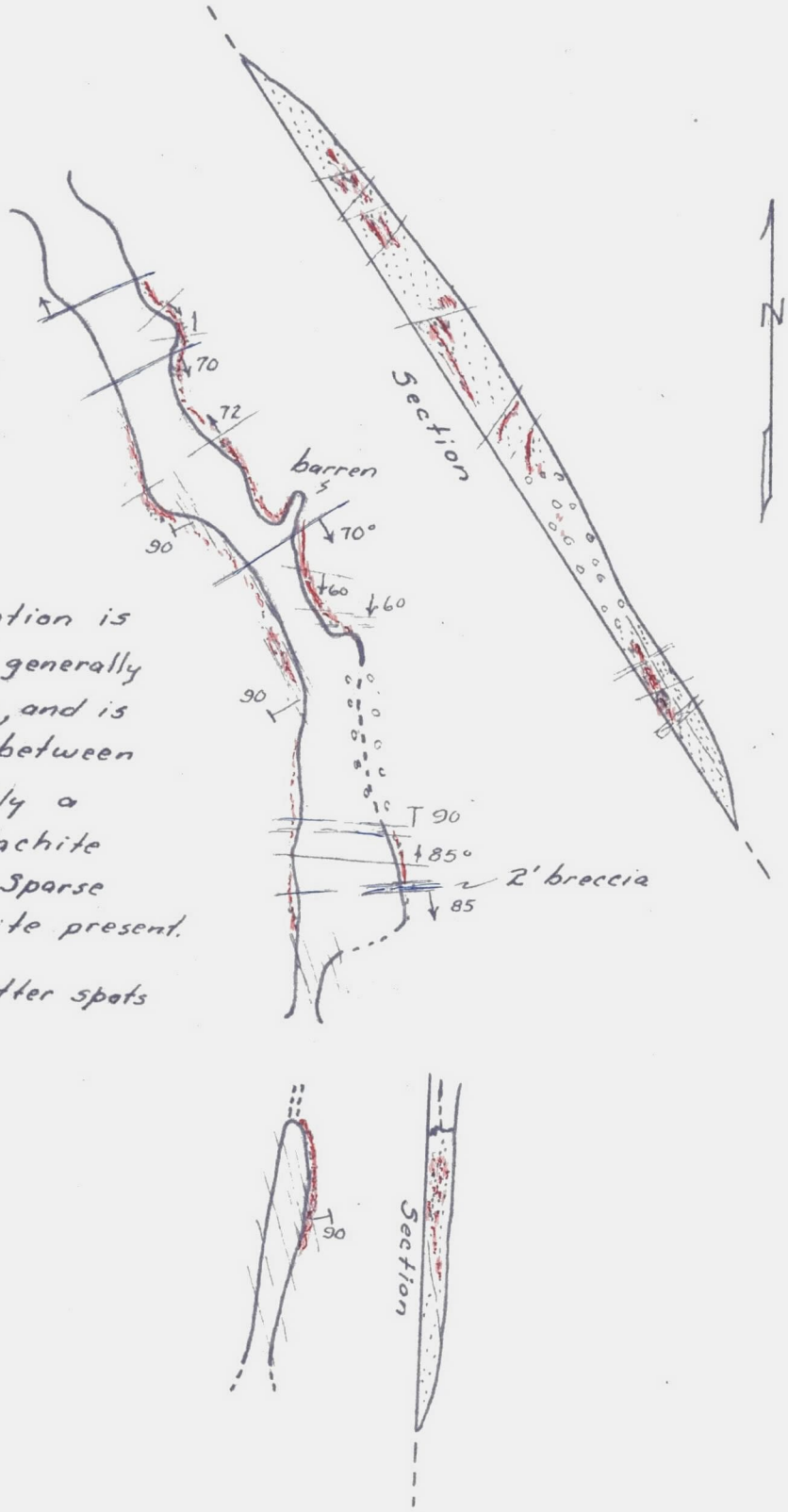
COPYRIGHT BY RAND McNALLY & COMPANY, CHICAGO



Most fractures weak.

Copper mineralization is in x-b Navajo ss, generally related to fractures, and is bluish-green filling between sand grains, possibly a mixture of malachite and chrysocolla. Sparse nodules of chalcocite present.

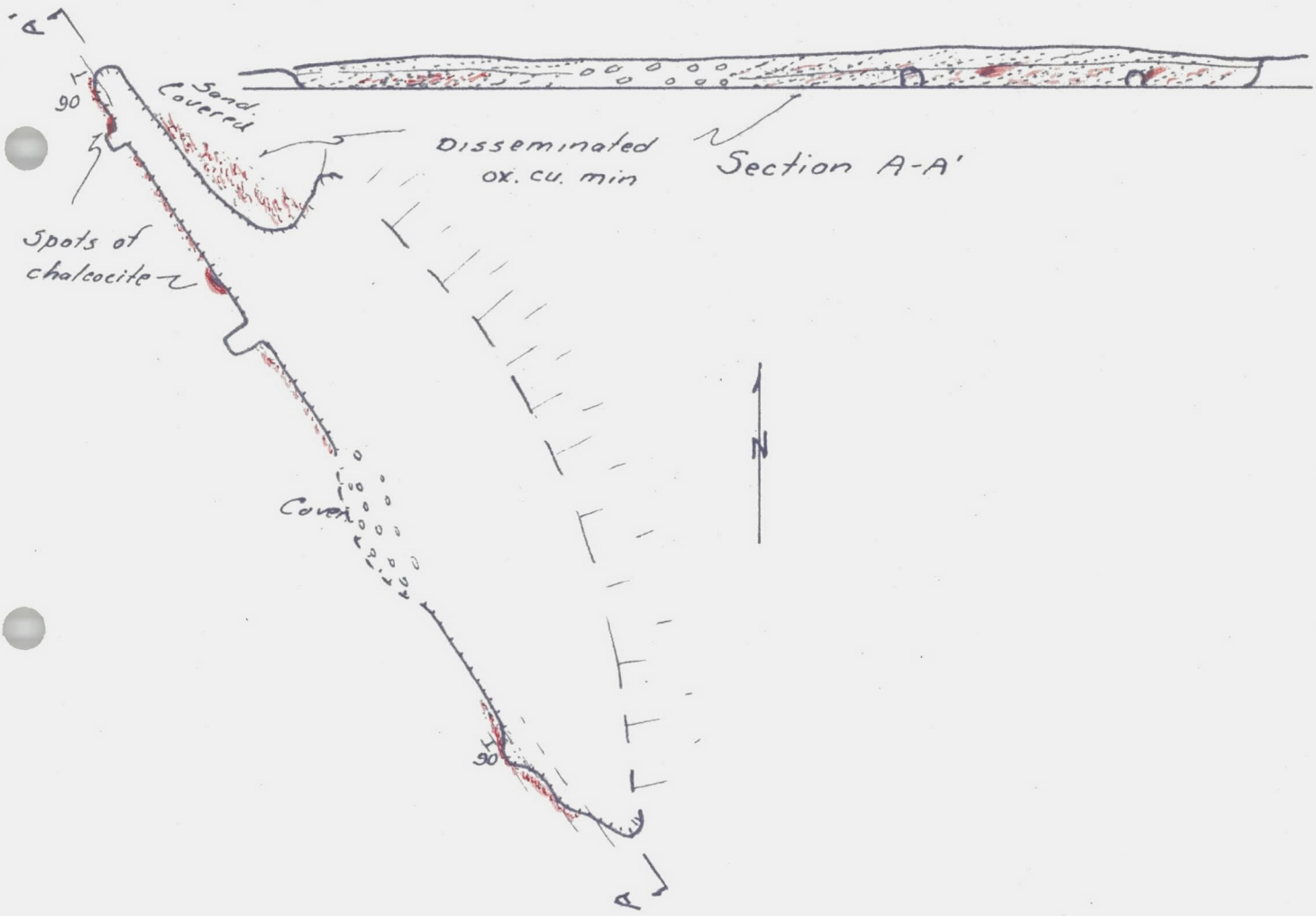
Average grade of better spots may be 1-2% Cu.



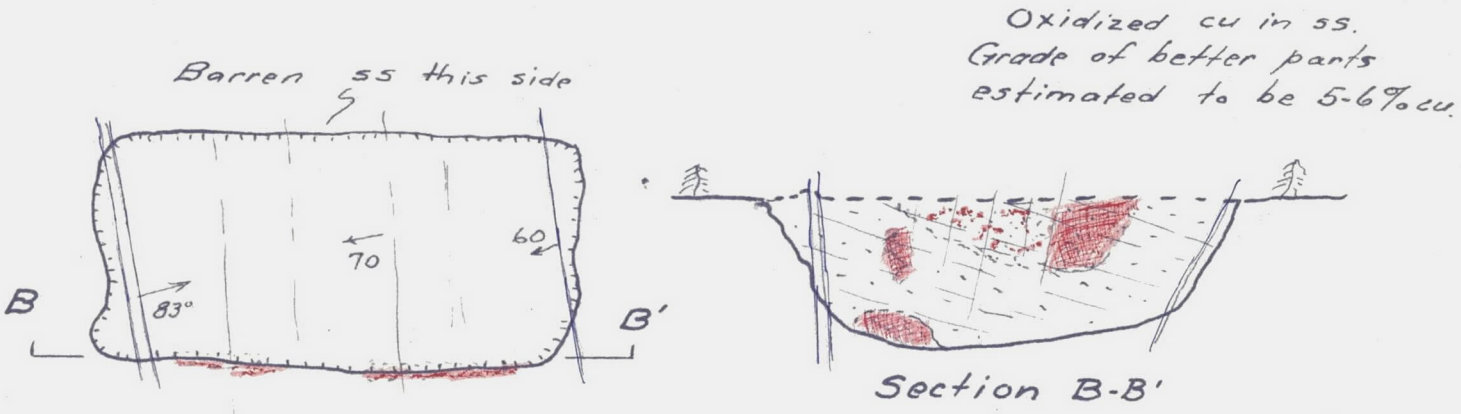
COPPER WORLD OPEN-CUTS
WHITE MESA DISTRICT
NAVAJO RESERVATION, ARIZONA
SCALE: 1" = 100'

R.K. Kirkpatrick
March 1957

Fig. 1



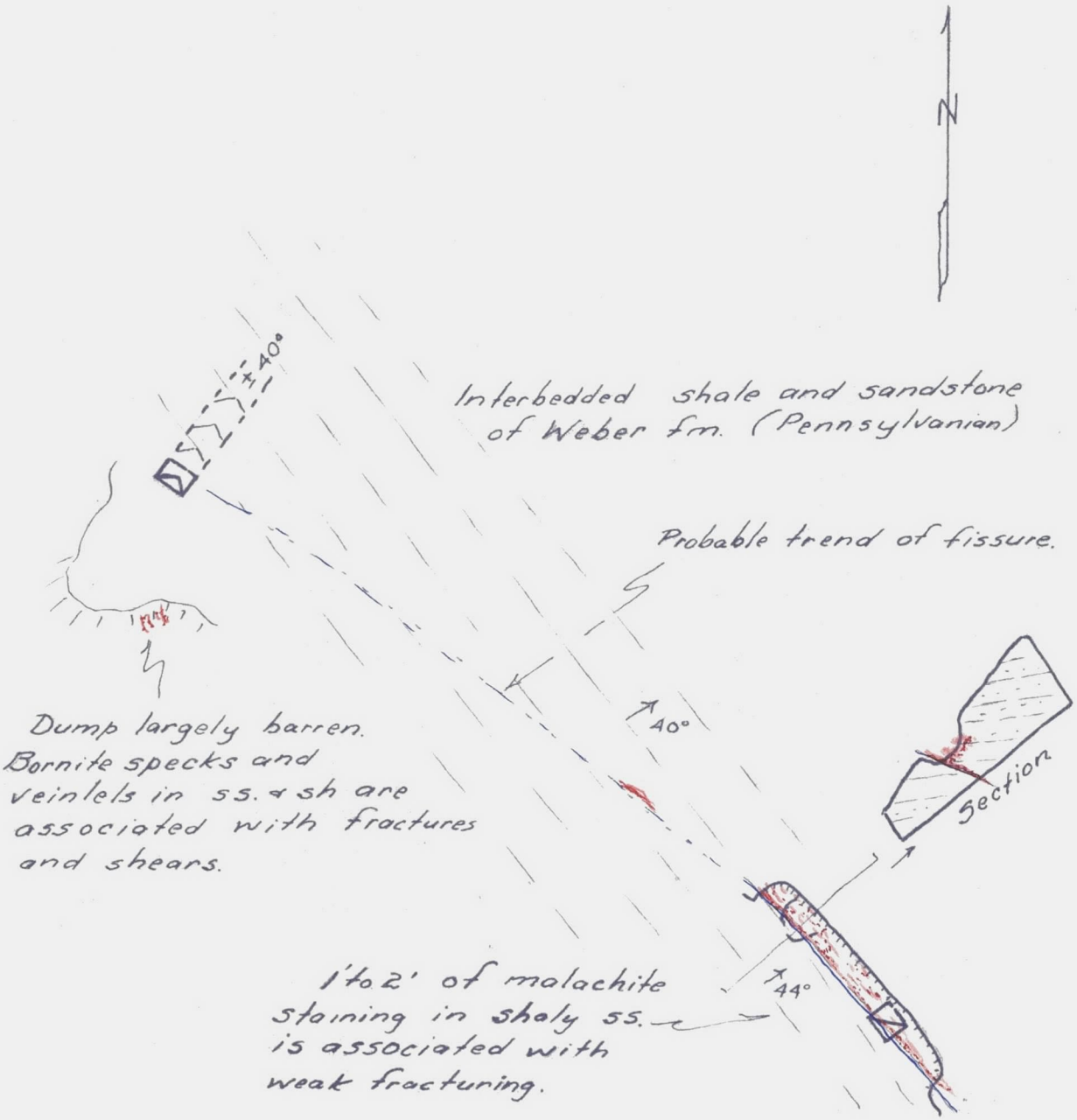
LITTLE DICK OPEN-CUT
SCALE: 1" = 50'



PROSPECT 1 MILE SE. of STORE
SCALE: 1" = 10'

WHITE MESA DISTRICT
NAVAJO RESERVATION, ARIZONA

Fig. 2

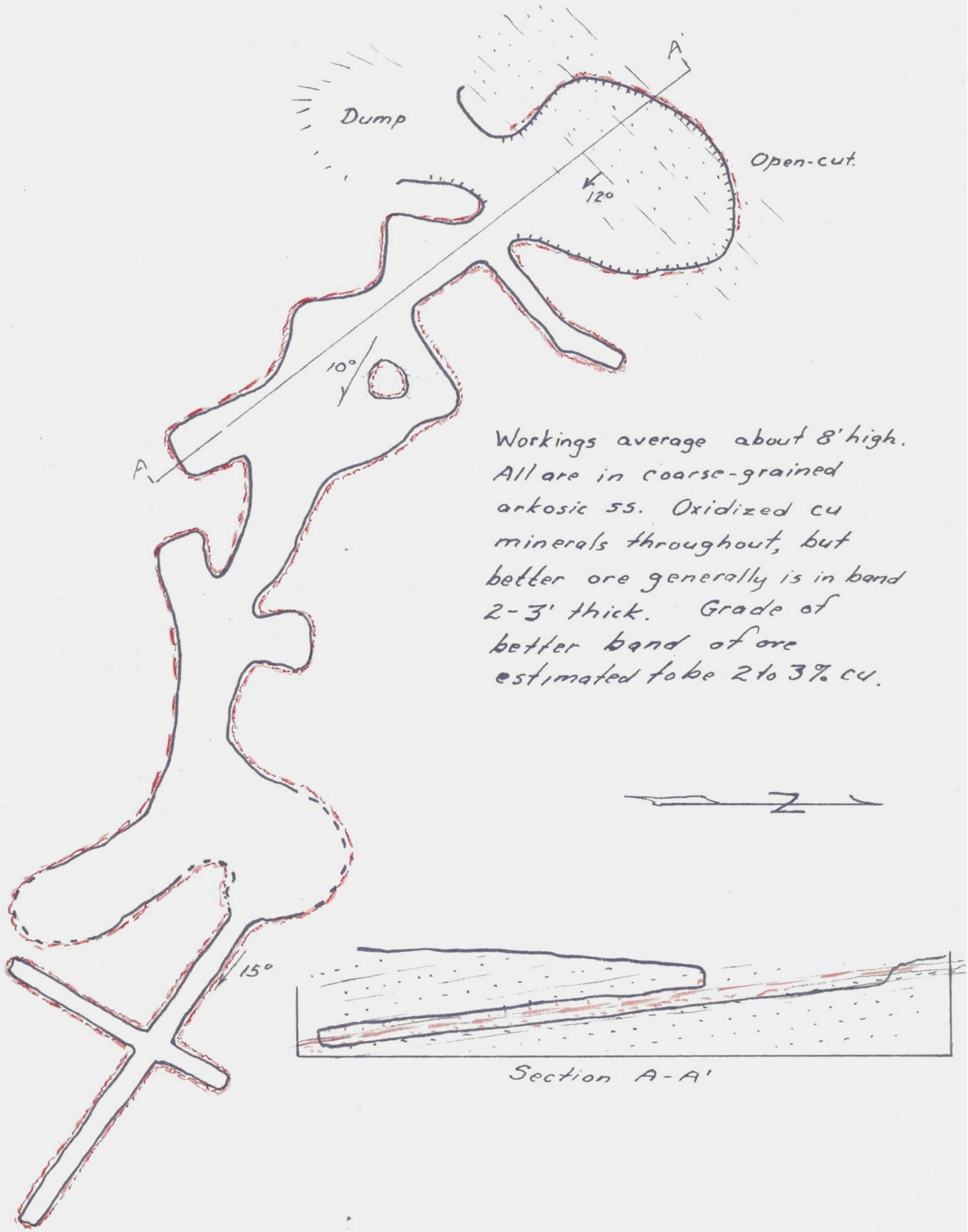


Although min. is in sediments, it is controlled by weak fissures that cross bedding at a slight angle.

FUTURITY MINE
CHAFFEE CO., COLORADO
SCALE: 1" = 50'

RKK, Oct. 1957

Fig. 3

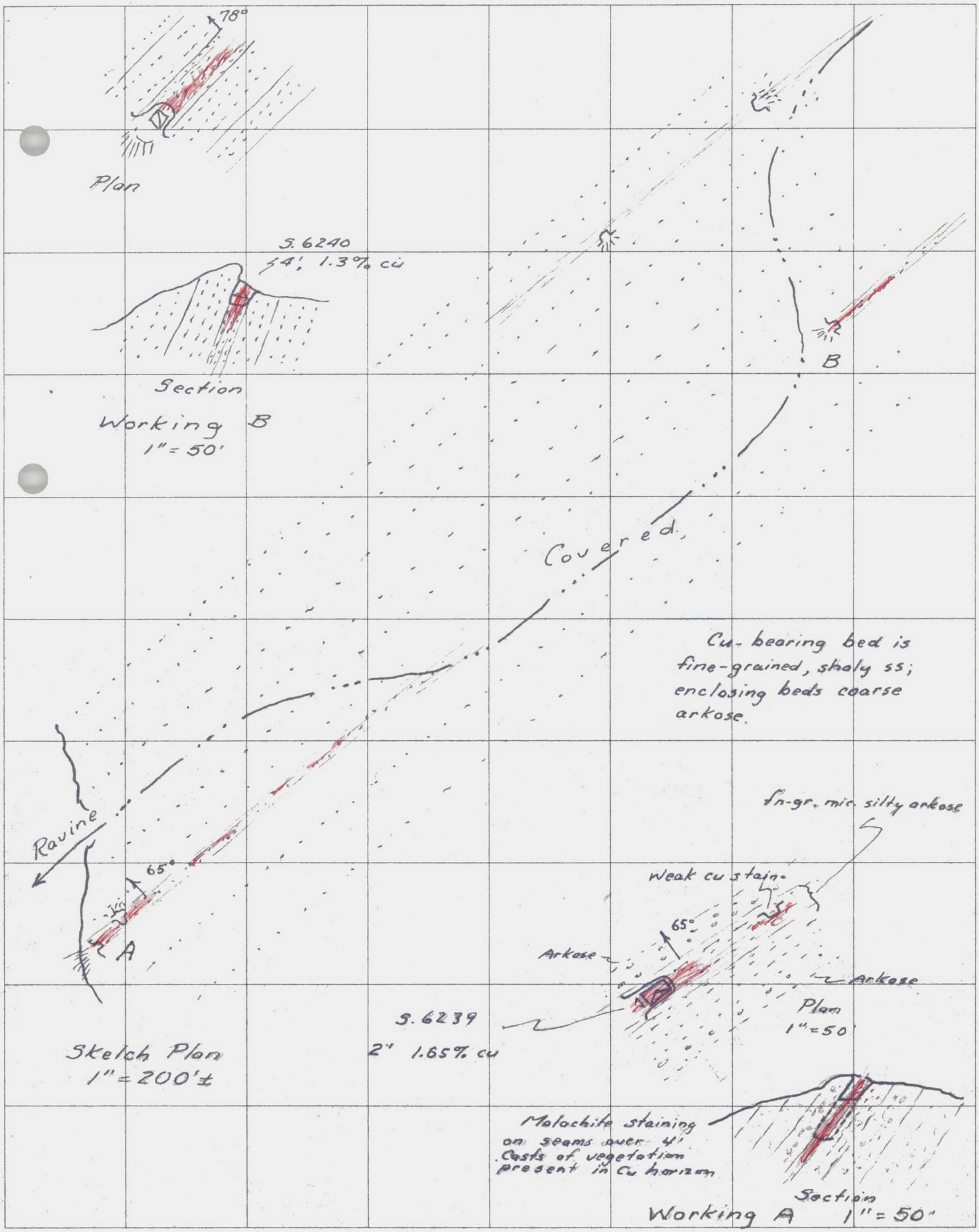


Note: Mine is approximately 2 mi. E. of McCoy and 1 mi. N. of point on R.R. known as Copper Spur.

COPPER KING MINE
NEAR McCOY
EAGLE Co., COLORADO
SCALE: 1" = 50'

R.K.K. Oct. 1957

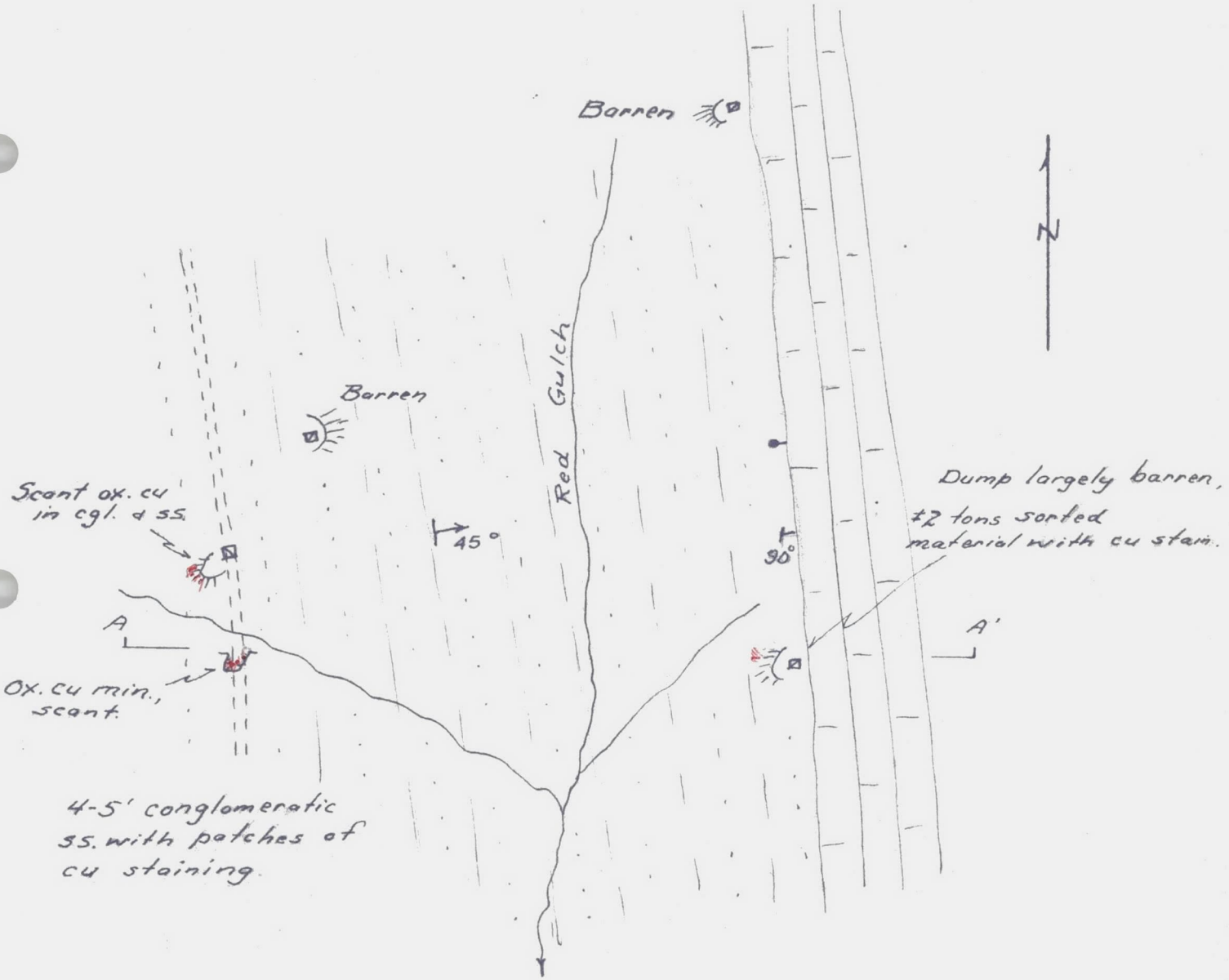
Fig. 4



MINE HOWARDVILLE DEPOSITS LOCATION CHAFFEE Co., COLO. LEVEL

GEOLOGY BY R.K.K. SURVEY SKETCH SCALE 1" = 200' 1" = 50' DATE 11-15-56

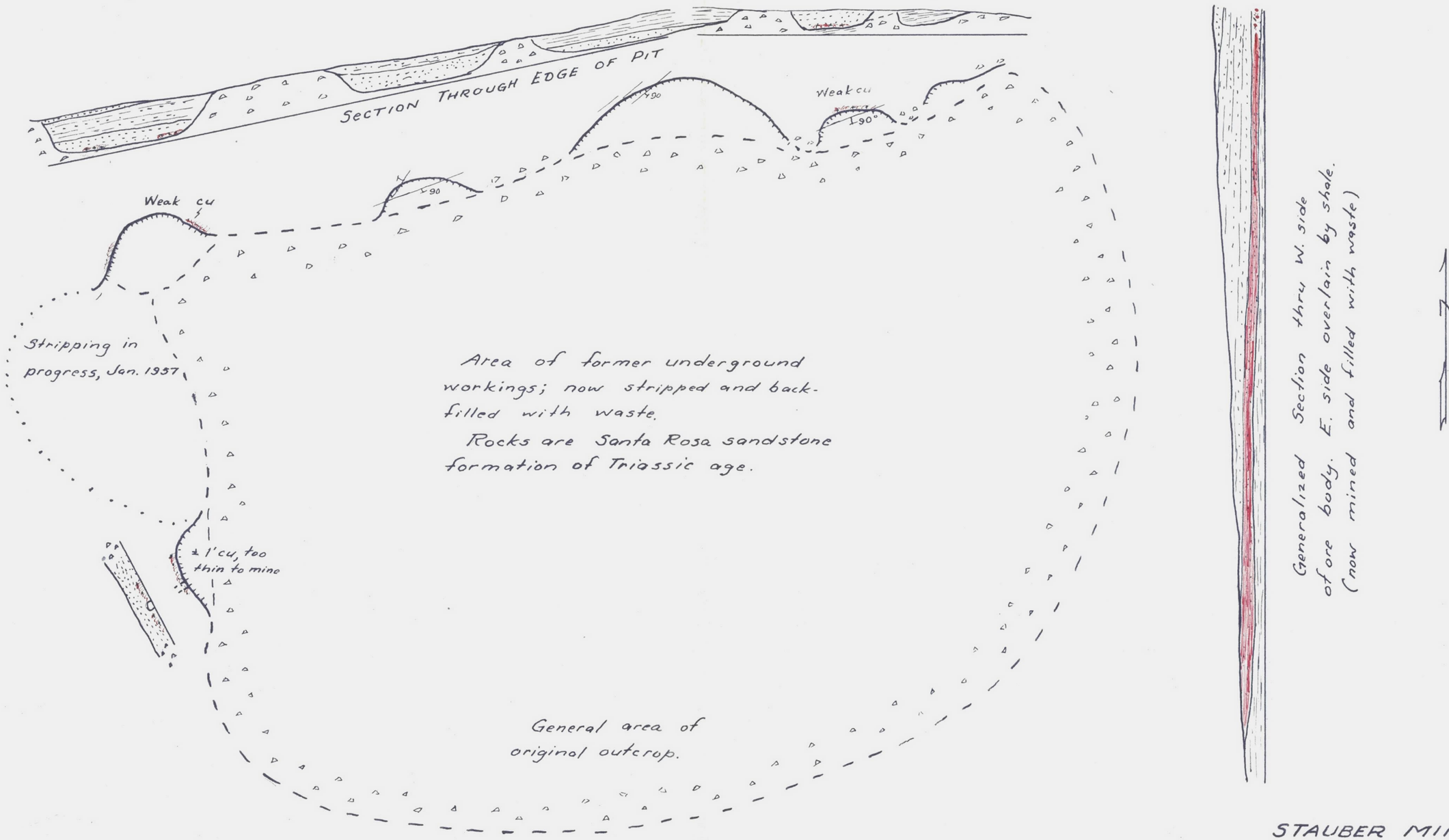
Fig. 5



Section A-A'

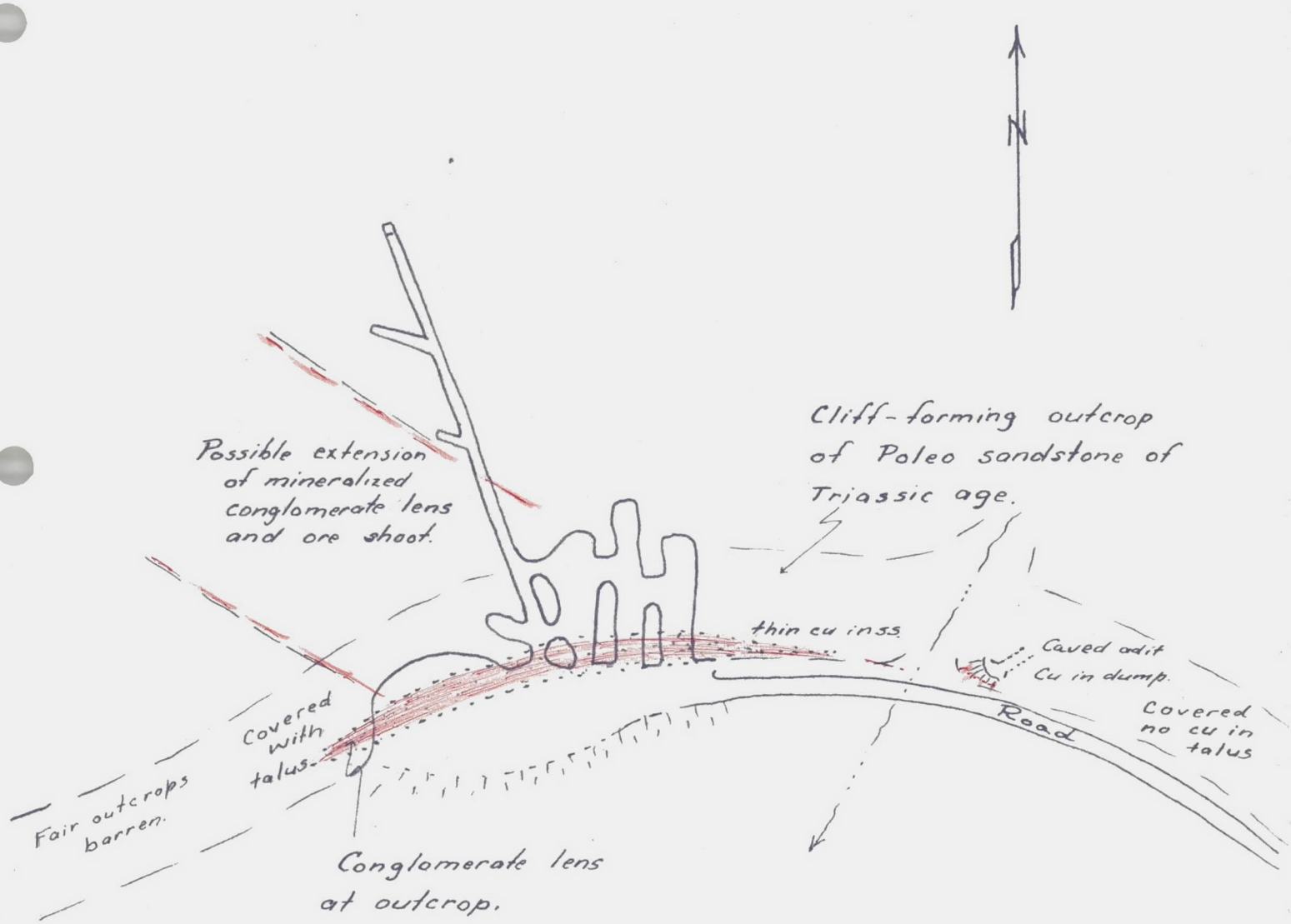
RED GULCH AREA
NORTH OF COTOPAXI
FREMONT CO., COLO.
SCALE : 1" = 500'±
R.K.K.-Oct. 1957

Fig. 5A.



STAUBER MINE
GUADALUPE Co., NEW MEX.
SCALE: 1" = 100'

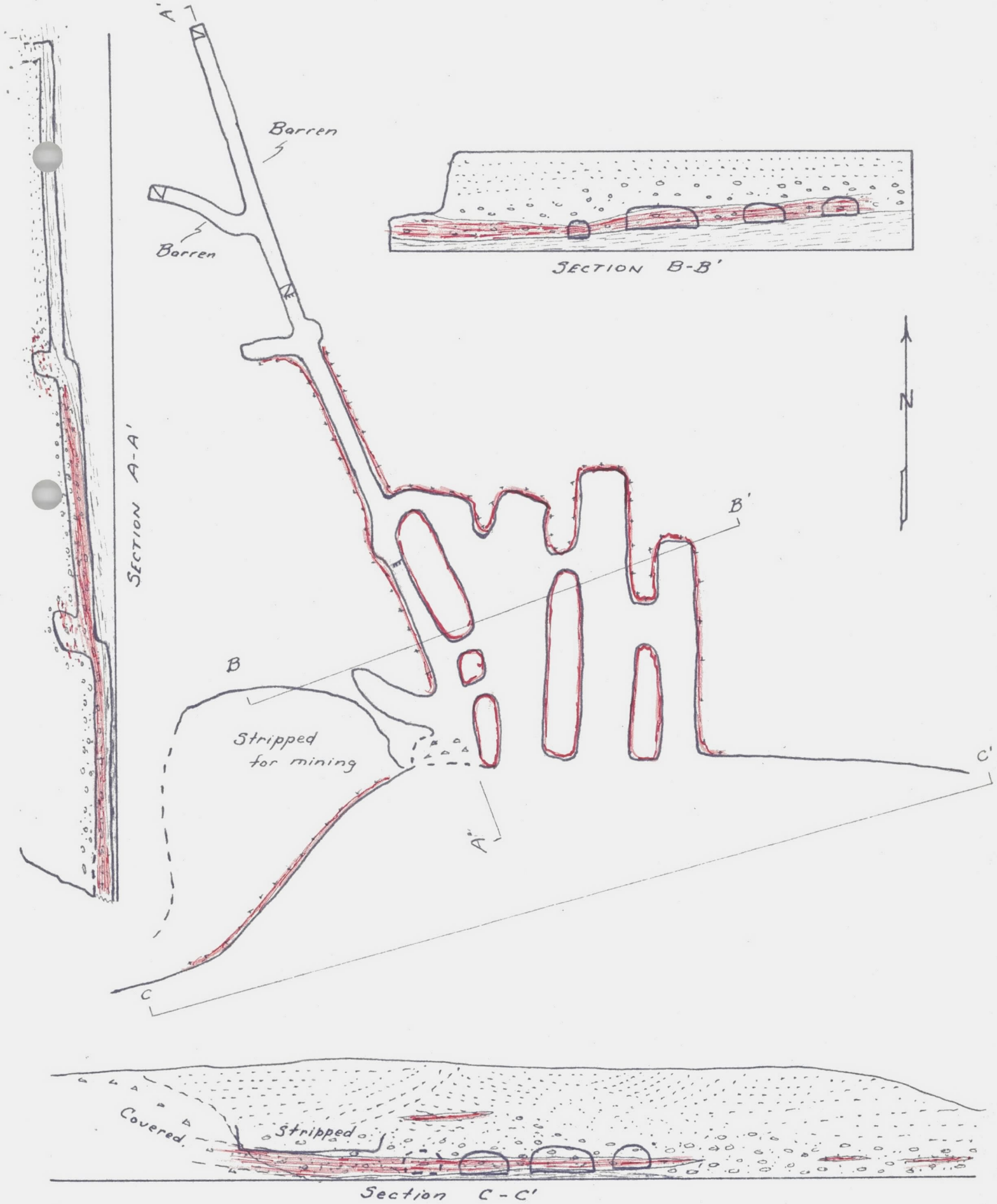
R.K. Kirkpatrick — Jan., 1957
FIG. 6



CONGLOMERATE MINE
RIO ARRIBA CO., N.M.
SCALE: 1" = 100'

R.K. Kirkpatrick
Jan., 1957

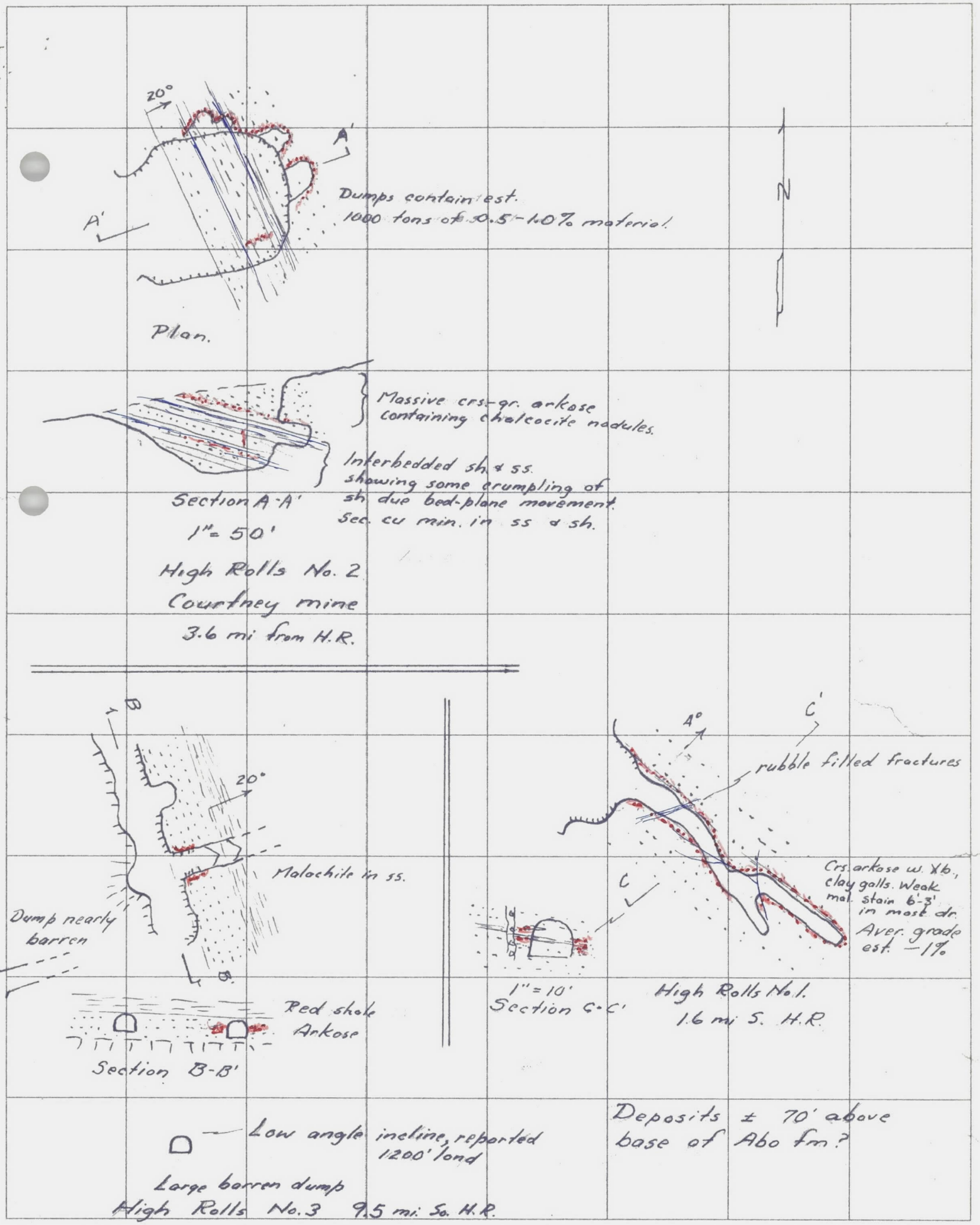
Fig. 7



CONGLOMERATE MINE
RIO ARRIBA CO., N.M.
SCALE: 1" = 50'

R.K. Kirkpatrick
Jan., 1957

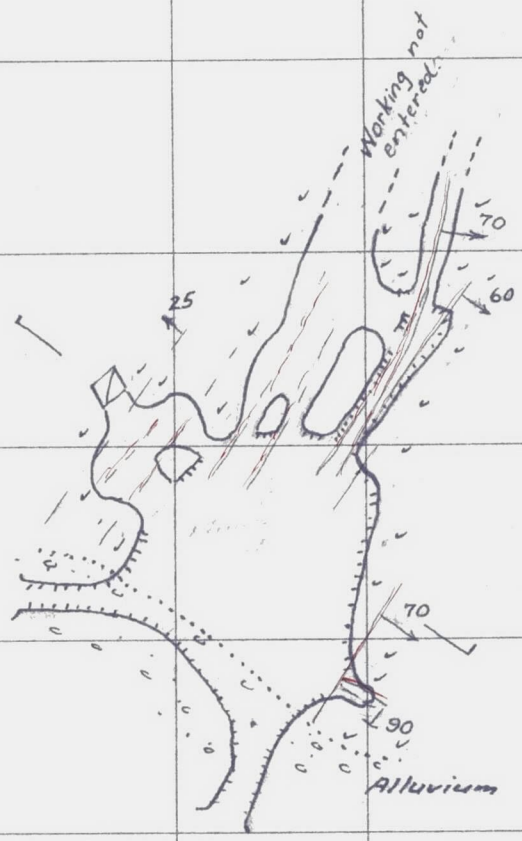
Fig. 8



MINE Ss-cu prospects LOCATION S. of High Rolls, Otero Co. N.M. LEVEL Surface workings

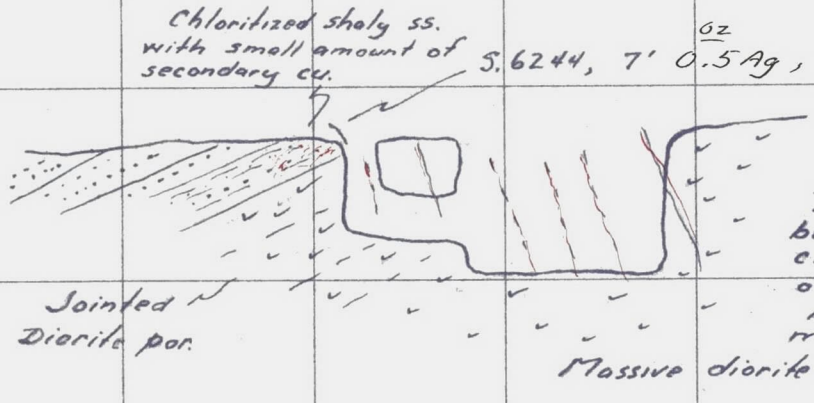
GEOLOGY BY R.K.K. SURVEY Brunton SCALE 1" = 50' DATE Dec. 5, 1956

FIG. 9



Mining at property was apparently confined to zones containing chalcocite-bearing stringers in diorite porphyry?

Large dump consists mainly of fresh barren diorite.
Ore produced was probably hand sorted.



$\frac{1}{2}$"-1" veinlets of barite dolomite containing chalcocite. Veins occur only in diorite.
Alteration very week, most of diorite is fresh.

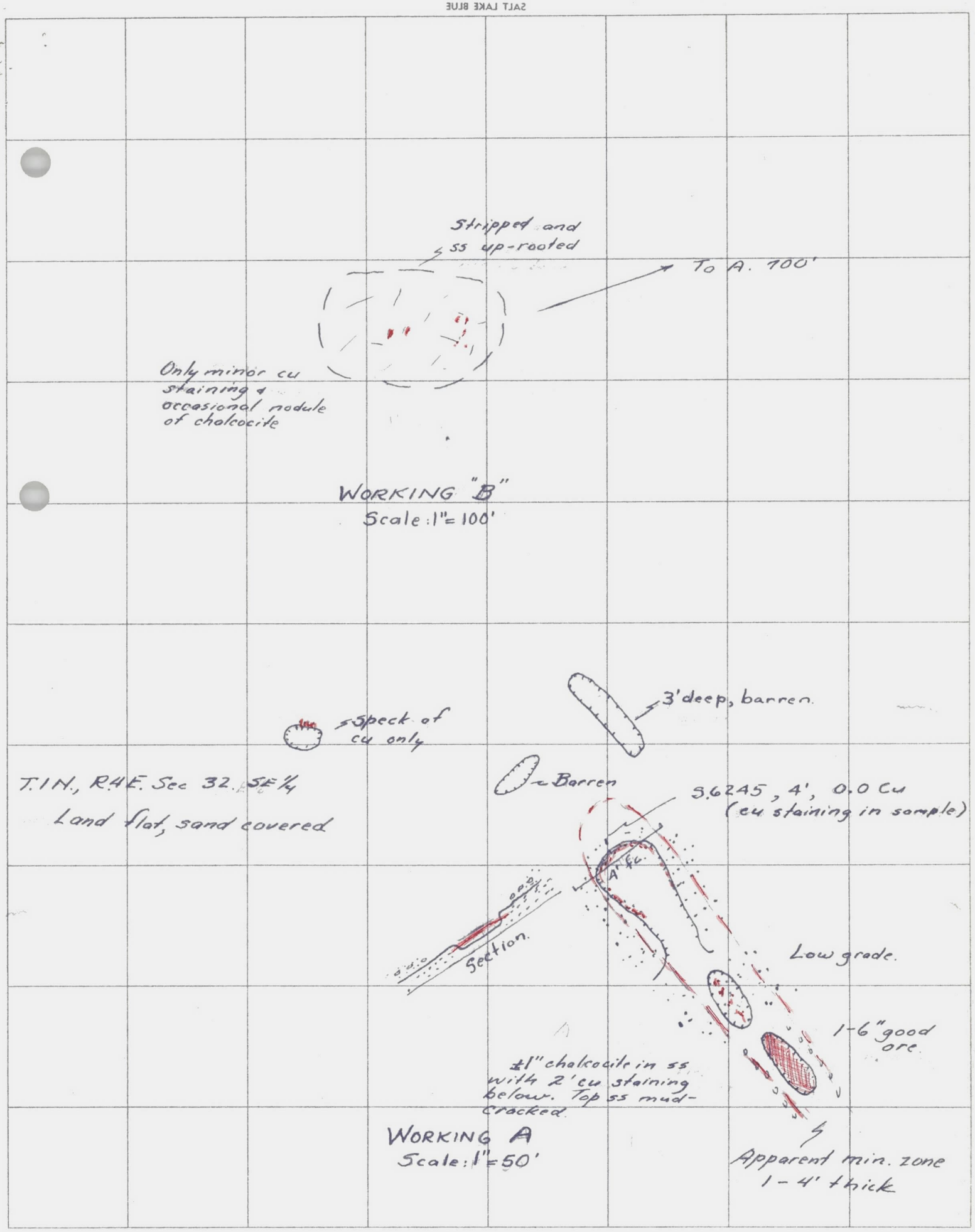
Sketch of open pit and near surface workings

Underground workings may be extensive

Near Bent
Otero Co., N.M.

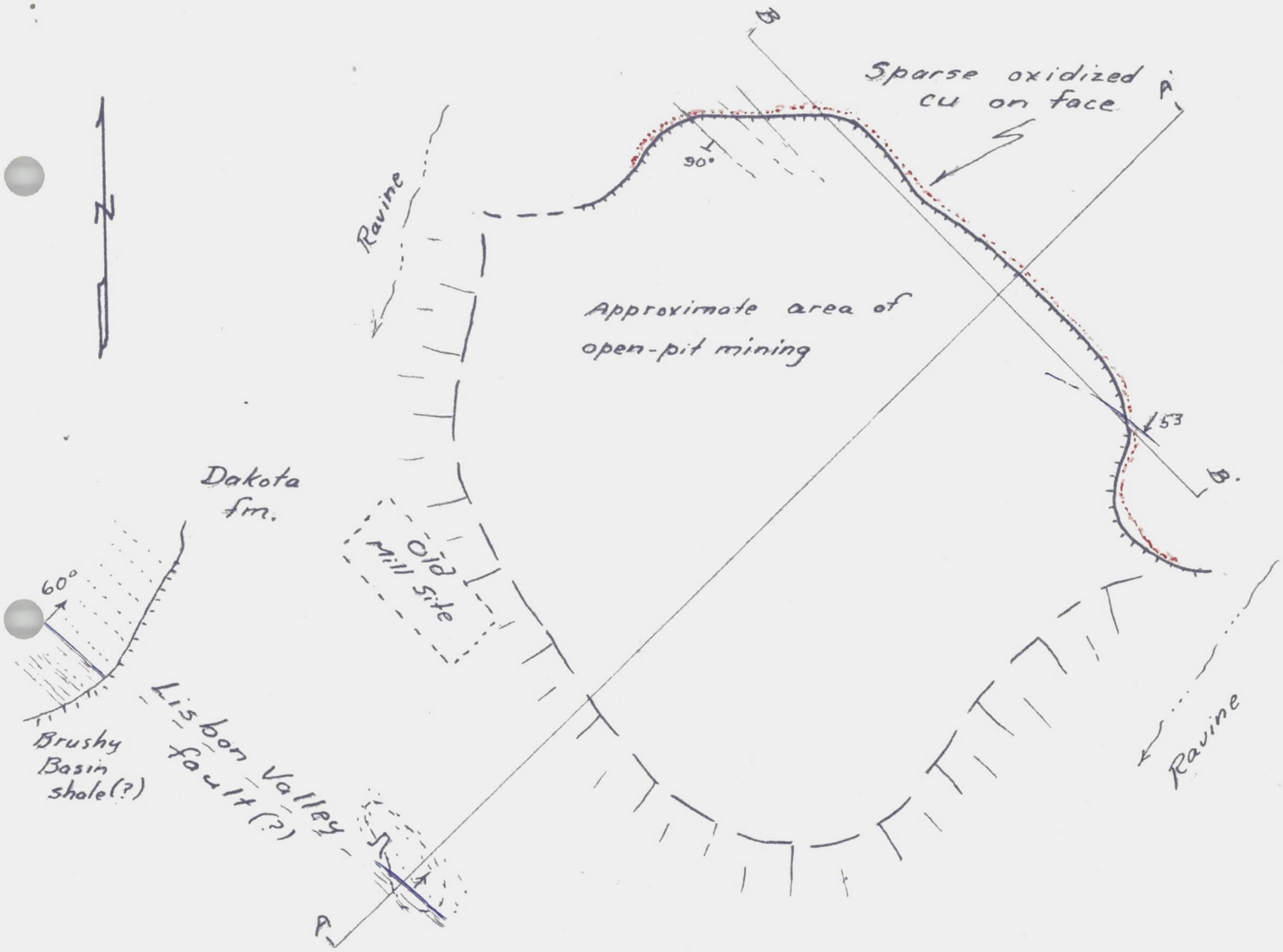
MINE Tularosa Copper LOCATION 1 mi. SE. of Bent. LEVEL Open pit.

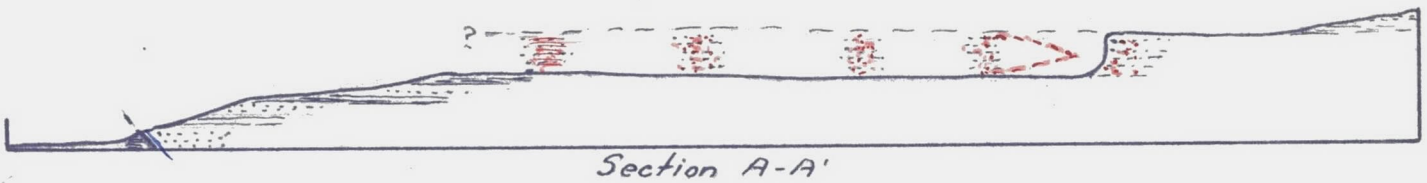
GEOLOGY BY R.K.K. SURVEY Sketch SCALE 1" = 50 DATE Dec. 4, 1956



MINE Meader Corp. Pits LOCATION Rayo Dist., N.M. LEVEL ss-cu prospects

GEOLOGY BY R.K.K. SURVEY Brunton-pace SCALE 1" = 50 DATE Dec. 1956





Section A-A'

Section B-B'

BIG INDIAN COPPER MINE

SAN JUAN CO., UTAH

SCALE: 1" = 100'

Fig. 12

R. K. Kirkpatrick
March 1957

JBS → gen. file. An-0.0.19H

ASARCO

Southwestern Exploration Division

April 28, 1976

F. T. G
APR 30 1976

TO: F. T. Graybeal

FROM: G. J. Stathis

"Stratiform Copper Deposits"
An Address by Felix Mendelsohn

Enclosed is a copy of an interesting Society of Economic Geologists Vice Presidential address entitled "Stratiform Copper Deposits" that was given by Dr. Felix Mendelsohn on February 23rd at the 1976 AIME Annual Meeting in Las Vegas, Nevada.

Dr. Mendelsohn is currently the SEG Regional Vice President for Africa. Apparently he has spent the bulk of his career working on Stratiform Copper deposits.

I have transcribed the address from the tape recording I made at Las Vegas. Editing of the text has been minimal.

G. J. Stathis
G. J. Stathis

GJS:lb
Att.

cc: JHCourtright - w/att.
WLKurtz - w/att.
JCBalla - w/att.

"STRATIFORM COPPER DEPOSITS"
by Felix Mendelsohn

I feel a bit hesitant about talking about stratiform copper deposits in sedimentary rocks, because from what I've heard this morning, it seems that I've stumbled onto a veritable hotbed of magmatic and hydrothermal activity. However, the stratiform copper deposits in sedimentary rocks are in fact a major group of deposits constituting perhaps 30% of the world's copper production. There are certain favorable point characteristics in these stratiform deposits. First, the important factors of size and range. These deposits range from 0.5% or less to around 5% copper and from rather small, less than 10 million tons, to approaching a billion tons. A substantial number of large deposits, having more than a hundred million tons, contain more than 1 percent copper. The first thing we notice about these deposits is that they occur dominantly, almost entirely, in intercratonic basins, so that there is very little association with geosynclinal or ? structural events.

The first of the sedimentary copper desposits are dominantly the rocks that were deposited in an ocean edge environment. In this, I include all deposits that could occur along close to the ocean edge dominantly though; they are lagonal or deltaic deposits and many of them are associated with conditions where circulation with the open ocean was somewhat restricted. As a consequence of the ocean edge depositional environment, we find that there is a distribution and, in many cases elongation, of the stratiform deposits along the ancient shorelines. A very important association is the red beds association. There has in the past been identified a small class of redbeds copper deposits, but I think, in fact, we have to abandon this particular group because it turns out that the majority of ore deposits of stratiform deposits are in association with redbeds, either above, below, or interlaminated. The redbeds themselves tend to be very low in copper and the redbeds are dominantly again in a continental environment. The ore deposits occur in the adjacent rocks, green to gray colored rocks, which again are those close to the continent and therefore in an ocean edge environment. One word of caution here and that is that where the rocks have been metamorphosed to a sufficient degree, the hematite that forms, or derives the red color, can become recrystallized and this color could be lost, so that the 75% of the stratiform deposits that are associated with redbeds is, I think, probably a minimum. Evaporites, anhydrite, gypsum, rock salt are very commonly associated with stratiform deposits and also such minerals as scapolite indicate probably high salinity.

Volcanics, as I indicated earlier, are remarkable by their absence. There are firstly very small amounts associated with some deposits, but there is one particular class that is very intimately associated with volcanics. Many of these deposits occur close to overlying either major or minor unconformities, or in some cases, both. Again, allied to this is the relationship to the Paleogeographic surface. Commonly stratiform deposits occur on the flanks or between basin and tides, that is depositional highs that could be ridges or isolated high points. This would be expected to affect particularly deposits occuring very close to the basin. The effect does seem to move somewhat up into the sequence, so that the deposits occuring some midway up still can seem to have a similar relationship.

The depositional environment is commonly that of a transgression. In other words, the ocean transgressing across the land and getting such stratigraphic sequences as conglomerate, sandstone, algal dolomites, stromatolitic structures, and shales commonly carbonaceous shales. In part, I suspect that this relationship with transgressions, particularly at the base of major sequences shown in the course to be transgressive sequence, is well preserved. If there was a regressive sequence at the base, it could happen that the rocks would be removed and would not be preserved. Another interesting observation is that the rock sequences, this is talking about the overall rock sequence, commonly exhibits cyclic sedimentation and this cyclic sedimentation could be on a major scale, thousands of meters down to tens of meters involving individual ore bodies or even less as approximate ore bodies. Associated with this in many cases, particularly with the smaller scale cyclic activity, it can be noted that there is an improvishment upwards, in other words, the lowest body associated with the lowest particular cycle is likely to be the biggest and best of the deposits found. There is one exception that I know of on a very larger scale, but we'll see that a little later. In general, stratiform deposits in sedimentary rocks are noteworthy for the broad equilibrium that is exhibited. The rocks and minerals, on the broad scale, exhibit a substantial degree of equilibrium quite different from that might be expected; e.g., such places as Butte where there is a marked change in and adjacent to the actual vein deposits.

The common minerals are chalcopyrite and bornite, quite often associated with pyrite, or in some cases chalcocite with copper. The noteworthy association of stratiform deposit host rocks is that of biologic activity. There is extensive and, in many places, intensive biologic activity associated with the deposition of these host rocks. We see these now in such forms as reefs, other bioherms, biostromes, algal mats, and carbonaceous formations. In general the organisms that created these formations in the Protozoic were blue-green algae. Whereas we move up into the Paleozoic, particularly after about 500 million years, the organisms instead of the plants are more animals, such as corals and stromatoporoids. There is a very strong time dependence of stratiform deposits of copper and of other minerals. I won't go into this in too much detail, except to point out there is a fairly strong antipathy between iron, particularly the chemical iron formations, and copper in these stratiform deposits. The ages, in which we find the host rocks do contain stratiform deposits, are the Lower Protozoic, the Upper Protozoic, there are a few in the Middle Protozoic, but not too many. Again there is a gap, and we find abundance again in the Upper Paleozoic, and then after another gap, during the Mesozoic, in the Cretaceous-Tertiary. And you will, I'm sure, all remember that in general iron is found in many of the vacant slots that I've mentioned there.

I have attempted to classify stratiform deposits. The reasons are several. Firstly, if it can be established that a particular type of stratiform deposit has a particular size and grade, then identification of the type will give an indication of the size and grade that might be found. Secondly, there is the innate desire to introduce some order and to look for the pattern that nature normally seems to establish. To allow comparisons between the deposits and, in the end, to discover more about them. The classification I suggest is based purely on the intrinsic properties and you will notice I haven't used the word genesis. I am avoiding this at this stage. The list of deposits or types of deposits that I have at the moment, which is purely provisional as any such classification must be, and runs into the normal problem of trying to fit into specific pigeon holes -- what is in

actual practice a widely and gradually changing spectrum of deposits. However, for what it's worth, the classification I am proposing, at this stage, is the African copper belt -- naming them after prominent examples; e.g., Jupferschiefer, Udokan, in Russiz Djezkazgan, White Pine, the intraformational -- this one would be not named after particular deposit, might be the middle ___?___. We will see something about these shortly. I basically refer to intraformational as those deposits which appear in the within rock sequence without any really obvious reason for their being there. No specific sequence of rocks or sedimentary phenomenon, etc. This is an interesting group. Most of them you do know. The broad size and grade that are likely to be found. The intraformational in general are small to medium in size and likely to be fairly low in grade, less than 1%. Another type is the Creta type, Colorado Plateau, and Mt. Isa. I will now show you a few slides. We can't go over all the detail characteristics of these deposits, but as we go thru the slides I think you will recognize many of the features that I am describing. First slide.

This is Africa, showing the distribution of the Katanga formations of Late Protozoic age. In the north there is the Congo Basin in which there are some small stratiform deposits. Farther south is the major African copper belt in Zambia and Aire, and this broad trough of deposition extends to the southwest underneath cover of Cretaceous and more Recent sands into Southwest Africa on the coast here.

This illustration is merely to show the broad stratigraphic division of these rocks. In yellow at the base - dominantly clastic formation. In the center, at different stratigraphic positions in different areas, but always in the approximate center of the formations of dominantly carbonate formations commonly with tillites. Two tillites in most areas, which are the ones that have provided the basis for this stratigraphic correlation. The upper part of the formations are fine clastic, shaly rocks. The stratiform copper deposits are dominantly in the lower clastic formations near the base. This is an illustration of Mufulira and the 3 superimposed bodies of the 3 Mufulira oreshoots. (Note: missed a sentence.) The 3 ore bodies, the major one is the "C" ore body and progressively they become smaller. The "C" ore body does continue off to the left. Specific rock sequence again; e.g., barren quartzite, quartzite, at the bottom argillaceous quartzite, and then an argillaceous greywacke overlain by limestone. The same sequence thru the "B" ore body with an algal bioherm forming on the one side, again, over the hill of schist in the basement. This is just an indication of the rock types found on the left of the barren footwall quartzites at Mufulira. On the right, massive, darker colored cupiferous sericitic quartzite. This is a reconstruction of the conditions of deposition of the footwall and the ore body formations at Mufulira. On the left is the uplift area where all the deposition was taking place and this deposition was dominated by currents down into the ocean and also longshore currents. An area can be seen of hills and basement or sand shoals formed the barrier between the depositional area and the local scene and the ____?____.

The footwall formations and then the carbonaceous quartzite of the ore body were deposited in somewhat restricted anerobic conditions. Moving across to the ore shale deposits. This is a section thru the ore shale at Roan Antelope. Quartzites on the left topped by a footwall conglomerate. Then an algal schistose, in this area, dolomite overlain by pyritic shale here and these well-bedded shales are the actual ore body containing about 3 to 4% copper. Again barren shales up on the right. This is a picture of the ore shale underground in a slightly more highly metamorphosed state than the one we saw in the previous picture. This

shows the ore shale itself, together with metamorphic veins and these veins contain feldspar, quartz, carbonate, and sulfide. Together with a good deal of biotite. You will notice that around the vein there is a leached area, and it is concluded that the material concentrated in the veins was derived from the wallrock in the immediate vicinity. So there has been a fairly limited amount of migration. This gives you a picture of a portion of the Roan Antelope ore body. A hill, a basement here, is overlain by this barren dolomite which is considered to be an algal bioherm completely barren of copper. Around that, very much richer than normal, we see a plume of bornite spreading across, followed by chalcopyrite, and eventually by pyrite which reaches the hanging wall and then occupies the whole thickness of the ore body. Near the base is an algal mat. Continuing out there which in other areas is mineralized _____?_____ . This is a picture at NKana mine showing, at the top, a pyritic, hanging wall argillite. On the right is the south ore body of carbonaceous shales. Elsewhere the normal argillaceous rock together with, colored in normal blue, the various types of algal siliceous dolomites.

This is a geologic map of the copperbelt. We have been looking at Mufulira over there and Roan Antelope and NKana on this side. And if we look along this side from the Roan thru NKana and up the Chambishi and probably beyond, we have a zone there with barren formations including stromatolitic bioherms on the right and then a zone about a mile wide containing the ore bodies and beyond that a zone about 10 miles wide consisting of barren, pyritic, carbonaceous shale. The distance from there to there is about 40 miles, and similar conditions probably carry on for another 30 miles. These rock types are described and the dimensions are very characteristic of the interdeltaic lagonal deposits and these are what I believe these formations consist of.

Moving north into Shaba province of Zaire; again the stratigraphic column, we don't need to look into it in detail. The bottom, in yellow, indicates arenaceous formations at the same stratigraphic horizon as the ones we just looked at in Zambia. About that is a dolomitic formation containing the lower ore body, then a stromatolitic dolomite which is barren, and over that an ore body of dolomitic shale very similar to that we looked at Roan Antelope. The red is the copper content. This is a photograph of a polished piece of the lower ore body -- a siliceous dolomite, finely bedded and cuperiferous _____?_____. We will refer back to this. This shows the internal dolomite -- with the stromatolitic dolomite -- with the reef that projects up into the dolomitic shale of the upper ore body. Now, moving up north in Europe, this illustration shows the depositional basin. Again, you will note the intracratonic depositional basin of the Permian Kupferschiefer. The ore bodies are distributed along the southern flank of this basin. From East Germany perhaps to Poland. This is a section thru the Kupferschiefer and underneath are the red rocks known as the Rote Faule which are barren and around which the copper is normally found. Again you will notice here the zoning of copper-rich to lime-rich, depending on chalcopyrite. Then lead and zinc moving commonly(?) into the basin. The Kupferschiefer is found at the base of one of the 4 major cycles in the Permian formations. Just above the Kupferschiefer is a second minor cycle which is different than the first. Of the major cycles, the Kupferschiefer occurs as the base of the lower one, the next one has some enrichment in copper but does not reach ore, and others are barren. That is just a picture of the footwall conglomerate, the Kupferschiefer itself is about 20 centimeters thick and underlying the limestones.

This is a picture in the Polish Kupferschiefer showing the varied construction of the depositional conditions. Note there is a lagoon with a highly saline little pond being relatively open ocean on the right and a sand barrier between

them. In the lagoon, lead and copper are present in this dolomitic formation at the base. Along the barrier, copper spreads down into the sands and forms very much thicker ore bodies than normal -- perhaps several meters, and copper then persists out into the open ocean which, in this case, must call for deuteric and reducing conditions to promote.

This is a section at Udokan. You'll notice the yellow is arenaceous formations which predominate the uncolored shales and schists and the blue dolomite. The 3 mineralized horizons. There is one there with the second body above. Again another with the 2 bodies there and here the main Udokan horizon. The scale here, each of these major divisions is several thousands meters thick. So this is quite large scale cyclic activity, and this is the one where I mentioned that the major ore body is in the top cycle. However, looking at each cycle, there is a copper ore body in deltaic formations overlain by shiny black here, a more widespread but lower grade lagonal deposit. The same there and the same thing occurred at the Udokan deposit itself. With a more widespread, lower grade lagonal deposit at the top. This does show some characteristics of the intraformational deposit. Although we get rather an unconformity there, but unfortunately I don't know enough of the detail of the geology to be able to analyze this any further.

Moving across the Udokan is another Protozoic deposit. One of the biggest of the Lower Protozoic copper deposits known. At Djezkazgan carboniferous formations, overlying limestones, contain the various copper horizons of which there are multiple horizons, 20 or 30. This shows a typical lens, a fairly large lens, but unfortunately I can't give you the exact scale. Redbeds and within the lens, conglomerates and the yellow showing sands and the various shades of orange and black, richer and richer ore grades. So that the ___?___ lens-like arrangement to the ore deposit itself. This just shows the detail section thru Djezkazgan illustrating the intimate interlaying of graybeds and cupiferous rocks. The stratibound ___?___ are cupiferous and the red color are the red beds.

Moving across now into North America, at White Pine we have the one class of deposits that is intimately associated with basic lavas, and overlying this, thick sequence of lavas and a conglomerate with another lavas intermingled. Within the first shaly sequence in this broad redbed sequence we find the White Pine deposits. This is a detailed section thru the White Pine just illustrating again the association between copper and these black beds and also the cyclic nature of the development with copper in the lowest redbed and lesser copper in the graybeds -- none in the sands and then again in the upper black formation representing the next cycle. This shows an interpretation of the depositional conditions at White Pine. With an alluvial fan on the southwest, a delta on the southeast, and a sand bar in the ore bed giving restricted circulation within this embayment. It is this embayment that contains the ore deposits of the White Pine mine. The embayment is opened to the northeast. The White Pine belongs to the vague or dominant group of Upper Protozoic stratiform copper deposits.

In the Belt, the lower Belt formations are virtually barren of copper, although we know they do have some lead-zinc deposits. The yellow coloring here indicates rather siltstones and sandstones and the deformation here which contains the one known ore body in the Belt, is also the base of most of the mineralization. All these black lines indicate some degree of mineralization. The Belt is one I would classify as an intraformational deposit. Although there are some doubts about the exact origin, as there are in many of these, there seems to be some fairly intense metamorphic activity which has also locally altered the red color to brown. This

is the same sequence of rocks in Canada and again equivalent to the Revett is the Valley group -- the Grinnell, again containing stratiform copper deposits with some zinc and iron in succession, again intimately associated with redbeds. However, stratiform small copper deposits.

The Creta deposit, Permian in age, in the Southeast is found in the Flowerpot shale, and this illustration shows the transgressive nature of the deposition of the Flowerpot; sea off to the left, land on the right, and transgressive sequence within which these little deposits are found. This is a photograph of the formations immediately above the Creta ore body, showing the finely bedded nature of gypsum and shaly redbeds with the copper deposit near the base of that sequence. Again this having intraformational characteristics with no particular reason that is evident as to why it should occur at this particular horizon. The several Colorado Plateau deposits which have been in the past referred to as the redbeds are found in the upper Paleozoic formations, Permian, and also Triassic within channels, sandy channels of fluvial or shoreline formation, and also in quartzites a little more wide and evenly spread, also in shoreline deposits.

NOTE: End of 30-minute tape run - changed tape.

.....47 copper. At the corner some of those blue colored formations I showed you. They contain 31.6 silica, 47.8 carbonate. At the Roan Antelope in a formation which again is very different in appearance, ranging from a biotite-tremolite schist to a paragneiss to a schistose dolomitic rock. We have very similar chemical composition again. So that the conclusion seems inescapable that these silica dolomite bodies are of similar origin. It has been suggested, for example, at Mt. Isa that these are in fact reef deposits or reef debris that has accumulated and, as you recall, I suggested a similar origin, in other words an algal origin for these formations in these other deposits. So it looks here as though we have a siliceous dolomite with algal affinities that does contain very substantial quantities (of copper ore?). At Mt. Isa, for example, there is perhaps 140 million tons that's quoted. At Shaba province, generally of Zaire, probably half their ore, which amounts to many hundreds of millions of tons, consists of the silica dolomite and at NKana, Roan Antelope, Chambishi. Again, there are several hundred million tons of this same ore.

what about hot spring?

These photographs have illustrated some of the characteristics of these deposits and I would just like to finish with a few, very few words on the genesis. The bulk of the evidence in a large number of these deposits indicates that the mineralization took place early in the history of the rocks. That is during deposition -- during diagenesis in the Main. There are some where later activity is also very obvious. Such as the Belt. So my conclusion is that we have, in fact, a fairly wide spectrum of genesis of these deposits; that you did, in fact, get syngenetic deposits or some sedimentary deposits that formed during the time of sedimentation of the host rocks. The copper could have been derived from volcanic emanations, hydrothermal emanations, or from the erosional hinterland. Subsequently, diagenetic processes could have led to the development of ore bodies, concentration of widespread copper in the rocks, and to modification of the existing ore bodies and similarly metamorphic processes, rather epigenetic processes could easily have resulted in ore bodies and could also have led to the modification of preexisting bodies of either the syngenetic or diagenetic type. So that, in fact, we have quite a range of possible genesis for these bodies and the only debate would be on a particular genesis for the individual bodies. Thank you.

One question from the floor (inaudible) .....during which the host rocks deposited in particular times tend to have concentrations of copper. There are, in general, other times during which there are concentrations of lead and zinc. So there is this differentiation and, although I wouldn't dare to substantiate this, I do have the impression that where there are lead and zinc deposits around, the silver is more likely to be with them. But where there isn't much in the way of lead and zinc, or particularly lead, there is more silver with the copper. But this is just an impression.

HAWLEY & HAWLEY
ASSAYERS AND CHEMISTS, INC.
1700 W. GRANT RD., TUCSON, ARIZONA 85703
BOX 5934 (602) 622-4836

Registered Assayers
OVER 50 YEARS

BRANCHES
Douglas
Hayden
Morenci
Inspiration
El Paso

J. H. C.

JAN 23 1970

BULLETIN NO. 6
January 20, 1970

COPPER
Oxide/Non-Sulfide Problems

W.E.S.

MAR 13 1970

Often geologists, or others, request oxide copper or non/sulfide assays, but do not specify the procedure for us to use which would be best suited to their particular requirements. (When the method is not specified we use a sulfurous/sulfuric acid leach, and determine the copper content of the resulting leach solution.) The geologist may also have other laboratories assay the same material, but unless the same method is used by each laboratory, the results most likely will not agree, and further, may not meet the mineralogical and metallurgical requirements of the customer. Metallurgist, mineralogist and assayer may unwittingly be working at cross purposes--hence this bulletin.

Since the copper oxide analysis is a metallurgical problem rather than one of strictly analytical chemistry, it must be approached on that basis. Each ore body is to a certain extent a new and distinct one and has its own particular mineralogical make-up. It therefore should become a joint effort of metallurgist, mineralogist, and assayer to determine what type of recovery can be expected from the ore, and therefore what oxide method should be used. correct!

When separating copper minerals by means of any solvent (H_2SO_4, H_2SO_3, etc.) that will dissolve some and not others, it must be understood that few such separations are absolute, and that practically any mineral present will be attacked to some extent. Also, in some copper mineral separations special solvents must be used that will dissolve the copper oxide minerals but not those of other elements. If compounds other than copper are dissolved, a new chemical can be formed which may dissolve minerals that should remain undissolved, or some constituent of the ore may re-precipitate the dissolved copper.

Two examples of this are as follows: In ore containing oxide and sulfide copper a good separation can be made by means of dilute sulfuric

acid, but if limonite, $2F_2O_3.3H_2O$, is present it will form ferric sulfate, which with the H_2SO_4 will leach both the oxide and sulfide copper minerals. The other example deals with failure to remove metallic iron in a non-reducing solvent. Sulfuric acid, as already mentioned, is a good solvent for copper oxide minerals, but any metallic iron present will re-precipitate the copper as metal almost as soon as it is leached from the ore. Hence, it either must be removed or some reducing agent such as sulfurous acid must be added.

The various copper oxide minerals do not leach equally well in any given solvent system. Cuprite, for example, is leached to the extent of 70% by dilute sulfuric acid, while azurite, malachite, and chrysocolla approach 100%. Cuprite, malachite and azurite approach 100% leachability in an ammonia-ammonium carbonate system, while chrysocolla is only about 60% soluble. (See tests on the following page -- U.S. Bureau of Mines RI 3228, prepared by W. A. Sloan of the Kennecott Research Center.)

This all points to the fact that only with advance mineralogical and metallurgical help can the analyst give a proper evaluation as to what constitutes a suitable copper oxide assay.

* * *

COPPER
Oxide/Non-Sulfide Determination

Weigh a 1 gram sample for concentrates, 2 grams for headings, or 3 grams for tailings into 250 ml beakers. Add 50 ml of concentrated ammonium hydroxide and 5 grams of small lump ammonium carbonate to the beakers and cover. Swirl the mixture in the beakers by hand every ten minutes for two hours. The leaching should be carried out at room temperature. Exactly two hours after the reagents were added, filter the samples through double Munktell #0 filter papers. Wash the samples from the beakers and wash on the paper five times with cold distilled water. Evaporate the filtrates to dryness on a hot plate. Dissolve the residue with 10 ml of concentrated nitric acid and a little bromine water, take nearly to dryness, and dilute with water. Add 1:1 ammonium hydroxide until a permanent blue color is obtained. Boil until all of the ammonia is removed, add 7 ml of acetic acid and heat until the copper is dissolved. Cool and titrate in the usual manner with standard sodium thiosulfate solution.

Comparison of Western Methods for Determination of Oxidized Copper in Ores. By W. A. Sloan RI 3228

A) 1 to 5 g, 75cc 10% (wt) H_2SO_4, just to boiling.

B) 2 g, 25cc 5% (vol.) H_2SO_4, 93°C, 20 minutes.

C) 2 to 5 g in 100 cc bottle, 10 cc 5% (vol.) H_2SO_4/g, agitate 1 hr., filter and wash hot water.

D) 2 to 5g, 250 beaker, 10cc H_2O, remove Fe with magnet, volume 75 ml, add equal volume 5% (vol.) H_2SO_4 sat. with SO_2, sit 1 hr. rm. temp. (80°F) stirring freq. Wash hot H_2O.

E) 5 g, 250 beaker, 50cc, 10% (vol.) H_2SO_4, bring to boil within 2-3 min. Boil 3 min, filter, wash.

F) 3 g ore, 10 g tail, 250 beaker, 40cc boiling H_2O, remove Fe with electromagnet, 10cc 20% H_2SO_4 (vol.) leach 1 hr. stirring freq. Filter and wash H_2O hot.

G) 0.5 to 5 g of ore or tails into 8 oz. oil sample bottle, 100cc 4-5% SO_2, cork tightly, roll 2 hrs, filter and wash with dil. SO_2.

H) 5 g ore or tails into 8 oz. salt mouth bottle and ground glass stopper, 100cc of 2.5 to 3% SO_2 stopper and seal with parafin. Agitate 2 hrs, filter and wash with SO_2 solution.

I) 2 g ore, 3 g tails, 1 g conc., into 250 beaker, 5 g lump $(NH_4)CO_3$, 50 ml NH_4OH, leach rm. temp. 2 hrs. shaking occasionally. Filter and wash well.

J) 2-5 g into 250 beaker, 50cc of 5% (vol.) H_2SO_4 sat. with SO_2 heat to boiling and boil 2 min. Filter and wash hot H_2O.

(See page 4 for solubility of selected copper minerals)

Total Copper Dissolved, %, by Method

Mineral Oxides	% Total Cu as Oxide	A	B	C	D	E	F	G	H	I	J
Azurite $2CuCO_3.Cu(OH)_2$	100.00	98.28	99.72	100.00	100.00	100.00	100.00	99.95	100.00	100.00	99.91
Malachite $CuCO_3.Cu(OH)_2$	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Cuprite Cu_2O	100.00	62.82	67.15	59.56	63.85	67.90	61.62	74.83	71.02	98.29	66.34
Chrysocolla $CuSiO_3.2H_2O$	100.00	99.66	100.00	100.00	100.00	99.66	98.66	99.00	99.68	59.20	99.33
Native Copper	3.72	4.47	5.95	3.82	3.62	4.90	5.19	9.80	7.25	15.10	4.46
Sulfides											
Chalcocite Cu_2S	1.55	1.42	1.42	1.16	1.66	1.68	1.94	1.42	1.16	3.24	1.55
Bornite Cu_5FeS_4	0.46	0.46	0.61	0.46	0.92	0.92	0.92	0.61	0.61	3.82	0.30
Chalcophrite $CuFeS_2$	0.35	0.95	1.27	0.63	0.32	0.63	1.27	1.27	1.27	2.24	0.63
Covellite CuS	3.15	3.31	3.77	3.16	3.31	3.77	3.92	3.63	3.46	4.67	3.16

which it is pumped to first stage strip- Johnson operation. END. ley. Dark band is wet dump.

Oak Flat
Porph Copper General

RECEIVED
SEP 22 1988
EXPLORATION DEPARTMENT

New York, N.Y., September 19, 1988

TO: R. L. Brown

Oxidized Porphyry Copper Deposits in the USA

In keeping with the enthusiasm for acquisition of open pit porphyry copper reserves in the USA, I attach a list of oxide (non-sulfide) deposits which might be candidates for in situ mining. I excluded deposits owned by Cyprus, Kennecott, Magma, and Phelps Dodge, small deposits of the type being reviewed by Mr. Anzalone, and oxide deposits which might be open pit mined (Blackwater).

The only deposit I could recommend acquiring with some enthusiasm is Casa Grande West. Ownership of Oak Flat would enhance our position at Superior East; whether Magma would be a potential competitor for this ground is unclear.

F. T. Graybeal

F. T. Graybeal

FTG:mc
Att.

cc: W. L. Kurtz (w/att.)

IN SITU MINING CONFIGURATION (OXIDIZED)

Deposit	Owner	Reserve (M=million)	Comments
Santa Cruz	Asarco-Freeport		
Casa Grande W.	Texaco-Simmons		Oxide reserves are present on ground E. of SC-58 which would not presently be acquired through the Simmons deal.
Sacaton	Asarco		
Van Dyke	unknown	100 M tons-0.5% Cu	May be some current leaching from underground workings, 1,000-1,500 ft. deep.
Miami E.	Miami (?)	200 M tons-0.6% Cu	Reserve unknown, 1,500 ft. deep.
Sanchez	unknown	130 M tons-0.4% Cu	1,000 ft. deep.
Superior E.	Asarco		Need Oak Flat.
Powers Gulch	unknown	100 M tons-0.39% Cu	2,000 ft. deep.
Florence	see open pit list		Oxides may be too shallow to keep leach solution confined.
Michigan Copper	UOP-Homestake	+100 M tons-+1%	Native copper.
Amoco	Asarco-Freeport trade to Simmons	+100 M tons-0.3%	500-1,000 ft. deep, thin zones of CuOx, not well drilled.
Yerington Area	unknown	unknown	Either Mickey Pass, Tailings Pond, or Ann Mason may contain large oxide copper tonnages.
Pine Flat ? where?	(?)	40 M tons-0.4%	Largely under post-mineral volcanics. May not be oxidized.

FTG-9/19/88