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

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CONFIDENTIAL

SULLIVAN JOHNSON ARIZONA COPPER DEPOSIT

KYPROS MINES INCORPORATED

Kypros Mines Incorporated

Scottsdale, Arizona & Winter Haven, Florida

KMI

KYPROS MINES INCORPORATED

P.O. Box 371, Winter Haven, FL 33882-0371
Telephone (813) 324-5268

April 8, 1983

E. Grover Heinrichs & Associates
1802 W. Grant Road
Tucson, AZ 85705

Subject: Arizona Acid Soluble Copper Deposit

Dear Grover:

I am forwarding to you a copy of the KYPROS Confidential and Supplemental Report on its Arizona Acid Soluble Copper Deposit following our telephone conversation, and a recent telephone conversation with Mr. Paul Eimon. The potential highly profitable situation lends itself ideally to the direction in which the copper industry is headed.

This confidential report is being sent to determine an interest by you or any interested third party; however, prior to transferring or reproducing this report for any third party, permission must be obtained from KYPROS and their identity revealed. If there is no interest by you or a third party, the report is to be returned to KYPROS immediately.

KYPROS is a relatively new Delaware Corporation, and authorized to do business in Arizona, and it has a program to acquire substantial mineral properties in south-central Arizona wherein extensive geophysical, geological, and core drilling data has indicated a contiguous minerals zone that could exceed 300,000,000 tons of leachable copper ore. That ore has an average grade of ±0.50% copper at a 0.30% cut-off grade; however, because we advocate and recommend an in-situ leaching mining operation the estimated tonnage figures increase to 817,000,000 tons at the lower cut-off grade of 0.10% copper.

The quantity of in-place acid soluble copper could vary from optimistically 4,902,000,000 lbs to pessimistically 1,745,000,000 lbs; hence, it could easily support a 50 STPD SX-EW operation at the

lower capacity, or a 100 STPD SX-EW operation at the higher capacity assuming reasonable copper recoveries of approximately 50%.

Most important, Grover, the potential profitability aspects are excellent. For instance, if a 60 STPD SX-EW operation was implemented and assuming the probable total capital outlay by a company is less than $20 million by leveraging the loan, the anticipated DCFROI is extremely high based on our estimated production cost of $0.45 per pound of cathode copper.

We emphatically believe that the companies that will be the copper leaders of tomorrow are the companies that will find ways today to get the bottom-line to a point where domestic copper will be competitive. We have to be competitive with the countries that are governmental producers of copper, countries that have considerably higher grades of copper, and with international producers who have no environmental and minimal labor problems.

KYPROS' oxide deposit is just such an opportunity, and coupling it with an in-situ leaching operation makes it an ideal investment situation. We advocate an in-situ leaching operation as it appears to be the most cost advantageous mining method to implement, probably using hydrofracturing, as the ore is located in fracture zones and bedding planes. Yes, acid consumption and its associated cost are a major concern; however, our investigation of this element indicates that highly favorable rates are obtainable. There have been tests performed on an adjacent ore deposit with the same geological characteristics and host rock conditions, and the acid consumption results of 12 lbs of acid per pound of copper were obtained on column leach tests. We feel that this rate can be improved upon.

We are not interested in a geological program that necessitates exploration funds for several years, but an equity partner who would be interested in implementing an immediate program to develop the property and take advantage of a copper cathode product that has an estimated production cost of $0.45 per pound. That production cost could possibly be less as more data is developed for the property. This property is potentially one of the largest acid soluble copper deposits in the United States, and could be a very highly profitable situation for the right technical, managerial, and financial team.

Grover, should you or a company that you are associated with have an interest in a corporate relationship to develop this ore deposit with KYPROS, I and my people would be happy to meet with you and your staff for an indepth discussion. I look forward to affirmatively hearing from you.

Sincerely yours,

Eugene T. Daum
President

ETD:bwd

EXECUTIVE SUMMARY

KYPROS MINES INCORPORATED is incorporated under the laws of the State of Delaware, and will have its principal executive office in Scottsdale, Arizona, with a mailing address of P. O. Box 3030, Scottsdale, AZ 85257. A secondary executive office is located in Winter Haven, Florida, with a mailing address of P. O. Box 371, Winter Haven, FL 33880. The telephone number of the Florida office is (813) 665-5658. KYPROS MINES INCORPORATED was organized to pursue minerals projects, primarily related to the base and precious metals industries. Its initial objective is to pursue a hydrometallurgical project for acid soluble copper ore.

KYPROS has initiated a program to acquire an interest in substantial mineral properties located at Johnson Camp, in Cochise County, Arizona. Extensive geophysical, geological, and core drilling data has indicated that these properties contain a contiguous mineral zone that could exceed 300,000,000 tons of copper ore with an average grade of ±0.50% copper at 0.30% cut-off grade.

It is the intention of KYPROS to establish an insitu leaching operation on this property and produce a high purity cathode copper product (99.95%), utilizing solvent extraction and electrowinning, both proven technologies.

The mineralization occurs in a paleozoic carbonate rock series that has been metamorphosed by a tertiary quartz monzonite stock. The minerals found are primarily chrysocolla, malachite, and azurite. The northwestern part of the deposit, the anticipated starting point, Section 36 has a minimum of 20,000,000 tons of ore grading 0.55% copper in excess of 200 feet in thickness, with the overburden being 200 to 400 feet.

The leaching is to be designed on the basis of a series of individual leach units, each unit will contain about 3,300 total tons of copper. This copper tonnage would require an ore block of 600,000 tons at 0.55% copper. The individual units will be drilled on 30 foot centers, with an 8 inch diameter hole to the bottom of the ore section. An ore leach unit of 268 foot thickness would typically contain 40 or more blast/injection drill holes, which after fracturing will be used for leach acid distribution. One or more large diameter leach liquor collection sumps, located down dip at the bottom of each leach unit mineral zone, would also be required. The rock will be fractured in place either by a controlled blast using A.N.F.O. or hydrofractured using oil field technology. The copper minerals are predominately located in fractures and between the bedding planes. Insitu blasting or hydrofracturing the rock should cause breakage along these zones of weakness, facilitating leaching.

A conceptional design base case is for a production facility to produce a minimum of 40 tons of cathode copper per day. The leaching will be by strong acid cure followed by a continuous weak acid rinse. A conservative estimate for design is to achieve 40% recovery of the available copper after four months of leach liquor recovery. It is expected that the rate of declining recovery per month will be 27%, 6%, 4%, 3%, of the total available copper. To meet the design production rate, at these recoveries, will require that a new leach unit will be placed into production each month. The first unit will then be taken out of the primary leach cycle after four months.

Given sufficient time under leach, each unit should ultimately reach at least 70% recovery of the available copper. With only 40% of the copper extracted during the four month leach period, it is to be expected that the remaining 30% could be leached if required during a secondary leach cycle within the next 24 to 30 months. The secondary leaching of each unit was not considered in the economic evaluation of the project; however, the first stage could approach the 70% recovery initially. The mine preparation costs will decrease in proportion per pound of copper produced as the recovery increases; however, if the recovery rate in leaching exceeds the 40% recovery rate per four month period, then the tonnage under leach per unit could be reduced if desired. Each leach unit will be designed to maintain a constant 1,000 gallon per minute pumping rate. Consequently, if the recovery rate were less than 40% during the period, then the tonnage under leach could be increased to maintain the same copper productivity. With four leach units in active service at any one time, the total flow rate to solvent extraction would be 4,000 gallons per minute. The copper loading of this flow would average 1.8 grams per liter at 40% recovery per primary cycle of four leach units. It would be expected that if the recovery rate increases, then the copper concentration of the leach liquor would also increase improving the productivity above the design of 40 tons per day of cathode copper. It is anticipated that full design production should be achieved within eight months of adding the first acid to start the cure.

A 4,000 gallon per minute single train solvent extraction plant has been proven historically to be the most efficient design size. The actual copper capacity through solvent extraction can be easily increased by holding the volume constant and increasing the grams per liter loading. The ultimate capacity of the total facility would be controlled by electrowinning.

The mine preparation cost per pound of copper extracted is a direct function of the recovery by insitu leaching. We have estimated that the cost based on 40% recovery could be as high as $0.188 to as low as $0.152 per pound of recovered

copper. As the recovery increases then these costs are reduced in direct proportion.

A survey was made of existing copper leach, solvent extraction, and electrowinning plants in Arizona. The typical cash costs for acid consumption, solvent extraction, and electrowinning did not exceed $0.33 and was as low as $0.24 per pound of copper produced. Based on our cost estimates and existing plant operating costs, it would be expected then that the total cost should not exceed $0.52 and could typically be less than $0.40 per pound of copper produced.

The October selling price quoted for cathode copper is $0.75 per pound. It has recently been as low as $0.67 and as high as $1.10 per pound. If we assumed a sale price range from $0.70 to $0.90 the gross profit margin per pound could vary from a high of $0.48 to a low of $0.18. At a minimum production rate of 40 tons cathode copper per day, or 14,000 tons per year, a gross profit margin of $13,440,000 to a low of $5,040,000 per year could be expected.

The long term growth projection for copper is 3% annually. Sulfide copper; copper requiring mining, grinding, flotation, and smelting; is the major source of copper in the world. The capital requirements for a complete grass roots sulfide mine and smelter are approaching and in some cases exceeding $1,000,000,000. Many industry experts have stated that no new facility for sulfide treatment can be justified in the United States, unless the price of copper increases to the $1.25 to $1.50 per pound level based on a typical large 0.50% copper ore body. The capital cost requirements for a solvent extraction/electrowinning complex of 40 tons per day capacity would be estimated not to exceed $30,000,000 excluding land, fees, exploration, working capital and pilot leach studies.*

The significance of the potential profitability of the leach operation at Sullivan Johnson can be appreciated if copper were to return to the 1980 price of $1.10 per pound. Then a maximum gross profit margin of $0.68 per pound or $19,000,000 annually could be expected for 40 tons per day capacity. With the restrictions on production growth imposed by the current price of copper, either copper must increase to the $1.25 or higher level or acid soluble leach copper must expand to fill the void to meet the long term growth rate. The KYPROS properties at Sullivan Johnson represent possibly the largest single source acid soluble copper deposit in the United States.

KYPROS is proposing to initiate a three phase program designed to ultimately place this large, presently unexploited, resource into a viable commercial production facility. The following sequence is required to meet this objective:

PHASE I

Collect all the geological, geophysical and core drilling

* See Appendix

data and cores produced previously. Establish a central work site so that all of this data can be analyzed to determine grade variations, host rock leach characteristics and other information required to construct a prefeasibility study.

PHASE II

With a positive prefeasibility study, fund and initiate the program recommended to compile the technical, economic, and environmental information required to produce a complete feasibility study that would be acceptable to financial institutions. This work would be composed of a developmental and exploration drilling program requiring possibly 20,000 feet of drilling and an experimental pilot scale insitu leaching program. This pilot program would evaluate drilling techniques, rock fracturing characteristics, acid consumption rates for the various horizons, and plant scale tests on the generated pregnant leach liquors. The scale of this pilot leach program would be of sufficient size to directly scale up to a commercial plant. Equally important during this phase would be environmental studies as to the long term impact of insitu leaching, and the marketability of the anticipated 99.95% cathode copper product.

PHASE III

Phase III would be the actual implementation of the commercial plant involving leaching, solvent extraction, and electrowinning scaled up from the feasibility study conclusions that indicated the most economically attractive operation for KYPROS and its associates.

PROJECT EVALUATION

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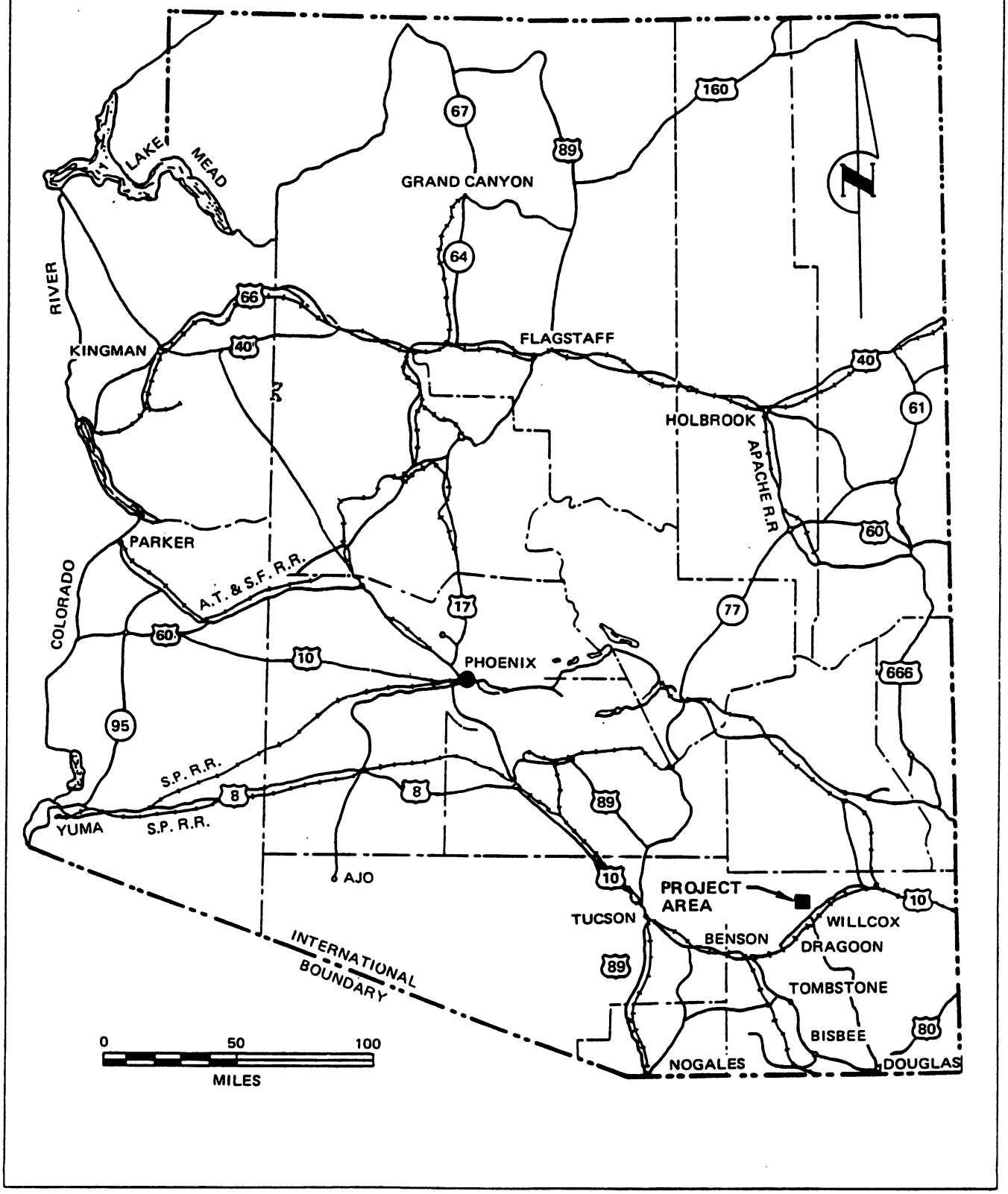
IV APPENDIX

III-2 INTRODUCTION

A. Location

KYPROS MINES INCORPORATED has initiated a program to acquire an interest in mineral properties located at Johnson Camp, Cochise County, Arizona, which will be referred to as the Sullivan Johnson Properties. The attached maps illustrate the location of this property and land holding status.

KMI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GRAND CANYON
RIVER
COLORADO
KINGMAN
PARKER
FLAGSTAFF
HOLBROOK
APACHE R.R
A.T. & S.F. R.R.
PHOENIX
S.P. R.R.
S.P. R.R.
YUMA
AJO
INTERNATIONAL BOUNDARY
PROJECT AREA
TUCSON
WILLCOX
BENSON
DRAGOON
TOMBSTONE
BISBEE
NOGALES
DOUGLAS
67
89
160
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60
10
95
8
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61
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80
0
50
100
MILES

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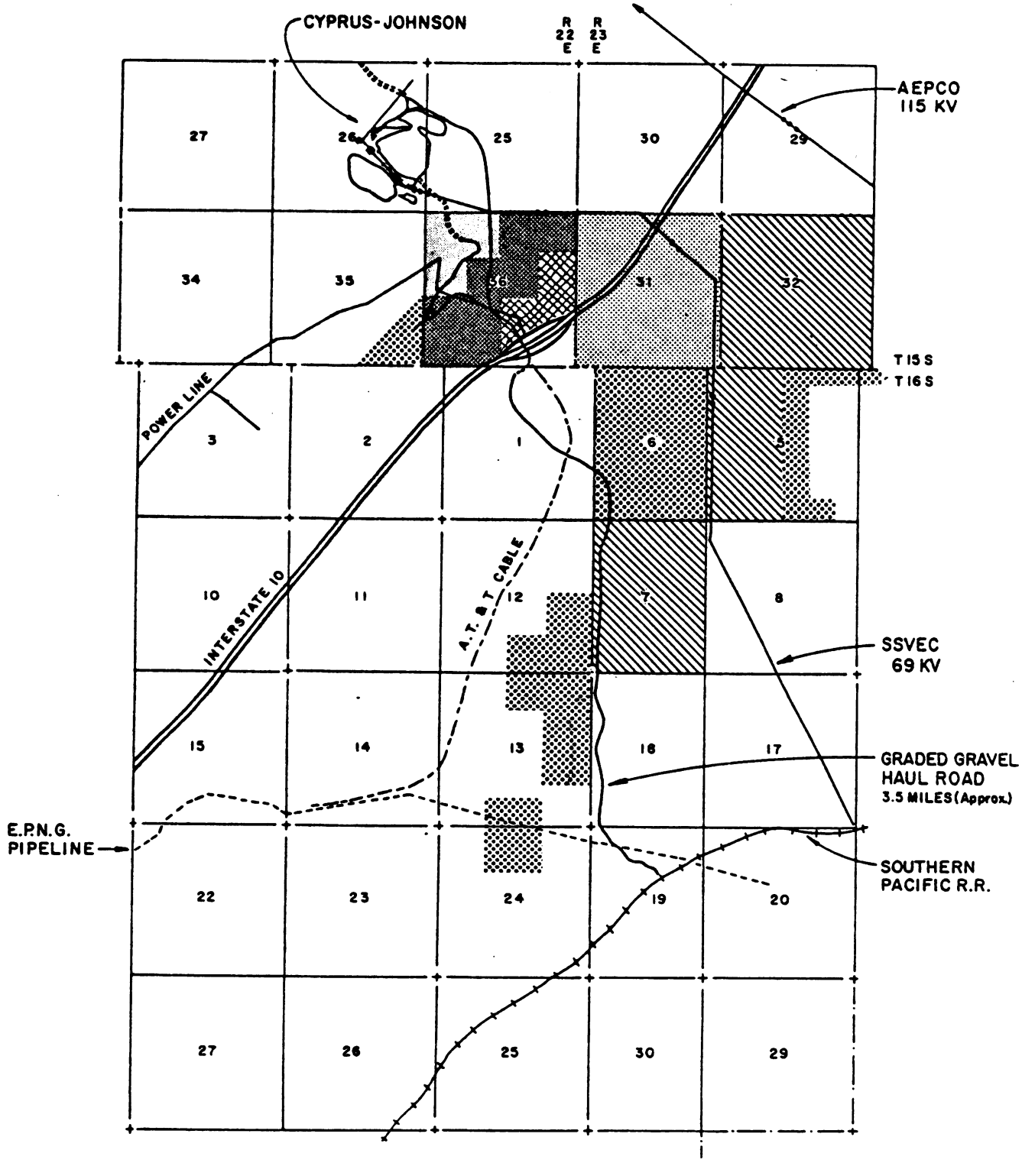


LAND STATUS MAP

- Federal Mining Claims
- State Mineral Leases
- State Prospecting Permits
- State Prospecting Permits App.
- Right to Acquire

B. District History

Mining started in the original Johnson Camp mining district, also known as the Cochise district, in the mid 1860's. Substantial production from the district commenced in the 1880's from high grade oxide mineral occurrences. Sulfide production from the district began after the turn of the century from high grade, small scattered copper zinc massive sulfide deposits. The mining of sulfides continued sporatically until the late 1950's. It was reported that 1,500,000 tons of ore valued at $25,000,000 was produced from the district.

The last sulfide producer in the district was the Coronado Copper and Zinc Company, a subsidiary of the Cyprus Mines Corp., that operated until 1958. Cyprus continued exploration in the district into the 1960's, which led to the development of the present Cyprus Johnson Copper Co. They have been operating successfully a small open pit mine, based on 22,000,000 tons of reserves, since 1975. The ore is processed by heap leaching, solvent extraction, and electrowinning. The plant capacity is 4,500 tons of cathode copper per year with the present open pit reserves expected to be depleted around 1985.

C. Reserve Potential

Copper mineralization, known as the Sullivan Johnson I-10 ore body, is buried under extensive alluvium and was not discovered until the early 1970's. Extensive rotary and diamond drilling and subsequent magnetic surveys has indicated that ±300,000,000 tons of ore averaging ±0.50% copper is present using a cut-off point of 0.30%. The most extensively drilled section in this deposit, Section 31, has an indicated reserve of 134,000,000 short tons at a grade of 0.47% copper. A drop in average grade from 0.47% to ±0.30% increases the reserve tonnage in this section to 360,000,000 tons, or 1.7 times as much contained copper at this reduced grade. If the reserves in Sections 36 and 6 were also dropped in grade to 0.30% copper, using the same ratios as used in Section 31, the tonnages could increase to 148,000,000 and 309,000,000 respectively for an overall total of ±817,000,000 tons from these three sections. We would assume that insitu leaching could be used on ore with a grade as low as 0.10% copper providing the acid consumption was not excessive.

III-3 MINERAL SOURCE

A. Geology

The geologic environment of the Cochise mining district has been thoroughly described by several authors, including Baker 1952-1953, Cooper 1957, Cooper-Silver 1964, Kantor 1975, and Clayton 1978.

The generalized cross section of the bedded sequence of the Paleozoic rocks in the Cochise district is as follows:

Tertiary and quaternary fill	0-600 feet
Horquilla Limestone-Pennsylvanian age	1600 feet
Black Prince limestone-Penn./ Mississippian age	125 feet
Escabrosa limestone-Miss. age	600 feet
Martin limestone-Devonian	200 feet
Upper Abrigo dolomite-Cambrian	60-120 feet
Middle Abrigo limestone-Cambrian	130-200 feet
Lower Abrigo limestone-Cambrian	200-300 feet
Bolsa Quartzite-Cambrian	180 feet

The underlying upper precambrian consists of diabase quartzites, conglomerates, and shale.

The host rocks were intruded by the Texas Canyon quartz monzonite stock of tertiary age (52 million years ago), which is the apparent source of the mineralization.

The upper Martin limestone, adjacent to the stock, has been metamorphosed to a tactite type rock consisting of diopside-hedenbergite and tremolite.

The upper Abrigo is composed of interbedded dolomitic sandstone and sandy dolomites. These beds were also altered to tactites by the stock and are now composed of diopside-tremolite-calcite rock.

The middle Abrigo is a crenulated limestone with intercalated lime sand and shale partings. These beds were altered to garnets, with lesser quartz and calcite. The shale sections were altered to hornfels consisting of biotite and feldspar. In some parts of the middle Abrigos both the limestone and shale units were garnetized.

The lower Abrigos consists of a gray fissile shale with intercalated carbonate and quartzite beds. The carbonates are dolomites near the base

and limestones near the top. The shale has been altered to a biotite hornfel that consists of biotite and variable potassium feldspars with lesser quartz, epidote, sphene, and chlorite. The ore for the Cyprus Johnson Camp leach operation is from this formation.

The underlying Bolsa Quartzite has not been appreciably affected by the metamorphism and helps to confine the secondary solutions from the metamorphism to the lower Abrigos.

The Texas Canyon stock is north east-south west elongated ovoid of tertiary "laramide" age that varies from a porphyritic to non-porphyritic quartz monzonite. The Paleozoic rocks tend to form a broad fold around the north east trending lobe of the stock.

The initial faulting of the Paleozoic formations occurred before the Texas Canyon stock intruded. A series of normal faults trending north east and east dipping to the north have been indicated by drilling and photo interpretation. A second series of faults referred to as the northwesters occurred either just before tilting of the Paleozoic or during the early stages of the intrusion.

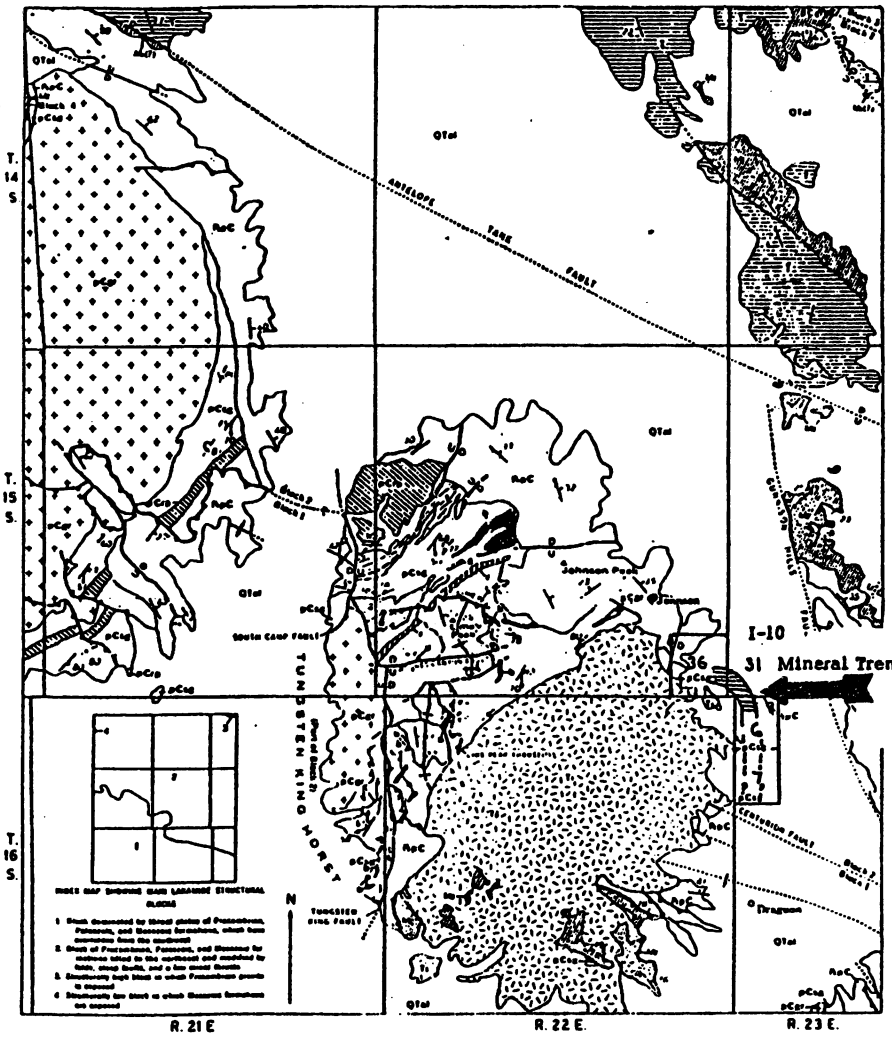
Most of these faults were pre mineral, but they did provide channel ways for mineralized solutions. The entire series was then subjected to intense faulting during the final stages of intrusion.

The Paleozoic host rock adjacent to the stock was intensely altered to the afore mentioned tactite zone by metamorphism. The garnetization provided the source rock environment for the mineralization to occur. The intense fracturing that resulted provided many receptive sites for the deposition of the primary mineralization.

Oxidation of the primary minerals has been stratigraphically controlled by the channeling of the secondary solutions down dip through the massively fractured Paleozoic rock. The impermeable underlying Bolsa quartzite restricted the fluids to the upper beds.

The mineralization of the deposit is primarily found in the Martin and upper Abrigos formation. The lower Abrigos contain a higher proportion of sulfide minerals with nonmineralized pyrite being present. The Cyprus Johnson mine is mining the ore from the lower Abrigos. The Martin and upper Abrigos have been either eroded away or have been displaced by faulting a that site.

UNITED STATES DEPARTMENT OF THE INTERIOR
GEOLOGICAL SURVEY



Compiled by J. R Cooper, 1956

GENERALIZED STRUCTURE MAP OF THE DRAGOON QUADRANGLE, ARIZONA

1 0 1 2 3 4 MILES

EXPLANATION

TERTIARY AND QUATERNARY

- Alluvium
- Sedimentary and volcanic rocks
- Quartz monzonite

MESOZOIC

- Sedimentary and volcanic rocks

PALEOZOIC

- Sedimentary rocks and diabase sills

PRECAMBRIAN

Upper Precambrian

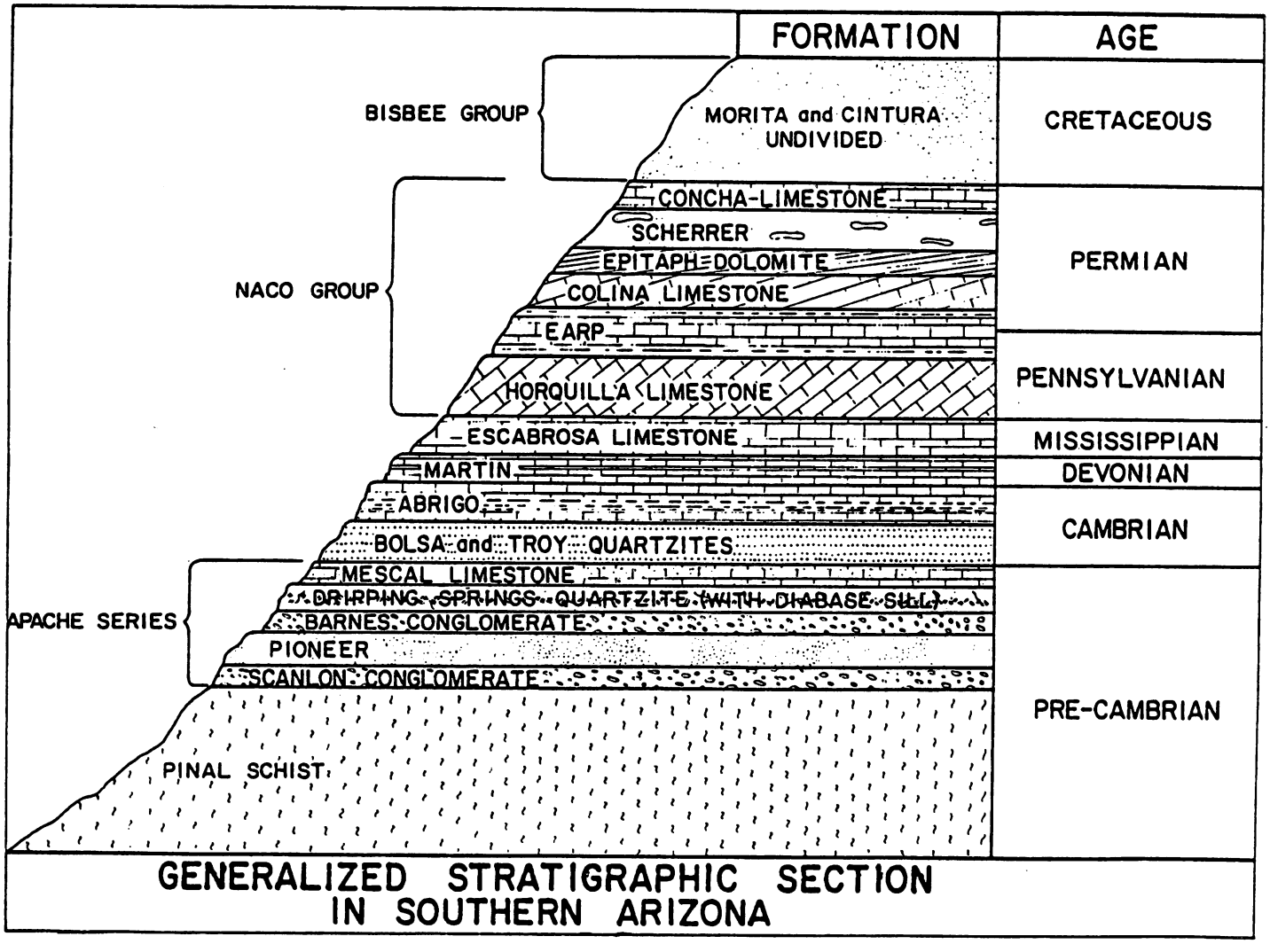
- Granodiorite and granite
- Rhyolite porphyry stocks and zones of intrusive sheets

Lower Precambrian

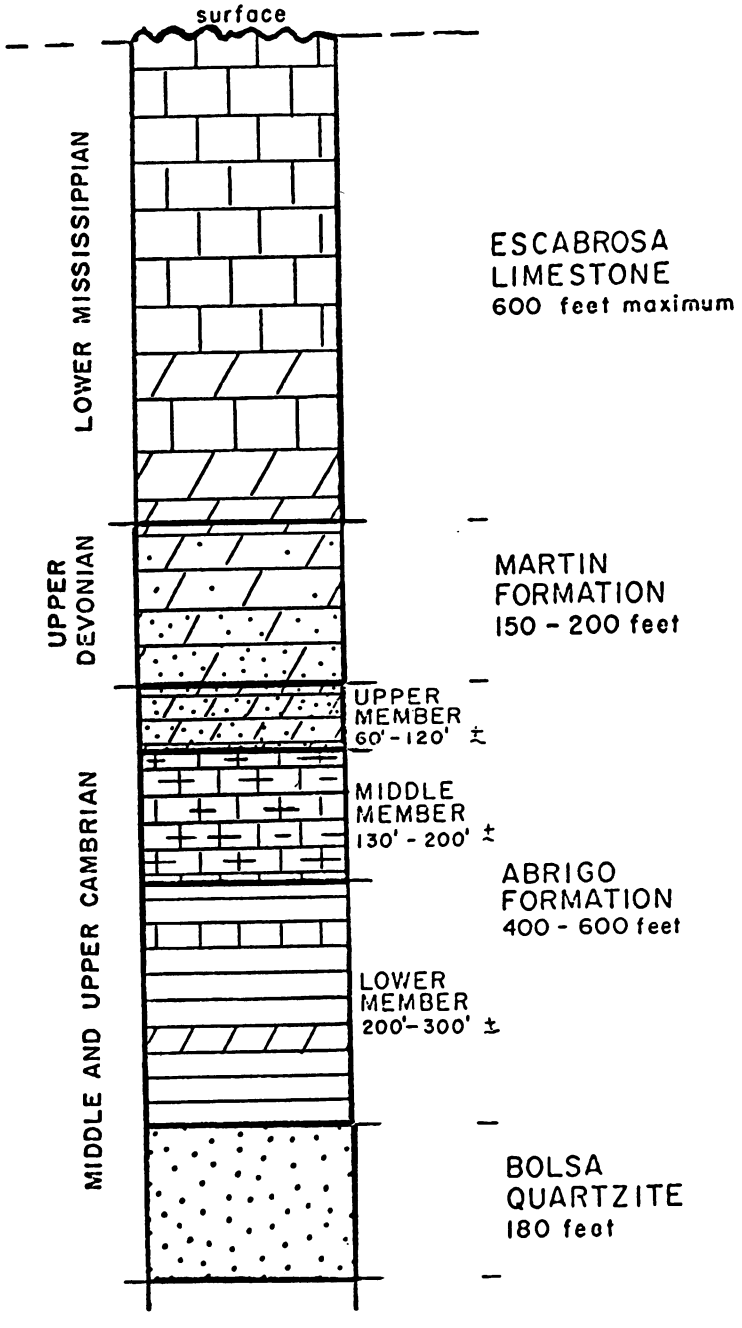
- Pinal schist

pCsg, sericite schist and metagraywacke.
s, sericite schist
g, metagraywacke
r, zone of rhyolite lava flows
b, metabasalt

- Contact
- High-angle fault
  Dashed where approximately located; dotted where concealed. U, upthrown side; D, downthrown side
- Thrust fault
  Dashed where approximately located, dotted where concealed. Sawteeth on side of upper plate. Queried where doubtful
- Syncline
  Dashed where approximately located, dotted where concealed
- Strike and dip of beds
- Strike and dip of overturned beds
- Strike and dip of schistosity and plunge of lineation
- Strike of vertical schistosity and plunge of lineation



GENERALIZED STRATIGRAPHIC SECTION
IN SOUTHERN ARIZONA



STRATIGRAPHIC SECTION

PALEOZOIC ROCKS IN THE SULLIVAN/JOHNSON I-10 PROJECT AREA

B. Mineral Reserves and Development

In the northwest part of the deposit, Section 36, KYPROS has the data and the core of four drill holes, and some information from two additional holes. All of these holes with one exception encounter copper mineralization, with a minimum of 0.40% copper for any significant length. The average intercept of the five holes that encountered copper was 0.51% with an average of ±180 feet in thickness with ±425 feet of overburden. An analysis of a magnetic survey which was undertaken after drilling was accomplished indicated that two of these holes are on the fringes of the ore body and that holes S-1 and S-4 best represent the bulk of the deposit. These two holes grade 0.56% and average 230 feet thick covered by overburden of 364 feet and could represent ±20,000,000 tons. The magnetics also indicated that substantial ore grade material should be found south of these two east-west holes, adjacent to Interstate 10, which bisects the property. A geologic projection based on thickness and structural relationships of the Paleozoic beds indicates that this northwest section could actually contain between 35,000,000 to 55,000,000 tons. The possibility of a northern extension that could increase this tonnage to past 80,000,000, at a somewhat lower grade, is also indicated.

The central part of the deposit, Section 31, has had 30 drill holes of which 16 were drilled on wide spread 500 foot centers and which indicated commercial quantities of ore. Three of these holes, CS-10, CS-7, and CS-2 running north-south, down the center of the deposit have 700 feet of overburden and average 515 feet of ore at a grade of 0.74% copper. A computer program based on the drill data had been prepared that indicated that the tonnages increase as the grade decreases at the following rate: 70,000,000 tons grading 0.58%, 134,000,000 tons grading 0.47%, and 360,000,000 tons grading 0.30%. KYPROS is in the process of acquiring all the technical data on these drill holes in this section.

South of Section 31 lies Section 6 which has had 18 drill holes. Core and other data is available on 11 of these holes that are in the mineral deposit. The average of two of these holes, which represent the best intercepts in this section, are 550 feet of overburden and 736 feet of

ORE RESERVES - 301 PROJECT - JANUARY 1973

TOTAL CU RESERVES (INCLUDES OXIDE CU)

TOTAL RESERVES

COMPUTER PROGRAM

SECTION 31 T-15-S; R-23-E

	ROCK TYPES										TOTAL	ACCUM	AVERAGE TOTAL CU	AVERAGE MOLY
	0	1	6	51	60	71	72	73	80	88	TONS	TOTAL	GRADE	GRADE
UNKNOWN	1146976.	9440.	240.	0.	0.	0.	688.	176.	240.	0.	1157760.	1646416.	.070	.004
GT 0.000 LE .050	0.	0.	30256.	8032.	0.	448.	23312.	448.	2016.	6352.	70864.	488656.	.234	.012
GT .050 LE .100	0.	0.	12944.	16416.	1440.	5088.	9904.	4976.	6016.	432.	57216.	417792.	.268	.013
GT .100 LE .150	0.	0.	0.	16304.	6304.	3536.	7520.	25872.	13248.	0.	72784.	360576.	.299	.014
GT .150 LE .200	0.	0.	0.	9456.	6096.	4128.	12560.	10032.	3680.	0.	45952.	287792.	.341	.015
GT .200 LE .250	0.	0.	0.	9136.	15056.	3200.	9312.	17296.	3664.	0.	57664.	241840.	.372	.016
GT .250 LE .300	0.	0.	0.	11424.	18320.	3488.	5152.	10336.	1680.	0.	50400.	184176.	.417	.017
GT .300 LE .350	0.	0.	0.	1824.	8816.	2720.	2112.	18944.	1504.	0.	35920.	133776.	.470	.018
GT .350 LE .400	0.	0.	0.	1152.	9776.	2896.	624.	13792.	160.	0.	28400.	97856.	.520	.021
GT .400 LE .450	0.	0.	0.	1616.	7856.	1584.	384.	3440.	0.	0.	14880.	69456.	.578	.024
GT .450 LE .500	0.	0.	0.	2512.	6240.	2160.	416.	912.	0.	0.	12240.	54576.	.619	.025
GT .500 LE .550	0.	0.	0.	1584.	1424.	2368.	656.	1120.	0.	0.	7152.	42336.	.660	.026
GT .550 LE .600	0.	0.	0.	4464.	2912.	352.	848.	528.	0.	0.	9104.	35184.	.687	.026
GT .600 LE .650	0.	0.	0.	8720.	512.	128.	160.	64.	0.	0.	9584.	26080.	.725	.024
GT .650 LE .700	0.	0.	0.	5168.	0.	0.	64.	0.	0.	0.	5232.	16496.	.780	.024
GT .700 LE .750	0.	0.	0.	2400.	112.	48.	0.	0.	0.	0.	2560.	11264.	.828	.026
GT .750 LE .800	0.	0.	0.	2256.	912.	0.	0.	0.	0.	0.	3168.	8704.	.857	.027
GT .800 LE .850	0.	0.	0.	1904.	736.	0.	0.	0.	0.	0.	2640.	5536.	.902	.031
GT .850 LE .900	0.	0.	0.	912.	0.	0.	32.	0.	0.	0.	944.	2896.	.975	.037
GT .900 LE .950	0.	0.	0.	592.	0.	0.	0.	0.	0.	0.	592.	1952.	1.025	.045
GT .950 LE 1.000	0.	0.	0.	112.	64.	0.	16.	0.	0.	0.	192.	1360.	1.070	.051
GT 1.000	0.	0.	0.	464.	192.	0.	512.	0.	0.	0.	1168.	1168.	1.084	.051
TOTALS	1146976.	9440.	43440.	106448.	86768.	32144.	74272.	107936.	32208.	6784.				

ALL TONNAGES IN THOUSANDS

COMPUTING ASSOCIATES TUCSON, ARIZONA

0.50% copper ore. These two holes where the grade is lowered to 0.40% copper have an average of 1010 feet of copper mineralization. This represents a remarkable thickness of ore. It should be further noted that these two holes also have a composite high grade copper section of 0.77% averaging 220 feet in thickness. The geologic inference of the tonnage of reserves in this section would be in excess of 50,000,000 tons grading plus 0.50% copper with a possible tonnage of 120,000,000 tons at some reduced grade in the 0.40% range, and 309,000,000 tons at an 0.30% grade if the same proportions used in the Section 31 study were applied.

A summary of these three sections indicates that the mining technique applicable to exploit these reserves will have a major factor as to what is considered as an economic lower grade cut-off. An evaluation was made comparing open pit, underground, and insitu leaching.

An analysis was made of the data from holes S-1 and S-4 in the northwest part of the deposit that indicated the minimum amount of overburden to determine if this area could be mined by open pit methods. The drill hole locations and their proximity to Interstate 10 made open pit mining highly improbable. Assuming that an open pit would require a minimum of a 1:1 pit slope, if the pit started right at the highway the bottom of the pit would be north of the east-west line of drill holes, leaving all the good ore indicated south of these holes unminable. The waste to ore ratio from an open pit in this area would be in excess of 4.5 to 1.

A similar open pit mining analysis was made on Section 6. The result of this study was the same in that using T-2 as typical, a total pit depth of 1200 feet would be required. The waste to ore ratio would be 2.9 to 1. The configuration of the indicated ore body and the drop in grade as the Paleozoic rock dipped off to the east however, made open pit mining seem most impractical.

With open pit mining evaluated, the next consideration was to study the least expensive form of underground mining. In terms of tons moved, block caving is the cheapest form of underground mining. The rock did not appear to be of a type that would cave so sublevel stoping was next considered. Recent data has indicated that a large mine moving about 7,000,000 tons per year would cost about $4.25 per ton. With rock averaging 10 lb. copper/ton with a maximum of 7 pounds recoverable, the cost per pound would exceed $0.60 per pound. This eliminated underground

mining at current copper prices.

After careful review of the other alternatives it became apparent that insitu leaching was the only practical way to exploit this very large resource economically. Insitu represented a method to maximize the recovery of copper at the least possible cost. It also was the most environmentally acceptable since no massive excavations would be made, surface pollution would be minimized, and no unsightly waste dumps would be left. From a commercial standpoint, insitu leaching is not capital intensive in the mining sector. Depending upon the reserves available, the production cost for a small plant per pound of copper produced would be similar to a large plant. The cost per ton of ore placed under leach is the same regardless of scale. This greatly increases the flexibility of a mine.

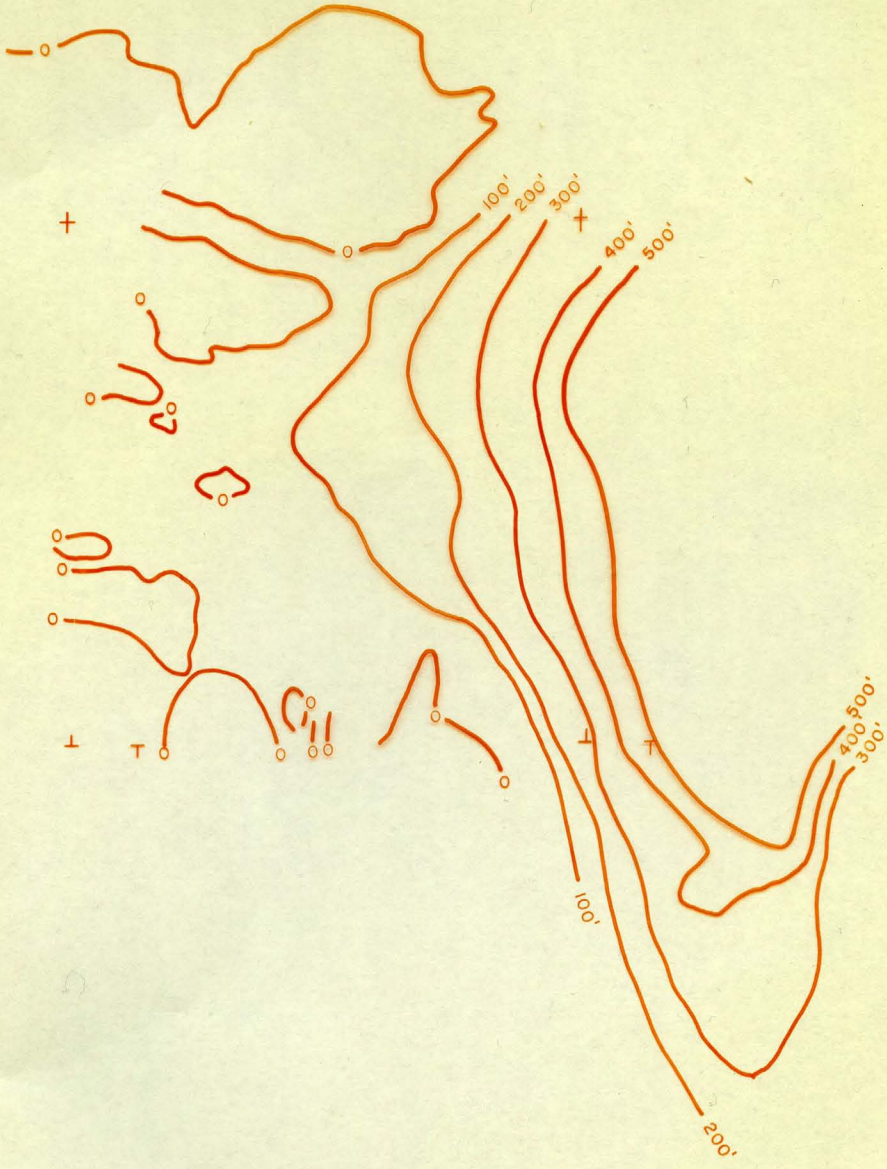
SULLIVAN JOHNSON PROPERTIES

BASED ON 0.30% CUT-OFF

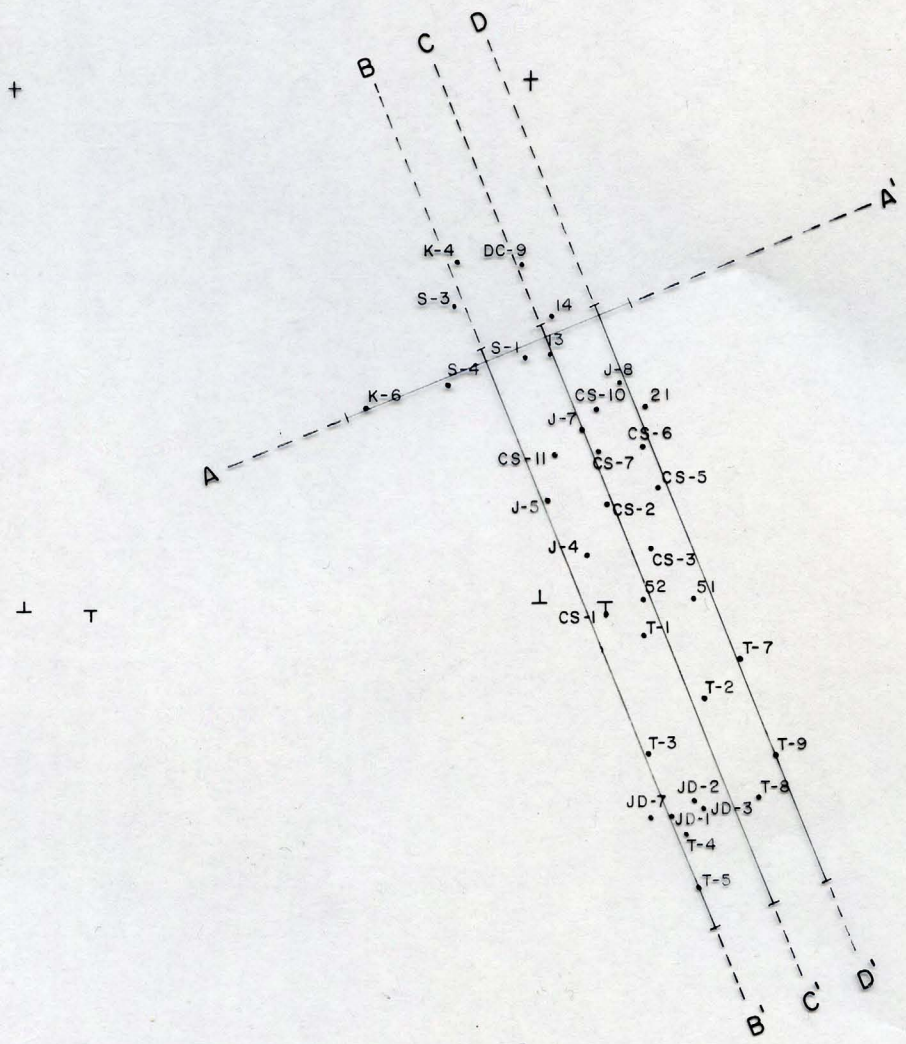
DATA SUMMARY

I	LOCATION	Johnson Camp Mining District Sections 36, 31, and 6 Cochise County, Arizona		
II	POSSIBLE QUANTITIES OF ACID SOLUBLE COPPER ORE RESERVES IN SHORT TONS	Section 31 -	0.47%	134,000,000 ST
		Section 36 -	0.56%	55,000,000 St
		Section 6 -	0.53%	115,000,000 ST
			TOTAL	304,000,000 ST
III	POSSIBLE TOTAL QUANTITIES OF ACID SOLUBLE COPPER METAL RESERVES IN POUNDS	Section 31 - 134,000,000 @ 0.47% = 1,259,000,000 lbs.		
		Section 36 - 55,000,000 @ 0.56% = 616,000,000 lbs.		
		Section 6 - 115,000,000 @ 0.53% = 1,219,000,000 lbs.		
			TOTAL	3,094,000,000 lbs.
IV	MINIMUM PROBABLE QUANTITIES OF LEACHABLE COPPER IF INSITU LEACHING MINING METHOD IS USED.	Section 31 - 82,000,000 @ 0.55% = 902,000,000 lbs.		
		Section 36 - 20,000,000 @ 0.51% = 204,000,000 lbs.		
		Section 6 - 71,000,000 @ 0.45% = 639,000,000 lbs.		
			TOTAL	1,745,000,000 lbs.

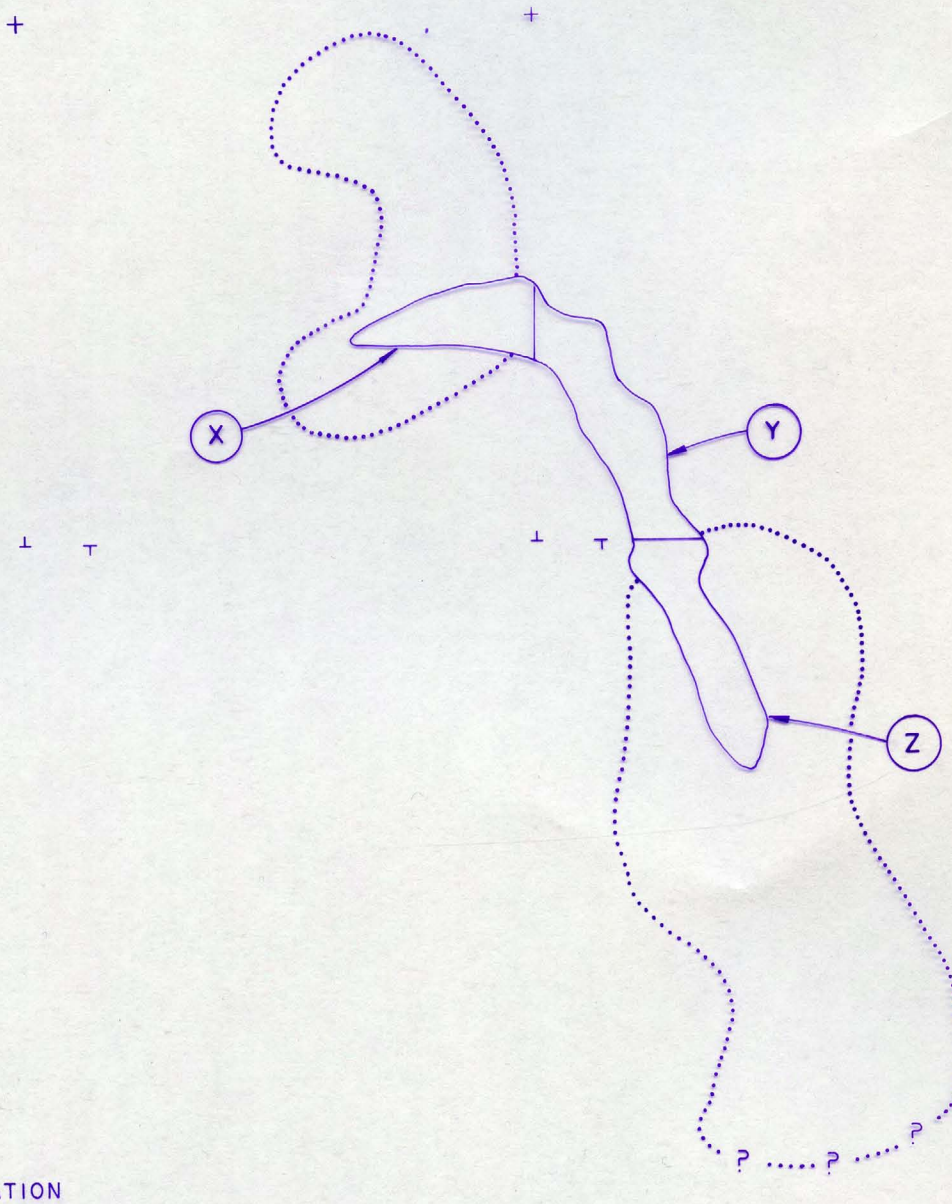
Notes
* Items I, II, III are from G. S. A. letter dated January 21, 1982
* Data for Item IV is from recently obtained and KYPROS interpolated information
* All capacities are in-place values



THICKNESS OF VALLEY FILL

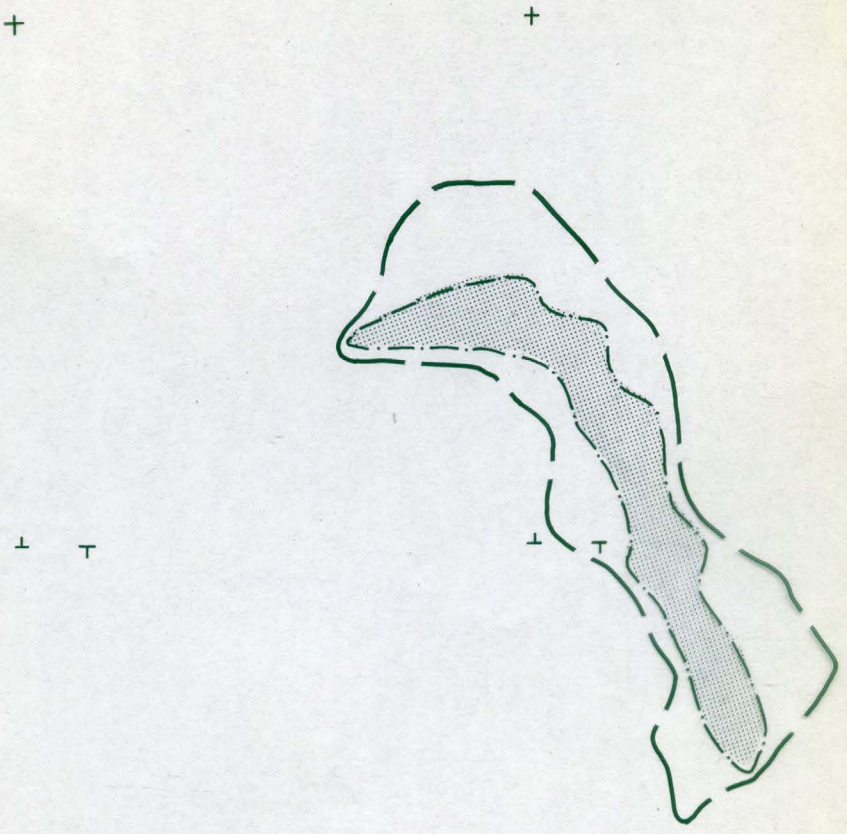


CROSS SECTION LOCATIONS



PROPOSED EXPLORATION
TARGET AREAS TO DETERMINE
THE POSSIBLE EXTENSION
OF EXISTING MINERALIZATION

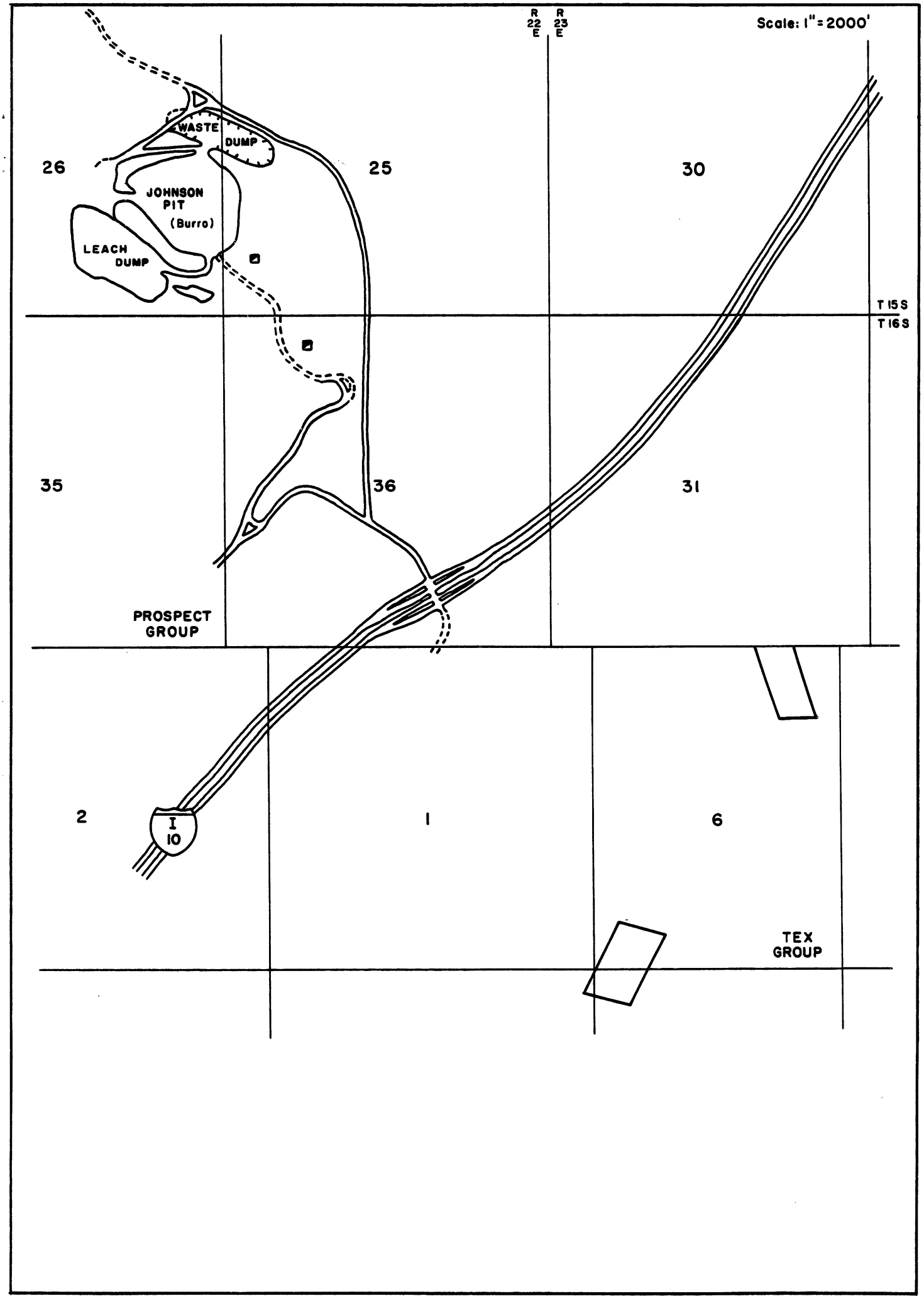
(X) SECTION 36 $+20 \times 10^6$ TONS AT .51% Cu
(Y) SECTION 31 $+82 \times 10^6$ TONS AT .55% Cu
(Z) SECTION 6 $+71 \times 10^6$ TONS AT .45% Cu



APPROXIMATE OUTLINE
OF MINERALIZATION

CONTAINING A
POTENTIAL RESOURCE OF 300×10^6
TONS OF COPPER MINERALIZATION
GREATER THAN .30%

HIGH GRADE LEACH ZONE AREA
INFERRED 173×10^6 TONS AT .51 Cu


R 22 E
R 23 E
Scale: 1" = 2000'
WASTE DUMP
26
25
30
JOHNSON PIT (Burro)
LEACH DUMP
T 15 S
T 16 S
35
36
31
PROSPECT GROUP
2
I 10
1
6
TEX GROUP

III-4 INSITU LEACH DEVELOPMENT

A. Leaching Concept

Insitu leaching is a process in which an acid such as sulfuric is allowed to perculate through rock, dissolving minerals to be recovered. The resultant pregnant leach liquor then migrates to a central collection point, from which it is pumped to a processing plant. After the mineral is extracted, the resultant regenerated acid or raffinate is then supplemented with additional acid to compensate for the acid consumed by the host rock and the whole process is repeated.

Before insitu leaching can be conducted, several variables need to be evaluated. The primary concern is to determine if the mineral will be dissolved preferentially to the host rock. The unaltered paleozoic rock at Johnson Camp is primarily composed of carbonates which in their natural state are high acid consumers. The rock adjacent to the Texas Canyon stock has been highly metamorphosed. The tactite or skarn zone has been subjected to high heat and pressure. The heat has been sufficient to physically evolve the CO_2 of the host carbonate rock in places. This process has been referred to as garnetization and has created a substantially different suite of minerals. In general the resultant rock has a much lower reaction to sulfuric acid. The greater the distance from the intrusion the higher the CO_2 content becomes and the higher the acid consumption. It is assumed that not all copper bearing rock from this deposit is leachable by sulfuric acid.

The analyses made previously have indicated that the principal copper minerals chrysocolla, malachite and azurite are concentrated within fracture zones and along bedding planes of the host rock. This is an advantageous situation since acid contact needs to be only in these areas rather than throughout the rock as would be required if the rock had a porphyritic texture. It has been estimated previously that crushed rock would have about five times the surface area as rock left insitu. With an unknown quantity of acid consuming constitutes still remaining in the garnetized paleozoic host rock, by maintaining minimum surface areas, acid consumption would be expected to be significantly less. This rock when crushed could have a much higher acid consumption than when left intact. Any leaching tests on this rock while crushed from core samples would give misleading results with possibly five times as much acid consumption

when conventional leaching is considered, and should only be used to show relative reactivity.

The highly fractured host rock would have a relatively high degree of porosity because of CO_2 evolution. It may be necessary, however, to apply pressure on the rock to refracture the mineral cemented joints and cracks to allow the migration of the acid leach liquor. The energy to create this pressure can be provided by a slow controlled explosion or by a high energy hydrofracing process using a non acid consuming medium and 60 - 80 mesh silica to help retain the porosity. A large blast would be counter productive since it would expose fresh non-mineralized surfaces that could increase acid consumption. The copper minerals were originally deposited from solutions, or gases migrating along conduits created by fractures and faults. The leaching process is simply to allow soluble copper dissolving acids to migrate down dip along these same channelways to a central collection sump for eventual recovery.

The process of leaching copper minerals has changed significantly over the past 5 to 8 years. Historically an acid of 3 to 8 grams per liter is sprinkled on a "run of mine" heap or dump and allowed to perculate through to an impervious layer at which point it is collected, processed, and recycled. Recoveries of copper in many cases never exceeded 50% in several years of this type of leaching. In the middle 1970's the leaching technique had evolved to a much more sophisticated technology called pad leaching. The scientific approach applied called for sized crushed rock placed on pads of uniform level thicknesses in which high strength acid, 20 - 30 grams per liter, were applied to initiate the chemical reaction, processed and recycled. After the extraction reached a steady state the acid strength of the leach liquor was reduced to below 10 grams per liter. When the actual recoveries had deminished, usually in 3 to 9 months, the leaching was stopped and a fresh pad was placed on top of the original pad and the process was repeated. The underlying pad would continue to have copper extracted, but at a much slower rate and be cumulative with the new pad pregnant leach liquor. This process is repeated several times.

The pad leaching process greatly accelerated hydrometallurgical copper recovery, which helped reduce acid consumption by host rock reactions. A very significant improvement was made to the pad leaching process in the late 1970's that further significantly reduced acid consumption and further accelerated copper extraction. This process called for a high strength acid, 100 to 200 grams per liter, to be placed on an ore to saturate

it and be allowed to "cook" for several weeks. After a two to three week period a weak acid rinse is added, 8 to 10 grams per liter, which displaces the cooked solution which is then processed and recycled as a weak acid. This process has been found at other mines to leach in excess of 70% of the contained copper values within four months of extracting the initial leach liquor. The cycle is stopped when the grams per liter of the leach liquor drops below some predetermined concentration. Operational results have indicated that this process has reduced acid consumption from pad leaching by over 60%.

Insitu leaching of Johnson Camp rock by this latest process appears to be potentially an extremely viable technique. The high strength acid when injected into the previously fractured rock should rapidly react with the copper bearing minerals in the voids. The acid would essentially be consumed by the copper bearing minerals and subsequently be displaced by the fresh weak acid solution. The weak acid rinse would displace and dilute the acid cure causing it to migrate to the collection point. A potentially important phenomenon for speeding up the chemical reaction would be the exothermic reaction created by contacting the strong acid by the weak acid or water which would create a driving force into the mineralized fractures.

III-4 INSITU LEACH DEVELOPMENT

B. Operational Criteria

No definitive design can be made for an insitu leach system until pilot scale test work is conducted. The following conceptional system is suggested based only on what limited information is available at the present time. Many broad assumptions have been made that may or may not be verified by actual test results.

The most practical size capacity for a single train solvent extraction plant is 4,000 gallons per minute (gpm) aqueous feed. This size plant would be capable of extracting copper from solution in excess of 40 short tons per day of cathode copper. The tonnage capacity is a function of the grams per liter copper concentration of this aqueous flow.

Using the 4,000 gpm as a design criteria of pregnant liquor and 40 short tons per day cathode copper production, the mine design can be back calculated using an assumed recovery rate for acid cure leaching. Previous experience with the acid cure at an operating mine has indicated that an extraction rate in excess of 70% of the available copper, mixed oxide-sulfide, in about four months can be achieved. If we assumed, however, that only 40% can be extracted in four months, then a per month extraction rate curve can be established based on typical copper leaching decreasing recovery rate curves. The resulting recoveries based on four months would indicate 27% the first month, 6% the second, 4% the third, and 3% the fourth month.

The sequence is best described by the table below:

MONTHS	1st	2nd	3rd	4th	5th	6th
New Tons/Month	Recoveries					
600,000	27	6	4	3	--	--
600,000	--	27	6	4	3	--
600,000	--	--	27	6	4	3
600,000	--	--	--	27	6	4
600,000	--	--	--	--	27	6
600,000	--	--	--	--	--	27
	27	33	37	40	40	40

With a four month primary life cycle per leach unit, it would be assumed that four units need to be under leach at any one time. With four units being leached then each one would be designed for a 1000 gallon

per minute leach liquor flow rate. A 40 short ton per day production rate would require that each unit needs to produce an average of 10 short tons per day per unit for four months. or 1200 tons of copper. For a leach unit to produce 1200 tons at 40% recovery would require that 1333 tons of copper be extracted at 90% recovery in solvent extraction. This would require that a leach unit contain at least 3300 tons of metal copper.

Until additional information is obtained, we have to assume that Section 36, the northwest part of the deposit, is the best starting point to base a mine design upon. Two drill holes, S-1 and S-4, have an indicated grade of slightly higher than 0.55% copper. For a unit to contain 3300 tons at 0.55% would require 600,000 tons of ore to be under leach at any one time. Drill hole S-1 has a thickness of 268 feet of ore. Assuming a bulk density of 146 pounds per cubic foot the horizontal surface area would be about 31,000 square feet, or a circle with a 100 foot radius.

The area under leach must be prepared so that the insitu rock be cracked or fractured in such a manner that the unfractured rock surrounding it will act as a barrier restricting the migration of solutions to within the area under leach. Until more detailed information is obtained, we would assume that a central collection sump be provided in the approximate center of the leach unit. The pregnant liquor will flow by gravity or by the drawdown created by pumping similar to a well field, to this sump.

Hydrofracing and blasting studies have indicated that the area of influence around each hole would be about 15 feet. This would indicate that a drill hole every 30 feet would be sufficient to break the rock adequately. With a 30 foot spacing in a 31,000 square foot area, up to 44 holes could be required per leach unit. Of the 44 holes, 43 would be used to distribute the 1000 gpm leach liquor uniformly over the leach area and one hole would be used as the collection sump for the pregnant liquor. Each hole will need to be capable of receiving 23 to 25 gallons per minute plus the proportional percentage of leach solution lost. The size of the drill hole would be dependent on how the rock was to be refractured. If conventional blasting were required the hole size may need to be as large as 8 inches in the mineral or ore section. To prevent valuable acid loss in the overburden material, a plastic pipe would need to be inserted in the holes to the bottom of the overburden and above the broken rock area. This pipe would not need to be any larger than 2 inches to handle the flow.

The cost of drilling an 8 inch hole 677 feet should be in the

range of $8.00 or less per foot if production scale mine owned equipment were used. It would be expected that if a 5 or 6 inch hole would be sufficient, then the cost per foot could be reduced to $5.00 or less per foot. The central hole that would be used as the sump would have to be cased with an 18 inch pipe to accommodate a multistage turbine pump capable of elevating 1000 gpm against about 1000 feet of head. This would require about 400 brake horsepower for each leach unit. The center hole would need to be re-excavated after blasting to accommodate the casing.

At a cost of $8.00 per foot for 43 holes, a 677 foot deep mineral deposit would cost $233,000 per leach unit for the blast/leach solution holes. The central sump would need to be a 22 inch hole which would cost about $30.00 per foot for 677 feet plus redrilling the 268 foot mineral zone for a total of 945 feet at a cost of $30,000. The casing cost for the central sump at $30.00 per foot would be about $20,000. The leach hole piping would not be expected to cost more than $1.00 per foot, or an additional $20,000. The pregnant liquor pump plus piping to the surface would not exceed $20,000.

The total cost for the drilling of 8 inch holes and casing a 409 foot overburden, 268 foot mineral zone would be ±$325,000 at an $8.00 per foot drilling cost. If the hole size were reduced to 5 inches, this cost could be reduced to $240,000. (Again it should be noted that this drill cost example is with the use of inhouse drilling equipment.)

The rock breaking cost will depend upon the method used. If A.N.F.O. were used, the loading should not exceed 20 pounds per foot of mineral (roughly 0.4 pounds per ton of ore if sump breaking included) or 240,000 pounds of explosives. At a delivered price of $8.00 per 100 pounds the explosive cost would be about $20,000. The labor, detonators and cord would cost about $10,000 for a total rock breaking cost of $30,000. This number could be appreciably higher if the rock fracturing requirements were for a more exotic type explosive. The economics could indicate that a smaller or greater spacing between holes loaded with a much higher energy explosive would be more cost effective.

The total cost of preparing a leach unit for production based on the above would be about $355,000 if the large 8 inch hole was required or $270,000if the smaller 5 to 6 inch hole were adequate. With a calculated recovery of 1200 tons of copper, the mine preparation cost would be $0.148 per pound of copper for an 8 inch hole and $0.112 for a 5 inch

hole. The effect of leaching recovery on these costs can be seen when a 50% recovery was achieved (1500 tons), then these costs would drop to $0.118 and $0.09 respectively. The most optimistic example that could be considered that would actually be achievable would be an overall 70% recovery and $3.00 per foot drilling cost. In this case, the per pound of copper cost would be reduced to $0.05 per pound ($210,000 ÷ 4,160,000 pounds).

The operating costs to produce from a leach unit would be $25,000 in labor and $75,000 in power over a four month period. At a 40% recovery this would be $0.04 per pound of copper. At 50% recovery it would drop to $0.033 per pound. When a total recovery of 70% is considered using primary and secondary leaching this cost per pound could actually increase since the total recovery would be achieved in substantially longer than four months increasing pump operating costs.

The effect of the above analysis indicates that the total mining development and operating cost for a single leach unit should not exceed $0.188 per pound of copper produced at 40% recovery and could be as low as $0.13 at 50% recovery. If 50% recovery were considered at the potential low drilling price then this total cost would be further reduced to $0.11 per pound. The drilling cost has a major effect on the thickness of the ore body or the overall depth. If, for example, a 600 foot section were leached the actual drill holes required would be significantly reduced.

III-5 PRODUCTION FACILITIES

A. Solvent Extraction

KYPROS plans to include a solvent extraction (SX) plant, together with all the necessary associated facilities, on the Sullivan Johnson properties in Arizona to process the pregnant liquor from the leach area.

The use of solvent extraction for the recovery of non-ferrous base metals, particularly copper, has increased rapidly during the past eighteen years. This is due mainly to the development of selective extraction reagents, which in the case of copper will extract the copper from the aqueous solution leaving the stripped liquor to be returned to the leach area.

The success of such early installations as Ranchers Exploration and Development Company at their Bluebird Mine, and Cyprus' operations at the Bagdad Mine significantly contributed to the confidence and success of processing copper via the solvent extraction-electrowinning method.

The solvent extraction system will be designed to handle a pregnant liquor solution from the insitu leaching mining area, which would assume to be 4000 gpm aqueous flow at an average loading of 1.8 grams per liter of copper for a 40 STPD plant. The copper extracted from the leach liquor by the SX plant will be transferred to a tankhouse in the form of an upgraded solution as the electrolyte. The copper is then recovered in the tankhouse as electrowon cathode copper at 99.95% quality.

The SX plant design will be based on well proven operating experience and existing technology. The SX plant will be complete with all necessary connections and tie-ins, and will include two extraction mixer-settlers, two stripping mixer-settlers, solution and reagent handling equipment, a raffinate pond, a sludge treatment circuit, a fire protection system, and a control room. The critical materials of construction would be defined during the Phase III, Project Implementation, facet of the project.

The proposed plant design will allow for future expansion to increase capacity, because flexibility in the plant design will be recommended and endorsed by KYPROS.

III-5 PRODUCTION FACILITIES

B. Electrowinning

The tankhouse will be designed to efficiently handle the electrolyte, copper bearing solution from the Solvent Extraction Plant, at a production rate that balances the capacity of this and the preceeding plants.

KYPROS' initial intent would be to build and operate an electrowinning facility with a current density of 20 to 23 amperes per square foot. Careful consideration will be given to optimizing the efficiency of the electrical circuit by preferential arrangement of the cells and its electrical tie to the transformer circuit.

Electrolytic circulation, ventilation, starting sheet production, minimum bus bars runs, crane sizes, and numerous other problems will be addressed during the Phase III part of the project. There are several engineering firms that have excellent qualifications to design and build an Electrowinning Tankhouse; therefore, KYPROS will utilize its inhouse talents and that of a qualified company to design this facility.

III-5 PRODUCTION FACILITIES

C. Infrastructure

The Sullivan Johnson ore body is ideally located to all the critical infrastructure items required to produce electrowon cathode copper at this site. Critical items such as water, power, rail, federal highways, etc., require minimal capital expenditure as all of these items are accessible.

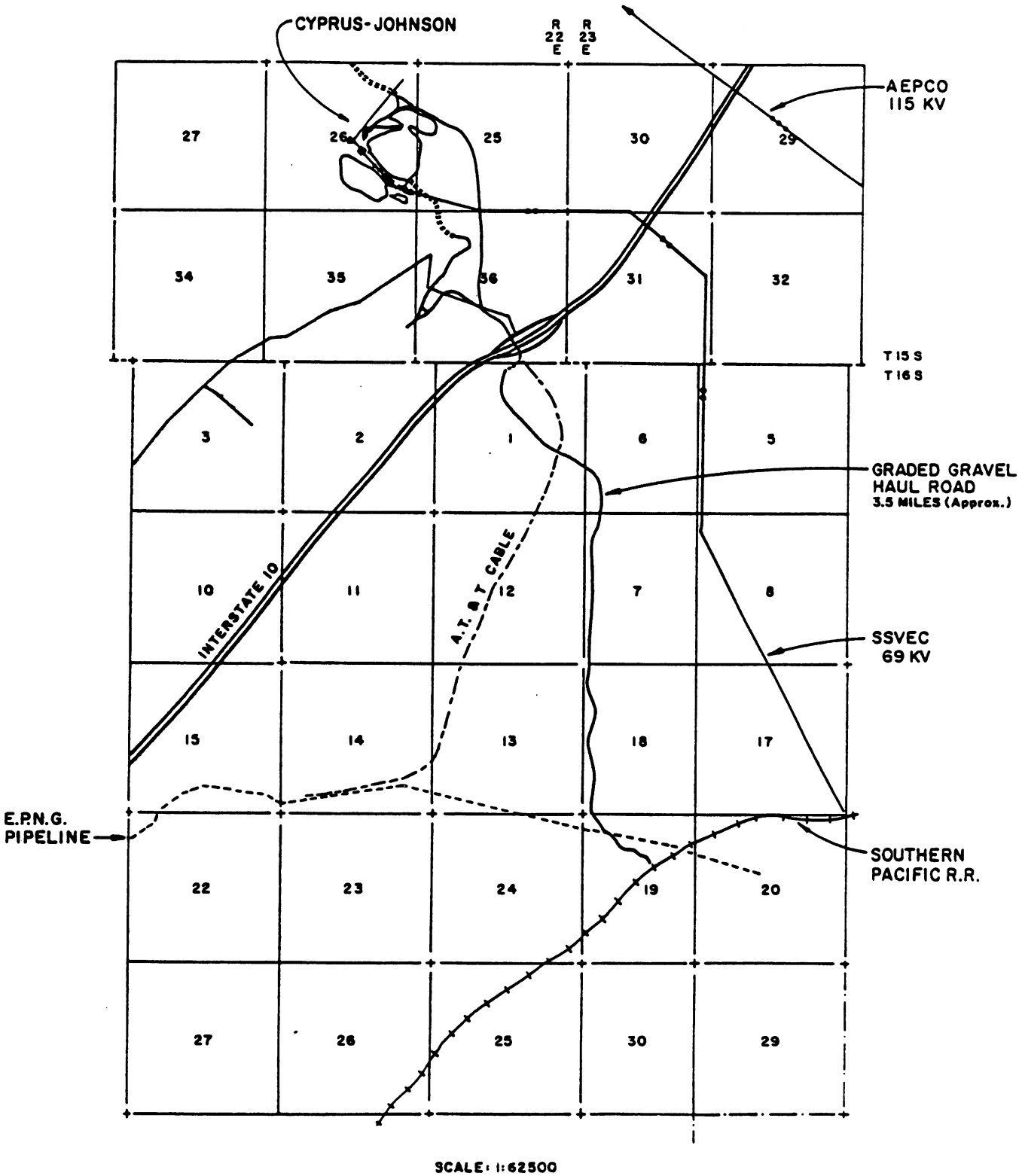
Southern Pacific Railroad Co. has a line complete with a siding within three miles from the center of the ore body. Federal highway Interstate I-10 intersects the property through Sections 36 and 31, making it easily accessible for plant personnel to commute from nearby towns and to obtain required supplies from terminal points such as Tucson and Phoenix.

There is a 69 KV line that runs along the east and northerly boundries of the property, with a 115 KV line within two miles from the property.

Water is available for a similar type operation a few miles away. Future hydrology studies will identify the most economical and practical water source that KYPROS would use or recommend for the selected operation.

A dirt haul road to the railroad siding at Dragoon, Arizona, is available if it is required for an access road. A mining camp facility would not be required as two adequate sized towns are within twenty miles of the deposit; hence, experienced personnel to operate the mine is also readily available.

Pertinent infrastructure is shown on the attached map.



INFRASTRUCTURE OF THE JOHNSON CAMP AREA

III-6 DEVELOPMENT PROGRAM

A. Phase I - Prefeasibility Study

Currently the cores and drill cuttings from 54 holes drilled on the properties are stored at various locations in southeastern Arizona. These holes have been logged and analyzed by different geologists with a variety of objectives, which influenced the manner in which they were logged. Collectively, these holes represent corporate and individual investments that would range from $4,000,000 to $6,000,000 in replacement value over the past 12 years.

KYPROS' objective is to implement a program of collection of these cores and to standardize the information from them such that an intelligent appraisal can be made of their characteristics. Various geological consultants familiar with the previous work would be retained to help in this process of standardization. All previously published and corporate geological reports will be collected and incorporated with new raw geological data obtained from previously generated drill cores or from any new cores to establish a comprehensive report.

The thrust of the evaluation of these cores will be towards obtaining the information received to properly evaluate the viability of insitu leaching. The information gained from these cores and the other geologic data will be used to evaluate host rock acid reaction variations, drillability of the rock, explosive reactivity, amenability to oil field fracing technology, plus hydrological and environmental considerations. Consultants will be utilized where applicable to evaluate these various components. A major impact will be made during the collection of the original information and its evaluations by the management and directors of KYPROS, who have collectively approximately 100 years of broad based successful experience in exploration, process development, project management, and operations of mineral projects.

The second phase program will be designed based on the results and conclusions formed during this first stage program. KYPROS' intent is to be totally objective in its evaluation of the data generated, and that the decision to advance to Phase Two will be made because the deposit warrants further investigation and that technological capability is available to resolve any unproven component required for mine development success.

B. Phase II - Feasibility Study

Currently, the indications are that additional diamond drilling will be required on the properties to fully define the ore body. Of the 54 holes drilled, 33 have encountered ore as defined by others. With insitu leaching the possibility remains that this classification could change.

The geological intrepretation of the previous work would indicate that an additional 20 holes would be required to block out, as proven ore, over 200,000,000 short tons in the three sections; however, the geological interpretation could indicate an additional reserve as high as 600,000,000 short tons or more of lower grade ore. This would be sufficient to justify an adequate size insitu leach, solvent extraction, and electrowinning facility. The indications are that substantial additional tonnages could be blocked out if economics justify a larger plant than now anticipated. Interpretation of any new core will be influenced by the requirements for an insitu leaching type operation, and information that will be obtained from these cores will be useful in designing a leaching system.

In addition to the diamond drilling, a pilot test leach unit will be designed and operated. The purpose in these tests will be to determine production scale drilling costs, rock breaking requirements, and drill hole spacing requirements. Perculation or migration rates of the leach liquor will be evaluated based on the optimum design of the rock breaking and hole spacing characteristics. Various configurations and patterns will be investigated to optimize leach solution recovery and minimize solution losses via faults, etc.

The acid consumption of the host rock under test will be a most important part of the development of the economics of insitu leaching. The acid cure approach has been proven to be a significant reducer of acid consumption on pad leaching tests. It is a key element in the establishment of a viable insitu leaching process, because of the reduced exposure time and increased copper solubilization rates. If the cure proves in practice that it can have the same relative effect on metamorphosed carbonate rock as it has shown at other copper leaching operations, then the viability of an insitu leach operation is greatly enhanced.

The solvent extraction and electrowinning phases of the project are both well proven techniques, and will not require the construction of a pilot facility. It is expected that some arrangement can be made with an operating SW-EW facility that can process the leach liquor generated by the leach liquor program, which already has all the required components. The essential item to be determined for the required solvent extraction plant design would be the grams per liter of copper resulting from the insitu leaching process.

III-6 DEVELOPMENT PROGRAM

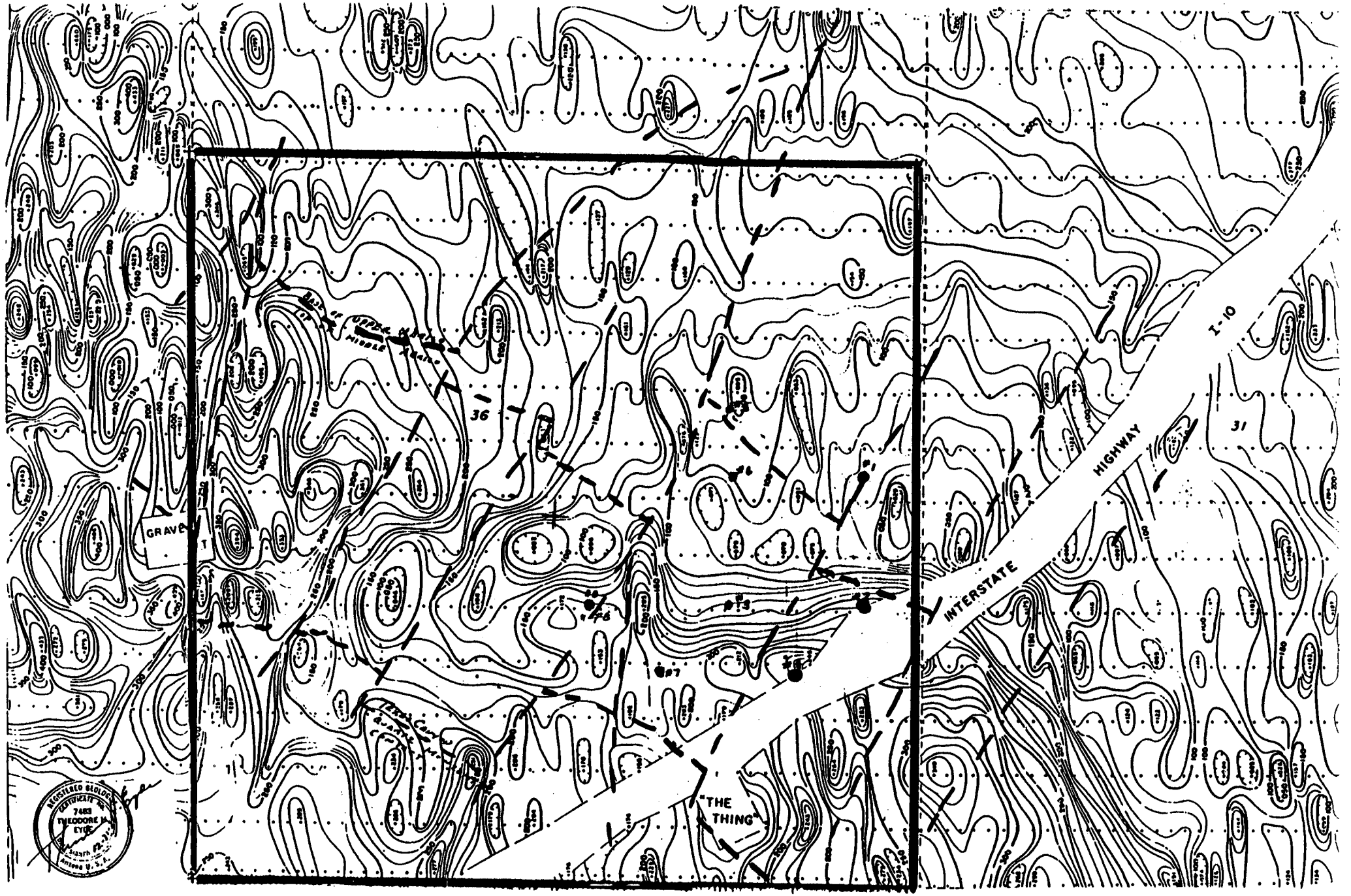
C. Project Implementation

After KYPROS and its associates or its joint venturers have completed the feasibility study and have reached an affirmative decision to proceed, the Phase III, Project Implementation phase, will be initiated. During the feasibility study, the necessary leaching criteria will be established and an optimum plant design for a given capacity will be arrived at for a production facility.

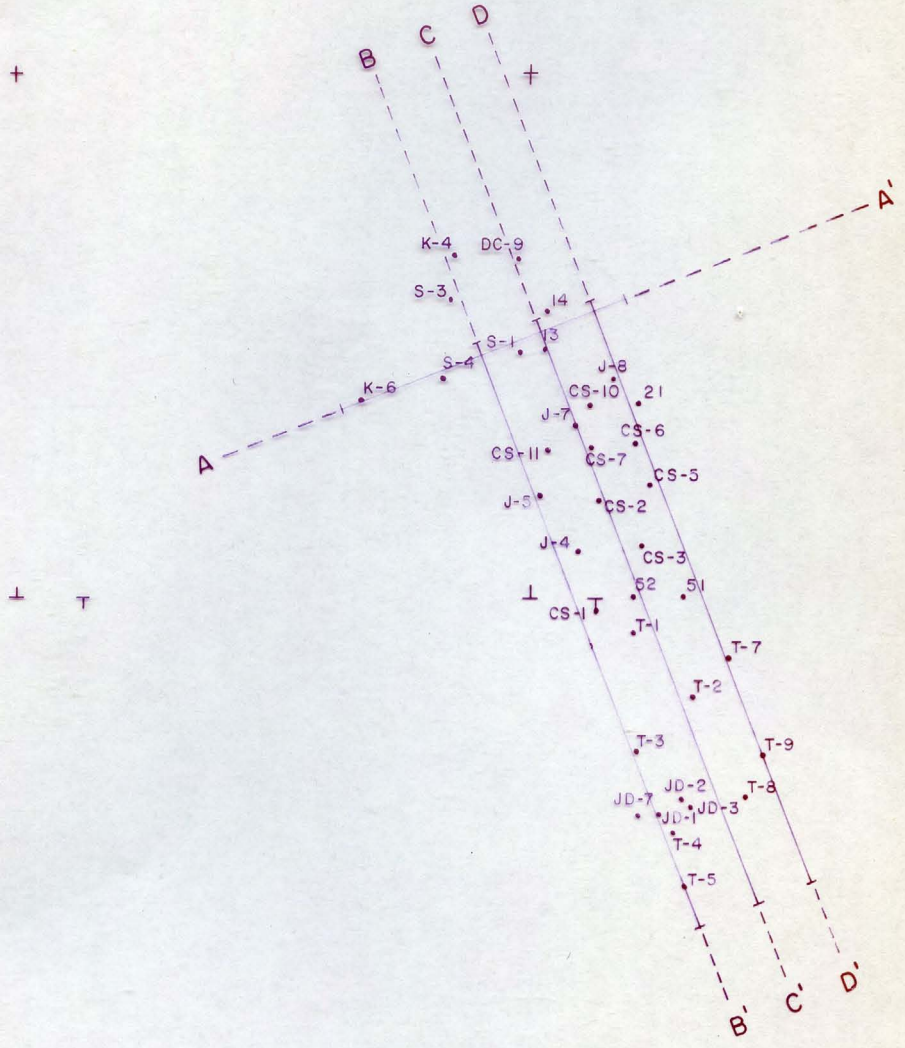
An engineering contractor will be selected to design, procure, construct, and start-up all the above ground facilities such as the solvent extraction plant, electrowinning plant, storage facilities, etc. Concurrently, the development of the mine will be implemented such that when the processing facilities will be completed the required quantities of pregnant leach liquor are available.

Required administrative and manpower personnel will be recruited, trained, and indoctrinated to efficiently and effectively operate the Sullivan Johnson facility. KYPROS recognizes that with the implementation of Phase III it must concurrently obtain all required permits to mine the soluble copper ore at this site. It would be the intent of KYPROS to implement the Phase III program that generates cash flow as quickly as possible.

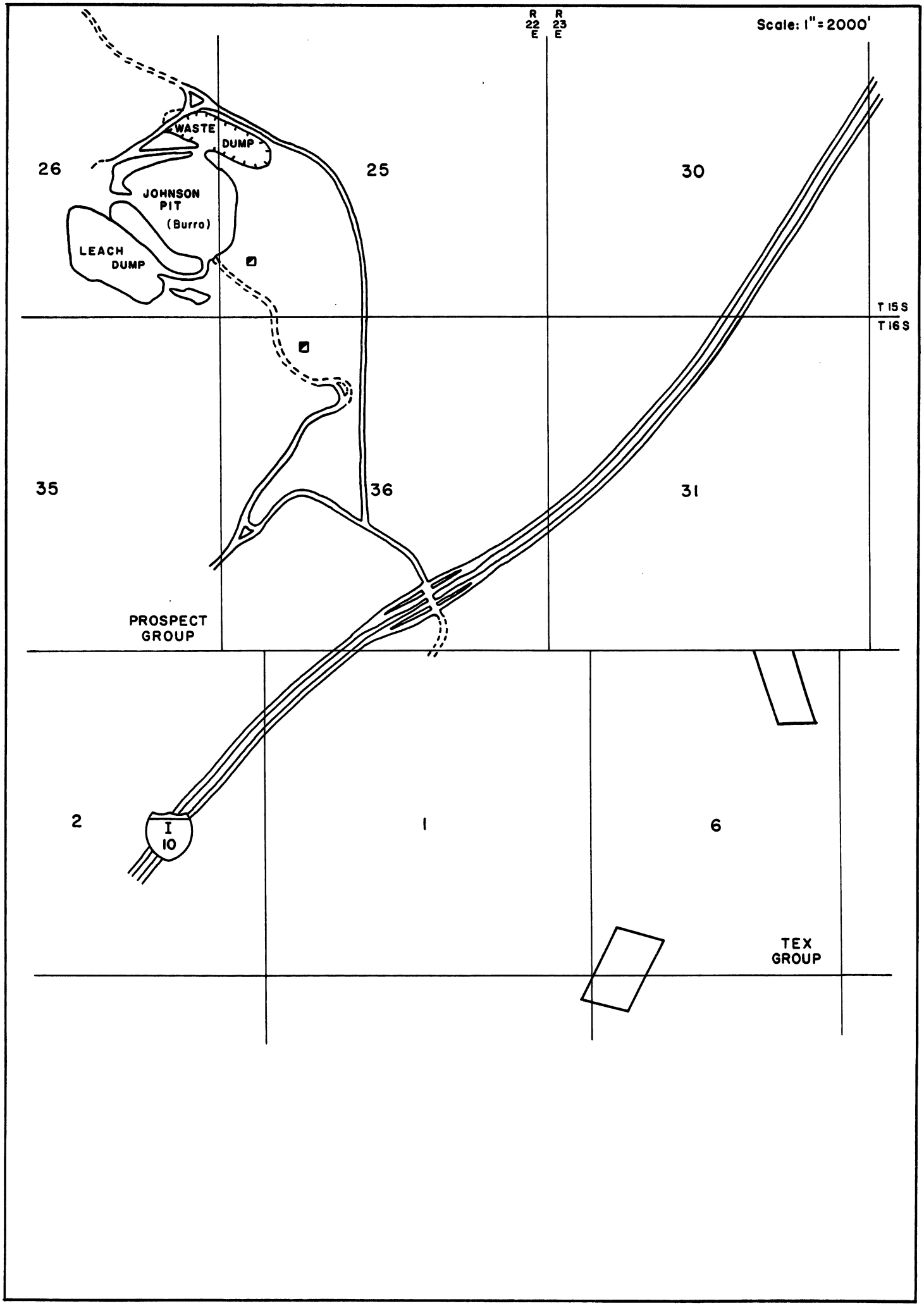
APPENDIX



Magnetic Survey



CROSS SECTION LOCATIONS


R 22 E
R 23 E
Scale: 1" = 2000'
WASTE DUMP
JOHNSON PIT (Burro)
LEACH DUMP
26
25
30
T 15 S
T 16 S
35
36
31
PROSPECT GROUP
2
I 10
1
6
TEX GROUP

DRILL SUMMARY

Drill Hole	Area Drilled -- Feet			% Cu	Formation
	From	To	Interval		
K-4	450	600	150	0.45	Escabrosa (?) and Martin
K-6	310	490	180	0.40	Lower Abrigo
S-I	409	677	268	0.56	Martin and Upper Abrigo
S-2	360 feet bedrock - 5 feet into rock				
S-3	Nil and low grade to 1223				
S-4	319	430	111	0.56	Martin Missing Upper Abrigo
	470	549	79	0.56	Lower Abrigo
	From 319 to 549 feet - 230 feet @ 0.49% Cu				
DC-9	630	732	102	0.57	
	Fault cut out part of upper and all of middle Martin				
CS-1	1300	1570	270	0.44	Lower Abrigo
CS-2	550	1600	1050	0.58	Escabrosa, Quartz Monzonite, Martin, Upper, Middle and Lower Abrigo and Bolsa
CS-3	710	1270	560	0.46	Escabrosa, Martin, Upper and Middle Abrigo
CS-5	870	1340	470	0.45	Escabrosa, Martin, Upper (some Middle) Abrigo
CS-6	717	980	263	0.32	Escabrosa
	1180	1380	200	0.46	Escabrosa and Martin
CS-7	596	1230	634	0.69	Escabrosa, Martin, Upper and Middle Abrigo

DRILL SUMMARY

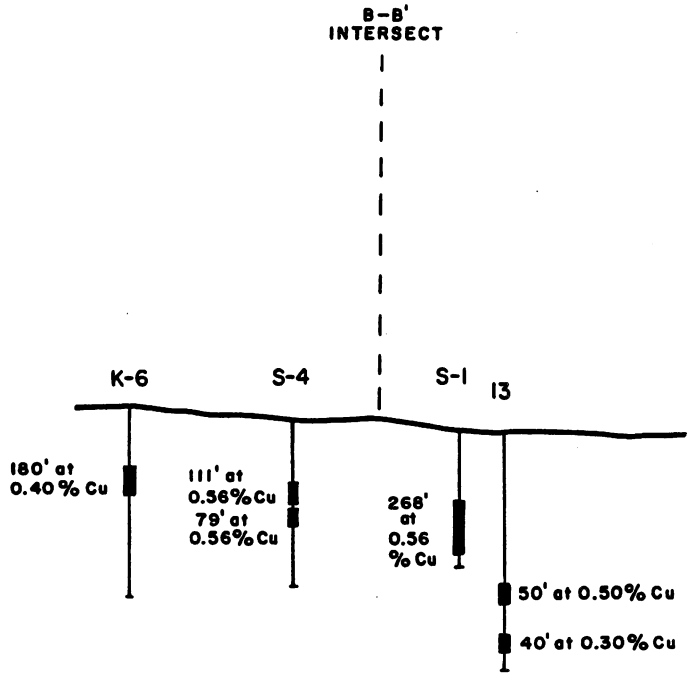
Drill Hole	Area Drilled -- Feet			% Cu	Formation
	From	To	Interval		
CS-10	720	1270	550	0.66	Escabrosa, Martin, Upper (some middle) Abrigo
CS-11	480	490	10	0.51	Martin
	610	880	270	0.31	Martin and Upper Abrigo (some middle) Abrigo
	1150	1200	50	0.30	Lower Abrigo
J-4	1020	1502	406	0.41+	Middle and Lower Abrigo
J-7	704	714	10	0.49	Upper, Middle and Lower Abrigo
	744	982	238	1.15	Upper, Middle and Lower Abrigo
	NOTE:	744 feet to 804 feet	60	1.67	Upper, Middle and Lower Ab rigo
J-8	660	725 (one section of 2')	65	Low grade 1.98	No Geological Log Available
	1125	1205	80	0.41	No Geological Log Available
13	700	770	50+	0.50	Escabrosa, Upper Abrigo
	1040	1100	40+	0.30	Lower Abrigo
14	466	480	14	4.20	No Geological Log Available
51	700	1490	790	0.41	Escabrosa, Martin, Upper, Middle and Lower Abrigo
52	660	710	50	0.42	Escabrosa
	960	1400	440	0.30	Upper, Middle and Lower Abrigo

DRILL SUMMARY

Drill Hole	Area Drilled -- feet From	To	Interval	% Cu	Formation
NOTE:	870	960	40	0.51	Martin
	1000	1310	310	0.23	
	1310	1400	90	0.44	
T-1	540	1191	651	0.44	Upper, Middle and Lower Abrigo
NOTE:	852	1068	216	0.79	
	1226	1430	204	0.51	
	1320	1430	110	0.66	
	from 540 feet to 1430 feet - 890 feet @ 0.443% Cu				
T-2	582	804	222	0.76	Martin
	884.5	1244.5	360	0.48	Upper and Middle Abrigo
	from 532 to 1464 feet - 1131 feet @ 0.37% Cu				
T-3	Nil and Low Grade to 1660				
T-4	385	486.5	101.5	0.40	Martin
	1301	1568.5	267.5	0.55	
T-5	992	1088	96.0	0.44	Lower Abrigo
T-6	215	421	No ore intervals		
T-7	843	928	85.0	0.36	Martin
	1072	1149.5	77.5	0.37	Fault zone
	1833.5	1874.0	40.5	0.67	Lower Abrigo
T-8	1007.5	1129	121.5	0.45	Upper and Middle Abrigo
	1199	1317.5	118.5	0.51	
	from 958.5 to 1397 feet - 438.5 feet @ 0.34% Cu				
T-9	1378	1429	51.0	0.44	No Geological Log Available
	includes one 4 foot run of			5.95	

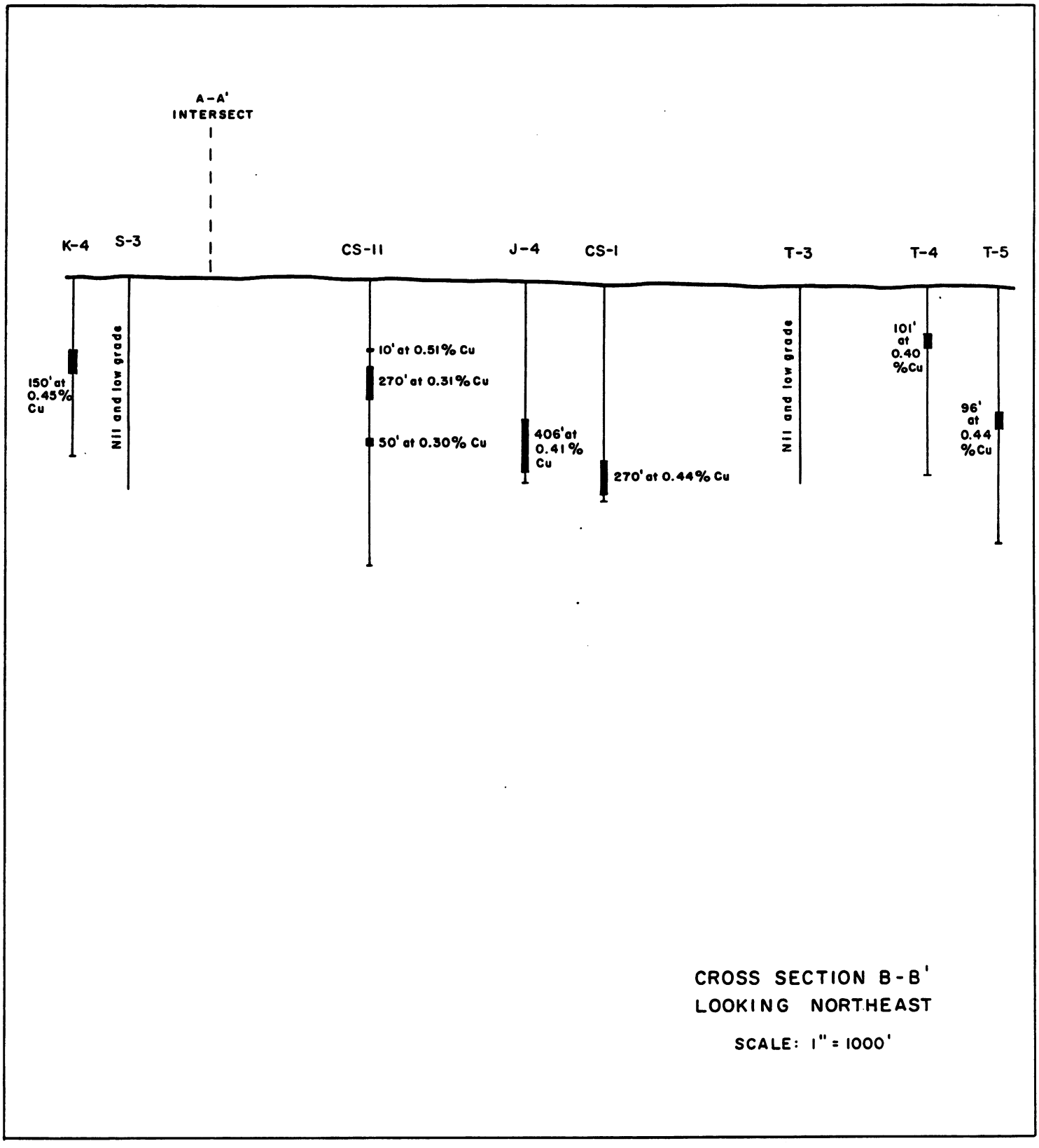
DRILL SUMMARY

Drill Hole	Area Drilled -- feet			% Cu	Formation
	From	To	Interval		
T-9 (Continued)	1378	1470	92.0	0.58	No Geological Log Available
	1578	1674	96.0	0.34	"
	1723	1740	17.0	0.39	"
	1772.5	1790	17.5	0.32	"
	1845	2011	166	0.42	"
T-10	300 Bedrock - 10 in Rock				
T-11	1060	1119	59.0	0.33	No Geological Log Available
T-12	1703	1741	38	0.33	"



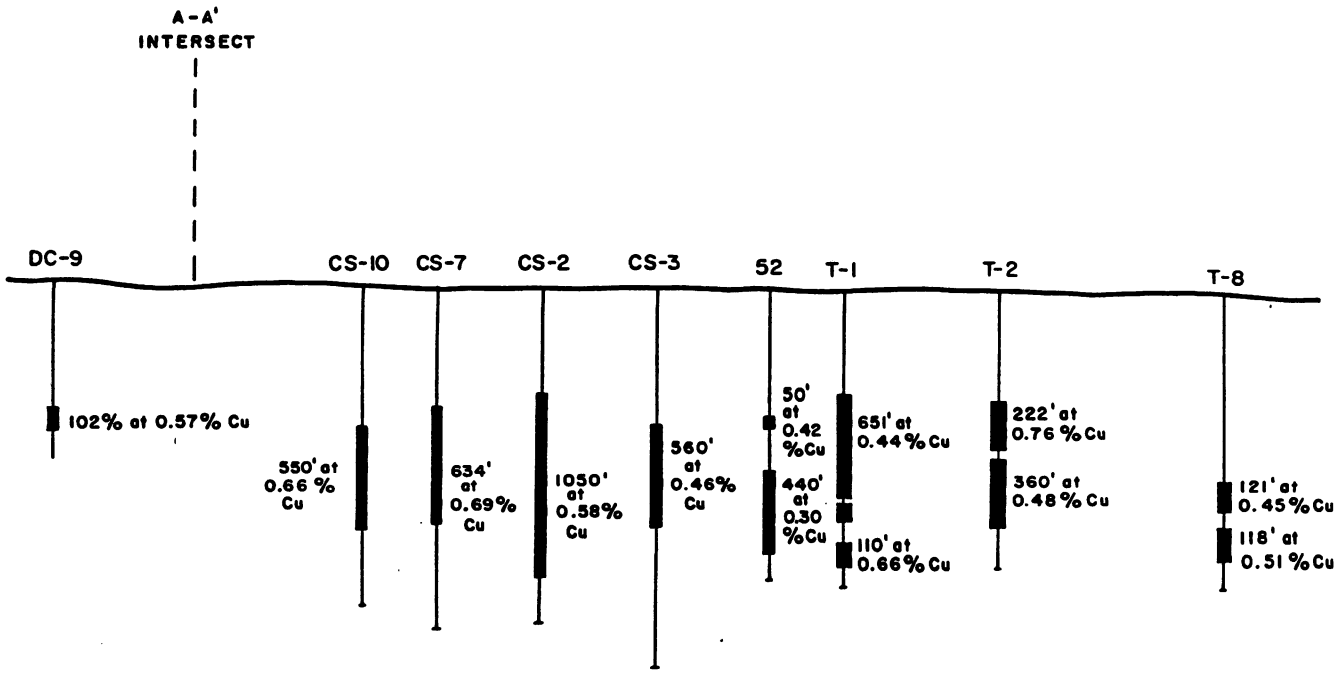
CROSS SECTION A-A'
LOOKING NORTHWEST

SCALE: 1" = 1000'



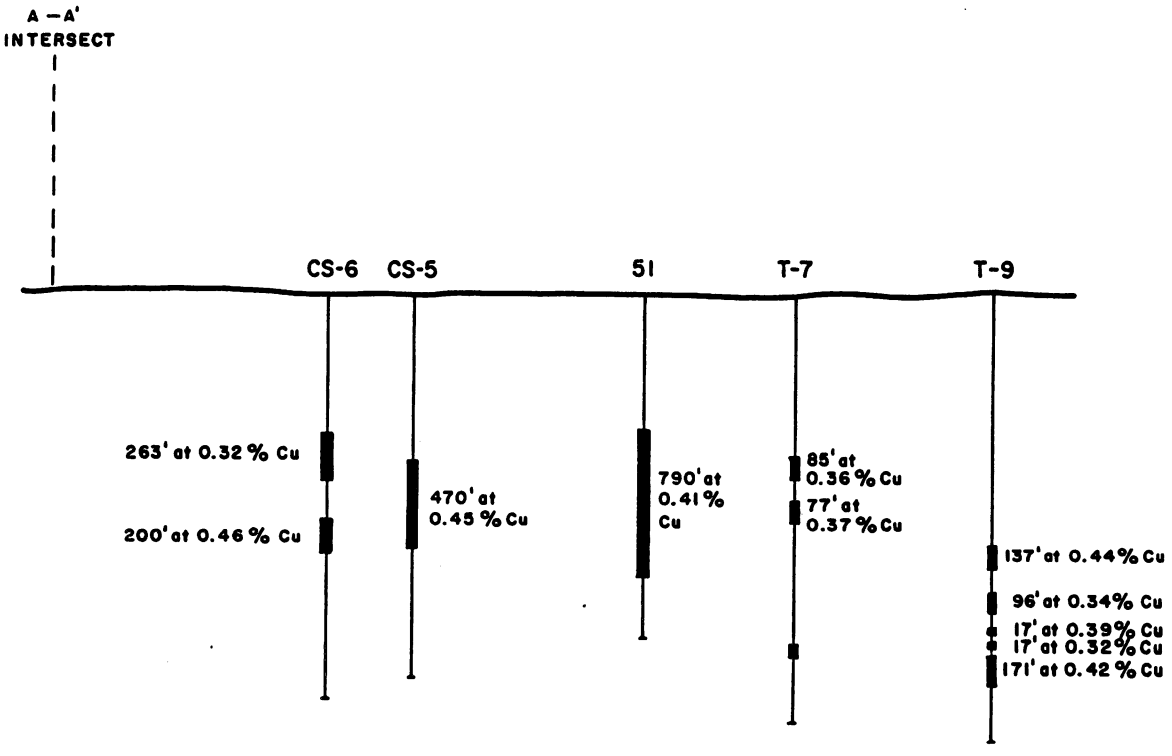
CROSS SECTION B-B'
LOOKING NORTHEAST

SCALE: 1" = 1000'



CROSS SECTION C-C'
LOOKING NORTHEAST

SCALE: 1" = 1000'



CROSS SECTION D-D'
LOOKING NORTHEAST

SCALE: 1" = 1000'

CAPITAL COST ESTIMATE

(Millions - U.S. Dollars)

Recovery		Capacity in S.T.P.D.		
%		40	60	80
• 40	A	$11.0	$11.0	$11.0
	B	38.0	45.4	52.9
	Total	$49.0	$56.4	$63.9
• 50	A	$11.0	$11.0	$11.0
	B	37.9	44.9	51.7
	Total	$48.9	$55.9	$62.7
• 60	A	$11.0	$11.0	$11.0
	B	37.6	44.0	50.4
	Total	$48.6	$55.0	$61.4

Notes:

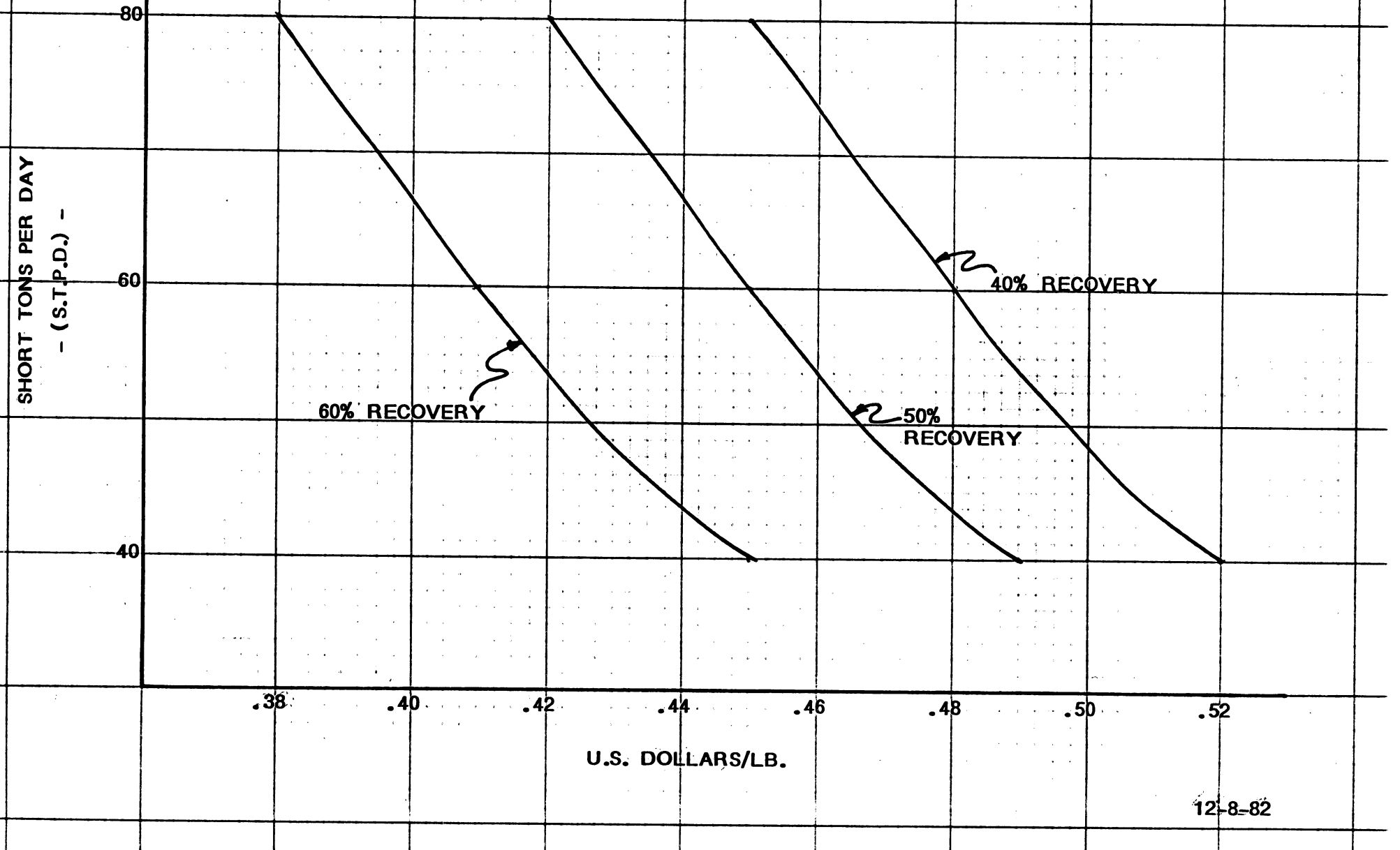
A Includes pilot plant, land acquisitions, and predevelopment cost.

B Cost includes mining, solvent extraction, electrowinning, working capital, and 10% contingency.

December 8, 1982

PRODUCTION COST ESTIMATE

SULLIVAN JOHNSON ORE DEPOSIT



12-8-82

KYPROS MINES INCORPORATED

PHASE I

SULLIVAN JOHNSON PROJECT

PREFEASIBILITY STUDY

GENERAL SCOPE OF WORK

1. Transfer all cores and core data to Phoenix, Arizona, from the Johnson Camp area. Construct core racks and skeletonize cores to reduce volume.

2. Relog and check all cores to confirm mineralization, metallurgical, and leachability data.

3. Review all previous ore reserve testing data and preliminary mining plans developed to date.

4. Drill 6" diameter spot cores in selected areas. Metallurgical testing on new and previously generated developed cores.

5. Review all metallurgical data and projected pilot test program.

6. Review hydrology, power, manpower, and other studies developed to date and related to the development of the Sullivan Johnson Project.

7. Study the metallurgical flow sheet options, and select the most promising method for further examination.

8. Develop a mining plan that would reflect the optimum development of the ore body on a short- and long-term basis.

9. Study options for various unit capacities.

10. Study options for various site locations and infrastructure requirements.

11. Develop preliminary project plan including marketing, manpower, and schedule for execution of the project.

12. Refine preliminary economic analyses of capital and operating costs for various alternative methods of unit capacity and mining plan.

13. Examine other factors that are important to the project as mutually determined by KYPROS and its associate.

14. Finalize and submit evaluation and prefeasibility report.

K[illegible]OS [illegible]ES [illegible]ORP[illegible]ATE

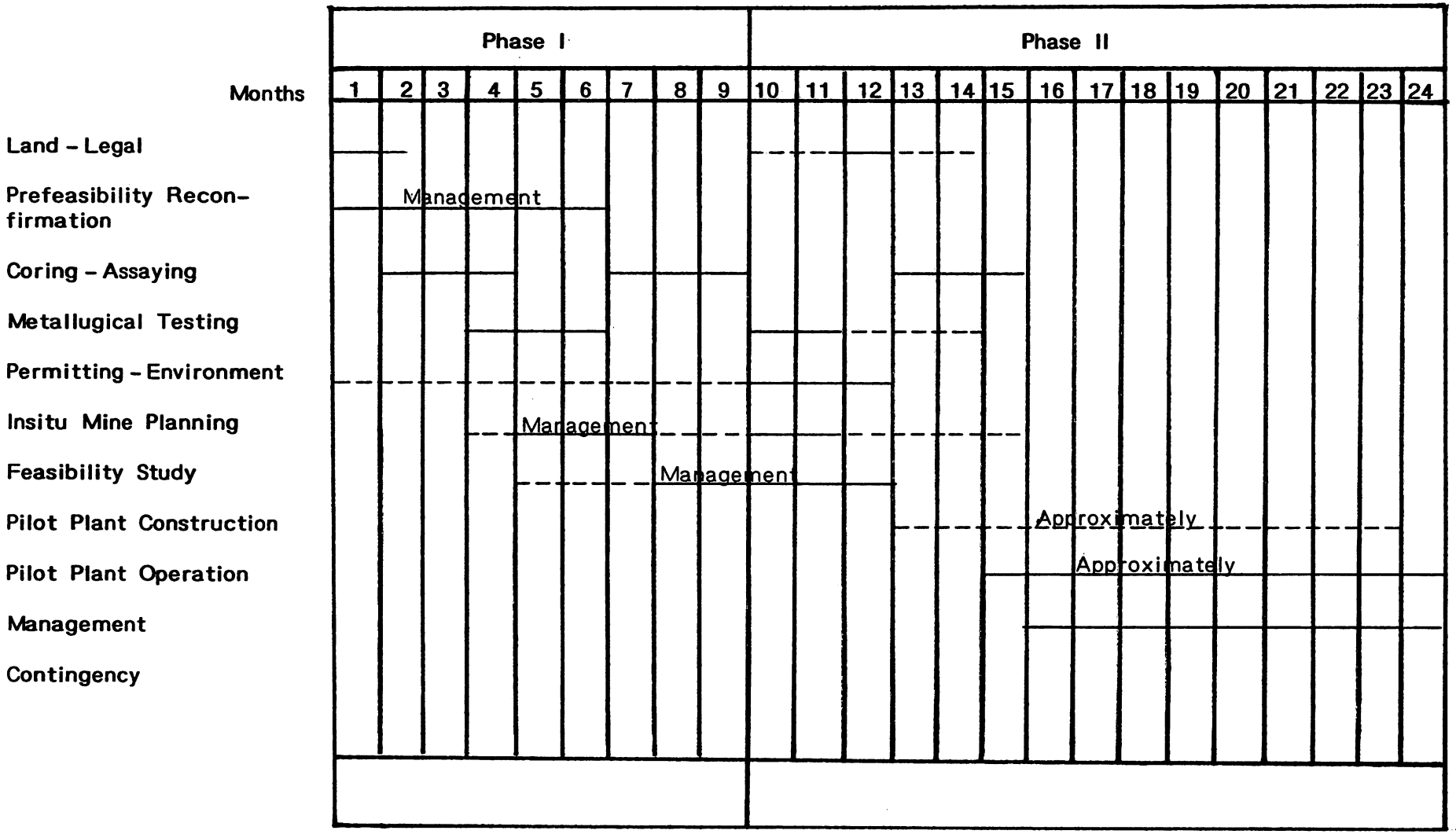
RESPONSIBILITY MATRIX

ITEM	PHASE I PREFEASIBILITY	PHASE II FEASIBILITY	PHASE III ENGINEER, PROCURE & CONSTRUCT	PHASE IV PRODUCTION
Financial	Review and Analysis ------(R)	Negotiate ----(R)	Finalize	------
Legal	Review Leases-(R)	Finalize ------(R)	-----	------
Market	Perform Prelim. Survey -(R)	Finalize -----(R)	-----	------
Ore Deposits	Geological Evaluation, Coring, and Review of Past Core Drillings	Additional Core Drill. ---(R)	-----	------
Mine	Evaluate Hydrology, Fracing, Blasting, Mining and Leaching	Dev. Plan	Finalize & Implement	Production
Process	Technical Review and Verification of Metallurgy	Pilot Plant	Finalize	------
Plants	Metallurgical --(R) Review & Eval.	Conc. Design -(R)	Design Procure Const. Start-Up ------(R)	Production
Infra-structure	Review and Evaluation	Dev. Plan	Design Procure Const. Start-Up	Production
Manpower	Survey --------(R)	Dev. Program -(R)	Recruit Train Incorporate	------
Adminis.	------	Survey -------(R)	Develop Programs & Systems	Implement
Environ-mental	Review -------(R)	Fatal Flaw ---(R) Analyses	Obtain Permits	------

D = Decision
R = Report
R c = Combined Report

December 8, 1982

BUDGET FORECAST



December 8, 1982

SUPPLEMENTAL REPORT

SULLIVAN-JOHNSON ARIZONA

COPPER DEPOSIT

KYPROS MINES INCORPORATED

REV. I

TABLE OF CONTENTS

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Letter of Transmittal	.
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TECHNICAL SUPPLEMENT AND SUMMARY

KYPROS, in its original confidential report, stated that an in-situ leaching process, which it advocates, is the preferred mining method. Our proprietary in-situ leaching method using high strength sulfuric acid to saturate the in-place fractured ore will permit the profitable economic recovery of acid soluble copper equal to or greater than what is currently achievable.

The process of leaching copper minerals has changed appreciably during the past two decades, particularly during the past 3 to 5 years. The probability is that the copper leaching process, as we know it today, will change considerably in the next decade as it, too, is an intregal part of the technical evolution taking place in the minerals industry today.

The Sullivan-Johnson (I-10) ore deposit, and ore deposits within an area of influence are unaltered paleozic rock primarily composed of carbonates, which in their natural state are considered to be high acid consumers. Further evidence indicates that favorable acid consumption rates in these situations are and have been obtainable. It is extremely important to recognize that the principle copper mineralization is concentrated within fracture zones and along bedding planes of the host rock; hence, preferential fracturing of the ore body is ideal and recommended. Exposing unnecessary acid consuming areas must and will be avoided to obtain the lowest

possible leachable mining cost. Tests conducted by some companies in the past on this ore have overlooked the importance of selecting the proper mining method, and have evaluated samples based on proposed agitation leaching, ammonical leaching, and pad leaching methods that have given unfavorable metallurgical and acid consumption results.

Within a few miles to the north of the Sullivan-Johnson (I-10) ore deposit, there are two properties having similar ore characteristics to KYPROS'. One of these is about mined out; however, the other has been tested for an agitation leaching process and for an ammonical leaching process. The following is item 5 of the summary of a major engineering company's report on the agitation leaching test:

> "A standard agitation leach test on oxide ore with sulfuric acid yielded extractions of 82.7% of the copper and 80.9% of the zinc. Acid consumption was 621 lbs/ton of ore, or 38.1 lbs/lb of copper extracted. Based on combined copper-zinc extraction values, acid consumption was 19.6 lbs/lb of combined metal."

Acid consumption of this magnitude is not only unacceptable, but most certainly not recommendable.

On a property a few miles south of the Sullivan-Johnson (I-10) deposit, a former owner had tests performed on "Bolsa and Abrigo" type ores. Incidentally, in the Sullivan-Johnson (I-10) deposit part of the mineralization occurs in the Abrigo zone, hence, similar

and most probably identical characteristics prevail. The conclusions by a major engineering and testing company to the column leach test they performed are as follows:

"1. Copper Extraction

A. The ore leached in the column contained 1.25% copper by assay and 1.21% by calculation. See Table VI.

B. About 40% of the copper was extracted in 50 days. The projected extraction would be about 50% in 100 days. These results would simulate a heap leaching condition on minus 12 inch materials. See Figure VII.

C. The above extraction amounts to about 60% of the acid soluble material in the ore and examination of the residue indicated that the surface copper mineralization had been solublized but not the major portion within the pieces larger than 2".

The relationship between copper extraction and time vs. particle size is shown in Figure VIII.

D. This is confirmed by the Size-Assay analysis of the residue. These observations strongly indicate reasonable fragmentation of the orebody in "in situ" or on heaps is necessary for achieving higher extractions around 66% of the total copper content. See Tables VII and VIII.

E. The calculated leach rate amounted to 0.16 lbs. copper per tons per day or 0.64% copper extracted per day.

2. Acid Consumption - The acid consumption required to achieve the above extraction was 120 lbs/ton of ore leached or 12 lbs/lb copper extracted.

> The relationship between copper extraction vs. acid consumption is shown in Figure IX. On the other hand, the relationship between acid consumption vs. time is shown in Figure X. In this case, the data from agitation and bottle leach tests previously performed by Metcon is included to show the relationship.
>
> The relationship between acid consumption vs. particle size is illustrated in Figure XI while the relationship between solution grade and pH vs. time is shown in Figure XII." a

The aforementioned Abrigo column leach test was performed on a grade of 1.25% copper by assay, and they obtained a recovery of 40% of the copper in 50 days. The projected copper recovery was stated about 50% in 100 days in the report, and these results would simulate a heap leaching condition on minus 12 inch material. KYPROS, by leaching in-place and fracturing the ore within fracture zones and along bedding planes, intends to lower the acid consumption even further to below 10 lbs of acid per pound of copper and probably to 8 lbs per pound of copper.

An in-place test simulating in-situ leaching can be immediately performed to determine acid dissolution and recovery on high grade and average ores. Test units have been made available to KYPROS to perform this test, and a program can be implemented to confirm the practical achieving of low acid consumption leaching results on Abrigo type ores.

The proposed test using the United States Bureau of Mines

a. See Appendix

"Huff and Puff" in-situ drilling unit, assuming that the unit is completely adaptable to KYPROS' situation, follows:

Huff and Puff Test

A. Mobilize

1. Transfer unit to Arizona.

2. Check and adapt unit to meet KYPROS test conditions for depts, pressures, etc.

3. Open selected core holed, if necessary.

4. Set-up test unit.

B. Perform tests as follows:

1. Saturate leaching area with high strength acid in the 250 to 300 gpl area, and let perculate for 15 days.

2. Inject a low strength acid rinse for 45 days.

3. Collect pregnant liquor and analyze daily for important characteristics.

C. Prepare report.

Additionally, KYPROS also recommends the implementing of an in-situ leaching test assuming favorable results are obtained in the previous test. An in-situ test would consist of the following:

In-situ Leach Test

A. Mobilize

1. Purchase test equipment.

2. Select and hire drilling contractor.

3. Select and hire engineering test company.

4. Fracture and drill one leaching unit consisting of a 5 or 7 hole pattern with the center hole being a recovery hole on a 30 or 50 foot pattern.

B. Perform tests as follows:

1. Saturate mine with high strength sulfuric acid, probably 250 to 300 gpl, and let perculate for 15 days.

2. Using a low strength sulfuric acid, inject and rinse the saturated area for 30, 60, 90, and 120 days.

3. Monitor and analyze results daily.

C. Prepare report.

Complete copies of the reports on the Strong-Harris and the Dragoon properties will be made available upon request, and only critical excerpts and conclusions have been used in this report or included in the appendix.

MOBILIZATION

KYPROS, when implementing any activity on its property, must, by its contractual obligations, physically move the cores located in the Sullivan Johnson (I-10) area to a new location. In the process of moving the cores, it is prudent to again spot check and analyze these cores for its mineralization characteristics.

Moving cores and other pertinent related data necessitates the renting of space, building permanent storage racks, transporting the cores, and relogging. This will bring forth numerous related problems to successfully implement a coordinated test program. The legalities and land acquisition problems will also require constant attention and objective direction to avoid any possibility of developing a problem for KYPROS.

KYPROS' mobilization includes a field and corporate program to assure continued continuity with its objective to bring into production profitable acid soluble ore deposits. Field mobilization will generally include a planned method to keep the cores for future use, plus augmenting the efforts of the test program people. Corporate mobilization will put into proper prospective the necessary legal, financial, and managerial aspects to assure corporate continuity in KYPROS' program.

BUDGET

Phase I - 12 Months

I Stage I - 60 Days (3/1/83 to 5/1/83)

• Land Acquisition	$148,720	
• Corporate Expenses	26,280	
Sub-Total		$175,000

II Stage II - 120 Days (5/1/83 to 9/1/83)

• Mobilization	$172,280	
A. Field		
B. Transport		
C. Warehouse		
D. Relogging & Checking Cores		
E. Metallurgical		
F. Computerizing Ore Body		
G. Consultants		
• H & P Leaching Test		
• Corporate Expenses	$ 67,720	
Sub-Total		$240,000

III Stage III - 180 Days (9/1/83 to 3/1/84)

• In-situ Test Program	$243,000	
• Corporate Expenses	92,000	
Sub-Total		$335,000

TOTAL $750,000

APPENDIX

PART II

ABRIGO ORE SAMPLE

CONCLUSIONS

Abrigo Column Leach (Detailed data and results shown in Table IX)

1. Copper Extraction

   a. The ore leached in the column contained 1.25% copper by assay and 1.21% by calculation. See Table VI.

   b. About 40% of the copper was extracted in 50 days. The projected extraction would be about 50% in 100 days. These results would simulate a heap leaching condition on minus 12 inch material. See Figure VII.

   c. The above extraction amounts to about 60% of the acid soluble material in the ore and examination of the residue indicated that the surface copper mineralization had been solublized but not the major portion within the pieces larger than 2".

      The relationship between copper extraction and time vs. particle size is shown in Figure VIII.

   d. This is confirmed by the Size-Assay analysis of the residue. These observations strongly indicate reasonable fragmentation of the orebody in "in situ" or on heaps is necessary for achieving higher extractions around 66% of the total copper content. See Tables VII and VIII.

   e. The calculated leach rate amounted to 0.16 lbs. copper per ton per day or 0.64% copper extracted per day.

2. Acid Consumption - The acid consumption required to achieve the above extraction was 120 lbs/ton of ore leached or 12 lbs/lb copper extracted.

   The relationship between copper extraction vs. acid consumption is shown in Figure IX. On the other hand, the relationship between acid consumption vs. time is shown in Figure X. In this case, the data from agitation and bottle leach tests previously performed by Metcon is included to show the relationship.

   The relationship between acid consumption vs. particle size is illustrated in Figure XI while the relationship between solution grade and pH vs. time is shown in Figure XII.

Summary of Acid Column Leaching of Abrigo Copper Oxide Ore
(See Table IX for details)

Leach Equipment: Vertical 2'-diameter rubber-lined steel column.

Ore Leached: Weight - 1986 lb. 10' column of chrysocolla-bearing oxide ore.
Particle size - Pieces of 6" drill core up to 1.5' in length.
Head Assay - 1.25% Cu (25 lbs/ton); 0.88% Ox Cu (17.6 lbs/ton).

Leach Conditions:

Leach type - Downward solution percolation at ambient outside temperature.

Solution Percolation Rate -

Average Flow Rate, cc/min	Leach Interval days	Total days
46*	0-2	2
56	3-17	15
66	18-19	2
80	20-43	24
106	44-45	2
42	46-50	5

* Approximately 0.004 gpm/ft^2 or 18.1 gpd.

Solution Acid Strength -

H_2SO_4, g/l	pH	Leach Interval days	Total days
20	0.65	0-14	14
15	0.75	15-50	36

Leaching Schedule -

Type of Solution Distribution	Leach Interval days	Total days
Percolation leach	0-43	43
Rest period	----	14
Percolation leach	44-50	7

Results: Copper Extraction - 9.91 lbs Cu/ton; 0.16 lbs Cu/ton/day; 0.64% Cu Extd./day.

Basis of Calculation	% Cu Extracted	% Ox Cu Extracted
Assayed Head	39.6	56.3
Calculated Head	40.9	47.4

Sulfuric Acid Consumption - 120.1 lbs acid per ton; 12.13 lbs acid per lb Cu extracted.

Metallurgical Balance:

Product	Weight/Volume	Assay, Cu	Content, lbs	Distribution, %
Preg. Solution	4608.0 l.	0.97 g/l	9.84	40.9
Leach Residue	1950.5 lbs	0.73 %	14.24	59.1
Total			24.08	100.0

Head (Calculated) - 1.21% Cu

Head (Assayed) - 1.25% Cu

Table VI:
Abrigo Head Ore Assays

Constituent	Southwestern Asseyers		Amer. Analytical		MSRD		Average	
	lbs/ton	%	lbs/ton	%	lbs/ton	%	lbs/ton	%
Cu	25.0	1.25	25.4	1.27	24.6	1.23	25.0	1.25
Ox Cu	19.6	0.98**	17.6	0.88**	22.6	1.13***	--	--
A S Cu*	--	--	--	--	21.4	1.07	--	--
$CaCO_3$	182.0	9.1	160.0	8.00	--	--	172.0	8.60

* By Duval Method.

** Assay by method using cold H_2SO_4 and SO_2.

*** Assay by method using 5% H_2SO_4 added to a boiling water solution.

Table VII:

Abrigo Leach Residue Assays

Size Fraction Inches	Weight %	Cu		ASCu*		OxCu		Fe		$CaCO_3$	
		lbs/ton	%	lbs/ton	%	lbs/ton	%	lbs/ton	%	lbs/ton	%
+6	48.0	16.8	0.84	15.0	0.75	11.4	0.57	174.0	8.7	196.0	9.8
-6+4	0.4	11.6	0.58	10.4	0.52	6.4	0.32	168.0	8.4	310.0	15.5
-4+2	13.9	16.6	0.83	15.2	0.76	12.2	0.61	200.0	10.0	118.0	5.9
-2	37.7	12.0	0.60	11.2	0.56	8.2	0.41	170.0	8.5	60.0	3.0
Calc'd. Composite	100.0	15.0	0.75	13.6	0.68	10.4	0.52	176.0	8.8	134.0	6.7
Assayed Composite**		14.6	0.73	13.6	0.68	11.2	0.56	128.0	6.4	156.0	7.8

*By Duval Method

**Contains <.001% (0.02 lbs/ton) copper that is soluble in pH 3.5 H_2SO_4 solution.

TABLE VIII:

DISTRIBUTION OF COPPER IN SIZE FRACTIONS OF ABRIGO LEACH RESIDUE

Size raction, Inches	Screen Analysis			Cu Distribution				Ox Cu Distribution			
	Weight Lbs.	Weight %	Cumulative Weight %	Assay %	Content Lbs.	% of Copper	Cumulative %	Assay %	Content Lbs.	% of Copper	Cumulative %
+6	935.5	48.0	48.0	0.84	7.86	54.0	54.0	0.57	5.34	53.2	53.2
-6+4	8.5	0.4	48.4	0.58	0.04	0.3	54.3	0.32	0.02	0.2	53.4
-4+2	270.5	13.9	62.3	0.83	2.24	15.4	69.7	0.61	1.66	16.5	69.9
-2	736.0	37.7	100.0	0.60	4.41	30.3	100.0	0.41	3.02	30.1	100.0
	1950.5	100.0		0.73	14.55	100.0			10.04	100.0	

Table IX:

Log of Acid Column Leaching of Abrigo Copper Oxide Ore

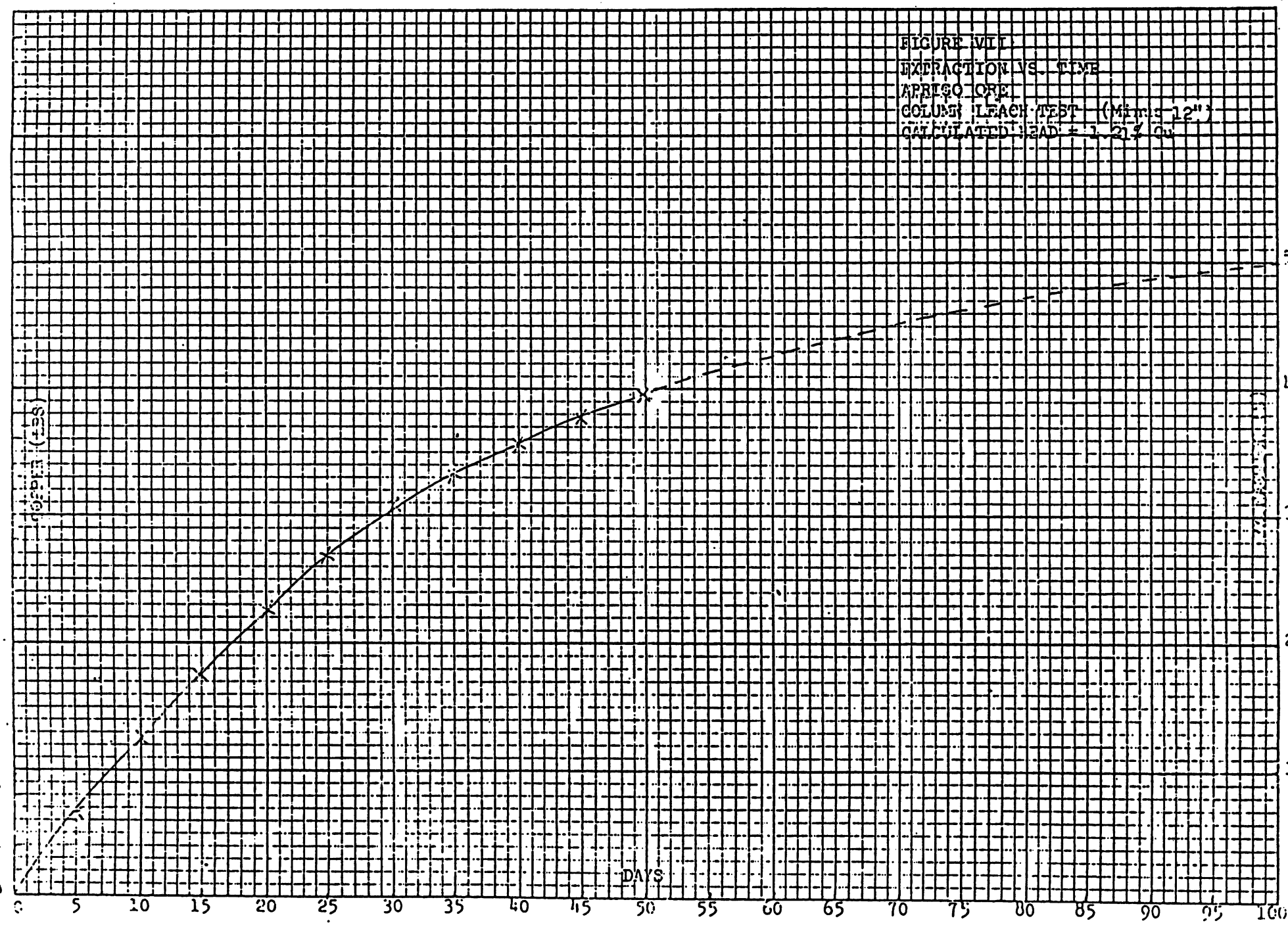
Days of each	Type of Solution Distribution	Fresh Leach Solution: Volume Consumed, gals	Flow to Col., cc/min	H_2SO_4 g/l	H_2SO_4 pH	Pregnant Leach Solution: Volume, gals.	pH Flowing Solution (Cum)	pH Storage Tank	Copper g/l	Content Cum. lbs.
	Percolation Leach	20	46	20	0.65	8.5	6.80	----	0.11	0.01
	"	16	46	"	"	15.5	4.90	----	0.28	0.04
	"	16.5	52	"	"	14.5	4.00	----	0.73	0.13
	"	18.5	58	"	"	17.5	3.20	----	1.88	0.41
	"	20.7	52	"	"	22.5	2.15	----	3.00	0.97
	"	19.5	58	"	"	19.5	2.15	----	3.03	1.46
	"	17.5	56	"	"	17	2.00	----	2.83	1.86
	"	19	57	"	"	21	1.50	----	3.64	2.50
	"	10.25	58	"	"	11	2.20	----	2.63	2.74
	"	21	57	"	"	20.5	1.15	----	2.43	3.16
	"	22	59	"	"	22	1.80	----	1.64	3.46
	"	22	56	"	"	22	1.40	----	1.15	3.67
	"	22	60	"	"	23	1.30	----	1.03	3.87
	"	19	57	"	"	18	1.20	----	1.14	4.04
	"	18	55	15	0.75	23	1.60	----	1.10	4.25
	"	22.5	53	"	"	22	1.70	----	1.12	4.46
	"	20	55	"	"	20	1.60	----	1.56	4.72
	"	22.5	67	"	"	22.5	1.70	----	1.38	4.98
	"	22	66	"	"	22	1.60	----	1.29	5.22
	"	34	75	"	"	33	1.60	----	1.44	5.61
	"	20.5	78	"	"	21.5	1.40	----	1.17	5.82
	"	29.5	77	"	"	29.5	1.30	----	0.77	6.02
	"	28.5	80	"	"	29	1.60	----	1.02	6.27
	"	29	77	"	"	30.5	1.30	----	0.77	6.47

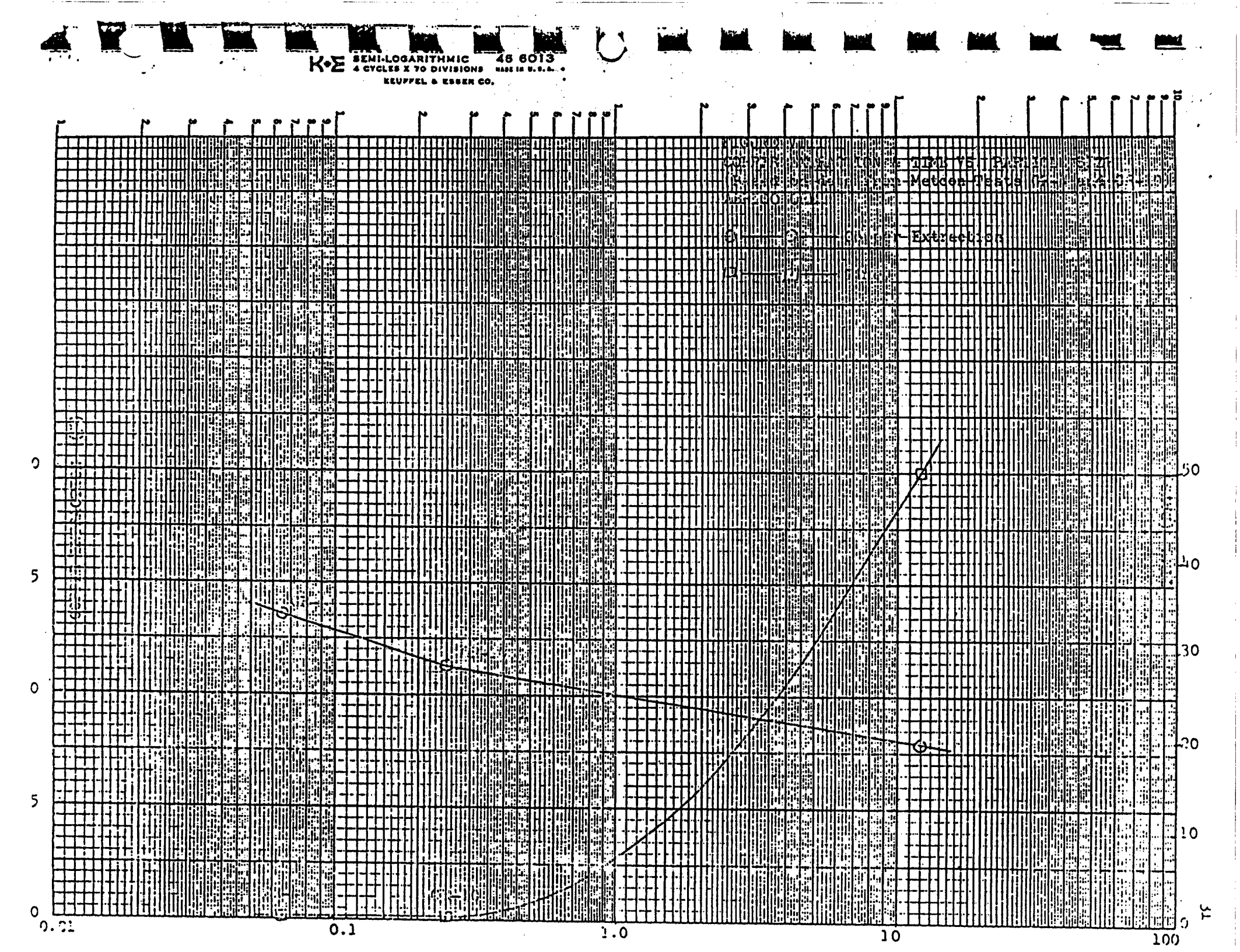
Table

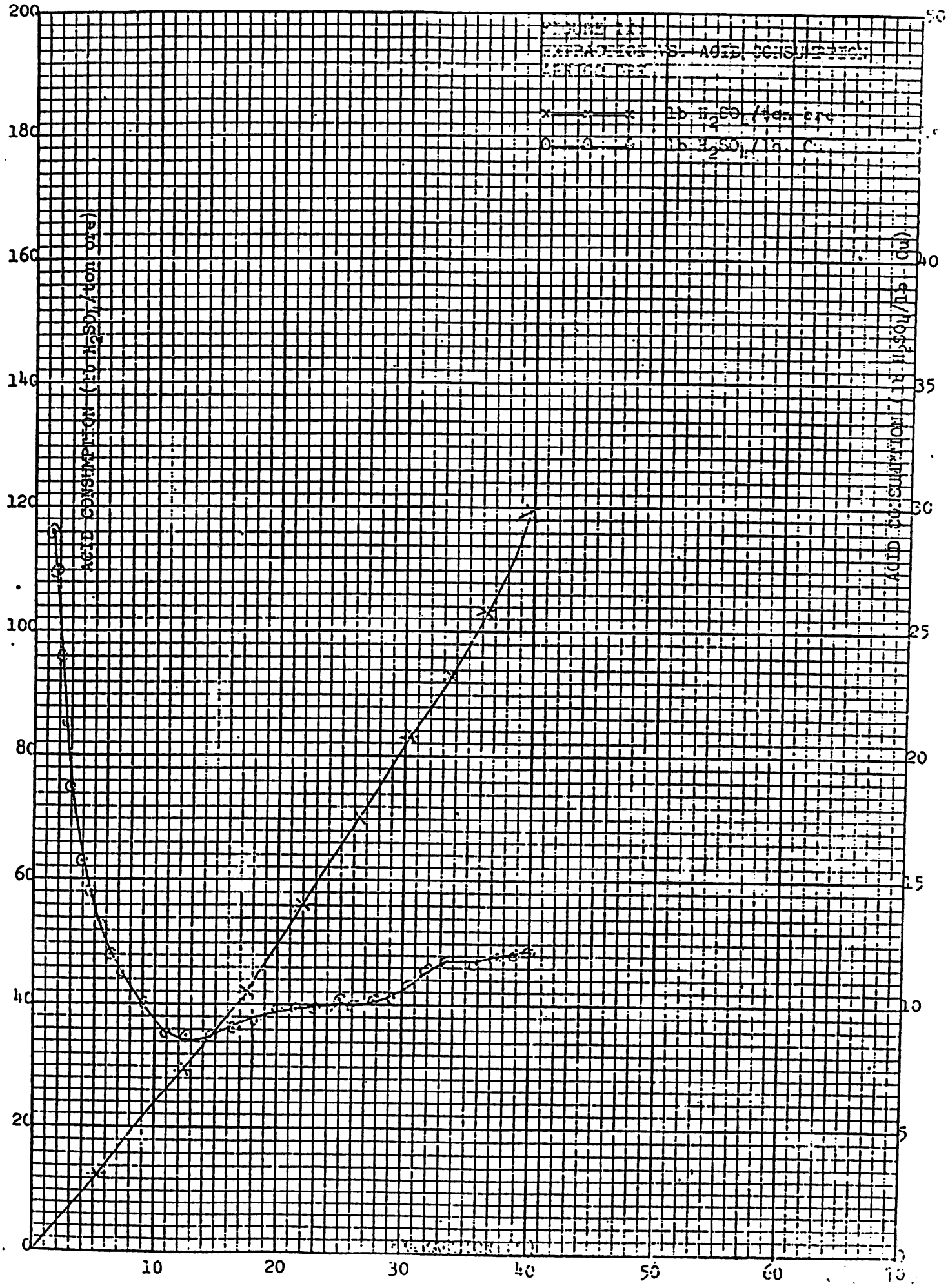
Log of Acid Column Leaching of Abrigo Copper Oxide Ore

Days of Leach	Type of Solution Distribution	Fresh Leach Solution: Volume Consumed (gals.)	Fresh Leach Solution: Flow to Col., cc/min.	Fresh Leach Solution: H_2SO_4 g/l	Fresh Leach Solution: H_2SO_4 pH	Pregnant Leach Solution: Volume, gals.	Pregnant Leach Solution: pH Flowing Solution (Cum.)	Pregnant Leach Solution: pH Storage Tank	Pregnant Leach Solution: Copper g/l	Pregnant Leach Solution: Content Cum. lbs.
25	"	32	78	"	"	34	1.25	----	0.69	6.66
26	"	31.5	81	"	"	29	1.25	----	0.91	6.88
27	"	27.5	80	"	"	31	1.40	----	0.94	7.13
28	"	27	83	"	"	27	1.70	----	0.63	7.27
29	"	28	83	"	"	27	1.15	----	0.58	7.40
30	"	32.5	80	"	"	34	1.20	1.15	0.56	7.56
31	"	29	81	"	"	29	1.30	1.15	0.55	7.69
32	"	28	82	"	"	27	1.20	1.15	0.76	7.86
33	"	31	75	"	"	32	1.20	1.20	0.58	8.02
34	"	27.5	78	"	"	27	1.40	1.00	0.68	8.17
35	"	28.5	80	"	"	28.5	1.00	1.15	0.54	8.30
36	"	30	80	"	"	30	1.00	1.10	0.46	8.41
37	"	28.5	82	"	"	30	1.10	1.10	0.62	8.57
38	"	28	82	"	"	29	1.00	1.20	0.52	8.69
39	"	29	80	"	"	30	1.20	1.30	0.46	8.81
40	"	27	80	"	"	31	1.10	1.10	0.40	8.91
41	"	28	82	"	"	32	1.20	1.20	0.34	9.00
42	"	25	85	"	"	25	1.10	1.10	0.34	9.10
43	"	----	----	"	"	7	----	1.10	0.47	----
----	14-day Rest Period	----	----	----	----	----	----	----	----	----
44	Percolation Leach	26	106	15	0.75	22	1.10	1.20	0.55	9.20
45	"	37.5	40	"	"	40	1.10	0.90	0.44	9.35
46	"	14.75	40	"	"	16.5	1.10	1.20	0.42	9.41
47	"	13.75	40	"	"	12.5	1.30	1.30	0.64	9.48
48	"	47	43	"	"	44.5	1.20	1.30	0.56	9.68
49	"	19.5	42	"	"	19	1.20	1.20	0.52	9.76
50	"	15	45	"	"	17	1.20	1.40	0.48	9.84

K+E 10 X 10 TO THE INCH 46 0703
7 X 10 INCHES MADE IN U.S.A.
KEUFFEL & ESSER CO.

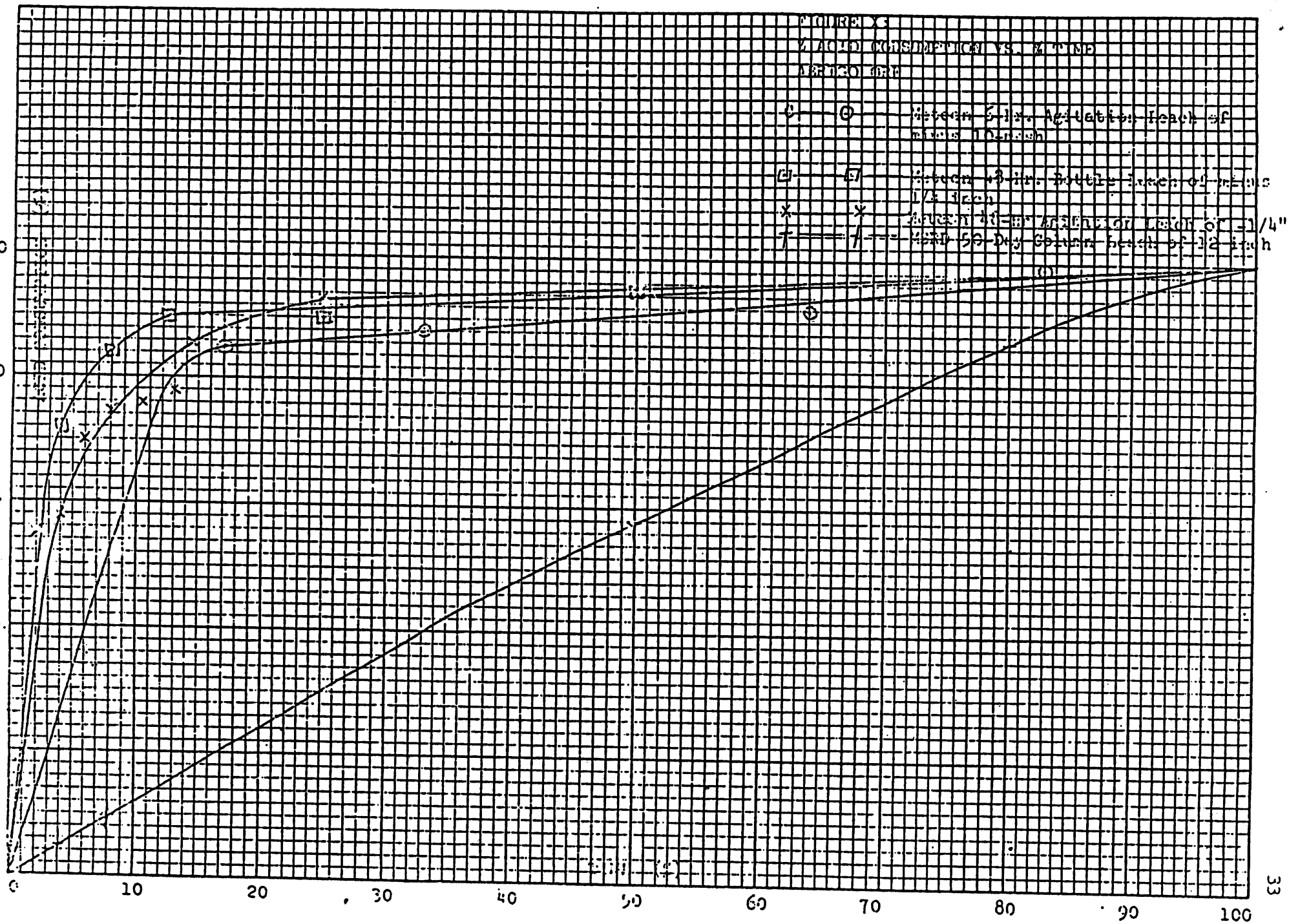


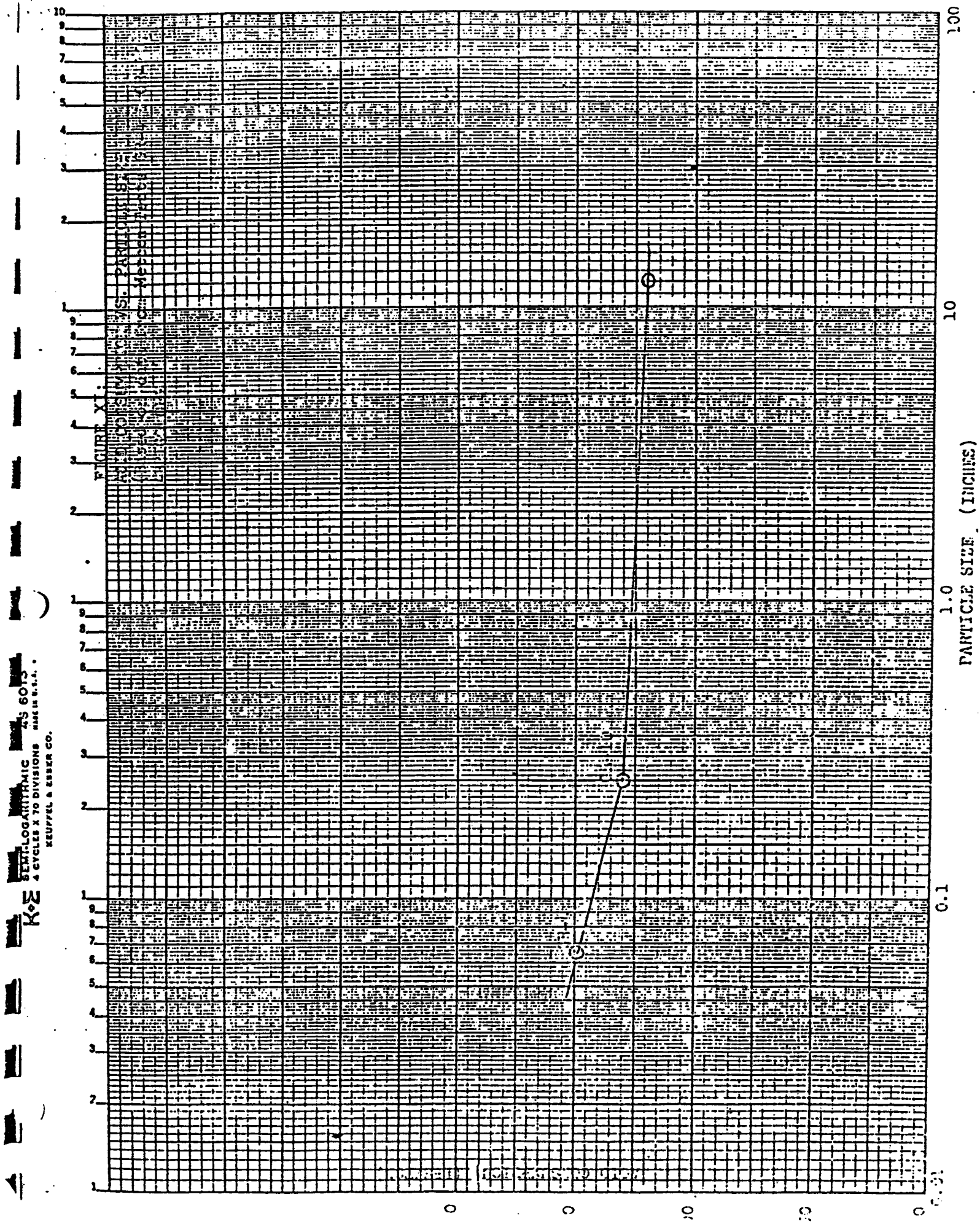
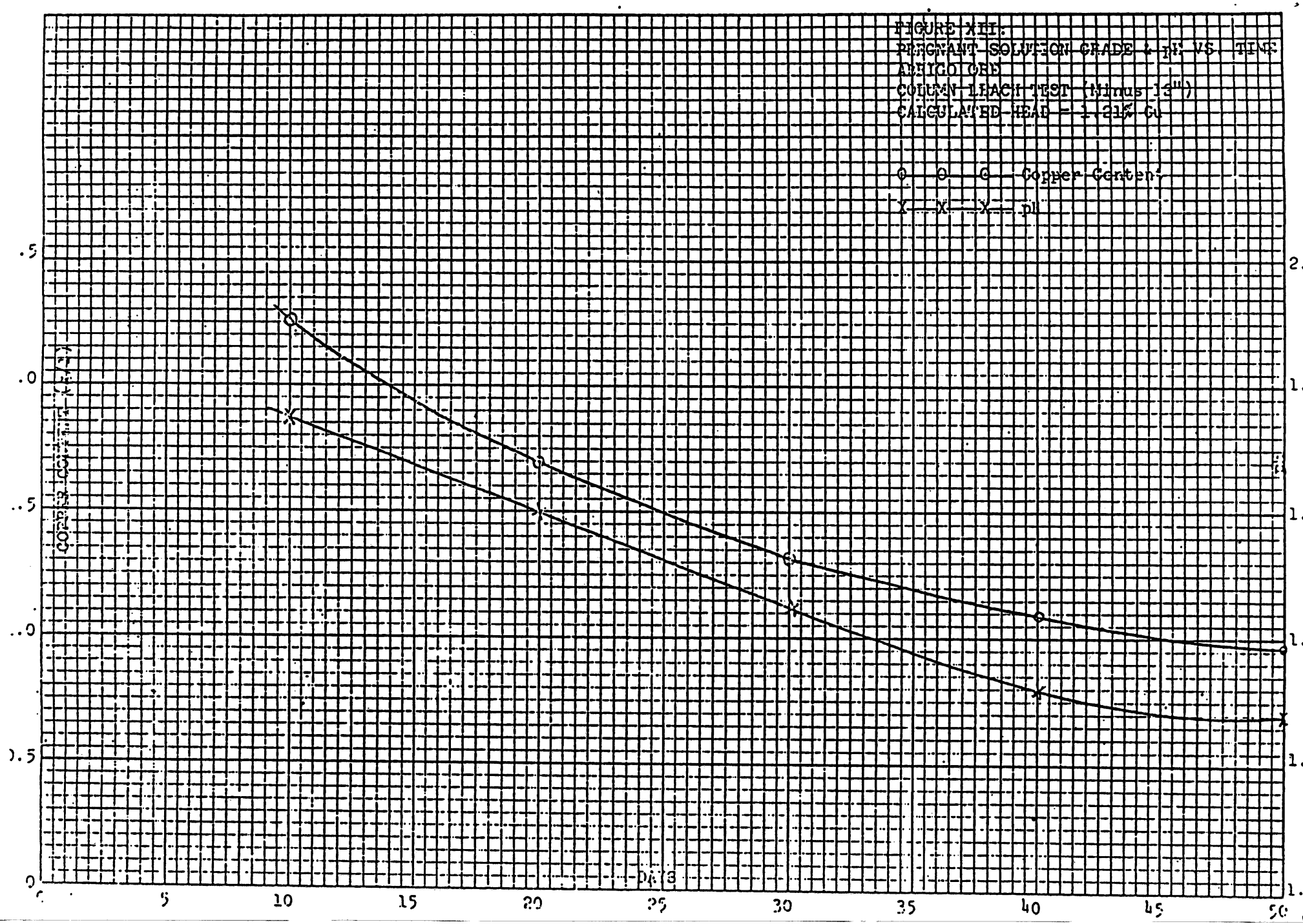

K+E SEMI-LOGARITHMIC 46 6013
4 CYCLES X 70 DIVISIONS MADE IN U.S.A.
KEUFFEL & ESSER CO.
Extraction
0.01
0.1
1.0
10
100
0
5
10
20
30
40
50



K+E 15 X 10 TO THE INCH 46 0703
7 X 10 INCHES MADE IN U.S.A.
KEUFFEL & ESSER CO.

K+E 10 X 10 TO THE INCH 46 0703
7 X 10 INCHES MADE IN U.S.A.
KEUFFEL & ESSER CO.




PARTICLE SIZE (INCHES)
100
10
1.0
0.1
K+E SEMI-LOGARITHMIC 4S 6013
4 CYCLES X 70 DIVISIONS MADE IN U.S.A.
KEUFFEL & ESSER CO.

K-E 10 X 10 TO THE INCH 46 0700
7 X 10 INCHES MADE IN U.S.A.
KEUFFEL & ESSER CO.