



CONTACT INFORMATION  
Mining Records Curator  
Arizona Geological Survey  
3550 N. Central Ave, 2nd floor  
Phoenix, AZ, 85012  
602-771-1601  
<http://www.azgs.az.gov>  
[inquiries@azgs.az.gov](mailto:inquiries@azgs.az.gov)

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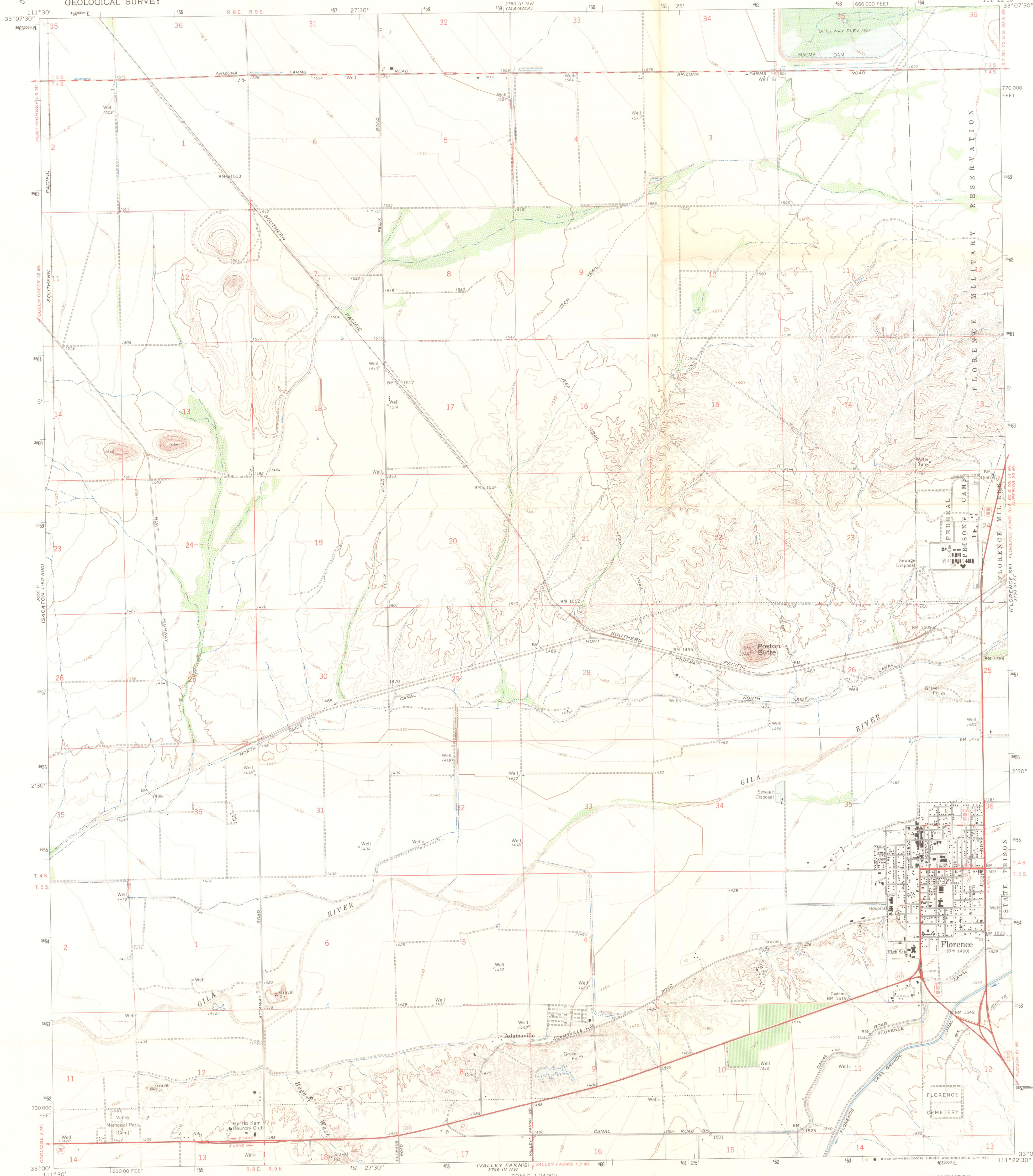
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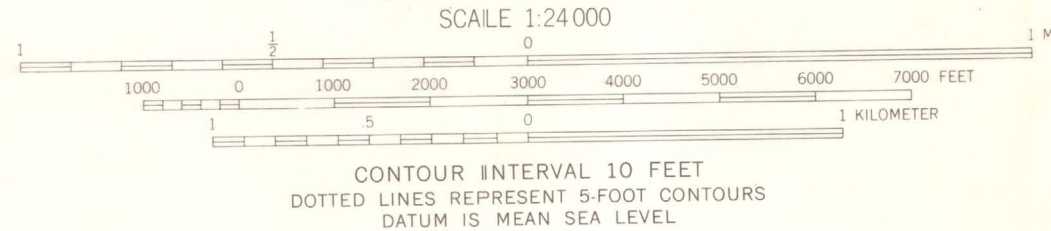
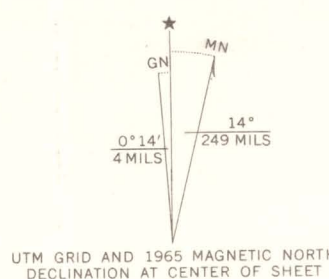
3600 LINE  
(SACATON NE)

UNITED STATES  
DEPARTMENT OF THE INTERIOR  
GEOLOGICAL SURVEY

FLORENCE QUADRANGLE  
ARIZONA-PINAL CO.  
7.5 MINUTE SERIES (TOPOGRAPHIC)



Mapped, edited, and published by the Geological Survey  
in cooperation with the Bureau of Reclamation  
Control by USGS and USC&GS  
Topography by photogrammetric methods from aerial  
photographs taken 1963 and planetable surveys 1965  
Polyconic projection. 1927 North American datum  
10,000-foot grid based on Arizona coordinate system, central zone  
1000-meter Universal Transverse Mercator grid ticks,  
zone 12, shown in blue  
Fine red dashed lines indicate selected fence lines  
Areas covered by dashed light-blue pattern  
are subject to controlled inundation



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ROAD CLASSIFICATION  
Heavy-duty ——— Light-duty ———  
Medium-duty ——— Unimproved dirt ———  
U. S. Route ——— State Route ———

FLORENCE, ARIZ.  
N3300—W11225/7.5  
1965  
AMS 3750 III SW—SERIES V898



a v o

TO H. Lanier LOCATION Ft. Wayne, Ind.  
FROM D.P. Bellum LOCATION Tucson, Ariz.  
SUBJ. Conoco's Florence Property DATE April 8, 1974

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I discussed the status of the Florence property with Adolph Mitterer on April 5.

Conoco's board has approved the third phase of development which consists of drilling a 400-500 foot shaft, underground development to obtain samples of both oxide and sulfide ore, and a pilot plant to process the samples. Teton Development Co. will drill the shaft, Centennial Development Co. will do the underground development, and Arthur G. McKee will build the pilot plant and surface facilities. The pilot plant will process 100 TPD of oxide ore with a LIX recovery circuit, but Conoco is planning to also test a chemical extraction method developed by their research department. The pilot plant will also process 50 TPD of sulfides in a conventional flotation circuit. The plans are to complete pilot plant work by mid-1976.

The present development and production plans are to start mine stripping and the construction of the oxide leaching plant with a LIX plant and electro-winning facility by mid-1976 with production of cathode copper starting in 1979. The leach plant capacity will be 30,000 TPD of ore and they expect another 10,000 TPD for dump leaching. At this time they have no arrangements for the sale of the cathode copper or for the purchase of acid. The concentrator construction will start in 1981 with start-up scheduled for 1984 when the sulfide ore is encountered. The concentrator will have a capacity of 50,000 TPD. The annual output of cathode copper will be about 40,000 tons and the annual output from the concentrator will be 50,000 tons of copper (650 TPD of concentrate).

A. Mitterer said that Conoco's plans are to develop the mine and construct the concentrator without partners. I could not determine if this is his desire, the desire of the Minerals Division management, or the decision of Conoco's corporate management. However, they are interested in a joint venture arrangement for a smelter and refinery. They would be particularly interested in a joint venture with a firm that also has concentrates to be smelted so that the smelter could be sized to produce at least 1300 TPD of acid for the leaching plant. They would like to have part of the smelter in production by 1979 so that they could obtain some acid from it for the start-up of the leaching plant with an expansion for Conoco's concentrates in 1984. They have been talking to Bagdad Copper about a possible smelter joint venture.

I believe that if Essex is interested in a joint venture arrangement for any part of the Florence project, contact should be made with the Conoco vice president responsible for the mining activities. I believe that J.S. Royds has this responsibility and he probably is located in their New York office.

DPB

DPB:td



AKV

GEOLOGY OF THE  
FLORENCE DEPOSIT,  
FLORENCE, ARIZONA

By

R. E. ANDERSON  
C. R. KNAPP  
J. D. LANGLOIS  
R. W. THRELKELD

December, 1971



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## INTRODUCTION

Florence, the Pinal County seat, is roughly halfway between Tucson and Phoenix in Southern Arizona, along the Gila River as seen in figure 1. It is a long established farming community, where the County Court House and the State Prison are located, never suspected of being economically mineralized until quite recently.

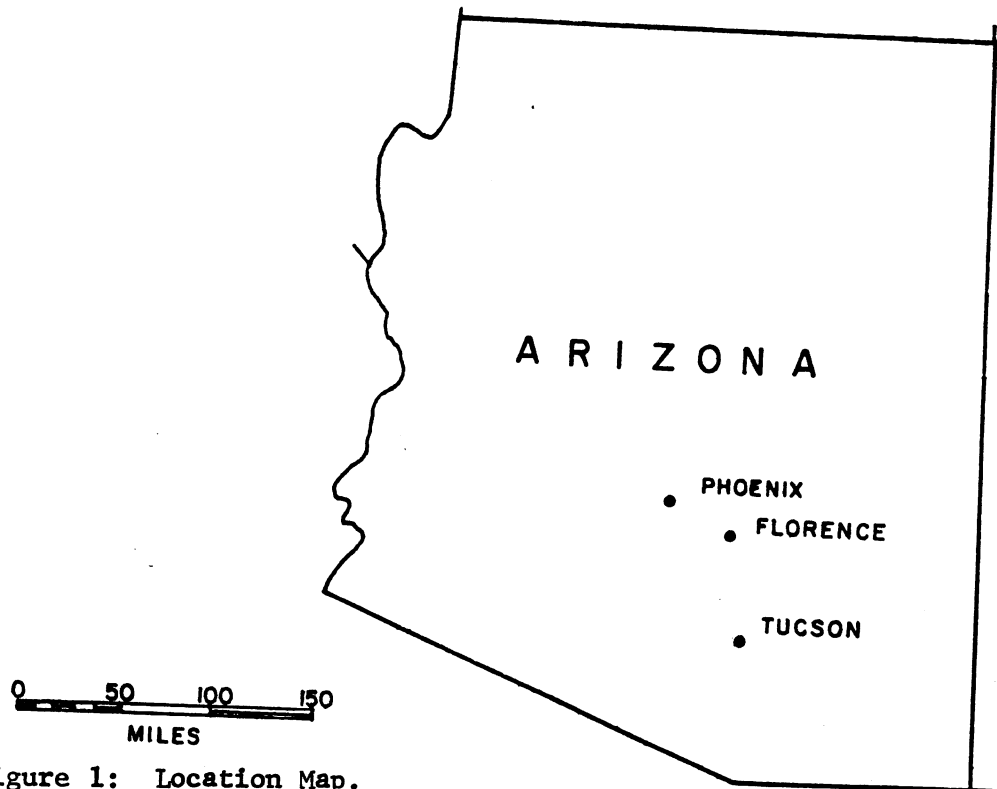


Figure 1: Location Map.

## RESULTS TO DATE:

Since CONOCO's initial discovery in February, 1969, 114,057 feet of core and rotary drilling have been done in the area up to the present time; of this, 64,949 feet were drilled in 1971. 43,653 feet have been drilled in and adjacent to the ore body on 1000' centers with some 500' centers.



21,296 feet were drilled in location of the ore body, perimeter drilling, validation, and assessment drilling. Detailed logging of drill cores is the only way to understand the size, shape, and direction of the ore body. Careful logging of the rotary drill cuttings has shown us the thickness of overburden, mineralization and the alteration zones. The ore body is outlined in figure 2.

During the early phase drilling, IP, gravitometer, and mercury vapor surveys were done to aid in location of and possibly outline the ore body. This year an IP survey was extended to evaluate the area around the Florence ore body and locate targets for exploration drilling. Similarly an MV survey was run covering much of the same area. About 99% of the areas surveyed are covered by overburden and conglomerate. Subsequent drilling on some of the MV and/or IP highs showed only a few areas which warrant further drilling because of the alteration or mineralization in the rock. The Florence ore body is entirely buried under at least 320 feet of alluvium, Gila, and Whitetail conglomerate so a great deal of drilling is needed during the next two years to even begin to truly understand this ore body, and arrive at an accurate estimation of tonnage and grade.

PLANS:

Our drilling program for 1972 will involve drilling approximately 83 holes for a total drilling depth of 96,900 feet. Of this, 76,900 feet are planned within the ore body and on its perimeter to get a more reliable calculation on the reserves. 20,000 feet are planned as validation, assessment, and exploration drilling. Isopach maps of ore thickness and ore grade are being used to prepare a priority of holes to be drilled. Eleven holes will be deepened; this is calculated in the predicted depth.

Our logging is done on a form for computer use, and a close correlation is kept between each geologist on the Florence project to make sure of uniformity.



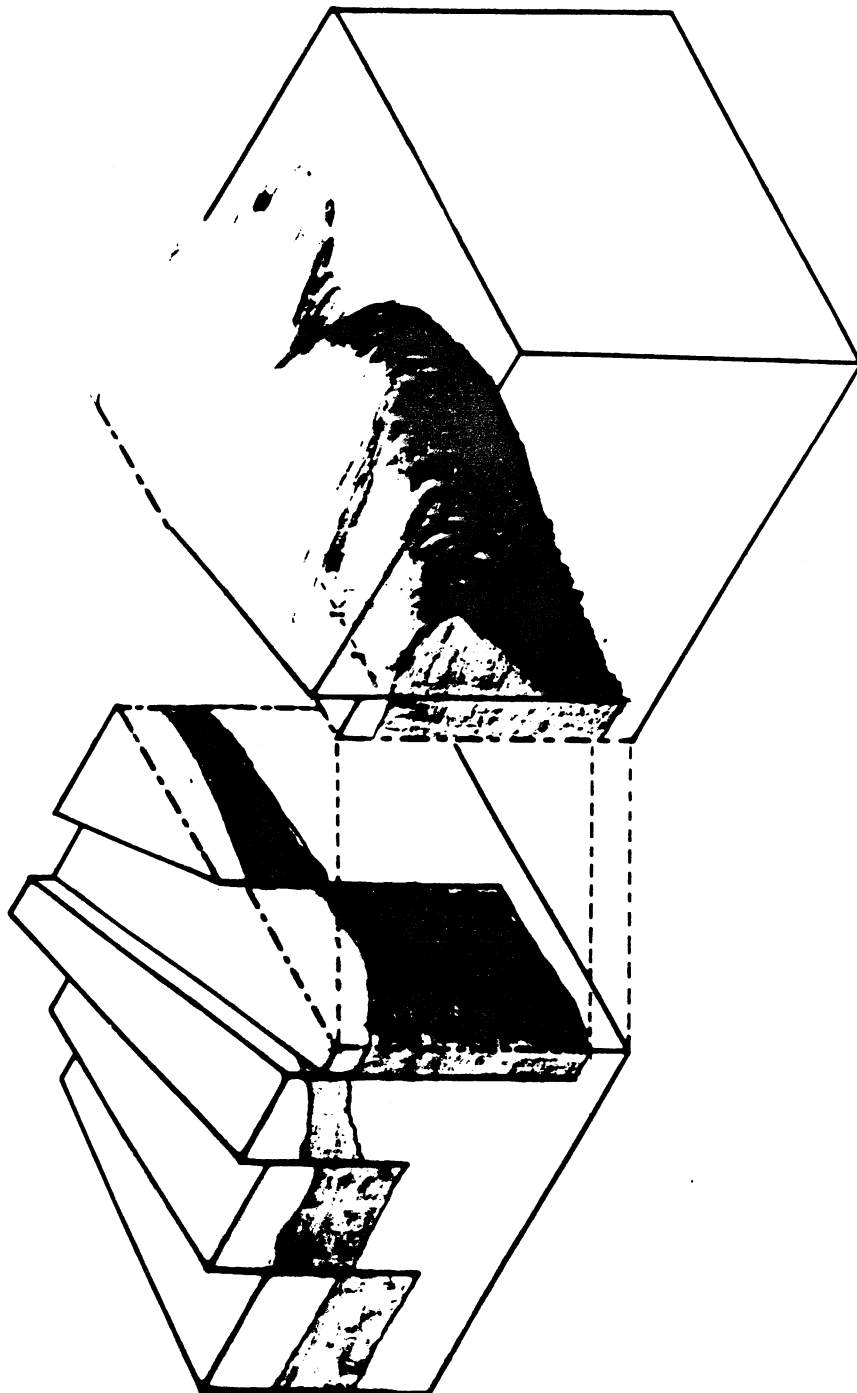


Figure 2: BLOCK DIAGRAM OF FLORENCE DEPOSIT



All the previous drill cores have been relogged onto this form. The computer interpretation on the geologic data is expected to be of tremendous help, both in a better understanding of the Florence ore body and for information that will aid future porphyry copper exploration. In 1972 we hope to develop a better understanding of the alteration sequence from fluid inclusion studies, and emission spectrographic analysis for a number of elements from each alteration zone.

HISTORY:

Approximately 1800 years ago, the Ho Ho Kam Indians moved into the Florence area, settled and began farming right through the area of our ore deposit. Their settlement lasted about 1250 years until 1450 A. D. when they left the Gila Valley, and completely disappeared. One of the speculations on what happened to the tribe is that they joined another tribe, and became the present Pima-Papago Indian tribe. If this is the case, they became a more nomadic, less civilized group.

Toward the end of the 17th Century, a missionary group led by Father Kino came through the area and visited the Ho Ho Kam ruins at what later became Casa Grande National Monument. In 1853, the United States purchased from Mexico what they named the New Mexico Territory and included what ten years later became the Territory of Arizona. About 1866, the town of Florence came into existence, making it the fifth oldest settlement in Arizona. It began as a farming community, which was occasionally attacked by the Apache Indians, and frequently harassed by them until close to the turn of the century. The United States Cavalry had troops stationed in the area following the Civil War adding some protection for the early settlers. Florence enjoyed a surge of growth in 1908 when the State Prison was placed there. Subsequently, it appears to have changed, but slightly.

Much of the Florence mineralized area is held by CONOCO at the original exploration suggestion of Bill Kern and has truly proved out to be an ore body.

Bill was led to Florence in the early 1960's following the Ray? Lineament, and was encouraged by the outcrop on Poston Butte. At that time ASARCO held the property. ASARCO drilled several holes around the edge of the ore body, none of which were in the better portion. Subsequently, they dropped their leases and permits. In 1969, Bill came with CONOCO and quickly picked up all the available State leases. Drilling began under the direction of Ray Barkley and quickly indicated that Cecil England's and Earnest McFarland's properties were essential. At this point ASARCO decided they wanted all the Florence property they could get back again. A nail-biting negotiation began with former Governor, Senator, Judge McFarland with CONOCO finally getting the lease option on his property. ASARCO picked up a lease option on the property to the west of McFarland and CONOCO got all the rest.

#### GENERAL GEOLOGY

Drilling to date has shown that the Florence ore body nicely fits a porphyry copper deposit as described by Lowell and Guilbert (pp. 374-375):

"A porphyry deposit is here defined as a copper and/or molybdenum sulfide deposit consisting of disseminated and stockwork veinlet sulfide mineralization emplaced in various host rocks that have been altered by hydrothermal solutions into roughly concentric zonal patterns. The deposit is generally large, on the scale of several thousands of feet, although smaller occurrences are recognized. The relatively homogeneous and commonly roughly equidimensional deposit is associated with a complex, passively emplaced stock of intermediate composition including porphyry units. It contains significant amounts of pyrite, chalcopyrite, molybdenite, quartz, and sericite associated with other alteration, gangue, and ore minerals and metals including minor lead, zinc, gold, and silver. Mineralization and alteration suggest a late magmatic-mesothermal temperature range. The deposit is generally associated with breccia pipes, usually with a large crackle brecciation zone, and is surrounded by peripheral mineralization.

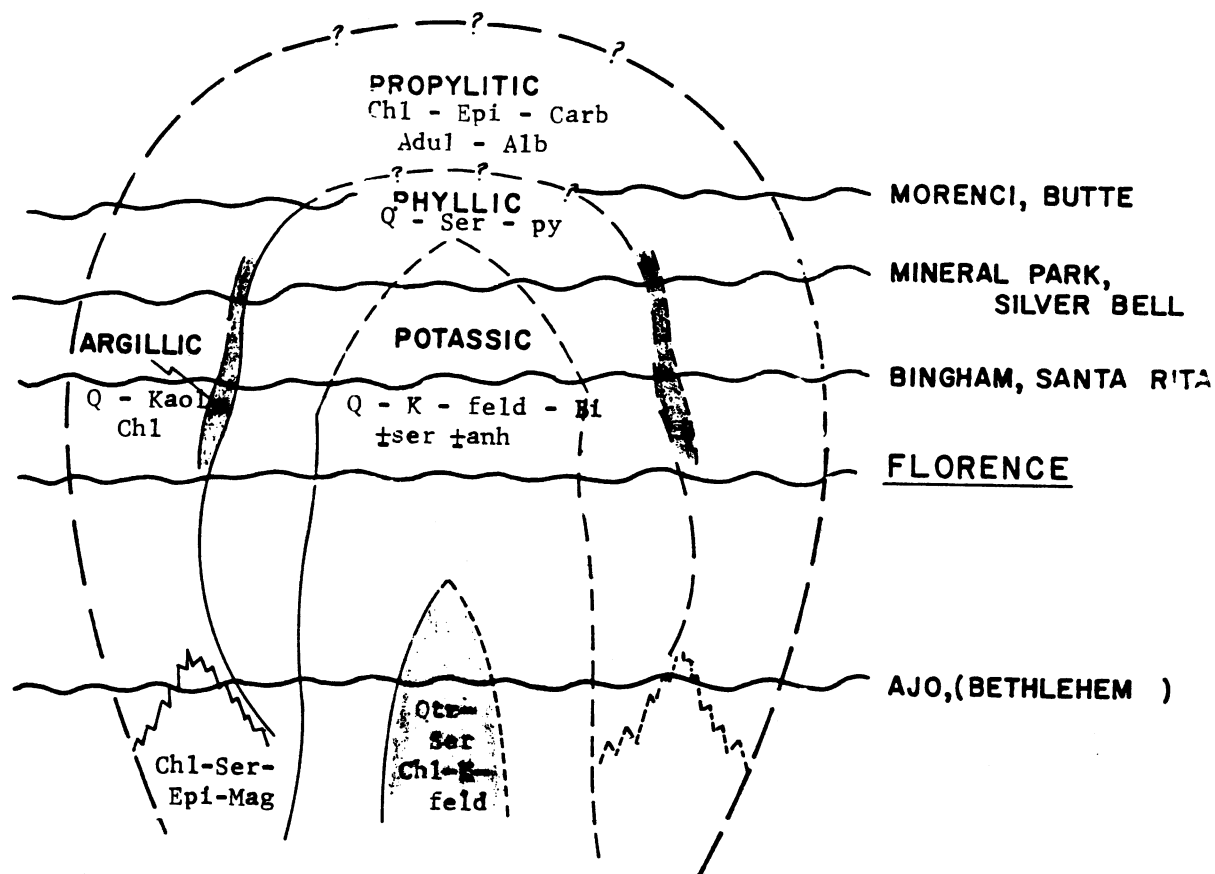
The grade of primary mineralization in typical porphyry copper deposits ranges up to 0.8% Cu and 0.20% Mo, ..." Figure 3 shows the alteration zoning.



Figure 3

## ALTERATION ZONING IN PORPHYRY ORE DEPOSITS

Derived from Lowell &amp; Guilbert (p. 405)



## Abbreviations:

Adul. --- Adularia  
 Alb. ---- Albite  
 Bi. ----- Biotite  
 Carb. --- Carbonate  
 Chl. ---- Chlorite  
 Epi. ---- Epidote  
 Kaol. --- Kaolinite  
 K-feld. - Potassium feldspar  
 Mag. ---- Magnetite  
 py. ----- Pyrite  
 Qtz. or Q Quartz  
 Ser. ---- Sericite

The Florence deposit's beginning dates back to emplacement of a Precambrian granitic batholith. This major granitic batholith, the Oracle Granite, dated in other areas as 1.3 billion years old, is probably the basement quartz monzonite porphyry of the Florence Deposit. It possibly intruded lower Precambrian Pinal (?) Schist which has been located about three miles off to the northwest. Mafic dykes and sills intruded the porphyry, probably in late Precambrian.

During late Precambrian and early Paleozoic, most of Arizona was subjected to regional stress which brought about a number of lineaments. Two and possibly three of these lineaments crossed in the Florence area creating a crustal weakness for the Laramide intrusive and hypogene activity that formed the ore body as shown in figure 4.

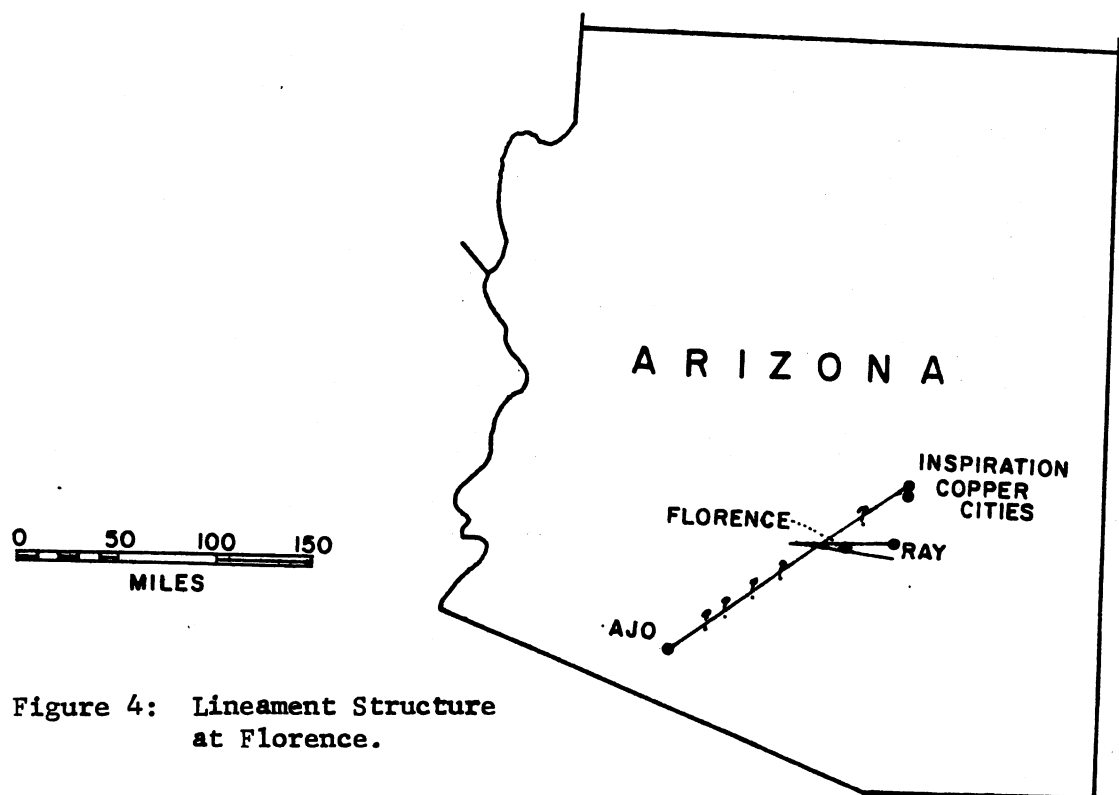


Figure 4: Lineament Structure at Florence.

The Ray Lineament trending N70 E goes through the Ray Deposit, Florence Deposit, Blackwater Deposit, and the Sacaton Deposit. The Florence Deposit appears to trend east-northeast. There is a N70 E vein-dyke system two miles northeast



of Florence on the Aztec Property. The Apollo 9 infrared photo over Florence recorded a N70 E structural feature just north of Poston Butte apparent for over thirty miles. Another vein-dyke system on the Aztec Property trends east-west. This east-west lineament? is supported by high altitude (60,000') photos showing this trend is at least twenty miles wide. An east-west vein-dyke system is on the Red Hills Prospect nine miles to the east. A north 45 E possible lineament is also exposed by a vein-fracture system on the Aztec Property. This lineament? if extended, goes through the Superior, the Globe-Miami districts to the northeast, and the Ajo Deposit to the southwest. The Magma mine in Superior reports both east and northeast trending faults of an "old system", (Hammer and Peterson, p. 1294). The northeast system appears to be similar in Globe-Miami which "... suggests that the faults in the two areas are part of the same system," (Hammer and Peterson, p. 1292). At the Copper Cities Mine, "... the Lost Gulch Quartz Monzonite - the host rock for the Copper Cities Mine - is intruded along a northeastward zone ..." (Simmons and Fowells, p. 152). At Ajo, "The Able fault is a very strong premineral fault that cuts across the northwest corner of the pit. The fault strikes N40 E ...", (Dixon, p. 126).

If deposition took place during Paleozoic, it was completely eroded away by early Tertiary times throughout the area. During Late Paleozoic, the Precambrian-early Paleozoic stress fields were reactivated, creating a zone of weakness for the Laramide Orogeny. At Florence, during Laramide time, a granodiorite porphyry complex and associated sills and dykes were intruded into the quartz monzonite porphyry. The drilling to present indicates that associated breccia pipes may have developed. Toward the end of and following the intrusion, hypogene alteration and mineralization occurred. Hypogene mineralization generally preferred the quartz monzonite porphyry as the host rock but alteration was not limited to this rock type.

the oxide zone. At depth, drilling in the sulfide zone, then oxide zone has been encountered, going back into the sulfide zone. Low angle faulting associated with high angle graben-horst faulting is seen in a number of porphyry deposits in Arizona. A generalized picture of what may have happened in Florence is shown in figures 5a and 5b. During oxidation and faulting in Florence, one block probably fluctuated bringing about an intermixed oxide-sulfide zone. The oxidation occurred principally before the uplifting, which brought about the most recent graben-horst faulting, and rapid erosion. The most recent structural trends are to the north and north-northwest indicated by present drilling and an earlier gravitometer survey as shown on figure 6. From the 60,000' photos, a N25 W lineament or major structural feature south of the Gila River may also cross through the ore body. It appears much narrower (one mile) and more prevalent, indicating more recent movement. The Apollo 9 infrared photo over Florence recorded this N25 W structural feature north of the River, an estimated two miles west of Poston Route. The bottom bed deposited during this period of recent faulting is probably the post Laramide, Tertiary, Whitetail or Yellow Peak conglomerate described by Blucher, (p. 72) which fills the downdropped blocks and is capped by a flat-lying reddish-brown sandstone (Phil Sterling). Lying unconformably on this is the Gila or Rock Peak conglomerate (Sell, p. 73) if our interpretation is correct. Recent alluvium covers the conglomerate in the Florence area.

The ore body now lies entirely beneath the water table which will cause some mine operation problems of an opposite nature from most desert mines continually searching for water.

#### ROCK TYPES OF THE DEPOSIT

Mineralized rock types associated with the Florence deposit are intrusive and hypabyssal in origin. Approximately 90% of the mineralization is contained in quartz monzonite porphyry and granodiorite porphyry. Other igneous rocks include diabase, diorite, basalt, andesite, dacite, latite, rhyodacite, and



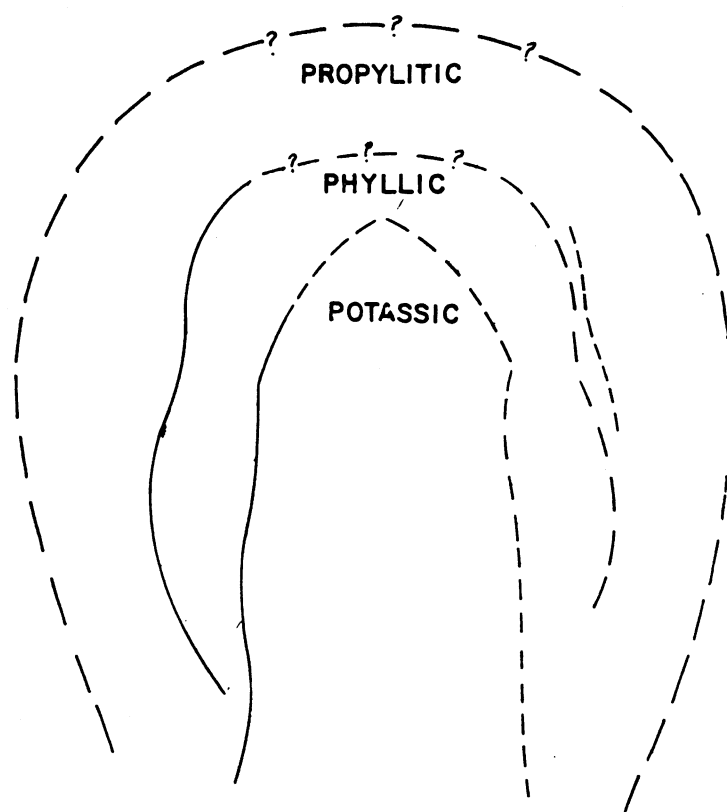
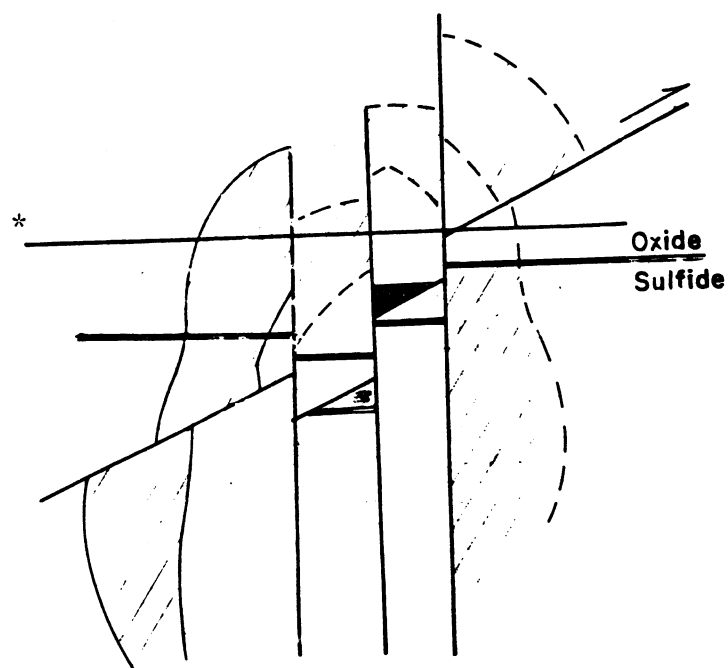


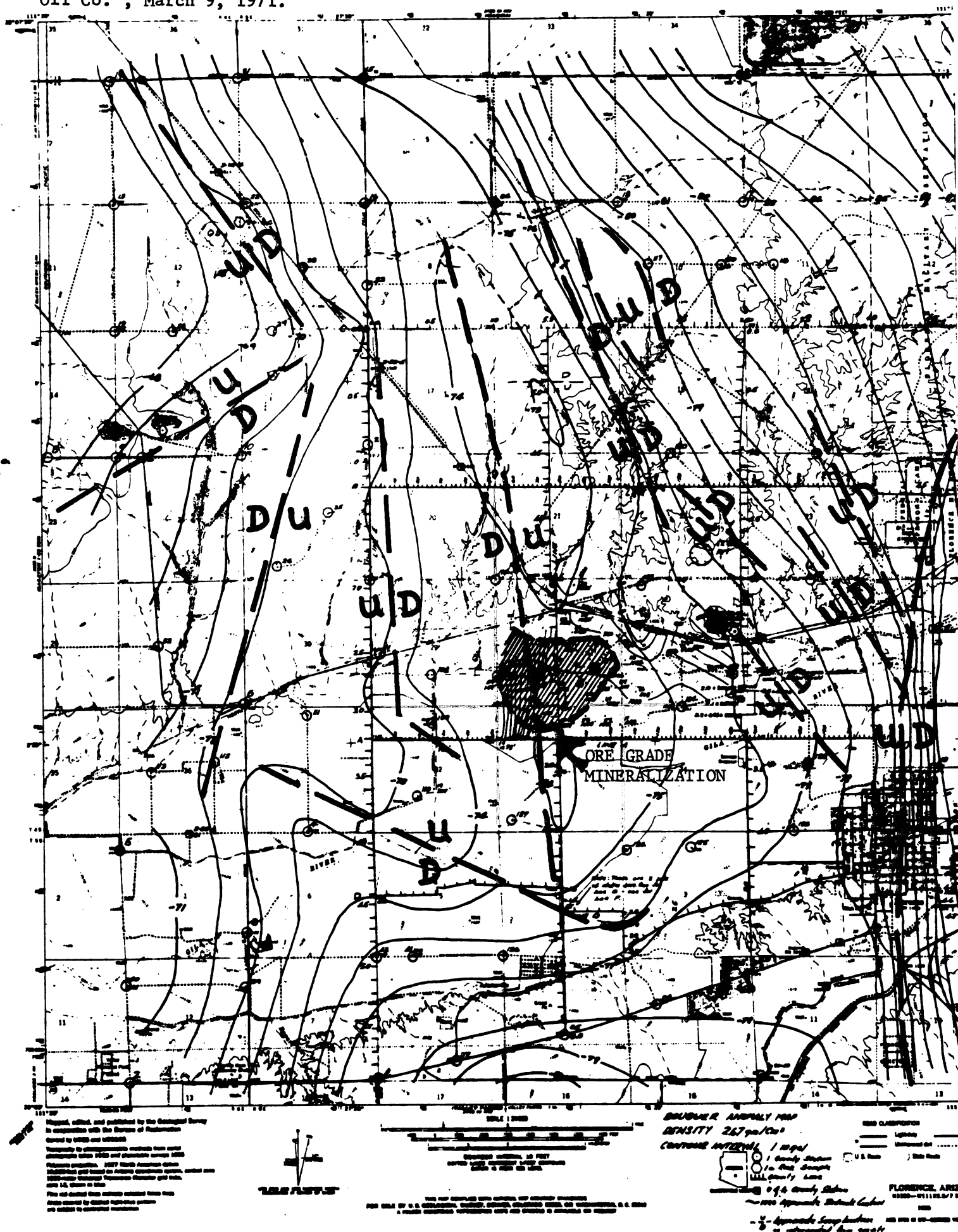
Figure 5a



\* Bedrock surface before most recent faulting.

Figure 5b. Post-Mineral Horst-Graben and Related Faulting.

### Figure 6



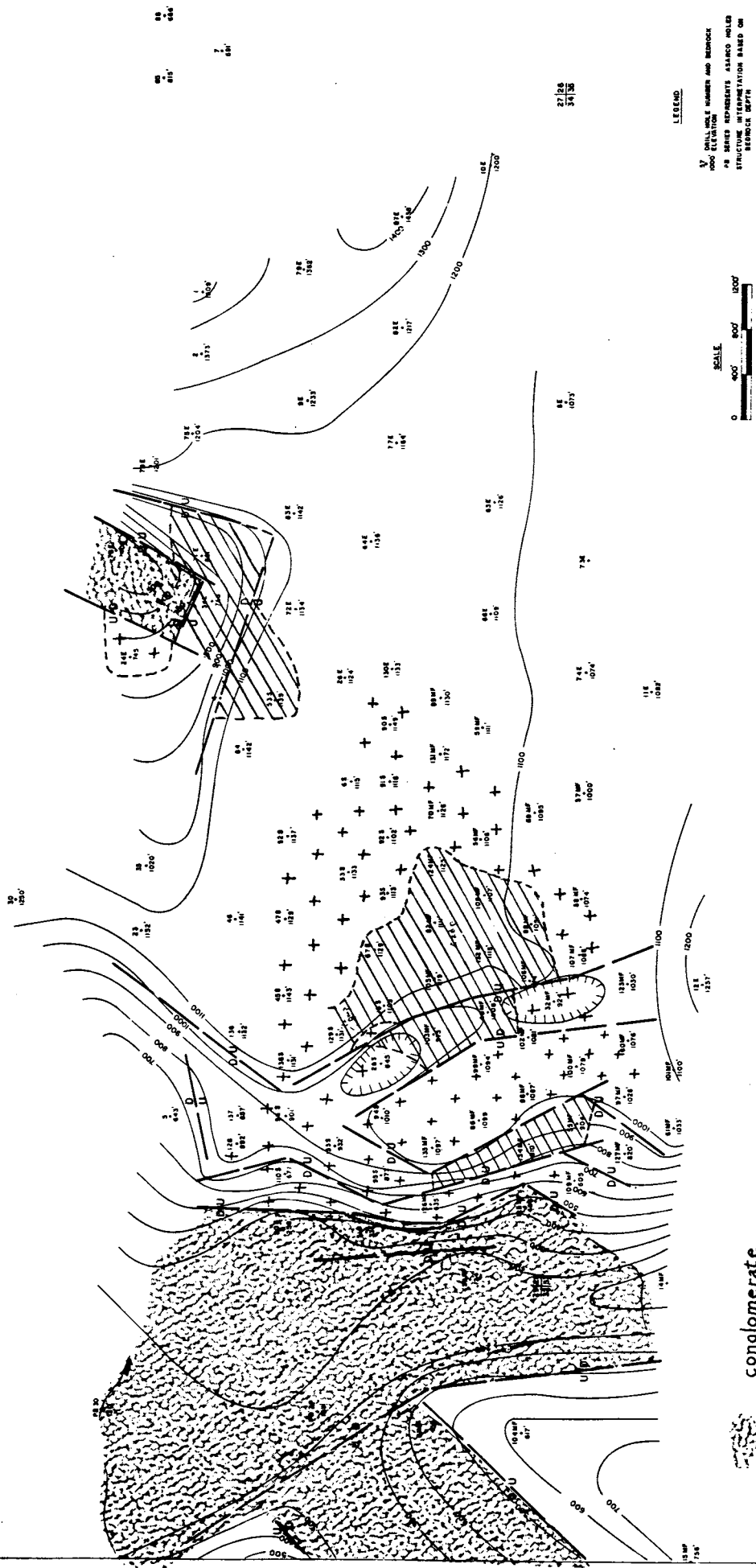


aplite. The ages of these intrusive rock types are imprecisely known, but based on comparison to other deposits and knowledge of Arizona geology, three ages of igneous activity are postulated - Precambrian, Laramide, and Tertiary-Quaternary. Subsequent to crystallization of these intrusive rock types a final period of erosion, faulting, and deposition occurred which resulted in burial of the deposit by Tertiary-Quaternary conglomerate and alluvium. A geologic map at the 500' level of the deposit is presented in figure 7 and a cross section is shown in figure 8.

Precambrian rock types include an intrusive quartz monzonite porphyry mass and dikes of diabase and diorite. The quartz monzonite porphyry is volumetrically the largest mineralized rock type and is correlated with the Oracle quartz monzonite porphyry batholith. Aplite dikes occur within the intrusive mass with gradational to sharp contacts. These contacts indicate a close genetic association with the late stages of quartz monzonite porphyry formation.

The quartz monzonite porphyry intersected in the Florence core consists of coarse grained, subhedral, twinned K-feldspar phenocrysts (25-35%) embedded in an hypidiomorphic to xenomorphic, medium grained matrix of quartz (15-30%), plagioclase (20-35%), and biotite (5-8%). Locally graphic and myrmekitic intergrowths are developed although these could be a result of hydrothermal alteration. K-feldspar phenocrysts in this rock are generally dusty textured, pink colored, slightly perthitic, and locally may show some microcline twinning. These K-feldspar phenocrysts may range up to two inches in size and include plagioclase and biotite crystals. Plagioclase matrix crystals are commonly zoned and generally range in composition from andesine ( $An_{38}$ ) to oligoclase ( $An_{26}$ ). Accessory minerals include apatite, rutile, sphene, zircon, and magnetite.

The quartz monzonite porphyry is the major host for copper mineralization.



conglomerate  
 granodiorite porphyry  
 quartz monzonite porphyry

Figure 7: 500 Level Geology Map

CONTINENTAL OIL COMPANY		BEDROCK ELEVATION MAP	
MINERAL EXPLORATION DEPARTMENT		ADJUSTED TO SEA LEVEL	
METALLURGICAL SECTION		SHEET NO. 14	
DATE	10/1/50	PROJECT	500 LEVEL GEOLOGY
BY	W. H. HARRIS	SCALE	1" = 100'
CHECKED BY	W. H. HARRIS	DATE	10/1/50
APPROVED BY	W. H. HARRIS	LOCATION	500 LEVEL
REVISIONS		REVISIONS	
1. 10/1/50		1. 10/1/50	
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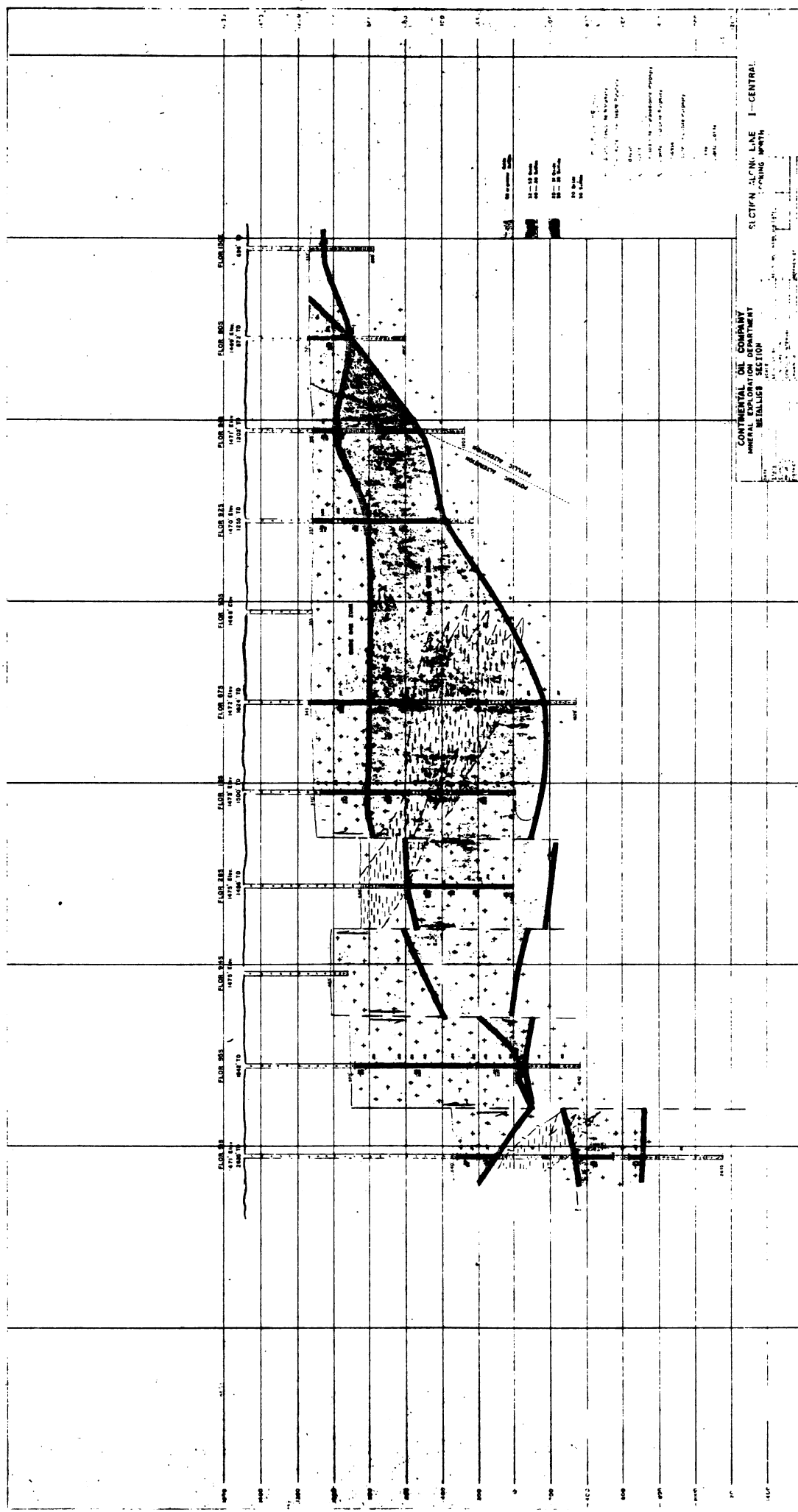


Figure 8



Fracturing, brecciation, and development of permeable zones related to emplacement of Laramide granodiorite porphyry controlled the distribution of mineralization. Chalcopyrite as the major primary copper sulfide occurs in veinlets and disseminations with biotite in the quartz monzonite porphyry. Adjacent to contacts with granodiorite porphyry the quartz monzonite porphyry is generally more intensely mineralized with chalcopyrite.

Diabase and diorite apparently occurring as dikes or sills are probably in part of Precambrian age. The exact age of these more mafic rocks relative to quartz monzonite porphyry is unknown. Inclusions of some varieties of these mafic rocks are found in quartz monzonite porphyry but in other cases, some contacts indicate a younger age for the mafic rocks. The diabases and diorites are fine grained, dark, and dense, ranging in core intercepts from a few inches to tens of feet. Petrographically these rocks consist of plagioclase laths embedded in a mafic matrix. Although these basic rock types are volumetrically low in abundance, the diabases and diorites are highly mineralized. Locally the altered varieties of these rocks contain 10-15% chalcopyrite.

Laramide igneous activity is represented by granodiorite porphyries. The granodiorites associated with the Florence deposit are subdivided into three petrographic varieties. The exact relationship between the two major varieties is uncertain although there is a genetic relationship. In some cases inclusions of the lighter, more coarse grained granodiorite type 1 are present in the darker, finer grained granodiorite 2, but in other cases the contacts are gradational. This would suggest that granodiorite type 1 is a slightly earlier phase of the Laramide intrusive system than type 2 granodiorite. The relationship of granodiorite porphyry 3 to the other varieties is unknown.

Granodiorite porphyry 1 is a light gray, medium to fine grained rock with quartz, plagioclase, and biotite phenocrysts set in a xenomorphic matrix of quartz and K-feldspar. Modal ranges are quartz (15-20%), plagioclase (40-50%), K-feldspar (15-20%), and biotite (5%). Myrmekitic and graphic textures are locally developed. Plagioclase phenocrysts tend to be compositionally zoned and range from  $An_{30}$  to  $An_{40}$ . Accessory amounts of rutile, apatite, zircon, and magnetite are also present.

Granodiorite porphyry 2 is a medium gray, more mafic and finer grained rock than granodiorite porphyry 1. Quartz, biotite, and plagioclase phenocrysts are set in a very fine grained xenomorphic matrix of quartz, K-feldspar, biotite, and plagioclase. Modal composition is quartz (20%), plagioclase (50%), K-feldspar (15-20%), and biotite (5-10%). Plagioclase phenocrysts are oscillatory zoned with an average composition in the andesine range. Accessory minerals include zircon, apatite, and magnetite.

Based on current drill hole data, the thickest portion of the granodiorite porphyry mass is present in the vicinity of holes 48-MF and 103-MF (1100' thick). Fingers of granodiorite porphyry extend to east-northeast and to the northwest of this thickest section. Another fairly thick portion of granodiorite porphyry extends to the northeast of the ore body and is associated with mineralization in holes 34-E and 71-E.

Both granodiorite types are generally weakly mineralized in comparison with quartz monzonite porphyry. This relative lack of mineralization may be a function of composition or structure. In general, the granodiorite porphyries are less intensely fractured than quartz monzonite porphyry. Mineralization when developed in the granodiorite porphyry is mostly veinlet controlled. In terms

of geometry of granodiorite porphyry in relation to ore and the nature of contact mineralization the granodiorite porphyry must have played a fundamental role in the development of porphyry copper type mineralization. Following the intrusion and partial crystallization of granodiorite porphyry, fractures developed in the largely consolidated crystal-magma system. Probably at this late stage of crystallization, saturation occurred with a resulting migration of water to zones of weakness. This migration of water and dissolved constituents was responsible for the deposition of copper sulfides and hydrothermal alteration.

The third variety of granodiorite porphyry is greenish gray, very fine grained, and contains only plagioclase and quartz phenocrysts. Granodiorite porphyry 3 appears to be restricted to dikes or small intrusive masses. A modal composition for this mineralized variety is quartz (15-20%), K-feldspar (15-20%), biotite (5-10%), and plagioclase (40-55%).

Laramide igneous activity may also have been responsible for the development of mixed or intrusive breccia zones. These breccia zones commonly consist of mixtures of quartz monzonite porphyry, granodiorite, diorite, diabase, and other rock types set in a fine grained, finely comminuted matrix. These zones may also be intensely mineralized and resemble breccia pipes.

Following the intrusion of granodiorite types, dikes and sills of Tertiary hypabyssal intrusive rock are intersected in core. These rocks include latites, dacites, andesites, basalts, quartz latites, and diorites. These rock types are devoid of primary porphyry copper type mineralization. However, during subsequent oxidation of sulfide copper mineralization, latites, andesites, and basalts locally acted as precipitating hosts to the copper bearing solutions. Thus, these rocks locally form rich zones of exotic oxide copper mineralization.

ROCK TYPES OF THE DEPOSIT, con't.

During Tertiary-Quaternary time faulting and burial of the deposit occurred. The thickness of this overburden is variable ranging from 320' to 1225' in the area of the ore body. The depth to the pre-overburden surface along the eastern half of the ore body varies from 320' to 350' but along the western portion the depth of burial is greatest as a result of faulting. This overburden consists of partially consolidated conglomerate and alluvium.

STRUCTURE

Structural interpretations within the Florence deposit have been made entirely on the basis of drill hole information. As the deposit is covered by a minimum of 320' of conglomerate and alluvial cover, structural interpretations have been based on depth to basement rock, structures encountered in core drilling and alteration changes.

Two sets of post-mineralization faulting based on strike trend associated with mid-Tertiary (?) Basin and Range orogenesis have been recognized at Florence as indicated by figure 9. The major set is a series of north-south to north 25° west striking normal faults. The west edge of the deposit is bounded by at least three normal step faults downdropping the west side approximately 700 feet as shown in figure 8. A graben in the center of the deposit 1000' long and 400' wide has been downdropped a maximum of 266'.

Possible lateral movement has occurred along the west side of the graben. Large thicknesses of granodiorite porphyry have been encountered within the graben and can be correlated to the east. The upthrown area west of the graben contains small amounts of granodiorite porphyry as dikes or sills. Closer spaced drilling is needed within the upthrown block near the west edge of the graben and to the northwest and southwest portions of the deposit to determine direction and distance of lateral movement.



CONTINENTAL OIL COMPANY MINERAL EXPLORATION DEPARTMENT		BEDROCK ELEVATION MAP ADJUSTED TO SEA LEVEL	
METALLICS SECTION		POST-LARAMIDE STRUCTURE	
DATE	REVISION	SHEET NO. 12,408	
5/20/61	1	SHEET NO. 12,409	
5/20/61	2	SHEET NO. 12,410	
5/20/61	3	SHEET NO. 12,411	
5/20/61	4	SHEET NO. 12,412	
5/20/61	5	SHEET NO. 12,413	
5/20/61	6	SHEET NO. 12,414	
5/20/61	7	SHEET NO. 12,415	
5/20/61	8	SHEET NO. 12,416	
5/20/61	9	SHEET NO. 12,417	
5/20/61	10	SHEET NO. 12,418	
5/20/61	11	SHEET NO. 12,419	
5/20/61	12	SHEET NO. 12,420	
5/20/61	13	SHEET NO. 12,421	
5/20/61	14	SHEET NO. 12,422	
5/20/61	15	SHEET NO. 12,423	
5/20/61	16	SHEET NO. 12,424	
5/20/61	17	SHEET NO. 12,425	
5/20/61	18	SHEET NO. 12,426	
5/20/61	19	SHEET NO. 12,427	
5/20/61	20	SHEET NO. 12,428	
5/20/61	21	SHEET NO. 12,429	
5/20/61	22	SHEET NO. 12,430	
5/20/61	23	SHEET NO. 12,431	
5/20/61	24	SHEET NO. 12,432	
5/20/61	25	SHEET NO. 12,433	
5/20/61	26	SHEET NO. 12,434	
5/20/61	27	SHEET NO. 12,435	
5/20/61	28	SHEET NO. 12,436	
5/20/61	29	SHEET NO. 12,437	
5/20/61	30	SHEET NO. 12,438	
5/20/61	31	SHEET NO. 12,439	
5/20/61	32	SHEET NO. 12,440	
5/20/61	33	SHEET NO. 12,441	
5/20/61	34	SHEET NO. 12,442	
5/20/61	35	SHEET NO. 12,443	
5/20/61	36	SHEET NO. 12,444	
5/20/61	37	SHEET NO. 12,445	
5/20/61	38	SHEET NO. 12,446	
5/20/61	39	SHEET NO. 12,447	
5/20/61	40	SHEET NO. 12,448	
5/20/61	41	SHEET NO. 12,449	
5/20/61	42	SHEET NO. 12,450	
5/20/61	43	SHEET NO. 12,451	
5/20/61	44	SHEET NO. 12,452	
5/20/61	45	SHEET NO. 12,453	
5/20/61	46	SHEET NO. 12,454	
5/20/61	47	SHEET NO. 12,455	
5/20/61	48	SHEET NO. 12,456	
5/20/61	49	SHEET NO. 12,457	
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5/20/61	77	SHEET NO. 12,485	
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5/20/61	79	SHEET NO. 12,487	
5/20/61	80	SHEET NO. 12,488	
5/20/61	81	SHEET NO. 12,489	
5/20/61	82	SHEET NO. 12,490	
5/20/61	83		

The second set of post-mineralization faults are those striking north  $30^{\circ}$ - $35^{\circ}$  east. Northeast faulting within the deposit is known to occur in the northwest portion truncating the  $N25^{\circ}W$  striking graben. A northeast striking fault is suspected to terminate the ore zone on the south side. Drilling in the area has shown intense brecciation in the oxidized zone. The encountered brecciation in conjunction with a change from potassic alteration to propylitic alteration over a horizontal distance of 800 feet southward shows that there is a good possibility of a fault with no apparent change in elevation of bedrock.

The problem of low angle faulting within the Florence deposit is a subject of wide speculation and difficult to determine at this point in our drilling program. Low angle faults have been mapped on other porphyry districts in Arizona and have been shown to be part of the tensional system which produced north-south striking horst and graben structures common to the districts. The displacement on these faults has been normal to the strike of the north-south striking grabens.

Using this information and figure 10 one can see in the northeast portion of the deposit the bottom of the ore zone is dipping approximately  $40^{\circ}$  to the southwest normal to the strike of the graben in the center of the deposit. A breccia zone 10' to 55' wide intruded by a post-mineral basalt was intersected in three drill holes in this area. A three point solution produced a strike of north  $40^{\circ}$  west and dip of  $35^{\circ} \pm 5^{\circ}$  to the southwest normal to the graben and step-faulting within the deposit.

Alteration changes from potassic to sericitic with depth have also been noted in this portion of the ore body. Below the low angle fault sericite and pyrite increase while potassium-feldspar decreases.

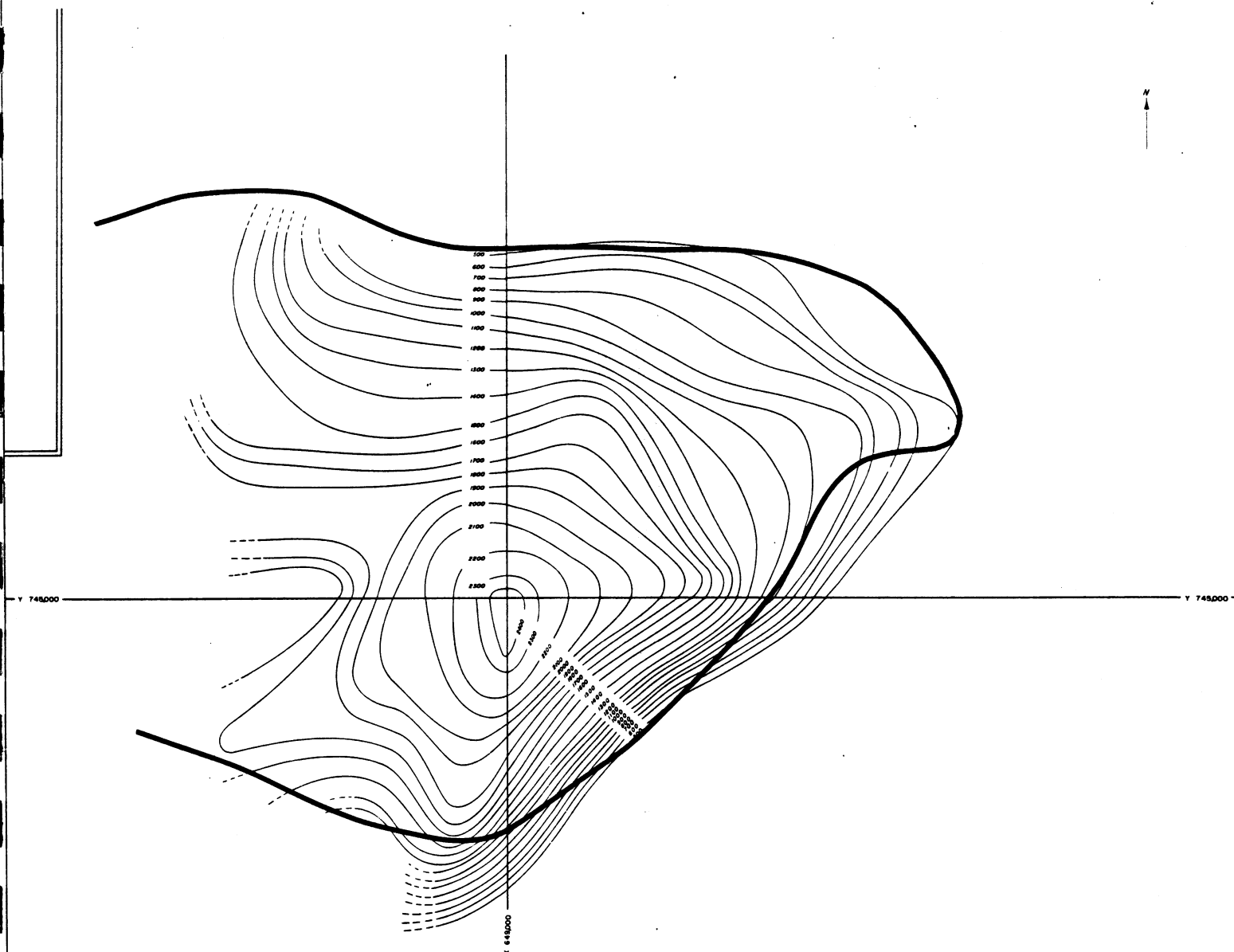


Figure 10

CONTINENTAL OIL COMPANY		ISOPACH OF TOTAL DEPTH	
MINERAL EXPLORATION DEPARTMENT		FROM SURFACE TO BOTTOM OF ORE	
REDAUCE SECTION			
DATE	BY	APPROVED BY	REVIEWED (NAME AND DATE)
PROJECT	LOCATION	SECTION	
WELL NO.	WELL NO.	WELL NO.	
WELL NO.	WELL NO.	WELL NO.	

These three bits of evidence show low angle faulting may have taken place within the Florence deposit and may be a major control over the depth of ore grade material in the northeast portion of the deposit. Displacement along the fault may have moved the hanging-wall to the northeast.

The greatest structural feature and most important for localization of ore within the Florence deposit is the intrusion of Laramide granodiorite porphyry along the N70 E striking Ray Lineament. Figure 11, with the shaded area representing the greatest thickness of Laramide granodiorite porphyry shows the elongation of the granodiorite to the east-northeast with the thickest sulfide mineralization peripheral to the granodiorite. North of the granodiorite sulfide mineralization reaches a thickness of 600 feet. To the south it reaches a maximum of 1400 feet.

Control over formation of sulfide mineralization peripheral to the granodiorite is thought to be a combination of fracturing produced by forceful intrusion of the granodiorite porphyry. Zones of intrusive (?) breccia have been encountered in drilling along contact with quartz monzonite porphyry.

A hypothetical structural sequence for the Florence deposit is postulated as follows: During Laramide orogeny (Late Cretaceous to Early Cenozoic) granodiorite porphyry intruded the Precambrian Oracle quartz monzonite porphyry along the pre-existing Ray Lineament. Mineralization and alteration followed. During Oligocene uplift and erosion occurred with the possibility of tilting to the northeast exposing the ore body. Evidence for tilting has not been specifically found around Florence but regional geology has shown tilting has occurred thirty miles to the northeast. Oxidation subsequently occurred during exposure of the ore body. During Mid-Tertiary, perhaps Miocene-Pliocene, Basin and Range faulting



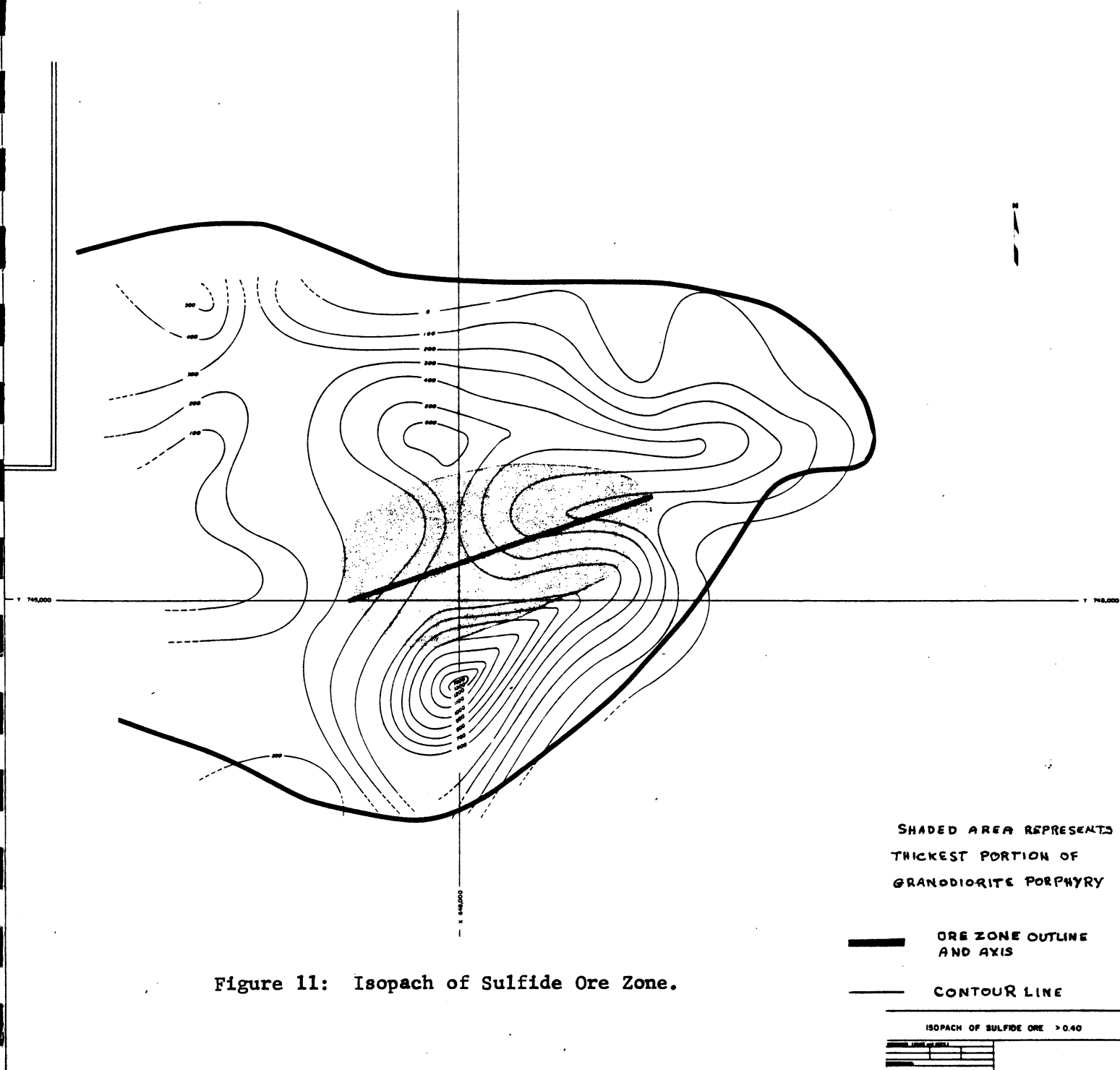


Figure 11: Isopach of Sulfide Ore Zone.

occurred downdropping the west side of the ore body. Deposition of Whitetail conglomerate occurred during, and continued after, Basin and Range orogenesis in Miocene time. Intrusion of volcanic dikes occurred from Mid-Tertiary to as late as Pleistocene(?). No definite age can be correlated as yet. Deposition of Gila (?) conglomerate occurred during Pliocene to Pleistocene times.

#### MINERALIZATION

The Florence deposit contains substantial tonnages of both oxide and sulfide ore types. In general these ore types are distinct from a geological and metallurgical view. Zones of mixed oxide and sulfide are noted only in a few holes. Total tonnage of both ore types amounts to Cu. Ore is thickest in the vicinity of holes 32-MF and 108-MF and in general shows northeast to east-west trends as is shown on figure 12.

Mineralogy of the oxide zone consists predominantly of chrysocolla, tenorite, and medmontite. Additional minerals found in the oxide ore zone include cuprite, azurite, malachite, shattuckite, and native copper. Chrysocolla and tenorite generally occur along fracture planes as fillings and coatings although some chrysocolla does replace plagioclase (as shown by the slides). Medmontite, a montmorillonite type clay containing copper, is a more common replacement of plagioclase sites in the argillized rock. The thickness of the oxide zone is variable ranging from 0'-785'. From available drill hole data the ore averages 250'-300' thick. The thickest portion of the oxide zone lies in the vicinity of holes 86-MF--110-S and 124-MF as shown in figure 13.

The development of the oxide ore zone is related directly to supergene processes. The oxidation of the sulfide ore body and the accompanying argillization are a consequence of supergene alteration.

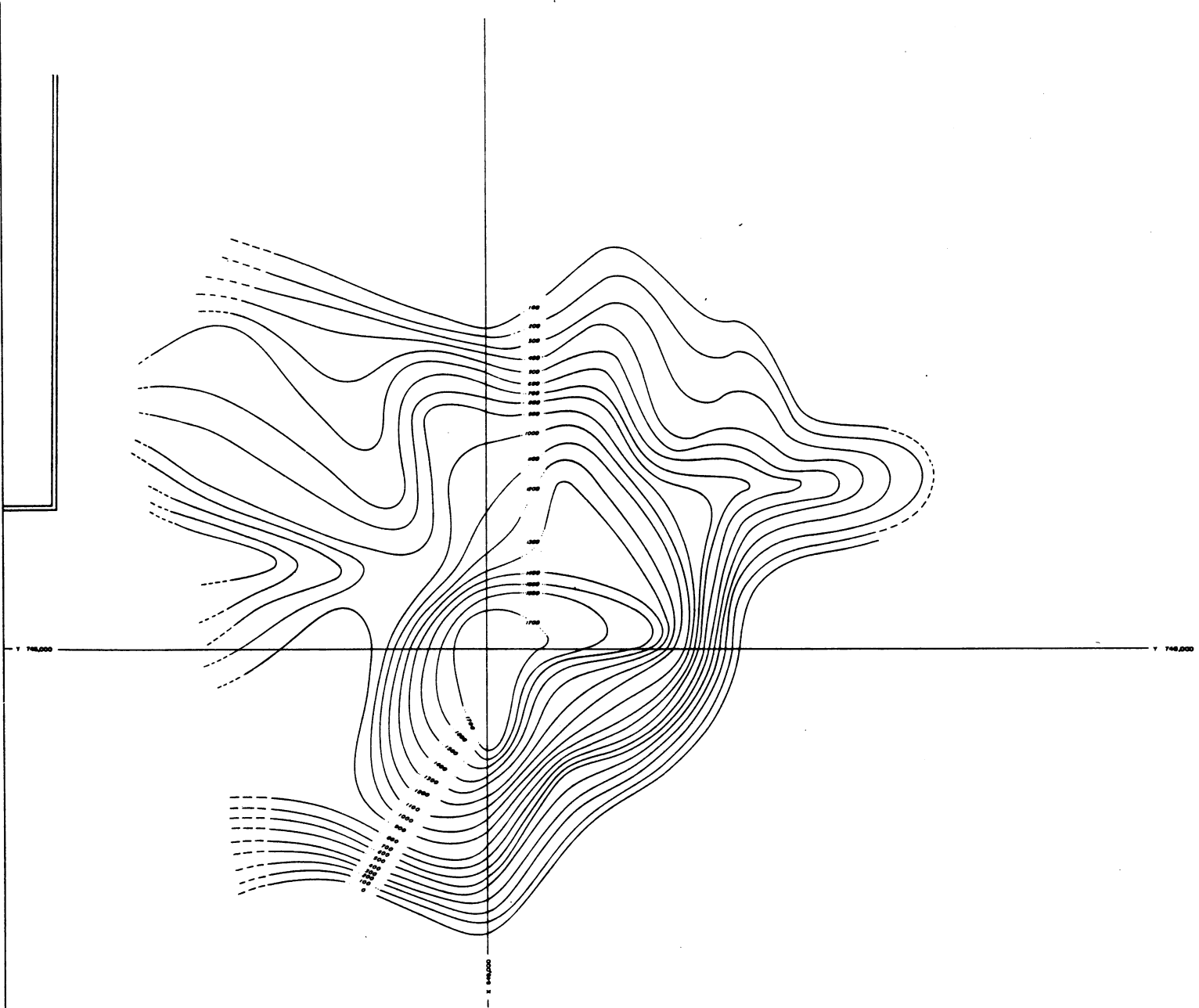


figure 12

CONTINENTAL OIL COMPANY			
MINERAL EXPLORATION DEPARTMENT			
ISOPACH OF ORE ZONE			
DATE	NOV. 11, 1951	BY	J. C. BEE
NO.	1000	REVISION	1
BY	J. C. BEE	REVISION	2
DATE	11/11/51	REVISION	3
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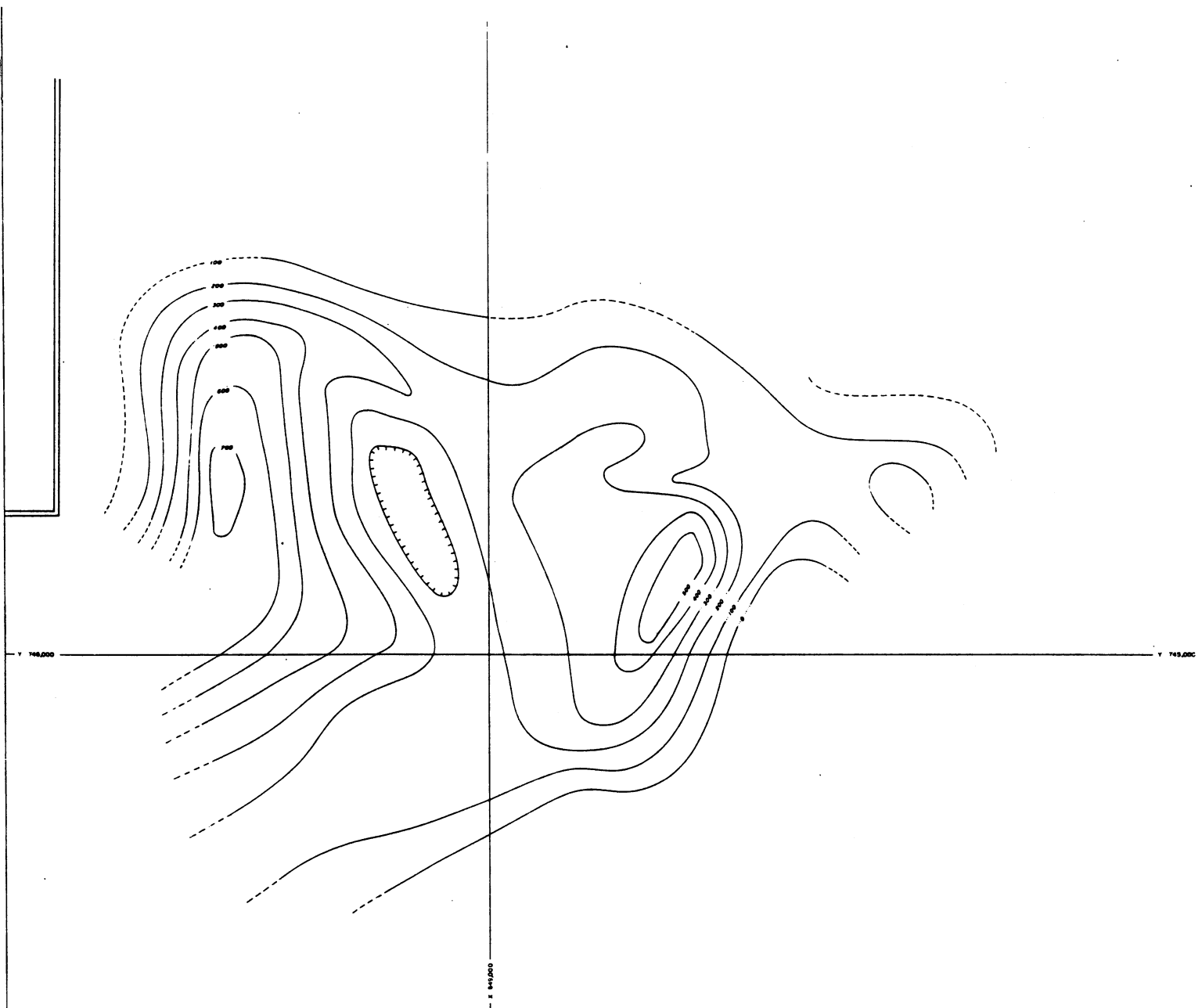


Figure 13

CONTINENTAL OIL COMPANY				ISOPACH OF OXIDE ORE 0.32	
MINING, EXPLORATION DEPARTMENT					
MATERIALS SECTION					
DATE	APR 21, 1961	BY	J. C. BRY	REVISION	1000 (1000' INTERVAL)
PROJECT	1000	APPROVED BY	J. C. BRY	DATE	11/11/61
REVIEW	1000	APPROVED BY	J. C. BRY	DATE	11/11/61
REVIEW	1000	APPROVED BY	J. C. BRY	DATE	11/11/61
REVIEW	1000	APPROVED BY	J. C. BRY	DATE	11/11/61

DATA TAKEN FROM  
FOOTING ASSAY MAP  
(100' ASSAY INTERVAL)



Rock composition controls somewhat the nature and intensity of oxide mineralization. The major portion of the oxide ore is contained in quartz monzonite porphyry and granodiorite porphyry but oxide copper development is most intense in more basic rocks such as andesite and basalt. The mineralogy of this oxide ore in the basic rocks consists of chrysocolla, tenorite, cuprite, and native copper generally averaging in grade between 0.60-1.00% copper. This oxide mineralization exists along fractures in post-sulfide, Tertiary intrusive dikes and sills. Apparently the more basic composition of the rock types resulted in the precipitation of exotic oxide copper minerals.

Metallurgically, three types of oxide ore are recognized at Florence, (Hazen Research Inc., 1971). They are:

- 1) Chrysocolla type
- 2) High Iron type
- 3) Mixed type.

The chrysocolla type is characterized by visible copper oxide minerals, chiefly chrysocolla and lesser amounts of tenorite. Iron oxides as goethite and/or hematite may be present in sparse to moderate amounts but never intense. Included in the chrysocolla type are copper stained clay minerals or other layer silicates along fractures and copper stained altered feldspar phenocrysts.

The high iron type contains moderate to abundant iron oxides with little or no visible copper oxide mineralization. Iron oxide mineralogy consists of goethite or hematite-commonly as a mixture. Minor jarosite may be present. The basis for recognizing this zone is that copper may enter the goethite structure and effect leaching behavior.

The mixed ore type is a mixture of both the chrysocolla and high iron types.

The mixed zone is recognized in two ways - as alternating intervals of chrysocolla type and high iron type not large enough to be mixed separately and as significant amounts of copper oxide minerals, chiefly tenorite and chrysocolla with intense iron oxide staining.

Leached capping is also a significant feature of the oxide zone. In general the nature of boxworks and the limonite coloration have proved to be reliable indications of mineralization at depth. In the potassic zone of the ore body boxworks are commonly observed to be only partially leached with the limonite color dominated by hematite indicating a high chalcopyrite to pyrite ratio. In the sericitic zone, the limonite coloration is dominated by goethite indicating the pyritic mineralization at depth.

Secondary sulfide enrichment is commonly thin in the Florence deposit. Oxide mineralization may be admixed in the enrichment zone in roughly equal amounts with sulfide minerals. Total copper and oxide copper assays are important in this transition zone to determine the exact copper extraction methods to be used. The mineralogy of this zone consists of chalcocite, covellite, chalcopyrite, and bornite. Chalcocite is the major component of this zone and generally occurs as coatings on pyrite grains. The thickness of the chalcocite zone is variable ranging from 0-600' and is thickest along the potassic-sericitic alteration boundary. The chalcocite zone does not appear to be related to post-sulfide structures. The relative thinness of the chalcocite zone may be a result of a number of factors such as 1) low total sulfide content, especially in the potassic zone; 2) lack of porosity and structure; or 3) subsequent destruction of chalcocite zones by faulting and/or erosion.

The hypogene sulfide zone is represented by pyrite, chalcopyrite, and molybdenite. Minor amounts of galena and bornite are also noted. The total sulfide content ranges from 2-3% in the potassic zone up to 5-10% in the sericitic zone. Primary sulfide ore is restricted to the hydrothermally developed potassic alteration zone and generally shows a high chalcopyrite to pyrite ratio.

Sulfide mineralization is related to structure and alteration. The mineralization is predominantly veinlet controlled. Higher grade mineralization is commonly associated with mixed zones which may be produced in part by structural brecciation. The proximity of sulfide mineralization adjacent to the granodiorite porphyry and the relatively low grade center that this intrusive outlines indicates that granodiorite porphyry is also a major control.

Sulfide ore is thickest in the vicinity of holes 32-MF and 108-MF. From the isopach of sulfide ore presented in figure 11, northeast and east-west trends are apparent. These trends may be a reflection of Precambrian structural patterns that were reactivated during the Laramide mineralization event.

Molybdenum, silver, and gold may also be recovered from the sulfide ore. The average for these metals in the ore is, 0.0101% Mo, 0.099 oz/ton Ag, and 0.0016 oz/ton Au. Mo shows a general correlation with Cu content. Zonation of Ag and Mo shows a general northeast to east-west trend paralleling the potassic sericitic alteration boundary along the eastern section of the ore body.

#### ALTERATION

Porphyry copper mineralization at the Florence deposit is associated with supergene and hypogene alteration zones. The supergene alteration is related to the amount of oxide mineralization and extensive argillization. Hypogene alteration is distributed in concentric halos which consist of potassic, sericitic, and propylitic alteration types. The exact dimensions of the alteration

zones are not known, but in plan view at the 500' level as shown in figure 14, the potassic zone is 3500' long in the north-south direction and >3000' long in the east-west dimension. Outward from this alteration zone, the sericitic alteration type varies from 500' wide at the southern region to 2500' long along the eastern edge. The propylitic zone apparently extends much farther outward and its dimensions are probably in the tens of thousands of feet. Alteration intensity at Florence is primarily veinlet or fracture controlled. Based upon the rough correlation between hypogene alteration boundaries and granodiorite distribution, especially along the eastern edge of the deposit, the granodiorite porphyry may have acted as the center of symmetry for the altering fluids.

Supergene alteration developed in response to oxidation of the sulfide ore deposit. This oxidation resulted in the development of clay minerals and iron oxides in addition to copper oxide minerals. The mineralogy of this argillic zone includes Kaolinite, montmorillonite types, halloysite, allophane, illite, quartz, chlorite?, vermiculite?, and hematite-goethite. Argillization primarily affects the plagioclase grains and to a lesser extent the mafic minerals. This argillization is commonly most intense along fractures.

The intensity of hypogene alteration and the resulting mineral assemblage depends on a number of factors such as temperature, pressure, composition of the altering fluids, time, and composition of the wall rocks. At Florence, the more mafic rocks such as diorite and diabase respond differently to the hypogene alteration processes than more felsic rocks such as granodiorite porphyry and quartz monzonite porphyry. Commonly the mafic rocks are more intensely biotized, chloritized, and epidotized than the felsic intrusives.

Alteration in the potassic zone is distinguished by the hydrothermal assemblage K-feldspar-biotite-sericite. This assemblage is distributed adjacent to



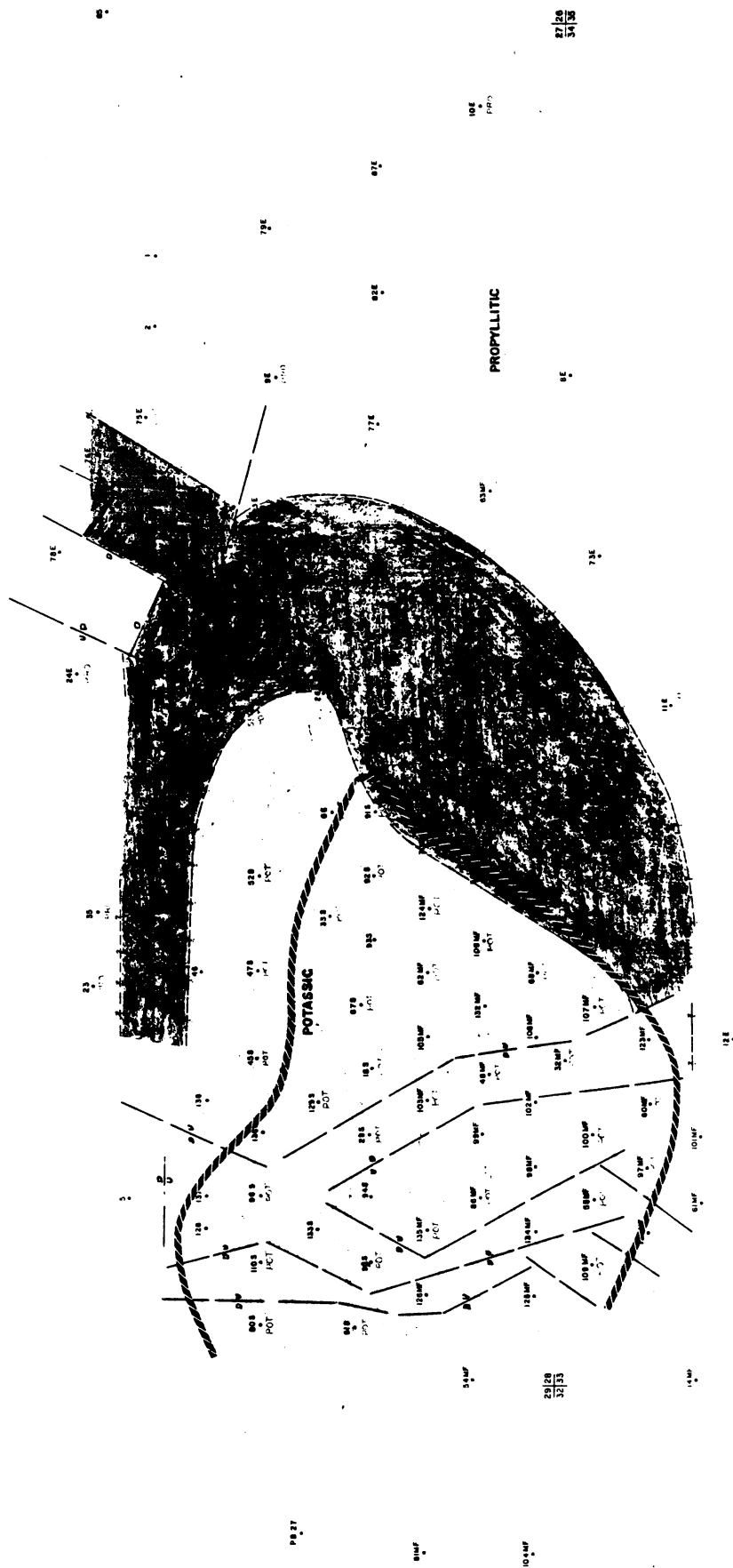
[illegible]

Figure 14

veinlets and also locally in a pervasive manner. Additional components of this assemblage are quartz, chlorite, calcite, gypsum?, anhydrite, sphene, rutile, and locally epidote. Magnetite is also a component of this assemblage and may occur as separate veinlets or in association with mafic grains. Sulfides associated with this alteration include pyrite, chalcopyrite, and molybdenite. In this zone, the total sulfide content is 2-4% with chalcopyrite exceeding pyrite.

Sericitic zone mineralogy is simple in nature. The bulk of the assemblage is comprised of quartz, sericite, and pyrite with some local quantities of rutile. Sericitic zone alteration is predominantly restricted to veinlets generally composed of quartz and pyrite. Total sulfide in this zone exceeds the sulfide quantities in the potassic assemblage and amounts to 5-10% sulfide content which consists mostly of pyrite.

The propylitic alteration zone represents the outer fringe alteration of the Florence deposit. Mineralogically this alteration consists of the assemblage epidote, sericite, carbonate, chlorite, quartz, rutile, magnetite, pyrite, adularia, and zeolites. In general, this alteration is weaker in degree of development than potassic or sericitic alteration and is lowest in total sulfides, averaging 1% pyrite.

Within the ore grade potassic zone of the deposit, a retrograde propylitic alteration apparently developed. This alteration is restricted to veinlets and consists of quartz, carbonate, and chlorite. This assemblage probably represents a later superimposed stage of alteration after potassic alteration.

#### GEOPHYSICAL AND GEOCHEMICAL EXPLORATION

The Florence copper deposit has been outlined with a fair degree of success with a few geochemical methods. The methods Continental Oil has used primarily

are gravity survey, induced polarization, and mercury vapor detection. The gravity survey was useful in determining general bedrock depths and displacements. The induced polarization was used to outline our deposit and other areas of interest. The geochemical method - mercury vapor detection - was also used to delineate the Florence deposit and to look for other promising areas.

#### GRAVITY SURVEY:

A gravity survey was conducted in the area by Robert West during August and September of 1970. The purpose was to delineate the subsurface scarp or scarps which downdrop the western edge of our deposit. Data was taken in the area of the ore body and outside of it. The data to the north has been used in exploratory drilling and has shown that gravity data can be a very effective tool.

The principle of the system is that variations in the density of rocks near the surface of the earth create small differences in the pull of gravity. These differences can be measured and interpreted providing several corrections are used in transforming the raw data into usable data. By mathematically "removing" the regional gravity effect, even better local anomalies may be observed and interpreted. The basis for interpretation of bedrock depths in this area is a bedrock which is more dense than the overlying, loosely consolidated sediments. Thus, the deeper the sediments, the less the pull of gravity compared with that of an equal amount of bedrock.

Using these principles, the gravity data was interpreted as shown on figure 6. The structural interpretation shown agrees very well with drill hole data to date. The most outstanding feature of this interpretation is a large horst which trends roughly NNW, truncated to the south, and thinning

GRAVITY SURVEY, con't:

to the north. This is the horst upon which the ore body sits. West of this horst is a graben which trends roughly north-south, and which is responsible for the groundwater flow to the north. East of the major horst is one, or more likely, a series of downthrown step faults.

As the exploratory drilling done in the area agrees with these interpretations, the gravity data has proved to be very useful in this area in interpreting bedrock displacements, and sometimes depths.

(West, 1970).

INDUCED POLARIZATION SURVEYS:

Two induced polarization surveys have been run in the area. The first survey, which was essentially a test of the effectiveness of I. P. over the ore body, was conducted in October and September of 1970 by Mining Geophysical Surveys. The second survey, which was a much more extensive reconnaissance survey, covering the ore body and much of the surrounding lands, was conducted during the spring of 1971 by McPhar Geophysics Incorporated. The results of this survey were used in exploratory drilling during 1971. The drilling results were discouraging, although preliminary.

Induced polarization is a method of locating scattered primary sulfide mineralization through electronically charging the ground by applying a current and then observing the electrical behavior or response. The principle is that an individual sulfide grain which exhibits metallic-like conductivity, will react much as a leaky condenser when current runs over its surface - ie., it will develop a small voltage potential. This behavior of the sulfides can be detected by observing the decay voltage after the current is shut off ( the current will "flow back" after the current is shut off) or by comparing the

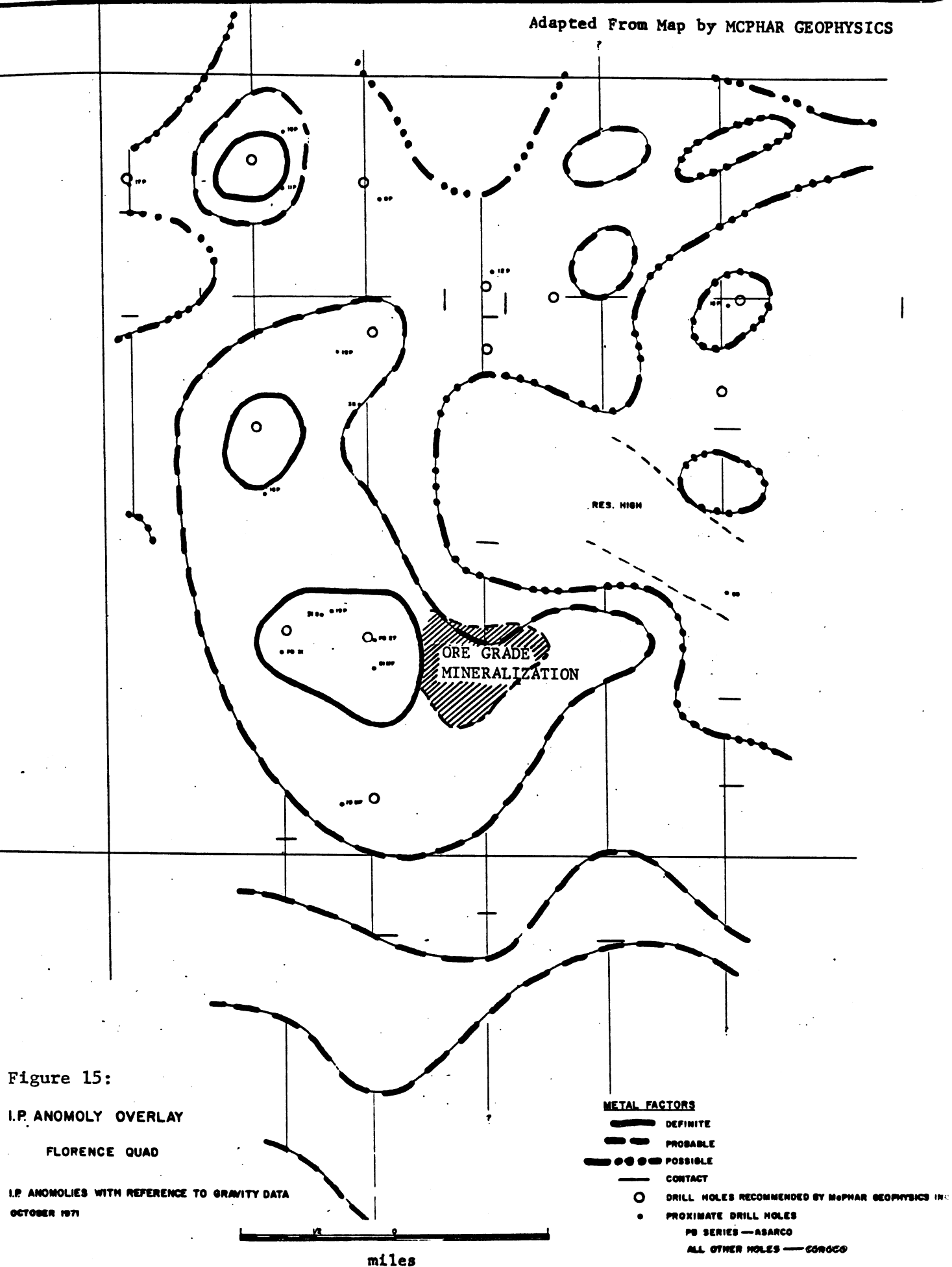
INDUCED POLARIZATION SURVEYS, con't.

impedence (resistivity) of the rock with change in A.C. current frequency. A complication to this method is that some non-metallic, non-sulfide minerals react in the same manner as do the sulfides - notably some of the clays such as bentonite and montmorillonite, plus some other minerals such as pyrolusite and magnetite. (Brandt, 1966).

The preliminary survey by Mining Geophysical Surveys indicated a moderate I.P. and resistivity response over the ore body area, indicating that I.P. would be useful in the area. Subsequently, the reconnaissance survey by McPhar Geophysics was conducted to attempt to better outline the ore body and to indicate areas of further interest. The results of this survey are shown on figure 15. As can be see , the deposit is reflected as a probable anomaly which is shown as an ENE extension of a major trending definite and probable metal factor anomaly. The NNW major trending anomaly has another ENE extension to the north which looks similar to the one in which the deposit is located. North of that there is another isolated "definite metal factor anomaly", and to the east are a couple of "probable metal factor anomalies." (Hauck, 1971).

During the latter part of 1971, most of these anomalies, and a good number of drill holes proposed by McPhar were rotaried to bedrock as an extension of our validation and assessment program. Holes were located so as to drill into either I.P. or MV anomalies and where possible, both. The results are discouraging overall. Ten holes were drilled, and out of these holes, only five seemed to show any hope of being in the vicinity of any mineralization. Those were holes 12-P and 13-P which were drilled mainly on an MV high; 15-P which was drilled on a possible P. anomaly; 19-P which was drilled into the definite I.P. anomaly just west of the deposit; and 30-P which was drilled in the west end of the

Adapted From Map by MCPHAR GEOPHYSICS





INDUCED POLARIZATION SURVEYS, con't.

definite I.P. anomaly west of the deposit. These holes (with the exception of 15-P) all showed a propylitically altered quartz monzonite porphyry, with slight to moderately high geochemical Cu, and sometimes anomalous Ag, Pb, Zn, and Hg values. Hole 17-P also had anomalous Ag.

However, there were several discouraging aspects of the I.P. data. None of the holes intersected any true verification of the I.P. anomalies (although some may lie at a greater depth - not on the bedrock surface). In addition, the axis of the NNW trending definite - probable anomaly has been shown to lie on or in the general vicinity of the large graben which trends north-south; and which drops the western part of the deposit. This suggests that resistivity lows, and possibly I.P. highs, are associated with the sediment filled graben (which has a high clay and groundwater content.)

In summary, the I.P. work has shown some very well defined anomalies which, so far, we haven't been able to entirely substantiate on the basis of having any mineralization. Future drilling will probably be done on these highs to fully evaluate this I.P. work.

MERCURY VAPOR DETECTION:

The Radiation Lab Group in Ponca City, under the leadership of Preston Gant, has been developing mercury vapor detection equipment and techniques right along with the copper deposit, and has compiled an enormous quantity of data from on and around the area. The detection methods and equipment are some of the best, if not the best, now used. These methods have also shown that mercury vapor detection can be a relatively fast and cheap exploration method. The Florence staff has used this data in exploratory drilling - mostly to the north of the deposit - with moderate success.

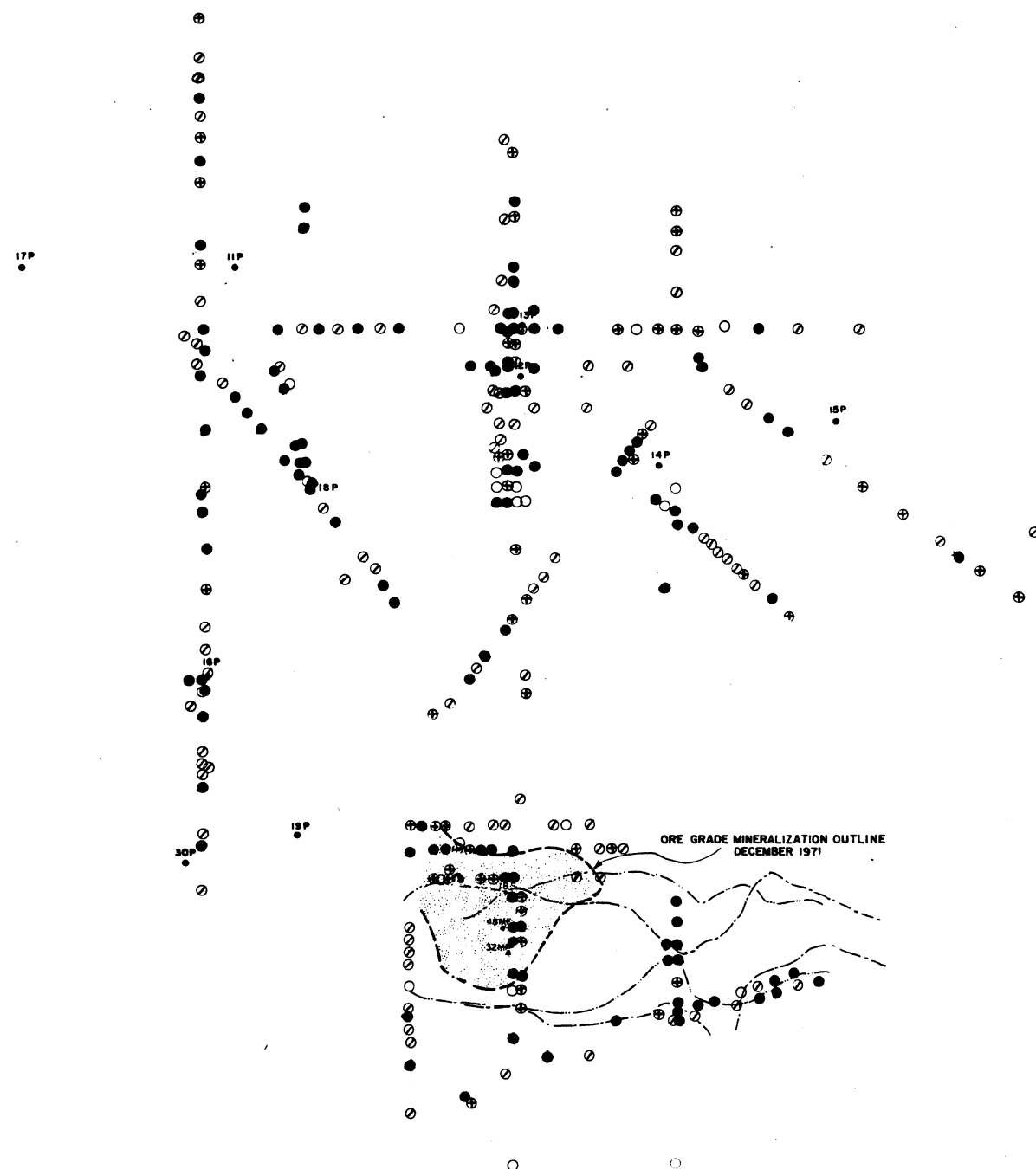
MERCURY VAPOR DETECTION, con't.

Anomalous mercury concentrations are known to be associated with porphyry copper type deposits, as well as disseminated gold, silver, zinc, and mercury deposits. Mercury, in the native state, is liquid, and maintains a high enough vapor pressure to release small but significant amounts of mercury vapor (measured in "nanograms" =  $10^{-9}$  grams). This vapor will migrate upwards from a buried deposit, and if sensitive enough equipment is used, can be detected on the surface - in the soil, the soil gas, or in the air above a deposit. (Gant, et.al., 1971).

The basic principle is to trap and measure this gas. The Ponca City group does this in a couple of ways, but the principle is all the same - by passing a large volume of air over a gold plated nichrome wire, the available mercury is amalgamated on the gold. By passing a current through the wire, the mercury is "burned" off and passed through a spectrophotometer which measures the amount of mercury there is in the air. In this manner, a very small amount of mercury may be effectively measured. (Gant, et. al., 1971).

A map of the MV results is shown on figure 16. The red-colored circles are those readings which are 1.5 times background. As can be seen, there is a general area of highs around the deposit, delineating the deposit fairly well. Also to be noted are the areas of general highs around holes 12-P, 13-P and 18-P. (sections 16 & 17

Almost all of the anomalies outside of the ore body have been drilled in conjunction with I.P. anomalies. Overall, the results are discouraging, although there are indications that MV highs were located over areas where bedrock was indeed anomalous in mercury content. Rotary hole 18-P was a big hope, since it is in the vicinity of both an MV high, and an I.P. anomaly which is similar to the one that the deposit is located on. Bedrock in 18-P was very deep (1586')



## INDIVIDUAL BUBBLE DATA

- BUBBLE STATION • DRILL HOLE
- > 1.5 x BACKGROUND
- ⊕ 1.3—1.5 x BACKGROUND
- ⊙ 1.0—1.3 x BACKGROUND

## ANOMALY OUTLINES

- MV ANOMALY OCT 3—17
- MV ANOMALY NOV 3—9

## SCALE



Figure 16:

MERCURY VAPOR DATA  
FLORENCE PROJECT AREAINDIVIDUAL BUBBLE DATA BY CLEO SCOTT  
& ASSOCIATES AS OF JUNE 3, 1971ANOMALY OUTLINES TAKEN FROM CONOCO  
RESEARCH REPORT NO. 1202-1-1-71:  
A MINERAL EXPLORATION SYSTEM —  
MEASUREMENT OF MERCURY VAPOR IN  
AIR AND SOIL GAS", BY GANT, MOTES,  
AND AYLOR, JANUARY, 1971

MERCURY VAPOR DETECTION, con't.

with no indications of mineralization. Holes 12-P and 13-P, however, are more promising. Both were shallow (210' and 180', respectively) with weak alteration and fairly heavy limonite and hematite staining at the bedrock-conglomerate contact. Hole 12-P was subsequently cored for 50 feet, and the geochemical assays were anomalous - mercury running an average of 225 ppb., with 325 ppb. high, and anomalous values of silver, copper, and zinc. Another hole, 16-P, was drilled on both an MV and an I.P. anomaly. Here the alteration was poor, but the spot core ran 217 ppb. Hg, which again, is an anomalous mercury value.

Therefore, while no direct indications of sulfide mineralization were drilled into, on MV anomalies to the north of the deposit, there is evidence from a couple of the holes that MV highs are associated with anomalous mercury concentrations in bedrock. This is important to establish, as it may be in some cases that MV highs could be attributed to other sources which could confuse the results. Possible sources of mercury other than bedrock might be from surface contamination, such as smelter fumes, fungicides, etc. or from some sort of subsurface source such as recent volcanic deposits, or perhaps from mercury which has migrated laterally from its source. The possibility that mercury may originate from other such sources should be kept in mind when using MV data.

The detection of mercury vapor, then, promises to be a good geochemical tool, to be used with the other available exploration methods. Since the use of mercury vapor detection is probably as yet in a developmental stage, and since exploration at Florence has not yet drilled out all of the MV anomalies, a complete evaluation cannot be made at this time.

HYDROLOGY

The Florence deposit is situated beneath the present flood plain of the

Gila River, within the groundwater underflow. Therefore, dewatering the mine area will be a problem in developing the Florence deposit.

SURFACE WATER:

The Gila River is a major river in south-central Arizona, draining some 18,305 square miles upstream from the Ashhurst-Hayden Dam. The dam, however, which is located 16 miles upstream from Florence, effectively eliminates any floodwater threat from this part of the river. A tributary to the Gila, the San Pedro River, with a drainage area of about 4,471 square miles, enters the Gila River downstream from the dam, however, and does pose a threat from floodwaters. Peak recorded discharge in the San Pedro has been 98,000 c.f.s. at Charleston and 90,000 c.f.s. at Reddington. (Halpenny, 1971).

Our consultant in hydrology, Leonard Halpenny of the Water Development Corporation, calculated that any threats to the mining operation could be effectively eliminated by the use of a dike around part of our pit. The dike would be 14 feet high at the narrowest point between our pit and the river, tapering down 4000' to the east and 4000' to the west to two or three feet in height.

SUBSURFACE WATER:

Subsurface water will be the prime concern at Florence. To date, most of the available information on groundwater hydrology is based on a quantitative pumping test conducted by the Water Development Corporation, some U.S.G.S. data (Hardt and Cattany, 1965), and some qualitative information gained from rotary and core drilling in the area.

Groundwater depth at the Florence Project is at about 190 feet - a depth that has been declining over a long period of time due to pumping rates which

SUBSURFACE WATER, con't.

are greater than replenishment rates. Groundwater is recharged primarily by the underflow of the Gila River. Groundwater is thought to be contained primarily by the recent alluvium, the Gila and Whitetail conglomerates, and the upper fractured and weathered portion of bedrock.

Groundwater flows generally to the west, at a gradient of about 40 feet/mile, until reaching the deposit. In the vicinity of the Florence Project, groundwater veers north, where the gradient apparently drops to 20 feet/mile. (Halpenny, letter and map of November 19, 1971). The northward flow of groundwater is in response to a lowered water table in the Queen Creek - Magma area to the northwest due to heavy pumping. Groundwater probably travels north through a relatively deep north-south trending graben which downdrops the western side of the deposit. The significance of this northward gradient, as explained by Halpenny, is that the effect of dewatering on the surrounding agricultural areas to the south will be less than ordinarily expected. (Halpenny, letter of November 19, 1971).

Preliminary pumping tests in three rotary holes by Elenburg Drilling Company, while not useful for quantitative determination of permeability and porosity of the conglomerate, provided some idea of relative water producing capabilities at various depths and locations in the deposit. In general, very little water was pumped above 250'. Also, it seemed that more water was produced with depth, as long as in the conglomerate.

From rotary drilling, it is also thought that in many areas, bedrock is overlain by a relatively loose sand which is probably a good "water-sand". Water wells in the vicinity are frequently partially in bedrock, probably producing some of their water from the upper fractured and weathered bedrock



SUBSURFACE WATER, con't.

surface. With this information, plus the drilling tests of Elenburg, it is thought that most groundwater is carried in the conglomerate, with some in the overlying recent alluvium, and some in the fractured, weathered top of bedrock. The deeper bedrock - particularly the sulfide zone - in all probability acts as a barrier, or as a "hydrologic bedrock", where little water can be stored or transmitted.

A pumping test was conducted in January of 1971 by Water Development Corporation to determine quantitative physical constants of the aquifers above bedrock. One well was pumped while two were observed. The coefficient of transmissibility was determined to be 525,000 gallons per day per foot, while short term coefficients of storage were determined to be 7 and 10 percent. The long term coefficient of storage as determined by the U.S.G.S. (Hardt and Cattany, 1965, p. 77) is 17 percent, which is confirmed by the calculated short term coefficients.

Using these physical constants, and some assumptions based on these, Halpenny has worked out dewatering programs for the Florence Project. A more thorough knowledge of the hydrologic constants will be needed for a more precise idea of dewatering procedures. Pumping tests to determine these are planned in 1972.

COMPUTER LOGGING

It was determined, once the Florence deposit was established as a possible ore body, that the use of a computer as an aid in sorting out the many variables which are encountered in the evaluation of such a deposit would be very desirable. Accordingly, the Florence staff, in conjunction with Joe Paden, Engineer of the Mineral Department in Denver, and Dale Cooper of Ponca City's Computer Department, has adopted a system of logging which puts the raw geologic data - including assays - in a standardized, easily codeable system. Of course, the reliability of the data

which is coded, and the reliability of the way it is coded is very basic to how reliable the computed results themselves are. The results that can be obtained from a good set of raw data, coupled with several good computer programs are staggering and promise to help the project immensely once they are in effect.

The logging system now in use is a method of recording several parameters plus comments which will be of value to the geologists. The comments will not be programmed, but will be punched for reproduction. The parameters are of two general groups: identifier information and geologic information. Two data cards are punched for each  $2\frac{1}{2}$  foot interval - one card with the program-able parameters, and one card with comments. The  $2\frac{1}{2}$  foot intervals were chosen in this instance to retain a detailed geologic picture with respect to structure, contact relationships, and alteration.

Of the two general kinds of parameters used, the basic ones are the identifying parameters. Identifying parameters now used are:

- Project Identification
- Drill Hole Number
- Coordinates or Location Code
- Elevation
- Inclination of Hole

The geologic information parameters are a bit more complex. The parameters now used are:

- Core Run and Recovery
- Rock Type
- Structure: Fracture Intensity
- Fracture Attitude
- Fault Attitude
- Alteration Minerals: K-feldspar, biotite, quartz, sericite, chlorite, epidote, carbonate, gypsum, anhydrite, magnetite, hypogene hematite, plus several variables.
- Oxide Zone Mineralization: Limonite, hematite, readily leachable copper oxide, or copper oxide in clays and feldspar sites.

Metallurgical Zone: Divides rock into zones which will probably be treated differently metallurgically - a single column is used to classify the five different metallurgical types.

Sulfide Zone Mineralization: Pyrite, chalcopyrite, bornite, molybdenite, covellite, plus several variables.

Assays: Total Cu, CuOx, Mo, Ag, Au, Pb, and Zn.

Each variable is assigned a particular column and the information used is coded as shown on figure 17. Rock type names are coded with an abbreviated 3 to 4 letter symbol. Fracture intensity is coded by numbers according to the quantity of fractures per foot (0 - 5 fractures/foot = 1), while dip is recorded according to the predominant dip angle (40 - 50 = 5). Alteration intensities are recorded with a scale ranging from 1 to 3; 1 = sparse, 2 = moderate, and 3 = abundant.

The remaining columns on our computer sheet are used for remarks. Important information which is not otherwise coded is recorded here. The illustration should be self-explanatory. This information is stored only - it cannot be used by the computer other than to print it out.

#### POSSIBLE USES OF COMPUTER:

The possible uses of this data by the computer are truly exciting, and while a real study of these are beyond the scope of this paper, a few possibilities are presented.

Statistical evaluation is the name of the computer game, and one of the exciting processes in a developmental stage, as we are now in, is a correlation analysis to determine mineralization trends and variability. By correlating

DRILL NO XYZ

PROJECT FLØR

LOGGED BY CRK

DATE 12/6/71 SCALE       

Page 47

CORE	Run	Depth Run Bottom	Rock	LITH	Tex.	Free	Fit.	COLOR	ALTERATION MINERALS	Oxidized ZONE	SULFIDE ZONE	ASSAYS	REMARKS
	2	1705	QMP			40		11 221					
	3	1710											
	4	1715											
	5	1720				26							
	6	1725				19							
	7	1730				5 9							
	8	1735				50							
	9	1740				42							
	10	1745				33							
	11	1750				14							
	12	1755				16							
	13	1760				37							
	14	1765				33							
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Figure 17: Computer Logging Sheet.

POSSIBLE USES OF COMPUTER, con't.

drill hole assays, and calculating the means and the variations for a certain population of the drill hole assays - e.g. for each drill hole - the computer will calculate mean assays with the variance (range) within certain limits. If the limits picked were 95%, then 95% of the time, the assays will fall within that variance. Thus, the predictability of the mineralization will be known for each area - areas that have a large variance and thus a low predictability will warrant more drill holes. Conversely, drill holes are saved where the predictability is good, and a great deal of time and money can be saved. Uses of this same principle should be entirely feasible on other variables for predicting certain values, with confidence limits. The computer can also plot these values on a map and contour them.

Variations of this idea, and the use of the computer for interpolation and extrapolation of all kinds of values, plus many other computational abilities, plus the ability to quickly plot values for easy interpretation make a computer assisted program very useful indeed. Other possible uses that could be of value at this time include:

- Exploration Evaluation
  - Correlation Analysis
  - Sequential Drilling Analysis
  - Preliminary Geologic Reserve Estimation
  - Compositing Drill Hole Assays
  - Level Maps and Cross Sections with Geology and Composites
- Pit Design, Tonnage and Grade
  - Calculate Mineable Ore Reserves
  - Pit Design - Including optimum bench height, elevation, etc.
- Financial Analysis
  - Grade Cutoff and Production Rate to Maximize Profit
  - Scheduling Pit Production
  - Equipment Selection and Process Optimization

These programs and more are now available for lease or purchase from various proprietary software firms, or perhaps could be converted from previous programs used in oil exploration, development, and production.

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Continental Oil Company  
P. O. Box 627  
Florence, Arizona 85232  
(602) 868-5852

December 7th, 1976

Dear Sir:

As promised, please find enclosed a copy of the requested paper.

Yours very truly,

J. J. Dworatzek  
Plant Superintendent

ss  
attachment



VAT AND AGITATION LEACH PILOT PLANT OPERATIONS  
AT CONOCO'S FLORENCE COPPER PROJECT

J. J. Dworatzek  
Plant Superintendent

December, 1976

## ABSTRACT

This paper describes the pilot test program for the treatment of oxide ores carried out by Continental Oil Company at their pilot plant in Florence, Arizona.

Ore feed to the pilot plant was obtained from the nearby underground pilot mine located close to the center of the orebody which contains about 500 million tons at 0.39 percent copper as sulfide ore and 250 million tons at 0.45 percent total and 0.32 percent acid-soluble copper as oxide ore.

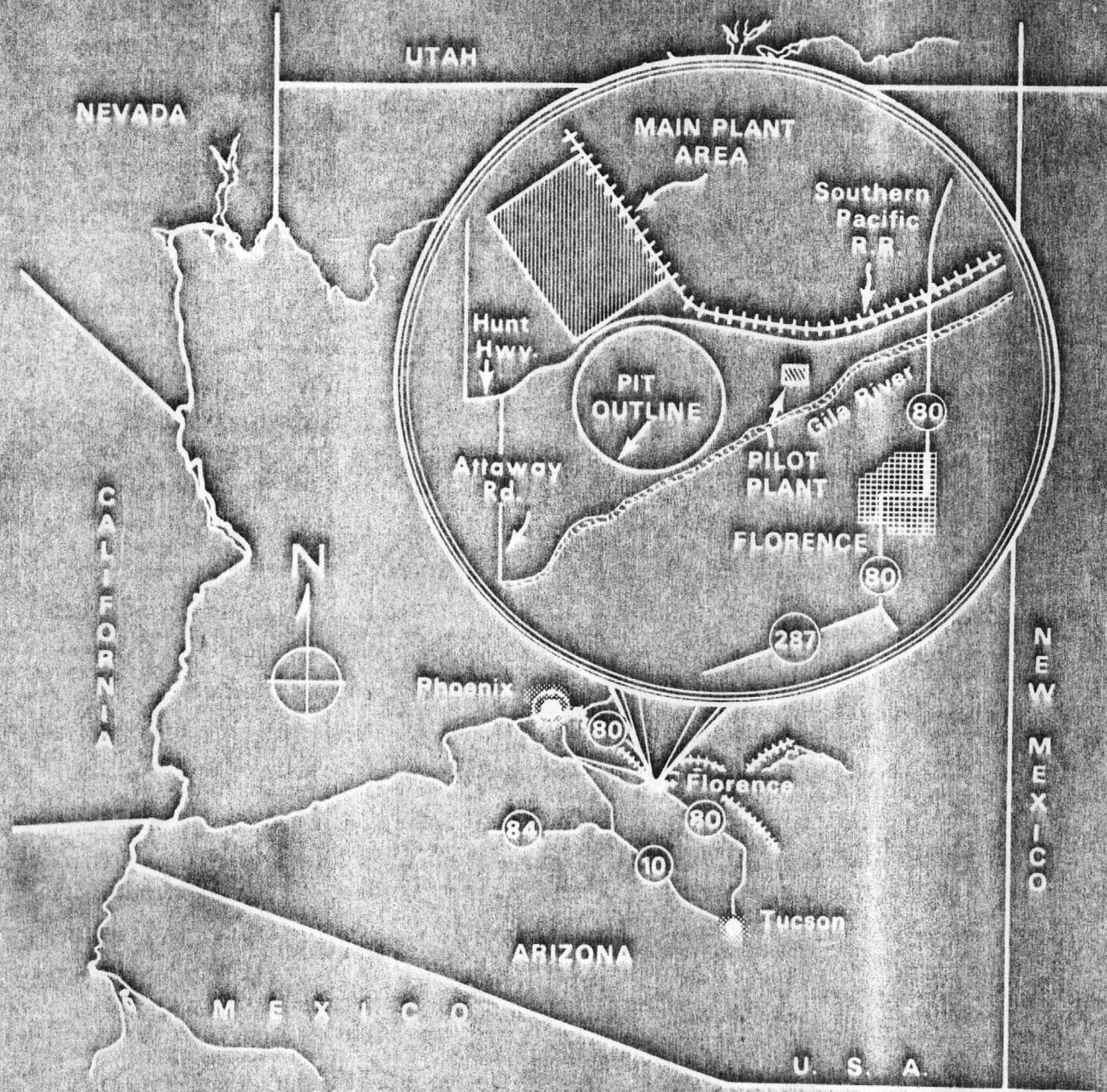
Two conventional methods were employed to evaluate the oxide ore metallurgically. One was in vats in a 100 ton per day plant and the other by agitation leaching in a six ton per day plant. Results showed that both processes are viable for treatment of this ore. Agitation leaching resulted in a higher extraction of copper values.

The final process type for production decision will have to be based on a combined evaluation of metallurgical recoveries and projected capital and operating costs.

## INTRODUCTION

Continental Oil Company owns a large low grade copper orebody approximately three miles northwest of the town of Florence in Pinal County, Arizona (see location map). The orebody is composed of about 250 million tons of oxide ore overlying 500 million tons of sulfide ore. The alluvium overburden is approximately three hundred and fifty feet thick. The copper content of the orebody amounts to 0.39 percent for the sulfide and 0.45 percent for the oxide ore. Of the latter, 0.32 percent is acid-soluble.

Ore grade mineralization is contained for the greater part in a Precambrian quartz monzonite and to a minor extent in Laramide granodiorite. Oxidation and weathering of the upper reaches of the sulfide deposit produced clays, iron oxides and



CONOCO COPPER PROJECT

LOCATION MAP

acid-soluble copper oxide minerals. The ore contains mainly chrysocolla, with minor amounts of cuprite, tenorite, malachite and copper-bearing clays. The sulfide minerals include chalcopryrite, pyrite, molybdenite and some bornite.

Both the oxides and the underlying sulfides are located in sharply defined discrete zones, with little evidence of mixing or secondary enrichment between zones.

Metallurgical testing of the ore began almost immediately after discovery in Conoco's research facilities in Ponca City, Oklahoma and in the laboratories of Hazen Research, Inc. and Mountain States Research and Development.

Results obtained from the laboratory work were very encouraging. Good recoveries and excellent finished products were obtained. Conoco therefore decided to continue with additional evaluations in a pilot plant. This work was conducted in 1975 and 1976.

#### SUMMARY OF RESULTS

The pilot plant test program established that good extractions of copper can consistently be obtained from oxide ores processed by vat or agitation leach methods.

Treatment of ore in vats resulted in copper extractions ranging from 94 to 107 percent of the acid-soluble copper content. The average leach extraction was 100 percent when treating oxide ore crushed to pass through a 3/8-inch screen. Soluble copper loss averaged 2.13 percent. Acid-soluble copper recovery prior to solvent extraction amounted to 98 percent.

Sulfuric acid consumption during vat leaching with raffinate return was 50 pounds of 100 percent sulfuric acid per ton of ore leached. The pregnant liquor contained 7.41 grams of copper and 5.7 grams of free acid per liter.

Overall vat leaching plant recovery into cathode copper amounted to 96 percent of the acid-soluble copper in the feed.



Leaching of ore in the pilot agitation leach plant resulted in extractions reaching as high as 110 percent of the acid-soluble copper contained in the ore.

Testwork established that two-stage leaching is preferable to single-stage leaching. Though two-stage leaching requires additional equipment, operating costs are lower and copper extractions are substantially higher than in single-stage leaching.

Optimum grind was found to be 95 percent passing minus 10 or 14-mesh. Acid consumption at these grinds ranged from 37.4 to 47.6 pounds of 100 percent sulfuric acid per ton of ore leached. Leaching retention time was 11 hours at ambient temperatures (29 to 34° C).

The pregnant solution contained 1.5 to 1.9 grams per liter copper and the pH ranged from 1.65 to 1.9. Lime was not required to adjust the pH of the solvent extraction feed solution produced by leaching at pH 0.8 and pH 0.9. However, leaching at pH 0.7 did require lime addition.

Recovery from the ore into cathode copper averaged 100 percent for the agitation leach circuit.

#### PLANT DESCRIPTION

The pilot plant was built approximately 2,000 feet east of the ultimate pit limit. Its primary function was to confirm metallurgical viability of the orebody, but it was also constructed to serve as a Research and Development unit for the Minerals Department as well as a plant to test oxide and sulfide ores from the Conoco mine.

#### CRUSHING PLANT

The crushing plant had two stages of ore reduction in a closed-circuit operation. A flowsheet for the crushing plant is enclosed (see Figure 1). Feed to the crushing plant was delivered



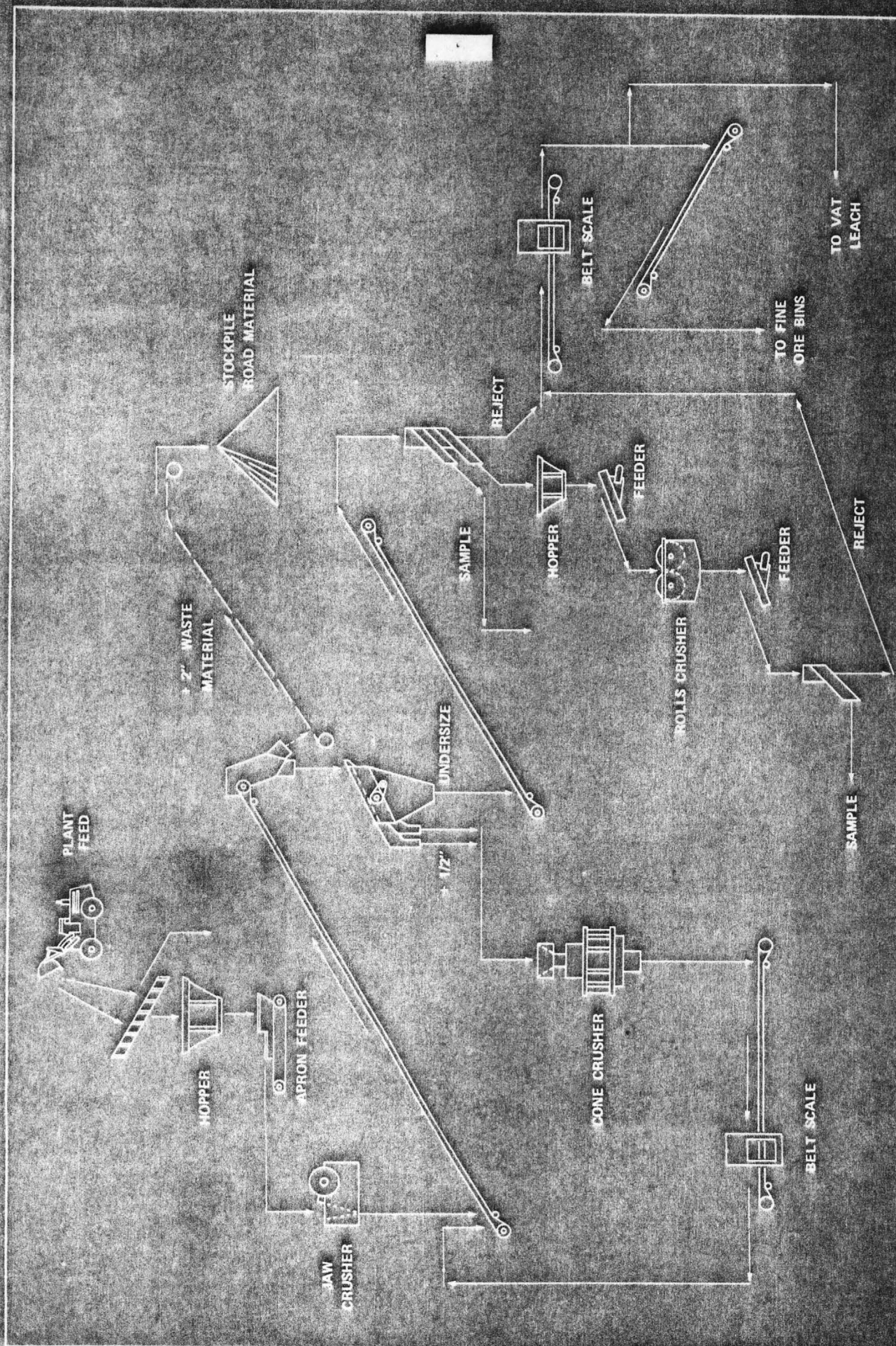


FIGURE 1 PILOT PLANT CRUSHING FLOWSHEET

by front end loader to a hopper equipped with an apron feeder. The feeder transferred material at a controlled rate to the primary crusher which was installed at a minimum height above ground. Product from the primary crusher at minus  $1\frac{1}{2}$ " discharged on to a belt conveyor having a magnetic head pulley for tramp iron removal. When waste material for roads was crushed, the discharge from this conveyor was diverted to a stockpile. After primary crushing, the conveyor discharged the ore on to a double-deck vibrating screen. Undersize material from the lower screen ranging from minus  $\frac{3}{8}$ " to minus  $\frac{5}{8}$ " in size was conveyed to the sampling tower. The oversize material from both decks fed the secondary crusher. A belt conveyor, equipped with a scale, returned the product from the secondary crusher to the primary crusher discharge conveyor. An automatic primary sampler cut the screen undersize product. This sample was further crushed in a rolls crusher and resampled to provide a final laboratory sample.

Provision had been made to divert part of the primary sample to a container for treatment elsewhere. The reject from the sampling tower was transferred to a conveyor and discharged via a rotary chute to the vat agglomeration conveyor or the fine ore bin feed conveyor.

The fine ore sampling system and the crushing plant chutes were designed for rapid self-cleaning because of the various ore types crushed. The crushing plant was not housed and a mobile crane was used for servicing the equipment. The fine ore bin feed conveyor was equipped with a rotary chute which discharged into the oxide or the sulfide bins.

#### VAT LEACHING PLANT

There were ten leaching vats, each measuring 10.75 by 10.75 feet inside by 19 feet deep. Crushed and sized oxide ore was fed directly from the crushing plant and distributed to the leaching vats by a shuttle conveyor. The ore was agglomerated by water sprays located at tumbling stations on the vat feed conveyor. A basic flowsheet for the vat leaching circuit is attached (see Figure 2).



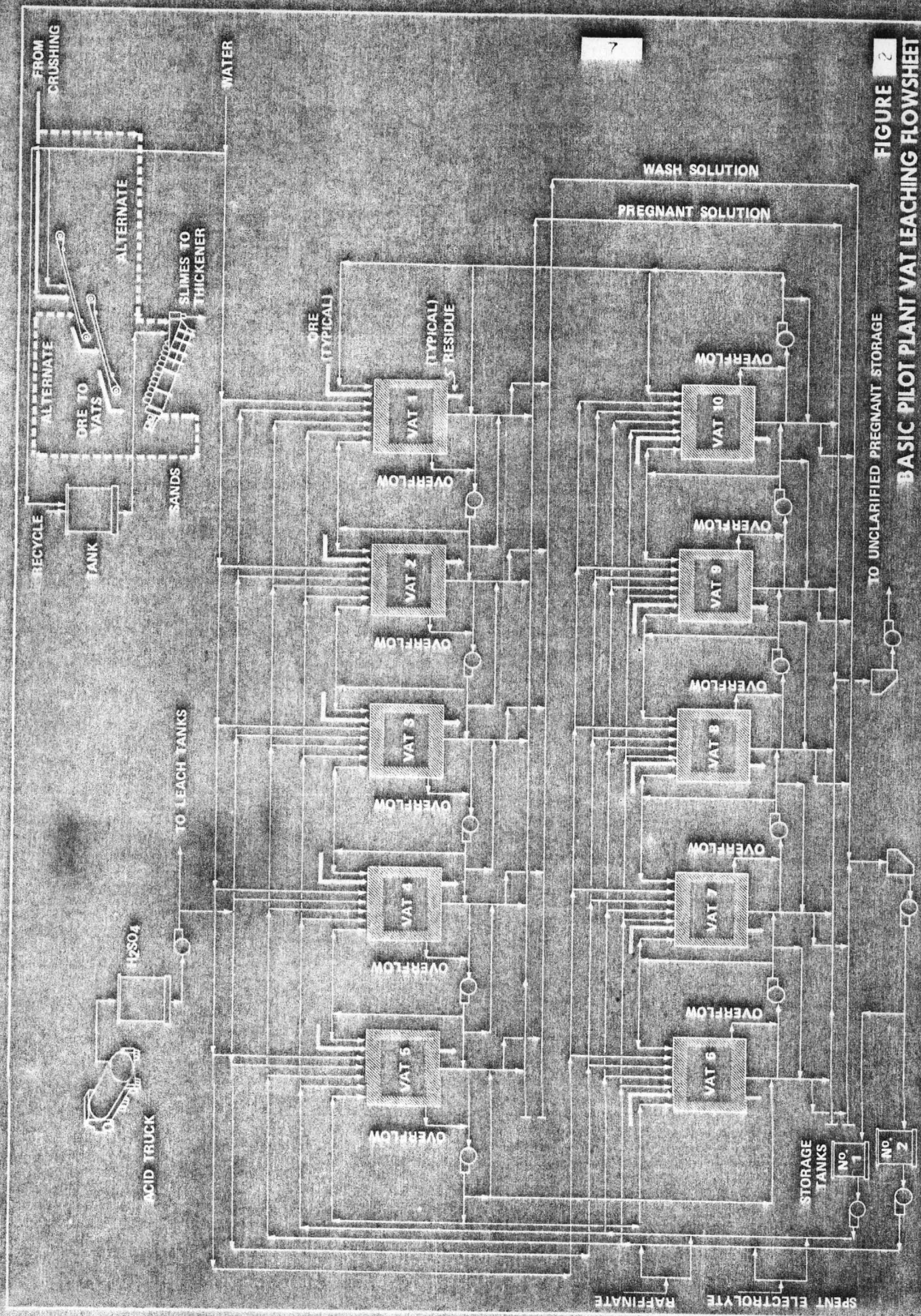


FIGURE 2  
BASIC PILOT PLANT VAT LEACHING FLOWSHEET

About 90 to 100 dry tons of ore were bedded evenly across a vat until the designed ore depth of 18 feet was reached. Loading was at the rate of 35 tons per hour. When a vat was completely filled, the shuttle conveyor was moved to the next one for loading.

Leaching vats were concrete. Three vats were unlined and the others were covered with various acid-resistant materials. Each vat had a false bottom of timbers and perforated planking. The ore rested on the planking. The spaces below the planking distributed the percolated leach solution to the vat outlet under downflow or to the ore under upflow conditions.

The vats had several pipe headers above and two lined concrete solution launders below. Two headers were used for the distribution of acid or water, the others distributed in sequence: raffinate, wash solution No. 1, wash solution No. 2, and water, in accordance with the vat leaching schedule. One launder was used to return wash solutions from the vats to either of the two separate wash solution storage tanks. The other launder was used to collect unclarified pregnant solution.

After a vat was loaded with ore, raffinate wash was introduced some six vats downstream. The solutions were displaced through the perforated false bottoms of all vats in the leaching cycle and through the newly ore-loaded vat and percolated up through the ore bed, covering it in about eight hours. During the next twelve hours, raffinate and wash water were added as indicated above and pregnant solution flowed from the newly loaded vat to the unclarified pregnant solution storage tank. The solution in each vat was then recirculated within the vat for four hours.

Leached ore was washed three times to remove copper-bearing solutions. Fresh water was used for the last wash. A vertical sump pump returned any spills to the leaching vats or to the oxide tailing pond. The vats were built on an acid-resistant coated slab which had solution launders and sumps cast into the slab. Operating platforms were made of steel and were coated with acid-resistant paint. Throughout the rest of the oxide plant, where corrosive service was encountered, fiberglass reinforced polyester or wood was used for tanks and polyvinyl chloride for piping. Platforms were wood and/or steel coated with acid-resistant paint.



Leaching residue was removed through a side door at the bottom of the vats by a front end loader, then loaded into trucks and transported to the oxide disposal area. The unclarified pregnant solution from the leaching vats was pumped to a surge tank feeding the clarification and solvent extraction circuits.

#### AGITATION LEACHING

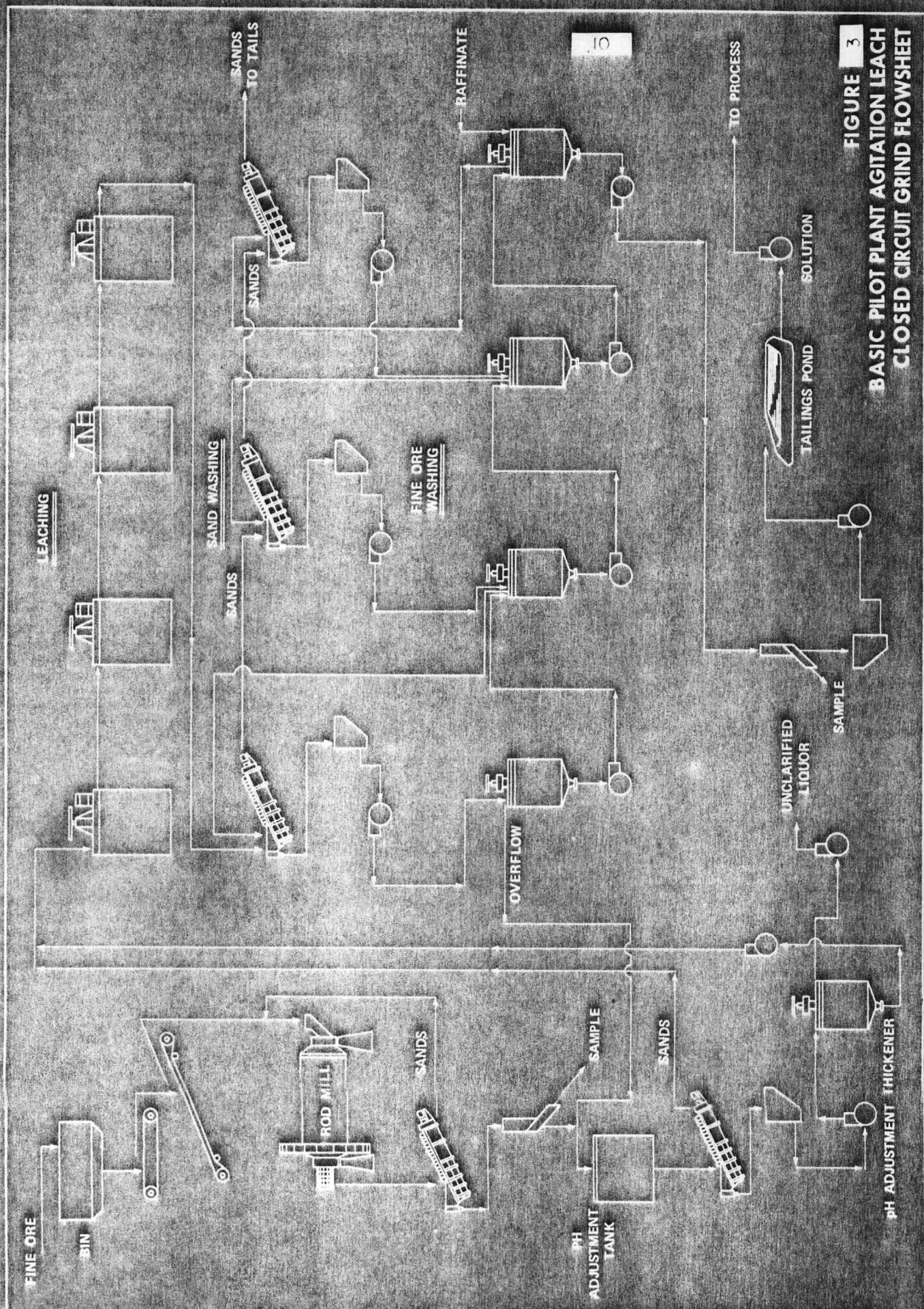
A flowsheet for the agitation leaching circuit is attached (see Figure 3). The agitation leach circuit was enclosed and designed to treat six tons of ore per day. The ore after fine crushing was reduced in a rod mill in open circuit.

With open circuit rod milling of agitation leach feed, the rod mill discharge flowed to the pH adjustment agitated tank. With closed circuit rod milling of the agitation leach feed, the rod mill classifier overflowed to the pH adjustment agitated tank. These circuits provided for a two-stage acid leach of the various possible feeds to the leach circuit. With a two-stage leach circuit, and when the rod mill discharge was fed directly to the pH adjustment tank, relatively coarse ore was leached. The use of two stage leaching allowed a high acid concentration in the second stage for improved copper extraction. At the same time, the system produced a pregnant leach liquor at a suitable pH for solvent extraction without use of chemicals for acid neutralization.

When leaching ore from closed or open circuit grinding, the pH adjustment tank discharge flowed into a screw classifier.

The classifier overflow was pumped to the pH adjustment thickener. The classifier sands dropped into the first tank of the leach section. The thickener overflow (unclarified pregnant solution), was pumped to a storage tank. Thickener underflow was pumped to the first tank in a series of the same four tank leach circuit that received the classifier sands where sulfuric acid was added to complete the leaching of the soluble copper values in the ore.

The leach discharge flowed by gravity to a spiral classifier, and the classifier sands were washed through a series of classifiers. The solids in the classifier overflow were washed through a four-stage CCD circuit. The combined tailing was pumped to an oxide tailing pond.



**FIGURE 3**  
**BASIC PILOT PLANT AGITATION LEACH**  
**CLOSED CIRCUIT GRIND FLOWSHEET**

## TEST PROGRAM

## VAT LEACHING OBJECTIVES

The major objectives for operating the pilot vat leaching plant were:

To confirm the favorable metallurgical results obtained during bench scale testing of drill core samples. Specifically, the work involved demonstrating on a continuous basis that a recovery of 98 percent of the acid-soluble content could be achieved.

To demonstrate that copper-bearing solutions obtained from vat leaching were amenable for the subsequent solvent extraction and electrowinning steps.

To demonstrate that high quality copper cathodes could be produced in the electrowinning cell on a continuous basis as direct feed to a rod plant.

To determine the size to which the ore has to be crushed for optimum copper extraction.

To determine the leaching time in days needed to achieve optimum copper recovery. The vat pilot plant had been built to determine if upward or downward solution flows were more advantageous in terms of copper recovery over a period of time.

To determine if excessive fines in the ore fed to the plant would have to be removed in a wash circuit and treated in a separate agitation leaching plant.

To determine the effect of solvent extraction raffinate return on copper solution losses.

To conduct concurrent laboratory tests as required to supplement data generated by the pilot plant.

To determine the effect of other operating variables such as acid consumption, stages of acid addition, flowsheet modifications and leaching time variations.

To determine which of several vat linings is best suited for concrete protection or if extra thickness of plain concrete is adequate and more economical.



## TEST SCHEDULE

A test program schedule commencing on April 1st, 1975, and continuing to the end of the year was planned. Actual pilot vat leaching operation began on March 31st and the program was completed by January 15th, 1976. However, as is common in pilot plant operations, the original program due to various circumstances had to be modified from the original schedule mainly because of the following reasons:

Late delivery of acid flowmeters which were not placed in service until September 2nd and without whom acid additions to the individual vats were too inaccurate.

Time consuming modifications to the seal arrangement on the vat doors. Solution losses due to door leakages were enormous and made meaningful interpretation of metallurgical results nearly impossible. It was not until about the middle of August that a method was developed which reduced seal leakage to zero for the balance of the test program.

Although most of the early work was useful in learning to operate the plant, for the above reasons it was decided to interpret and utilize only the metallurgical results obtained from September 2nd on.

The actual schedule achieved was as follows:

## CAMPAIGN No. 1, SEPTEMBER 2 to OCTOBER 15

During this period the ore was crushed to pass a 3/8-inch screen. Testwork concentrated on varying the amounts of stage fed sulfuric acid. Bedding of ore in the vats was being improved. Soluble copper losses in the residue were being reduced. A 50 percent solvent extraction raffinate return was begun on October 10 and resulting from this the final stage acid addition was discontinued.



## CAMPAIGN No. 2, OCTOBER 15 to NOVEMBER 6

The ore fed to the vats was crushed to pass a 5/8-inch screen. Concern was mainly centered on reducing soluble copper losses in the residue. To effect this the pregnant solution was diluted to decrease the copper content from about 8 grams to 3.5 grams of copper per liter. In that range the solvent extractant LIX64N improves copper removal and raffinates containing much less copper are produced.

## CAMPAIGN No. 3, NOVEMBER 6 to DECEMBER 1

The ore was crushed to pass a 1/2-inch screen before being fed to the vats. Main purpose of this campaign was to determine metallurgical results at this ore size. On November 24 a test with solutions flowing downwards through the vats was begun.

## CAMPAIGN No. 4, DECEMBER 1 to DECEMBER 23

Since best copper recoveries had been obtained from ore crushed to pass a 3/8-inch screen, the vats were again fed with ore of that size. Sulfuric acid concentrations in solution were optimized to provide a low acid pregnant liquor for solvent extraction. Up to December 12 downflow of solutions in the vats was practiced.

## CAMPAIGN No. 5, DECEMBER 23 to JANUARY 15

Certain parameters were rechecked and the test program was completed.

### AGITATION LEACHING OBJECTIVES

The major objectives for operating the pilot agitation leaching plant were:

To confirm the favorable metallurgical results obtained during bench scale testing of drill core samples and to demonstrate on a continuous basis that 98 percent of the acid-soluble copper can be recovered.

To demonstrate that copper-bearing solutions obtained from agitation leaching were amenable for the subsequent solvent extraction and electrowinning steps.

To demonstrate that good quality copper cathodes could be produced in the electrowinning cell on a continuous basis as direct feed to a rod plant.

To determine the size to which the ore has to be ground for optimum copper extraction.

To determine the leaching time in hours required to achieve optimum copper recovery.

To determine the number of extracting and stripping cells required in solvent extraction.

To conduct concurrent laboratory tests as deemed necessary to supplement data generated by the pilot plant.

To determine the effect of other operating variables such as acid requirement, need for additions of a neutralizing agent, variations in solution flows and general flowsheet modifications.

### TEST SCHEDULE

A test program schedule commencing on June 1st, 1975, and continuing to the end of the year was planned. Actual pilot agitation leaching operation began on June 5th and was continued to January 15th, 1976.

To confirm and consolidate the results obtained during the above period, the pilot agitation leaching operation was resumed on March 15th, 1976 and continued to June 29th, 1976.

In the evaluation of the metallurgical results obtained during the operation of the plant, the initial period from June 5th to July 30th was not included for the following reasons:

During that period inexperienced operators were being trained.

Minor modifications to the piping and pumping installations had to be carried out.

Additional acid and solution flowmeters had to be purchased and installed to obtain good metallurgical recording data.

The schedule achieved was as follows:

#### CAMPAIGN No. 1, JULY 30 to AUGUST 20

The ore was ground to 8 percent plus 35 mesh. Testwork was mainly conducted on optimizing acid addition, pulp densities and solution flows.

#### CAMPAIGN No. 2, AUGUST 20 to SEPTEMBER 10

Ore grind was maintained at 7 percent plus 20 mesh. Much the same testwork as in Campaign No. 1 was carried out, but at the coarser grind.

#### CAMPAIGN No. 3, SEPTEMBER 10 to SEPTEMBER 25

At 5 percent plus 10 mesh the ore was coarsened even more during this campaign. The impeller tip speed in the pH adjustment tank was increased to keep the solids in the pulp suspended. Other testwork concentrated on optimizing acid addition, pulp densities and solution flows.

## CAMPAIGN No. 4, SEPTEMBER 25 to OCTOBER 20

Attempts were made to coarsen the grind even further and several tests were carried out to leach ore ground to 2-3 percent plus 6-mesh. However, it was physically impossible to keep operations going at such a coarse grind and these tests had to be abandoned.

## CAMPAIGN No. 5, OCTOBER 20 to NOVEMBER 10

All raffinate was returned to the circuit as wash solution from this campaign on. Ore grind was maintained at about 4 percent plus 8-mesh. Other test investigations were pulp densities, solution flows and acid levels in the pregnant solution.

## CAMPAIGN No. 6, NOVEMBER 10 to JANUARY 15, 1976

Single-stage leaching was carried out during this period. Copper extraction, limestone and acid additions and retention time were the main test parameters investigated during this period. Grind during this time was carried out to about 2 percent plus 10 mesh.

## TEST PERIOD, MARCH 15 to JUNE 29, 1976

Several campaigns were conducted during this period. The main objective was to establish copper recoveries and acid consumptions in a two-stage leach system at grinds of 95 percent minus 10 or minus 14-mesh and to obtain consistent results on a continuing basis. Other objectives were to operate at minimal copper solution losses, high pulp densities and low pregnant solution turbidities. The solvent extraction plant and the electrowinning plant were also operated during this period. The objectives set for these plants were to improve and sustain good solvent extraction plant recoveries and produce copper cathodes of a quality acceptable to fabricators.

## METALLURGICAL RESULTS

## VAT LEACHING OPERATING PROCEDURE

In this section pilot plant metallurgical results are presented and interpreted. The data include results of optimum size determination for the ore fed to the vats, acid requirements and leaching time needed.

Three ore sizes, 3/8, 1/2 and 5/8-inch, were tested during the program. The screen analyses which follow, show the sizes obtained when crushing through screens with the above mentioned openings:

TABLE 1: SCREEN ANALYSES OF CRUSHED ORE

Analysis Weight, %	Crushing Screen Size, in.		
	<u>3/8</u>	<u>1/2</u>	<u>5/8</u>
+0.525 inch			nil
+0.371 inch		4.0	
+0.3125 inch	2.4		
+4 mesh	25.2	36.4	42.5
+14 mesh	38.6	30.6	28.1
+65 mesh	17.9	15.2	15.3
+200 mesh	5.3	4.0	5.4
-200 mesh	10.6	9.8	8.7

The following table presents the vat processing sequence. It was used in the pilot plant until raffinate was returned as wash solution. When raffinate was returned, the circuit was modified as described later in this section.

TABLE 2: VAT OPERATING SEQUENCE

Hour		Event	Solution Into and Exiting Vat	
Start	Stop		From	To
0.00	24.00	Load	-	-
24.00	30.67	Cover	Previous Vat	-
30.67	38.00	Recirculate	-	-
38.00	48.00	Advance	Previous Vat	Pregnant Storage
48.00	54.67	Advance	Previous Vat	Next Vat
54.67	62.00	Recirculate	-	-
62.00	78.67	Advance	Previous Vat	Next Vat
78.67	86.00	Recirculate	-	-
86.00	102.67	Advance	Previous Vat	Next Vat
102.67	110.00	Recirculate	-	-
110.00	126.67	Advance	Previous Vat	Next Vat
126.67	134.00	Recirculate	-	-
134.00	150.67	Advance	Previous Vat	Next Vat
150.67	158.00	Recirculate	-	-
158.00	168.00	Advance	Previous Vat	Next Vat
168.00	174.67	W <sub>1</sub> Wash	W <sub>1</sub> Storage	Next Vat
174.67	182.00	Recirculate	-	-
182.00	192.00	Water Wash	Storage	Next Vat
192.00	193.50	Water Wash	Storage	W <sub>1</sub> Storage
193.50	201.00	Drain	-	W <sub>1</sub> Storage
201.00	216.00	Unload	-	-



The pilot plant incorporated solvent extraction for copper recovery and purification of vat leach liquors. A low acid-bearing leach solution was required for satisfactory recovery in solvent extraction. To obtain such a solution without neutralizing by chemical means, fresh ore was used to consume excess acid. After a vat was loaded, solution was advanced to cover the ore. Advance was then stopped and recirculation was begun. No sulfuric acid was added to the vat during the first recirculation and the reaction of excess acid with the ore in the advancing solution was relied upon to reduce the acid content for the subsequent pregnant liquor displacement. After this initial recirculation, advance was again begun and solution was displaced to the unclarified pregnant solution storage tank. Advance was continued with solution until the next vat, which had been newly loaded with ore, was covered and the second recirculation was then started. Thereafter, advance and recirculation steps were carried out in several twenty-four hour periods, until the washing step began. Washing was initiated with a copper-acid-bearing wash solution from the wash storage tank. Wash volume was equal to the amount required to cover new ore. After the first wash the vat was again recirculated. Water was used for the final wash. The initial 10,800 gallons, which were equal to the amount of pregnant liquor displaced, were advanced and the remaining 1,632 gallons approximately equalled the amount of water contained in the leach residue less the water added during agglomeration. The volume of water was varied to maintain the solutions in balance. Finally, the vat was drained to wash storage. Throughout the test program a constant 10,800 gallons of pregnant solution were displaced from each vat though the quantity of ore loaded into the vats varied slightly from day to day. A vat solution balance is presented on the following page.

TABLE 3: VAT SOLUTION BALANCE  
(Based on 94.3 tons of dry ore)

<u>Solution Into Vat</u>	<u>Gallons</u>	<u>Tons/Ton of Dry Ore</u>
Agglomeration Water	1,776	0.079
Cover	7,200	0.318
W <sub>1</sub> Wash	7,200	0.318
Water Wash	12,432	0.549
	<hr/>	<hr/>
Total	28,608	1.264
 <u>Solution Out of Vat</u>		
Displacement	10,800	0.477
To Cover Next Vat	7,200	0.318
W <sub>1</sub> to Storage	1,632	0.072
Drain	5,568	0.246
Residue	3,408	0.151
	<hr/>	<hr/>
Total	28,608	1.264

Except for one test period, solution flow through the vats was maintained upflow at 18.0 gallons per minute (0.156 gallons per minute per square foot). This flowrate resulted in 6.67 hours for covering, 10.00 hours for displacement, and 7.33 hours for recirculation.

The schedule called for a total of 63 hours for loading, unloading and repairing a vat. For full-scale design this total time appeared to be adequate, but would have to be utilized as needed to accommodate each step in the operating sequence.

The vat leaching test program was carried out for the greater part with ore passing a 3/8-inch screen. Once data from this size were obtained, coarser ore sizes were studied.

Acid addition to each vat was controlled by flowmeters. On September 10th, acid addition points and quantities were fixed and not altered until later in the program when raffinate was returned from solvent extraction. Ore loading to each vat averaged 94.3 tons per vat during the period September 2nd to October 15th. Acid was added during the second through sixth recirculation at a rate of 0.7 gallons per minute. Procedure for acid addition in the vats is shown below and in Figure 4. Total addition amounted to 57 pounds of 100 percent acid per ton of dry ore.

TABLE 4: PROCEDURE FOR ACID ADDITION TO VATS  
(Addition rate: 0.7 gpm)

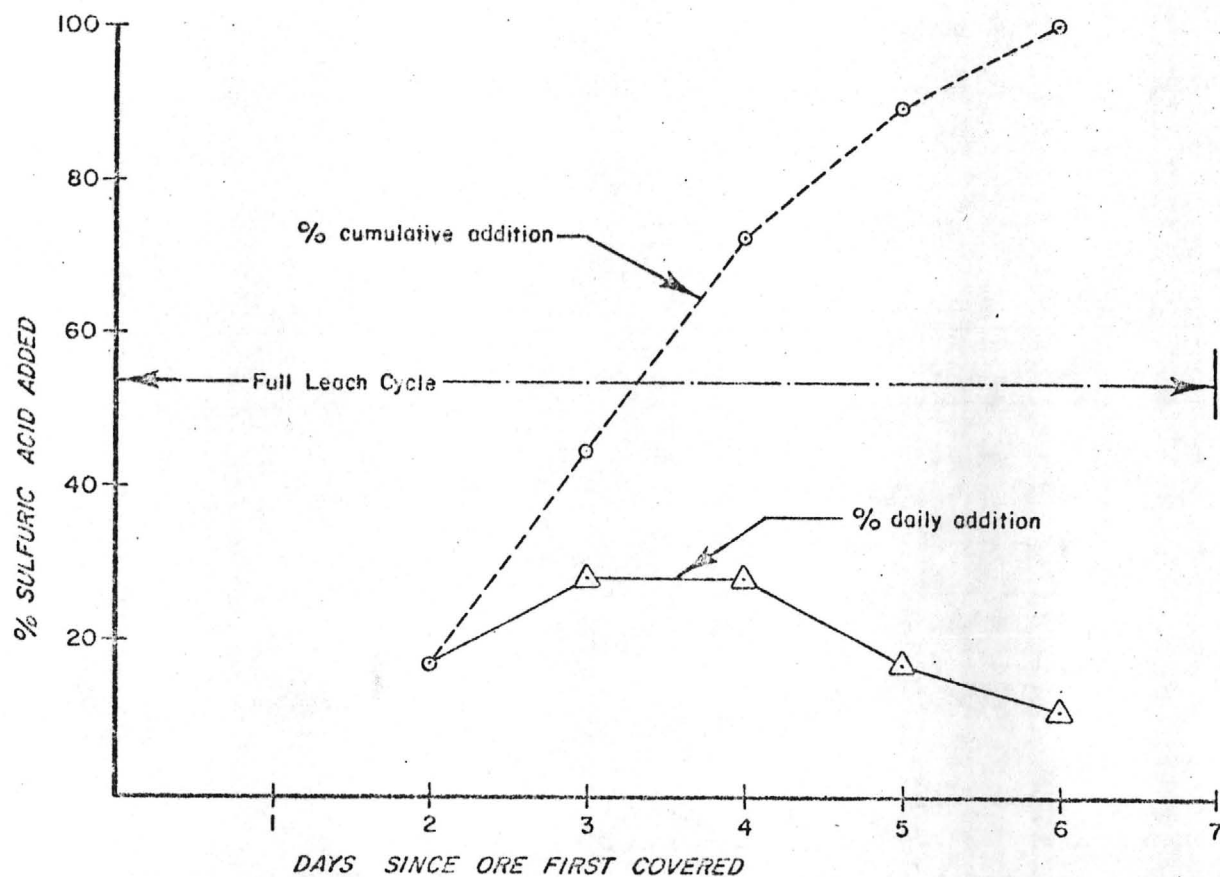
<u>Recirculation No.</u>	<u>Acid Addition</u>	
	<u>Gallons</u>	<u>% of Total</u>
2	63	16.7
3	105	27.8
4	105	27.8
5	63	16.7
6	42	11.0
Total	378	

Metallurgical accounting of copper and acid for each vat was monitored daily by sampling and analyses of solutions in and out of each vat. Though day to day accountabilities fluctuated widely, over an extended period of time they agreed very closely.

A 2.5 percent weight loss of the ore during leaching was allowed for throughout the pilot plant program.

LEACH CYCLE PERCENT OF ACID ADDITION vs.  
DAYS SINCE ORE FIRST COVERED

( Figure 4 )



CONOCO

FLORENCE, ARIZONA

Data By J.J.D.

Drawn By R.C.C., 10-13-76

## OPERATING RESULTS

During the period of September 2nd to October 15th, 1975, the dry ore had a bedded bulk density of 87.2 pounds per cubic foot and assayed 0.43 percent total copper and 0.31 percent acid-soluble copper. An acid-soluble copper extraction of 102.0 percent was obtained, when adding 55.8 pounds of acid per ton of ore. With a 4.5 percent soluble loss leach recovery was 97.5 percent. Soluble losses gradually decreased to two percent after the circuit came into balance.

Some compaction of the ore occurred during processing and the bedded bulk density of the leach residue was 92.3 pounds per cubic foot.

The pregnant liquor assayed 6.5 grams copper per liter and 9.3 grams acid per liter. Copper accountability was calculated as 101.4 percent during this period.

The highest copper recoveries obtained during the complete program occurred during the period September 25th to October 15th. Copper extraction amounted to 106.4 percent of the acid-soluble copper and after a 2.2 percent soluble loss, overall recovery was 104.2 percent.

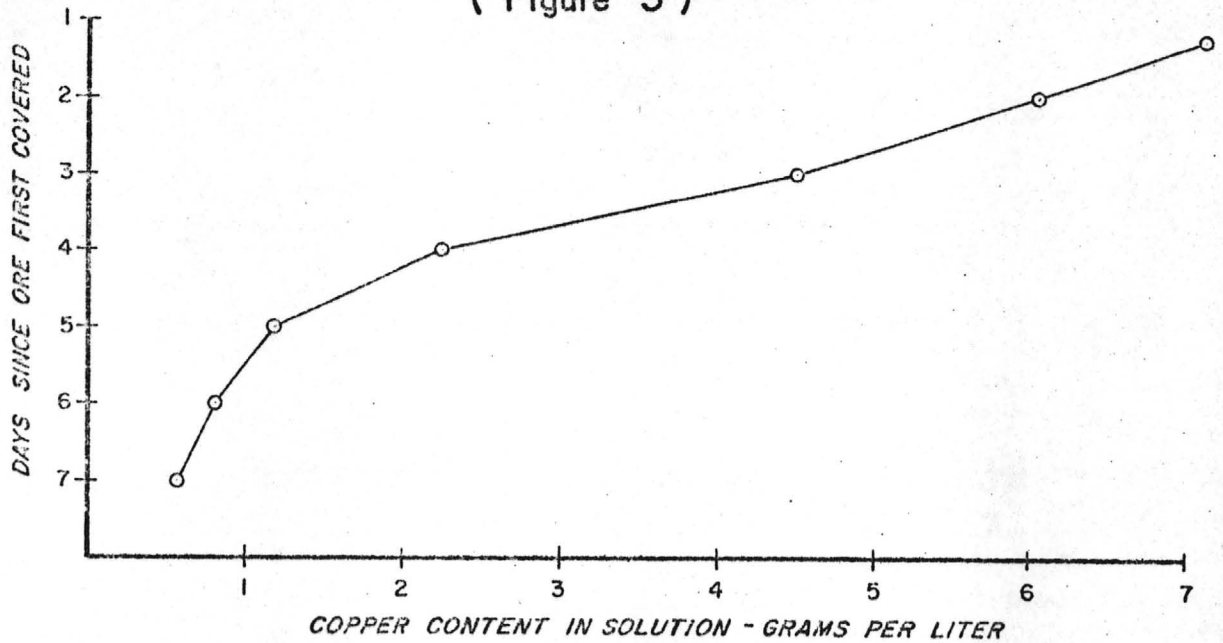
Copper and acid concentrations of the solutions leaving the vats (off solutions) are shown as follows for the September 25th to October 15th, 1975 test period (also Figures 5 and 6).<sup>1</sup>

TABLE 5: COPPER AND ACID ANALYSES OF VAT OFF SOLUTIONS  
9/25 to 10/15/75

<u>Day</u>	<u>Off Solution</u>	<u>g Cu/l</u>	<u>g Acid/l</u>
1	To Pregnant	7.14	7.2
1	To Cover	6.06	12.0
2	Advance	4.53	21.0
3	Advance	2.27	22.9
4	Advance	1.22	18.4
5	Advance	0.79	11.3
6	Advance	0.57	6.5
7	Advance	0.47	3.6

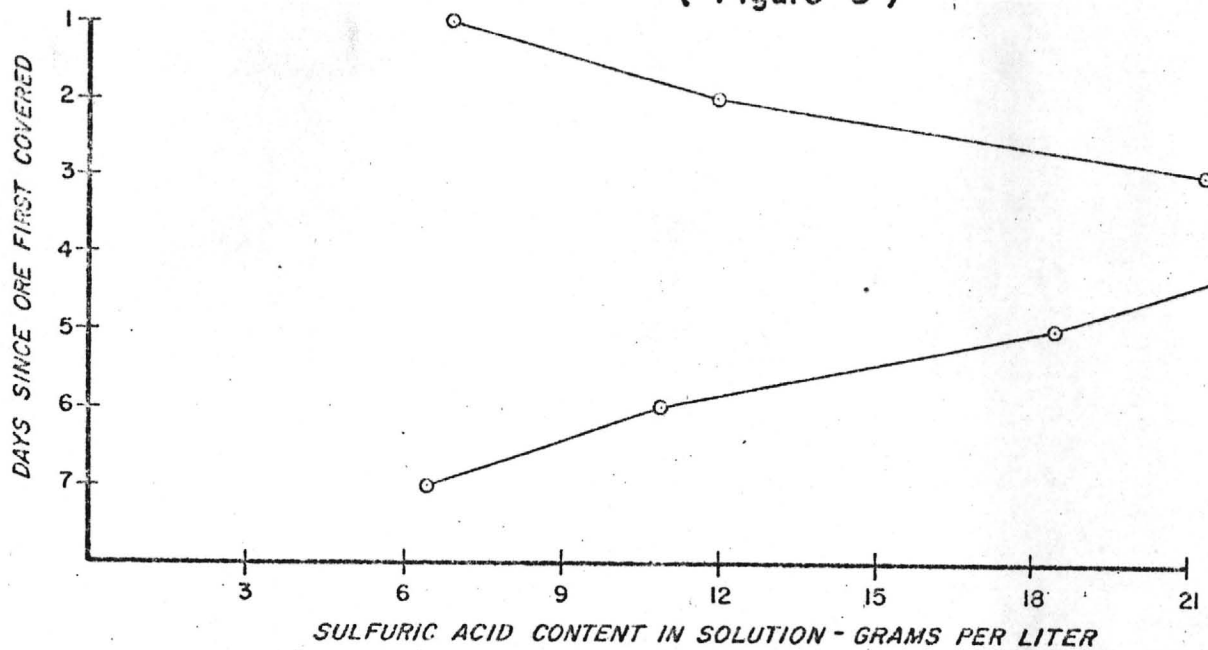
**SOLUTION STRENGTH IN GRAMS COPPER PER LITER vs.  
DAYS SINCE ORE FIRST COVERED**

( Figure 5 )



**SOLUTION STRENGTH IN GRAMS SULFURIC ACID PER LITER  
vs.  
DAYS SINCE ORE FIRST COVERED**

( Figure 6 )



**conoco**

FLORENCE, ARIZONA

Data By *J.J.D.*

Drawn By *R.C.C.* 10-13-76



Although the acid concentration in the pregnant liquor was too high for satisfactory recovery in the subsequent solvent extraction step, the acid gradient attained in the individual vats was close to the desired level. Highest acid strengths existed after the second, third and fourth day of leaching. A 50 percent raffinate return to the vats as wash was started on October 10th, 1975. Since the raffinate contained acid nearly equal to the quantity added during the sixth recirculation, the addition of acid at this point was discontinued and total acid addition was reduced to about 50 pounds per ton of ore.

With the addition of raffinate to the vats (5,400 gallons) the total wash water volume was decreased from 12,432 to 7,032 gallons. The raffinate was added just after the seventh recirculation and was then followed by a water wash. Raffinate return was continued for the remainder of the test program.

During the period of October 15th to November 6th, 1975, ore which passed a 5/8-inch screen was leached in the vats. Copper extraction amounted to 95.7 percent of the acid-soluble content. Soluble losses were high and copper recovery decreased to 89.3 percent. This was due partly to a high copper content in the raffinate and possibly due to not returning raffinate to a more advanced point in the washing cycle.

Researching published information on the use of LIX64N showed that improved copper extractions could be obtained when processing a lower grade pregnant liquor feed. It was therefore decided to produce a pregnant liquor of 3 to 3.5 grams copper per liter. This was to allow an additional 10,000 gallons of water to be used for washing the soluble copper from the residue. As a result more copper would be recovered. This concept was followed for the remainder of the program. The desired solvent extraction feed solution was produced by combining pregnant liquor as obtained from the vats with an equal volume of wash solution from the last vat. The vat leach operating schedule was changed and after the sixth recirculation 10,800 gallons of raffinate were introduced into the vat. This was followed by the normal wash and recirculation sequence and then by 12,432 gallons of water from which 10,800 gallons of solution were subsequently mixed with pregnant liquor.

From November 6th to November 26th, 1975, leaching was carried out with ore crushed through a 1/2-inch screen. During this period higher grade ore was treated which made it difficult

to compare results with those obtained during the other leaching studies. Leach extraction averaged 99.9 percent and plant recovery was 97.7 percent of the acid-soluble copper. Acid consumption amounted to 48.3 pounds.

Previous work had shown highest copper extractions were obtained with 3/8-inch ore (see Figure 8). Although equivalent extractions might be obtained from coarser ore by extending the leach time, economics favored retaining the ten vat cycle and process 3/8-inch ore.

Testwork with 3/8-inch ore was resumed on December 1st, 1975, and continued to the end of the test program. It was planned to confirm the high copper extractions that had previously been attained and to optimize acid concentrations so that a low acid pregnant liquor for solvent extraction might be obtained. The average for the period showed an acid-soluble copper extraction of 100.0 percent (see Figure 7). However, the average extraction for the first twelve days was only 94.3 percent, but it then improved to 106.7 percent. This extraction was comparable to that obtained earlier when treating 3/8-inch ore.

From November 24th to December 12th, 1975 the vats were operated downflow at 18.0 gallons per minute. During this period upflow was only used when a vat was covered. However, gas produced by acid reaction with the ore flowed countercurrent to the solution and hindered the downflow recirculation rate so that the desired flow of 18 gallons per minute could not be maintained. Downflow operation was therefore discontinued for the remainder of the test program.

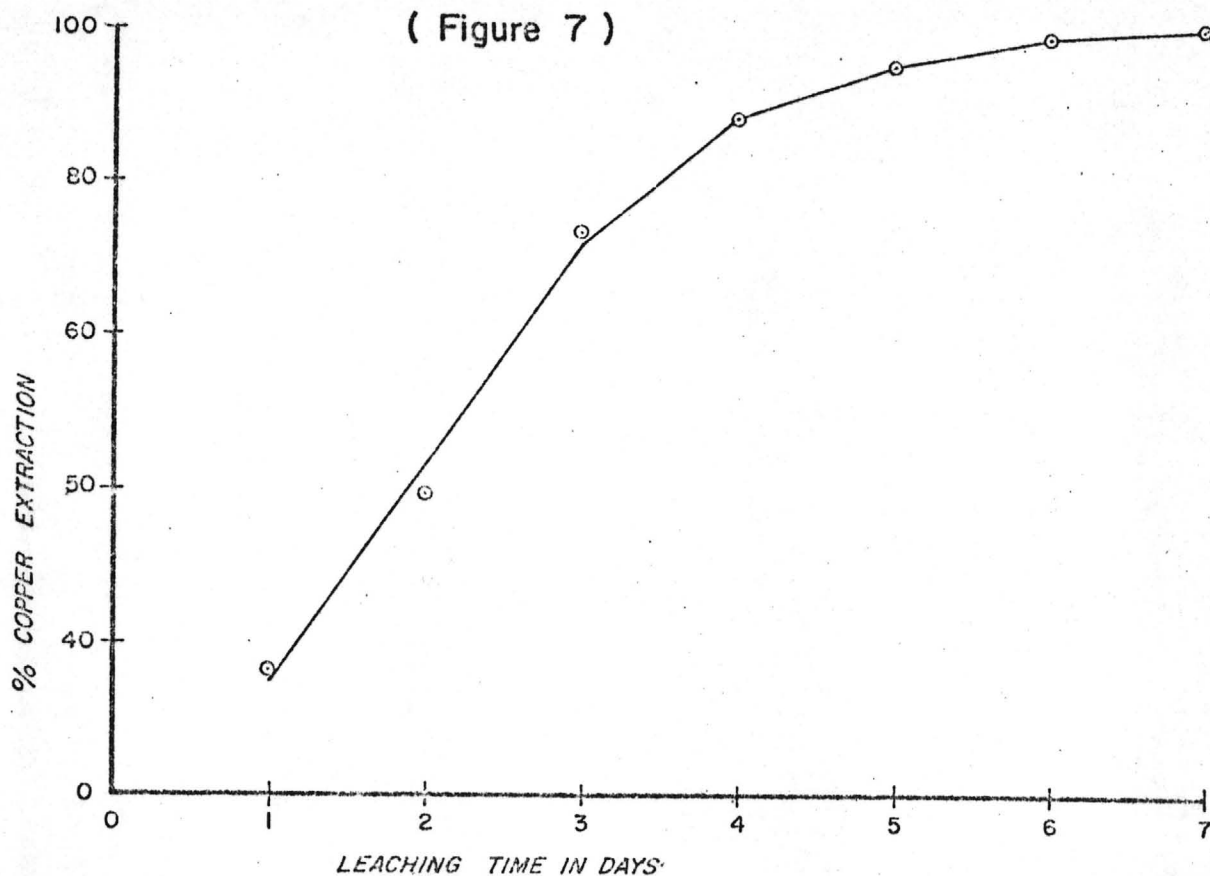
## AGITATION LEACHING

### GRINDING

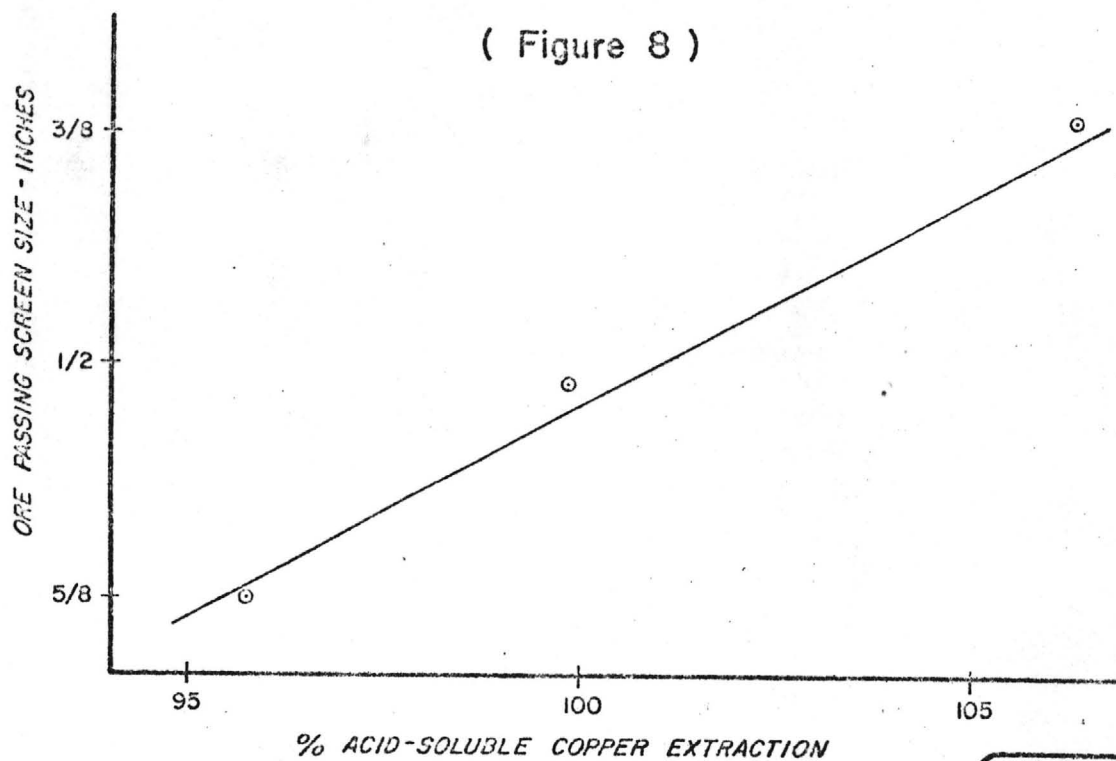
The grinding circuit consisted of a rod mill in conjunction with a spiral classifier. Feed to the rod mill was oxide ore which had been crushed through either a 3/8, 1/2 or 5/8-inch screen. Grind was controlled by adding or removing grinding rods. Generally a rod load of 180 to 250 pounds (5 to 9 rods) was all that was required for grinding of the ore. Grinding operations were carried out in open circuit with a 70 percent solids slurry in the rod mill and a 50 to 55 percent solids slurry in the classifier overflow. The ore fed to the plant contained from 3.3 percent to 5.4 percent moisture.

**% COPPER EXTRACTION vs. LEACHING TIME IN DAYS**

( Figure 7 )

**ACID-SOLUBLE COPPER EXTRACTION vs. SIZE CRUSHED**

( Figure 8 )

**CONOCO**

FLORENCE, ARIZONA

Data By *g* J.J.D.Drawn By *g* R.C.C., 10-13-78

## TWO-STAGE LEACHING

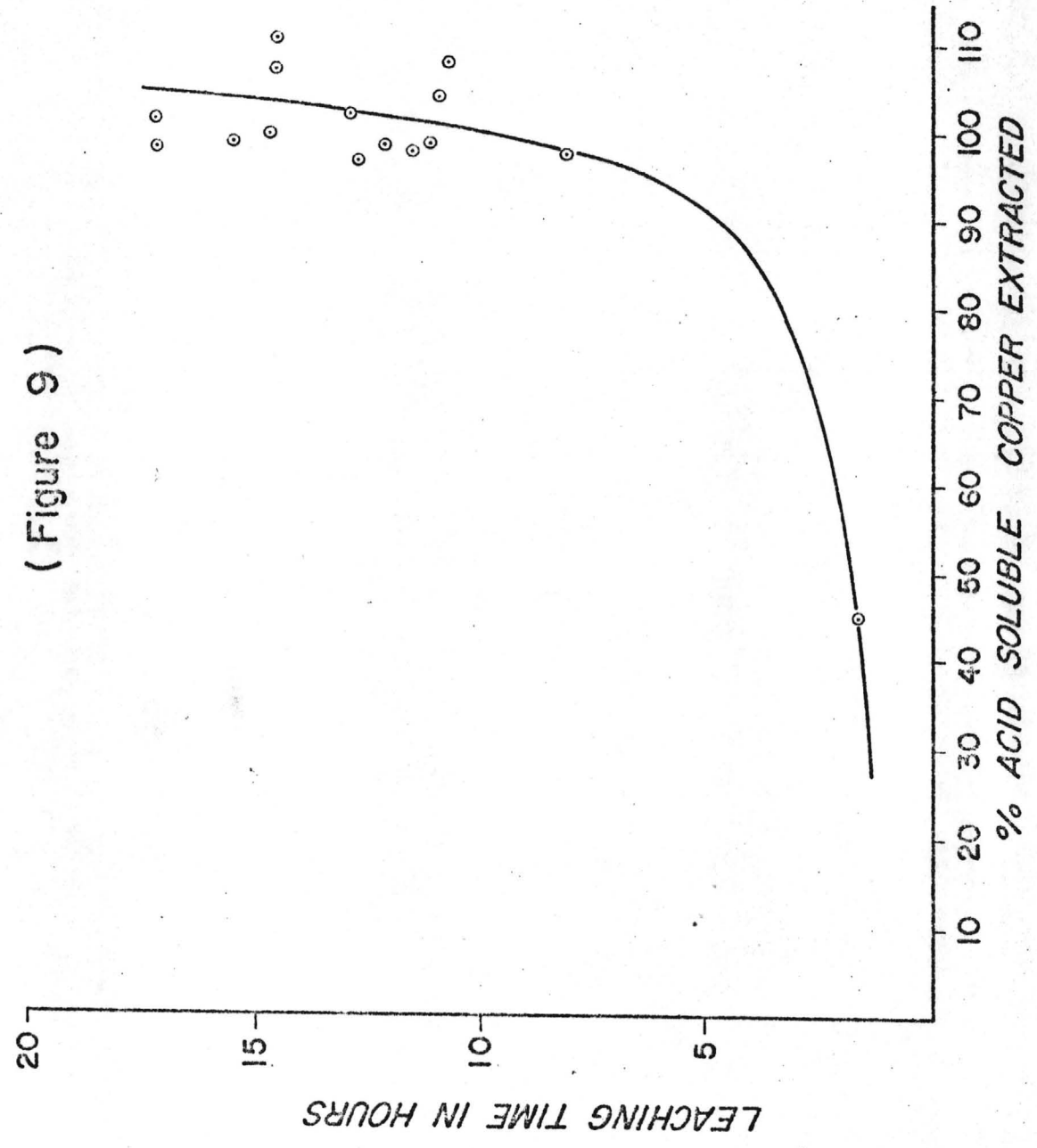
Accumulation of interpretative pilot plant data was begun on July 30th, 1975. By the end of July, the leach circuit was operated in a steady state situation. During the entire program acid was added to the first tank of the second-stage leach. To determine the time necessary for leaching, hourly samples of leach slurry overflowing the third and the fourth leach tanks were taken and immediately filtered and washed. Filter cakes for 24 hours were composited and assayed for total copper content. A 2.5 percent weight loss after leaching was determined during the test program.

The initial leaching study was conducted on a comparatively fine-ground ore (about 83 percent passing 48-mesh). The results for the period July 30th to August 20th, 1975, show a very high extraction of acid-soluble copper from ore which contained 0.48 percent total and 0.31 percent acid-soluble copper. In 10.9 hours of leaching extraction was 108 percent of the acid-soluble copper, and in 14.6 hours it reached 110 percent (see Figure 9). The final residue assayed 0.14 percent copper. Copper dissolved amounted to 6.90 pounds per ton of ore. Terminal pH of the leach was 1.0, the leach temperature was 36°C and the percent solids was 55. Difficulty was experienced in controlling acid addition until properly sealed acid-metering equipment was installed in August.

During this period the first-stage leach averaged 27.0 percent solids at a pH of 1.6. The pregnant liquor assayed 1.34 grams of copper per liter and a pH of 1.6. This test period gave the high copper extractions that can be expected from finely ground ore.

Subsequent leaching testwork was conducted on ore which was ground much coarser than in the initial leach study (see Figure 10). Ore ground to 2.5 percent on 14-mesh shows acid-soluble copper extractions of 104.8 percent and 105.7 percent in 11.0 and 14.7 hours, respectively. The ore assayed 0.47 percent total copper and 0.36 percent acid-soluble copper and the final leach residue contained 0.09 percent copper. Acidity in the leach circuit (0.9 terminal pH) was higher than in the previous test and acid addition amounted to 70.3 pounds per ton of ore. The acid addition rate was well-controlled. An abrupt decrease in acid requirements to about 45 pounds per ton occurred on September 9th while still operating under the same leach acidity. The lower acid consumption continued for the remainder of the two-stage leaching studies and is undoubtedly due to a decrease in acid consuming gangue materials in the feed.

Percent copper extracted as acid soluble copper  
Versus  
Leaching time in hours  
( Figure 9 )

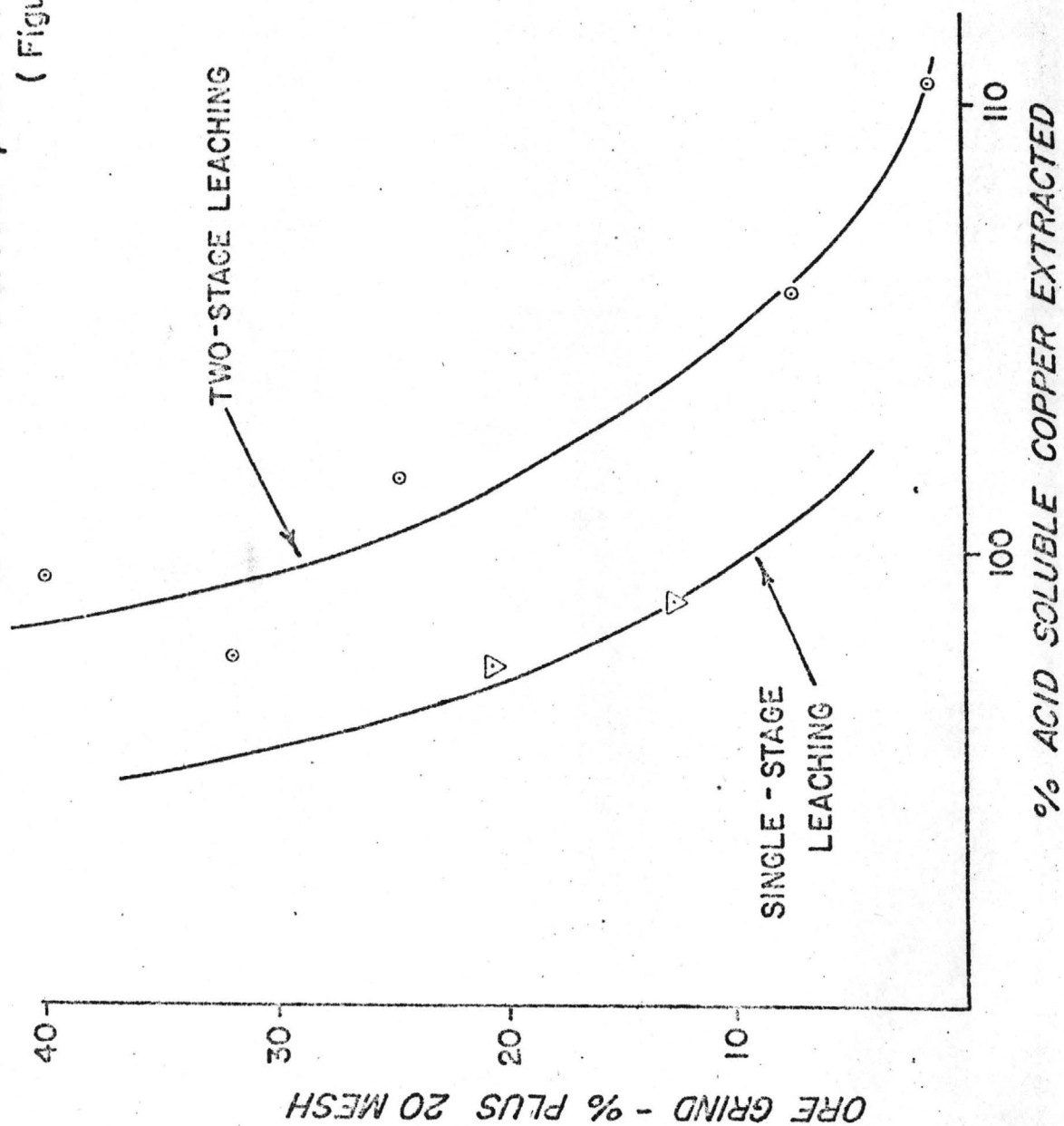


FLORENCE, ARIZONA

Data By: J.J.D.  
Drawn By: R.C.C., II-23-76



Percent acid soluble copper extracted  
Versus  
Percent plus 20 mesh ore grind  
(Figure 10)



CONOCO

FLORENCE, ARIZONA

Data By: J.J.D.

Drawn By: K.C.C., 11-24-76



The pregnant liquor assayed 1.54 grams copper per liter with a pH of 1.5.

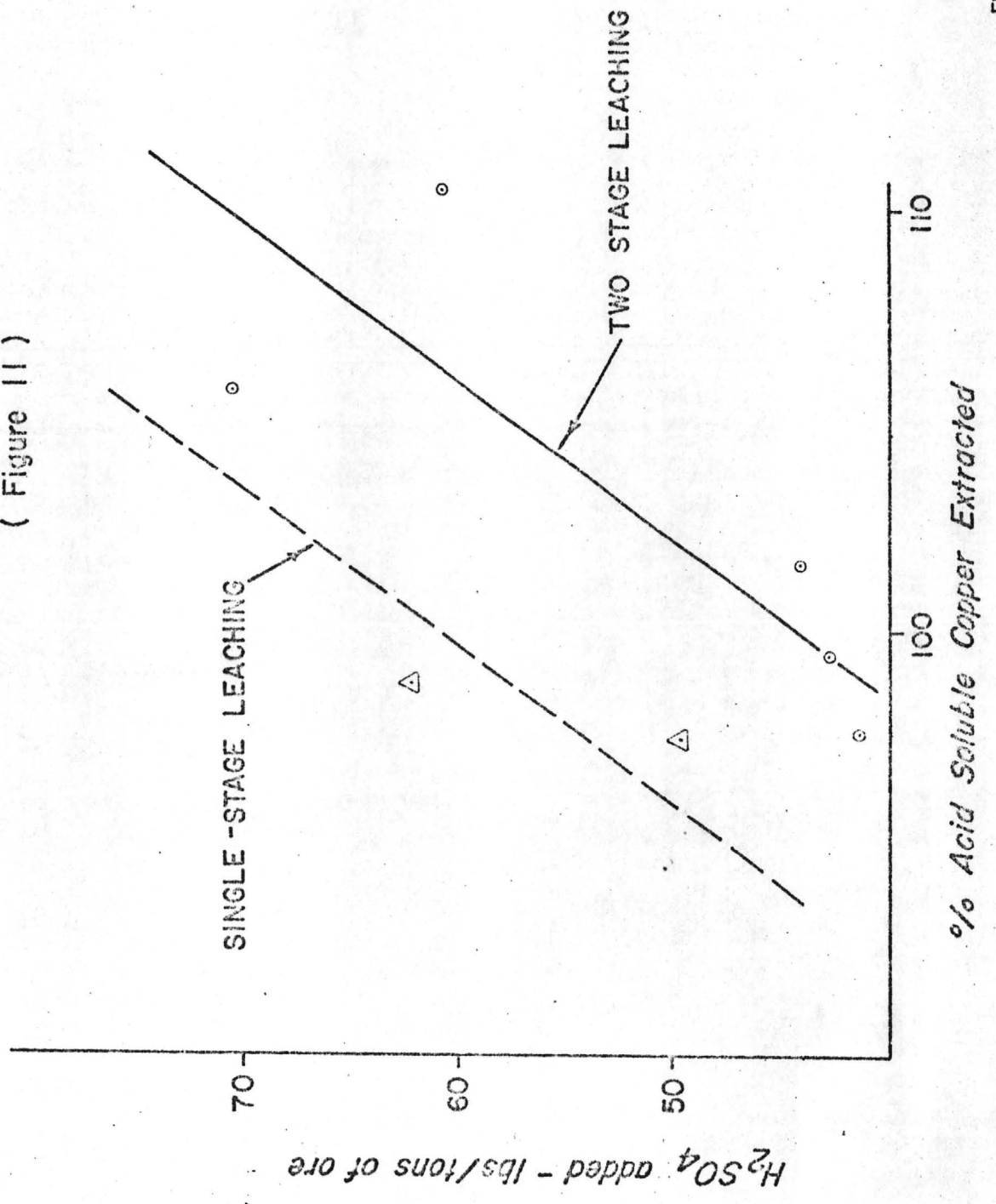
From September 10th to September 25th leaching was conducted on a very coarsely-ground ore (4.6 percent plus 10-mesh). During the course of this testwork, agitation intensity in the pH adjustment tank decreased to the point where essentially no agitation was occurring. The agitator speed was therefore increased by 65 percent to a peripheral speed of 1,843 fpm and suitable particle suspension was again attained. Pulp density in the first stage of leaching was 28.4 percent solids. The overflow of each leach tank was raised two inches to force the leach slurry up through the risers and reduce short-circuiting. Leaching time was thereby increased to 17.3 hours. On September 22nd an open-circuit grind with 70 percent solids was begun and was retained for the remainder of the program.

A final leach residue containing 0.115 percent copper was obtained from ore which assayed 0.43 percent total copper and 0.31 percent acid-soluble copper. Leach extraction amounted to 102 percent of the acid-soluble copper (see Figure 11). Acid addition was 44.3 pounds per ton of ore. The results showed the same extraction for either 13.0 or 17.3 hours of leaching. Percent solids in the first and the second stage leach averaged 29.6 and 60, respectively. PH of the first-stage leach was 1.7

Since by this time all leaching results with a coarse grind had shown good copper dissolution and a substantial reduction in acid consumption, it was decided to attempt to grind to a coarser mesh size (2-3 percent plus 6-mesh). However, this grind could not be consistently achieved. Coarse ore particles did not flow through the leach tanks and intensity of agitation became extremely poor. All leach tanks had to be emptied on several occasions. Failure to maintain the coarse grind continuously made it impossible to gather reliable leaching data and after nineteen days of effort this test was discontinued. Although the results for this period showed good leach extraction (99.4 percent of the acid-soluble copper), the data could not be considered as representative because of inconsistent operating conditions.

As of October 20th all raffinate from the solvent extraction circuit, which previously had been discarded, was returned as a wash solution to the countercurrent decantation circuit. The raffinate was metered into the last stage of washing at a

Percent copper extracted as acid soluble copper  
Versus  
Sulfuric acid consumption  
( Figure 11 )



FLORENCE, ARIZONA

Date By: J.J.D.  
Drawn By: R.C.C. , 11-23-76

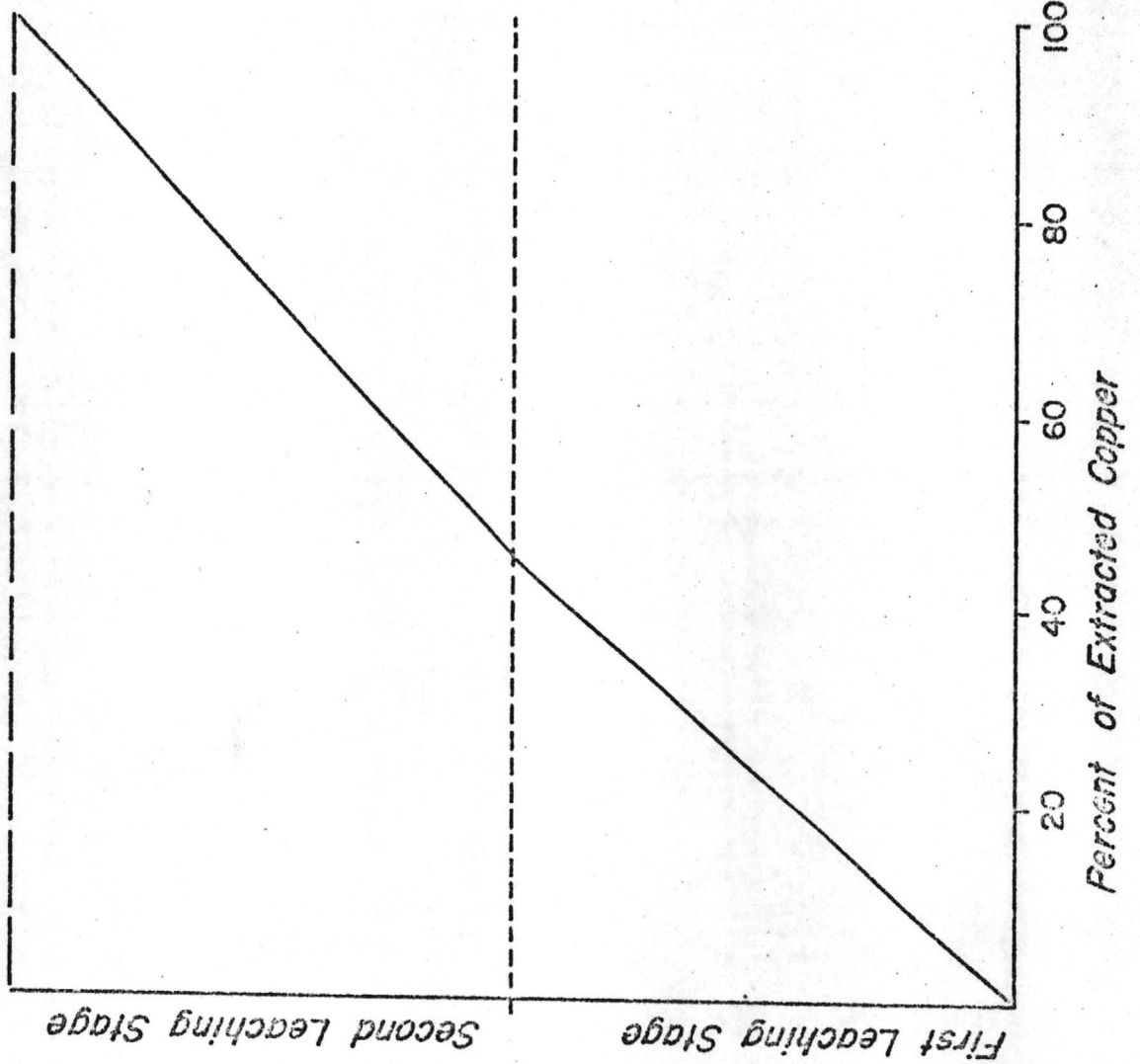
rate equal to the volume of pregnant liquor removed. Since this solution contained acid, its effect on the acidity of the pregnant liquor was closely followed. It was recognized that the oxide ore might not have sufficient acid consuming constituents to neutralize the acid and produce a pregnant liquor suitable for solvent extraction. It was also realized that the energy required to suspend a slurry of 27 to 32 percent solids with no segregation of coarse and fine ore particles would be much higher than that required for a slurry of 50 to 55 percent solids. Soon after the raffinate return was begun, the density in the pH adjustment tank was increased by diverting a part of the advancing wash solution to the pH adjustment thickener. It was felt that the by-passed solution would react in the pH adjustment thickener, and the resulting increased leach time obtained in the first-stage leach tank would provide adequate conditions to produce a pregnant liquor of low acidity for solvent extraction. However, with a first-stage leach time of 3.7 hours at 50 percent solids the pregnant liquor had a pH of 1.4. This was too acidic for acceptable copper recovery in solvent extraction and would require addition of a neutralizing agent. It was therefore decided to carry out single-stage leaching tests for comparison with two-stage leaching (see Figure 12).

During the period of October 20th to November 10th, 1975 a coarse grind (3.9 percent plus 8-mesh) was maintained with an acid addition of 41.3 pounds per ton (terminal leach pH = 0.9). Leach extractions of 95.7 and 97.7 percent of the acid-soluble copper were obtained in 12.9 and 17.2 hours respectively. The ore contained 0.47 percent total copper and 0.38 percent acid-soluble copper and the final leach residue assayed 0.10 percent copper.

#### SINGLE-STAGE LEACHING

For a successful leaching and solvent extraction operation it is most desirable to return raffinate to recover copper contained in this solution. Also of importance is consuming the acid contained in the raffinate on the ore during treatment and thus reducing the amount of acid required for leaching. In two-stage leaching of this ore, sufficient acid was not consumed by the ore when operating with the time and temperature conditions prevailing in the first-stage leach.

# Percent of Extracted Copper Versus Leaching Stages ( Figure 12 )



FLORENCE, ARIZONA

Data By: J.J.D.

Drawn By: R.C.C. , 11-23-76

The only practical method of returning raffinate was as wash to the countercurrent decantation circuit. In so doing a portion of it became part of the tailing solution and the remainder advanced and, along with the balance of acid from the second-stage leach, contacted the ore in the first-stage of the two-stage leach system.

Single-stage leaching followed by limestone neutralization of pregnant solution could possibly result in a more economic method for copper recovery. This circuit would eliminate the complete first-stage leach capital and operating costs connected with a two-stage leach system.

The single-stage leaching circuit was placed into operation on November 10th and the first three leach tanks were used for leaching. The fourth tank was used for neutralization with 97.5 percent calcium carbonate. The lime was supplied as a 70 percent minus 200-mesh product. Part of No. 1 countercurrent decantation thickener overflow was added to the rod mill discharge.

Results showed the same acid-soluble copper extraction of 97.4 percent in either 8.1 or 12.1 hours of leaching. Acid addition amounted to 49.8 pounds per ton of ore at a terminal leach pH of 0.9. Adding 30 pounds of limestone per ton of ore to the neutralization tank resulted in a pregnant liquor with a pH of 1.6. The pH of the neutralization tank was 2.9. The ore assayed 0.58 percent total copper and 0.39 percent acid-soluble copper, which was higher than the ore previously processed. The leached residue contained 0.21 percent copper.

Since the extraction of copper was too low with 12.1 hours of leaching time, a fourth leach tank was added and single-stage leaching was continued until operations were terminated on January 14th, 1976. The results of this last leach study showed copper extractions of 95.8 and 98.8 percent of the acid-soluble copper in 11.6 and 15.5 hours of leaching time, respectively. Sulfuric acid addition amounted to 62.1 pounds per ton of ore. Terminal pH was 1.0 and after addition of 34.3 pounds of limestone per ton of ore the pregnant liquor had a pH of 1.8. The ore contained 0.50 percent total copper and 0.35 percent acid-soluble copper. The leach residue assayed 0.16 percent copper.

The reason 62.1 pounds of acid per ton of ore were required during this period compared to the 49.8 pounds per ton used in the previous test could not be determined definitely, but was probably due to an increase in acid-consuming constituents in the ore.



## COUNTERCURRENT DECANTATION (CCD)

The circuit for washing soluble metal values from the leached ore consisted of three spiral classifiers for sand washing and four thickeners for slime washing. The dual washing system which separated the coarse and the fine fractions was necessary to avoid coarse sand from entering the thickeners. Washing was accomplished by adding wash solution to No. 4 thickener, and advancing the solution countercurrently to the solids flow. The wash solution advanced in sequence from thickener to classifier and finally overflowed the No. 1 thickener to the pH adjustment tank, where it contacted the freshly ground ore. Slurry exiting this leach tank entered a small classifier, where the coarse and the fine sands were separated. The coarse fraction entered the second-stage leach tanks, while the fine fraction in a slurry of low pulp density first settled in the pH adjustment thickener and then was pumped and recombined with the coarse fraction in the second-stage leach. Pregnant liquor was obtained from the overflow of the pH adjustment thickener. When single-stage leaching was studied, pregnant liquor was obtained from the overflow of the No. 1 thickener.

From July 20th to October 20th, water was used as wash solution. Starting on October 20th raffinate from solvent extraction plus make-up water was used for washing until the termination of the program. Wash solution flowrate was maintained at 2.0 gallons per minute (about 2 tons per ton of ore) during the entire program.

Tailing from the coarse and the fine ore washing was not combined before being sent to the tailing pond. Daily weights of the wet coarse sand tailing were obtained and the moisture content was determined. Based upon the dry ore feed to the pilot plant, the dry weight of the coarse sand fraction, and employing a 2.5 percent weight loss of the ore during processing, the daily quantity of fine tailing was calculated. Moisture and copper contents of the fine ore tailing as well as the copper content of the solutions in both the wet coarse and wet slime tailing were analytically determined. Total copper and soluble copper losses were determined daily from these weights, from analytical determinations and calculations. This method is valid for the various determinations only if carried out over a long period of time. The table on the following page and the discussions present the average data accumulated over extended time periods.



TABLE 6: PILOT PLANT TAILING DATA

	<u>Coarse Tailing</u>	<u>Fine Tailing</u>	<u>Total Tailing</u>
Wet Weight, lbs.	176,406	415,971	592,377
% Solids	81.0	31.9	46.5
Dry Weight, lbs.	142,843	132,653	275,496
% of Total	51.8	48.2	100
% Total Copper	0.088	0.191	0.138
Total Copper, lbs.	125.84	254.14	379.98
Solution, g Cu/l	0.20	0.12	0.13
lbs. of Copper	6.79	34.07	40.86
Soluble Loss,			
% of Total Copper	0.54	2.73	3.28
% of Acid Soluble Copper	0.76	3.79	4.55

Discarding of leach slurry during the coarse grinding tests which was previously mentioned in the Two-Stage Leaching section, has been accounted for. The results showed that of the total dry solids in the tailing, 51.8 percent was in the coarse fraction and 48.2 percent was fines. The wet coarse and the wet fine fraction contained 81.0 and 31.9 percent solids, respectively. This resulted in a combined tailing of 46.5 percent solids. The averages of hourly underflow solid determinations of the No. 4 thickener showed 1.0 to 1.5 percent higher solids content than the analytical determination. All four washing thickeners had essentially the same underflow solids content. The pH adjustment thickener underflow, however, had a higher solids content which resulted in a 55 percent solids content in the second-stage leach.

Leach extraction of the acid-soluble copper during the period of September 22nd to October 20th, 1975, as determined from the final leach residue was 99.7 percent. The tailing analyses showed close agreement at 101.0 percent extraction. An even closer correlation of

leach extractions was found over the period of August 20th to October 20th, 1975. From leach residues the extraction was 102.0 percent and from tailing analyses 102.7 percent.

During the September 22nd to October 20th, 1975 period the ore fed to the pilot plant assayed 0.33 percent total copper and 0.32 percent acid-soluble copper. The total soluble loss of copper amounted to 4.55 percent. Net copper recovery of the acid-soluble copper was 96.45 percent.

The copper content in the coarse tailing solution at 0.20 grams copper per liter indicated that washing efficiency of this fraction was lower than the efficiency of the fine ore washing. Solution in the fine ore tailing assayed 0.12 grams copper per liter, and if washing efficiencies had been the same, then the higher copper content in the coarse tailing solution would not have been experienced (see Figures 13 and 14).

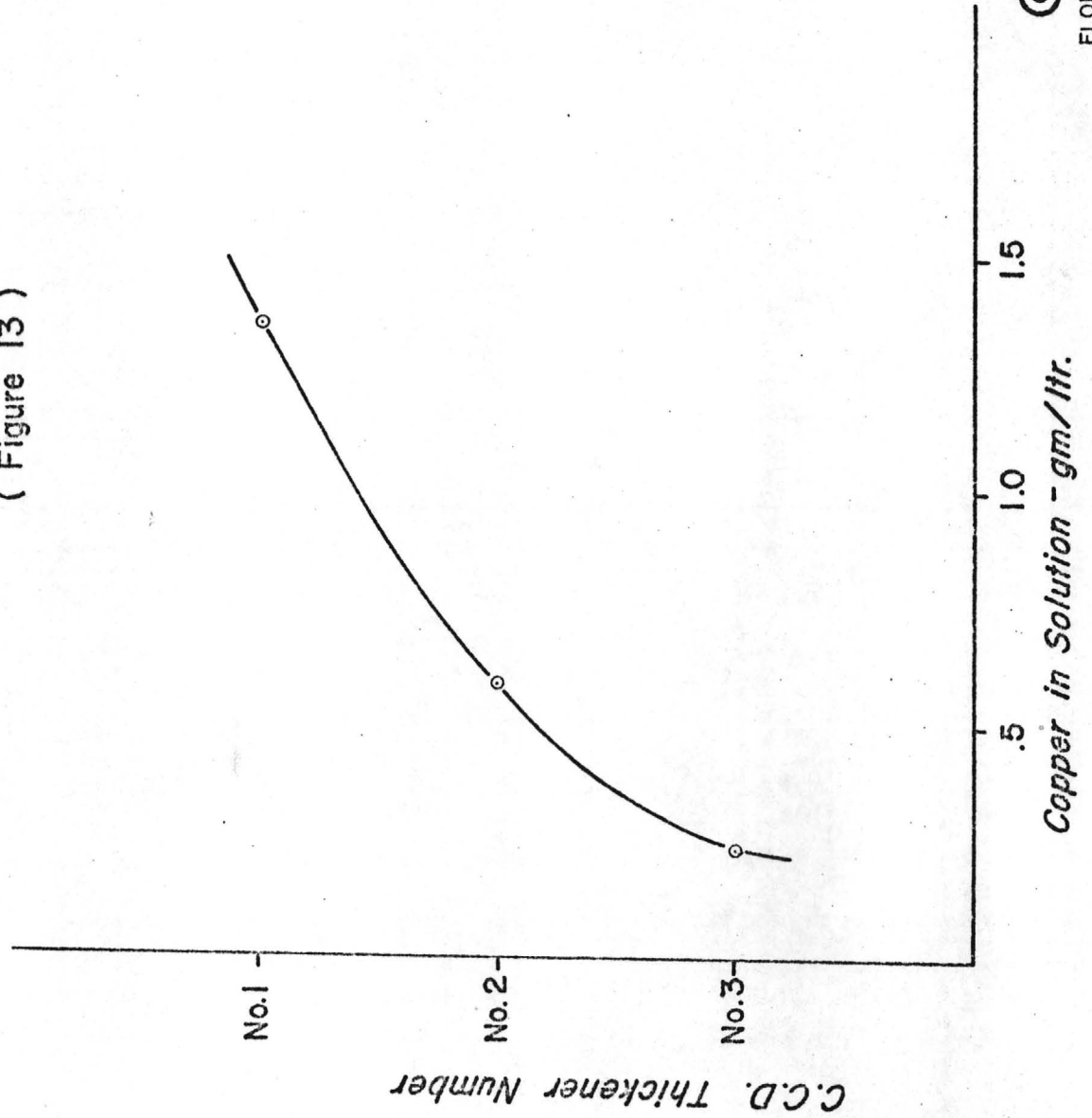
The soluble loss of copper in the total countercurrent decantation circuit could not be calculated theoretically since exact copper extraction in the first-stage leach could not be determined. However, a reasonable assumption was that 50 percent of the dissolvable copper was leached in the first-stage. With this assumption and with pilot plant determinations of 70 percent solids in the grinding circuit, 55 percent solids in the second-stage leach, 46.5 percent solids in the tailing, the same underflow solids content in all the washing thickeners, and with 2.0 gallons per minute of wash water, a theoretical soluble loss of the total copper dissolved was calculated at 6.02 percent. This was higher than was experienced in the pilot plant and could possibly have been due to addition of water in various areas of the pilot plant.

Throughout the duration of the pilot plant operation the quantity of pregnant liquor was much less by assay than calculated from both the theoretical and the actual soluble loss determinations. Pregnant liquor produced from the pilot plant was calculated to be 1.405 tons per ton of ore. When raffinate was returned as wash the flowrate was maintained at 1.4 gallons per minute. An additional 0.6 gallons per minute of water was added to give a total wash solution flow of 2.0 gallons per minute.

Flocculant was used to help the settling of fines in the thickeners. Dow Chemical's MG200 flocculant was used until November 17th, 1975 and thereafter American Cyanamid's Superfloc 127 flocculant was tested. The change to Superfloc 127 was made after

# *Copper Gradient in Wash Thickeners*

( Figure 13 )



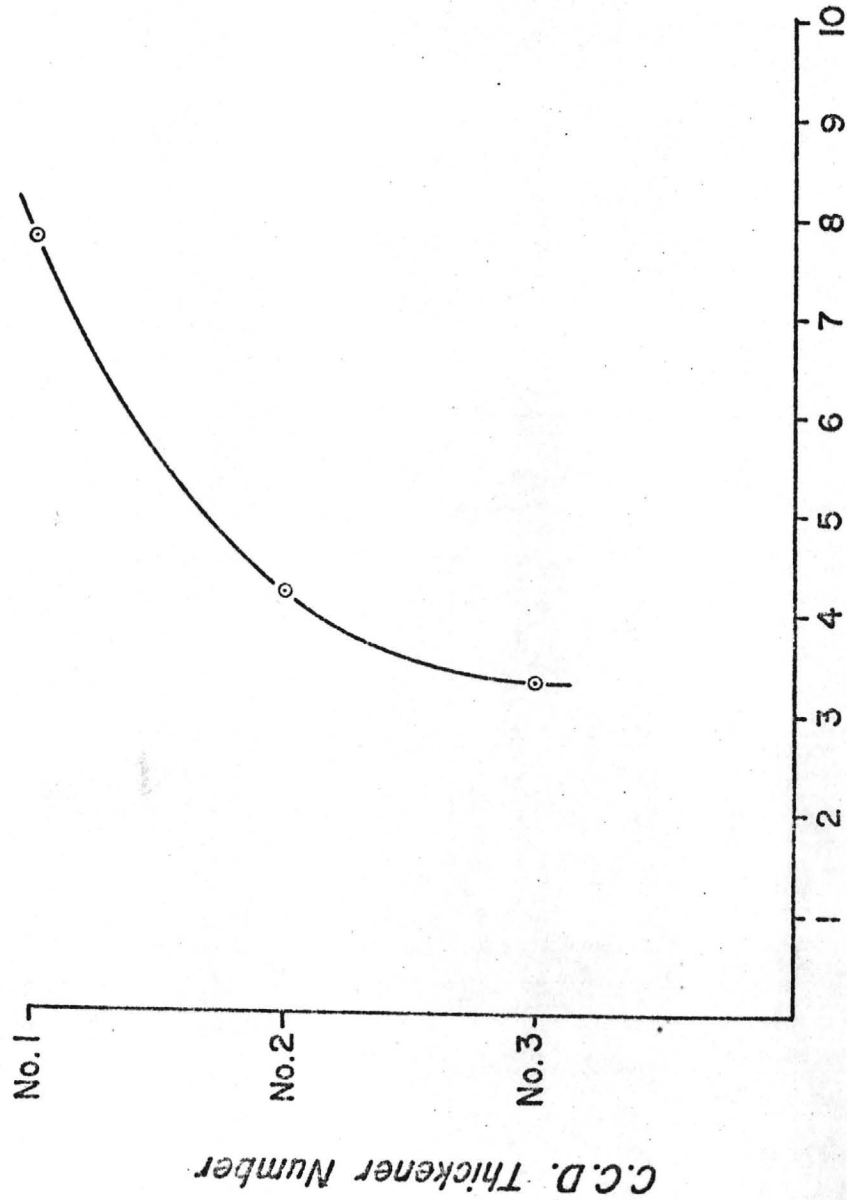
**CONOCO**

FLORENCE, ARIZONA

Data By: J.J.D.

Drawn By: R.C.C., 11-23-76

*Free Sulfuric Acid Gradient in  
Wash Thickeners  
( Figure 14 )*



*Free Sulfuric Acid in Solution - gm/ltr.*

**CONOCO**

FLORENCE, ARIZONA

Data By : J.J.D.

Drawn By : R.C.C. , 11-23-76

laboratory work showed a slight improvement in clarity of thickener overflow solutions and the same settling rate of the solids as with the MG200, while flocculant dosages were the same.

Initially about 0.1 pound of flocculant per ton of ore was added to the thickeners. Later on recycling of thickener overflows to mix and dilute the flocculant solution reduced this to 0.01 pounds per ton of ore.

The eight-foot diameter thickeners in the pilot plant had an area of 50.2 square feet. With a nominal feed rate of 6 tons of ore per day the unit settling area was 8.37 square feet per ton per day. It was soon recognized that the thickeners were too large to obtain scale-up data. A four-foot diameter thickener was therefore purchased and installed in place of the No. 1 CCD thickener. This smaller No. 1 thickener was capable of settling the ore on a unit area of 2.2 square feet per ton per day, when adding 0.01 pounds of flocculant per ton of ore.

A concentrated effort was made during the single-stage leach test period to establish thickener area requirements. Good data were acquired during a fifteen-day period when ideal thickener operations prevailed. Adding 0.012 pounds of Superfloc 127 per ton of ore, a thickener unit area requirement of 2.2 square feet per ton per day was determined. It was indicated that 0.01 pound of flocculant per ton of ore would be adequate. Additional settling rate determinations were carried out by Eimco Envirotech representatives.

Screen analyses were performed on various slurries during the test program. Analyses of the pH adjustment thickener feed and the No. 1 thickener feed showed a very high percentage of minus 325-mesh material, 63 to 64 percent, and only 9 to 10 percent of minus 200- and plus 325-mesh material. These analyses were performed when the ore was being ground in the rod mill to about two to four percent on 10-mesh.

Screen analyses of these same two slurry streams were also made on November 7th, 1975, when a 10 percent plus 10-mesh grind was being studied. These also showed a high percentage of minus 200-mesh material. Although minus 325-mesh analyses were not carried out, it can be assumed that most of the minus 200-mesh material was in the minus 325-mesh fraction.



Screen analyses of the ore before and after leaching, performed during the single-stage leach operation, showed that a considerable size reduction of the coarse fractions occurred during leaching and that most of the generated finer fractions reported into the minus 200-mesh product. It appeared that attrition during leaching caused this unusual size reduction.

#### 1976 AGITATION LEACH PLANT OPERATION

A second test program for the agitation leach plant was initiated on March 15th, 1976 and was continued to June 29th of this year (see Table 7). The main purpose was to confirm the optimum metallurgical results obtained during the previous year and to operate for lengthy periods of time under the favorable conditions previously established.

The main areas of study were to include:

The effect of acidity levels within a more narrow range than previously.

The effect of ore grind size on acid consumption and copper recovery.

Lime requirements for acidity control in the pregnant solution.

Flocculation and filtration tests for clarification of the pregnant solution.

A summary of metallurgical results obtained during this period is attached.

Throughout the test program the leaching time was held at 11 hours. Leaching was conducted at ambient temperatures which ranged from 29 to 34°C.

Grind sizes were held at 95 percent minus 10-mesh and 95 percent minus 14-mesh.

Acid consumption was essentially the same at both grind sizes. The finer grind improved copper extraction.



## 1976 Test Program

43

RUN	Start	1	2a	2b	2c (Totals)	3	4	5	6	7
<u>VARIABLES</u>										
Grinding	10-M	10-M	10-M	10-M	10-M	10-M	14-M	14-M	14-M	10-M
Leach, pH	1.0	1.0	0.9	0.9	0.9	0.9	0.7	0.9	0.8	0.8
Sand/Slime	63/37	63/37	63/37	63/37	63/37	63/37	56/44	56/44	56/44	63/37
<u>ORE TREATED</u>										
Dry Wt., Tons	59.0	81.2	58.8	61.9	49.0	169.7	52.0	58.9	54.5	54.9
Tot. Cu, lbs	531	715	519	543	442	1504	462	537	502	471
A.S. Cu, lbs	406	548	394	405	332	1131	347	400	365	354
Head, % Tot. Cu	0.45	0.44	0.44	0.44	0.45	0.44	0.45	0.45	0.45	0.43
% A.S. Cu	0.34	0.34	0.34	0.33	0.34	0.34	0.33	0.34	0.33	0.33
<u>LEACHING</u>										
Acid, lbs/Ton Ore	35.8	35.6	37.2	41.1	34.9	37.7	47.6	37.4	41.5	42.3
Lime, lbs/Ton Ore	-	-	-	-	-	-	4.26	-	-	-
Leach Sol., g/l Cu	1.7	2.6	2.8	3.3	2.8	3.0	2.5	2.6	3.3	2.9
Leach Solids, % Tot. Cu	0.14	0.15	0.16	0.15	0.15	0.15	0.13	0.14	0.14	0.13
% Solids	47	49	54	55	55	55	57	55	60	58
Temperature, °C	24	25	29	29	27	31	32	31	32	34
Pregnant Liquor, g/l Cu	1.6	1.6	2.0	2.0	1.9	1.9	1.7	1.8	1.5	1.6
<u>METALLURGICAL RESULTS</u>										
Insol. Loss, lbs Cu	149	211	107	144	115	365	106	108	131	116
% Tot. Cu	0.13	0.13	0.12	0.12	0.12	0.12	0.10	0.11	0.12	0.11
% Insol. Loss	28.00	29.48	20.53	26.53	25.85	24.30	22.95	23.65	24.34	24.68
Soluble Loss, lbs Cu	82	36	35	37	28	100	20	21	18	20
% Tot. Cu	16	5	7	7	6	7	4	4	4	4
% A.S. Cu	20	7	9	9	8	9	6	6	5	6
Total Loss, lbs Cu	230	247	141	181	143	465	126	129	151	136
<u>Cu Extraction</u>										
Dissolved, lbs Cu	382	504	413	399	327	1139	356	350	406	355
% Overall Recovery	56.6	65.4	72.8	66.6	67.8	69.1	72.8	71.9	71.8	70.0
% of Total Cu	72.0	70.5	79.5	73.5	74.2	75.7	77.0	76.4	75.7	73.7
% of A.S. Cu	94.2	92.0	104.7	98.4	98.6	100.6	102.6	104.0	101.6	101.3

Copper solubility appeared to be directly related to acidity of the leach solution. With the ore ground to 95 percent minus 14-mesh and leaching at a pH of 0.7 (47.6 pounds of sulfuric acid per ton treated), 104 percent of the acid-soluble copper was dissolved. This compares to an extraction of 101 percent at a pH of 0.9 (37.4 pounds of sulfuric acid per ton treated).

No lime was required to adjust the pH of the pregnant solution when leaching at a pH of 0.8 and 0.9. However, leaching at a pH of 0.7 required the addition of approximately five pounds of lime for every ton of ore treated to obtain a pregnant solution with a pH of 1.7.

The use of Polyox, a water-soluble resin produced by Union Carbide, resulted in relatively clear pregnant solutions. In addition clarification tests with an Immediumfilter, manufactured by DeLaval, resulted in a further reduction of suspended solids in the pregnant solution.

TABLE 8: COMPARISON OF PROCESSES  
A. Metallurgical

	<u>Vat Leaching</u>	<u>Agitation Leaching</u>
Ore Size	-3/8"	-10-mesh
Leaching Time	144 hours	11 hours
Sulfuric Acid Consumption, lbs/ton of ore	50	42½
Cu in Pregnant Solution, gm/l	7.4	1.7
% Cu Extracted	98	102
% Cu Recovered	96	100
% Raffinate Return	100	100

TABLE 9: COMPARISON OF PROCESSES

B. Capital Investment Requirement for  
30,000 tons per day Operating Plant  
(in MM\$)

	<u>Vat Leaching</u>	<u>Agitation Leaching</u>
Primary Crushing	10.5	10.5
Fine Crushing	12.0	10.9
Grinding	-	10.5
Leaching	40.8	12.2
CCD	-	13.5
SX	20.3	21.2
Electrowinning	15.6	15.6
Acid Handling & Storage	1.6	1.5
General Area	23.2	24.4
Miscellaneous	11.5	12.7
	<hr/>	<hr/>
Total:	<u>135.5</u>	<u>133.0</u>

TABLE 10: COMPARISON OF PROCESSES  
 C. Typical Direct Operating Cost Estimate  
 (cents per lb Cu)

	<u>Vat Leaching</u>	<u>Agitation Leaching</u>
Primary Crushing	1.2	1.2
Fine Crushing	2.9	2.6
Grinding	-	5.6
Leaching	14.7	11.5
SX	3.8	5.0
Electrowinning	9.9	9.8
Acid Handling & Storage	0.1	0.2
General Area	0.2	0.2
	<hr/>	<hr/>
Total:	<u>32.8</u>	<u>36.1</u>

### CONCLUSIONS

Metallurgical results obtained in the pilot plant indicate that agitation leaching and vat leaching are viable processes for the treatment of Florence oxide ores.

Copper recoveries were higher from ores treated by agitation leaching methods.

A compilation of the capital investment requirement for a 30,000 tons per day operating plant indicates the costs for a vat leaching plant to be slightly higher than those for an agitation leaching plant. On the other hand an estimate of plant operating costs shows them to be higher for agitation leaching.

The final decision on which process to employ in the production plant will have to be based on the combined effect of metallurgical recoveries and capital and operating costs.

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