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F (BS&K)

INDUCED POLARIZATION AND RESISTIVITY SURVEY
B. S. &. K. PROJECT
SILVER BELL DISTRICT
PIMA COUNTY, ARIZONA
FOR
THE HANNA MINING COMPANY

Canadian Aero Mineral Surveys Limited

HEINRICHS
GEOEXPLORATION CO.
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INDUCED POLARIZATION AND
RESISTIVITY SURVEY
BS AND K PROJECT
SILVER BELL DISTRICT
PIMA COUNTY, ARIZONA
FOR
THE HANNA MINING COMPANY

PROJECT 9618

CANADIAN AERO Mineral Surveys

INDUCED POLARIZATION AND
RESISTIVITY SURVEY

BS AND K PROJECT
SILVER BELL DISTRICT
PIMA COUNTY, ARIZONA

FOR

THE HANNA MINING COMPANY

PROJECT 9618

CANADIAN AERO *Mineral Surveys*

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1 Resistivity Contour Map

ABSTRACT

A strong IP response representing likely greater than 4% sulfides by volume has been outlined along the southeast side of the survey area. The response is most pronounced on line 52 where the zone has apparent characteristics of a flat-lying trend at a depth of 150'. Response decreases below 800' indicating an apparent thickness of 650'.

The anomaly lies parallel to the profile lines hence its lateral boundaries and depth extent cannot be accurately described by the present data. A suggested profile across the strike of the anomaly is considered a prudent step prior to locating a drill site.

Anomalous response characteristics along the south-east end of lines 48, 50, and 54 indicate that a broad area of response occurs to the south and east of the present survey area, and may extend easterly.

In the west half of the area a response layer occurs at an average depth of 1800' under the Tertiary-Cretaceous vol-canics. The response of 15 units could represent 1% sulfides by volume or a background varation due to change in rock type at that depth.

INDUCED POLARIZATION AND
RESISTIVITY SURVEY
BS AND K PROJECT
SILVER BELL DISTRICT
PIMA COUNTY, ARIZONA
FOR
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Part I

INTRODUCTION

At the request of Messrs Victor Mejia and Jackie Stephen of The Hanna Mining Company, personnel of Canadian Aero Mineral Surveys Limited were engaged in an induced polarization and resistivity survey in the Silver Bell District, Pima County, Arizona. The project was underway during the period from March 17 to April 10, 1969 and included 9 dipole-dipole profiles covering a total of 77,500 feet of surveyed line.

The survey was under the field direction of Avinash V. Hardas, Engineer for CAMS.

The major rock types that occur in the area include alaskite, granite, monzonite, Gila Conglomerate and the Tertiary-Cretaceous volcanics.

According to Victor Mejia, Gila Conglomerate occupies areas in sections 29, west 1/2 of 32, and southeast 1/4 of 31; flanked by intrusives (granite, monzonite, alaskite) to the east and Tertiary-Cretaceous volcanics to the west. Small patches of limestone occur as irregular isolated outcrops near the BS and K mine. A contact between granite and dacite porphyry is noted to occur between lines 52 and 54.

PURPOSE OF THE SURVEY

The purpose of the survey was to locate areas of mineralization near the BS and K mine and to describe the extent of related anomalies as being additional occurrences of sulphide mineralization. Also to determine if possible the thickness of overlying Tertiary-Cretaceous volcanic rocks in the western half of the project area; and the occurrence of IP response at depth below the volcanics.

SURVEY PROCEDURES

The induced polarization and resistivity measurements were made in the time-domain made of operation. A conventional system of measurements which uses a time cycle of 2.0 seconds "on" and 2.0 seconds "off", a 2.0 seconds "on" and 2.0 seconds "off" (current reversed), was employed.

The commencement of the measurement of the secondary voltage is delayed by 0.45 seconds to avoid coupling and other transient effects. The integration is preformed during the period from 0.45 seconds to 1.10 seconds after the cessation of current.

To conform to a standard presentation, the integral time constant is adjusted to give induced polarization readings equivalent to those obtained with transmitter cycles of 3.0 seconds "on" and 3.0 seconds "off" with integration of the secondary voltage during the first second of the "off" period.

Throughout the survey except line 44 and 46 a conventional inline dipole-dipole array of seven electrodes was used. For additional data coverage on lines 44 and 46, two additional electrodes were employed, there by obtaining additional coverage to the southeast. A dipole length of 1000' was used for the survey. This dipole length was recommended for greater penetration and greater lateral coverage.

Measurements were made for dipole separation factors (n) 1 to 6. Potential dipole setups occupied positions on both sides of the current-electrode setup. This resulted in obtaining a total line coverage approximately nine times the dipole length where 7 electrodes were used.

Apparent polarization responses is in units of millivolts-seconds per volt, or milliseconds, and apparent resistivity is in units of ohm-meters.

Part II

DISCUSSION OF RESULTS

Line 38: This line clearly indicates a change in rock type at about 500 feet southeast of the center; To the southeast the resistivity is of the order of 40-70 ohm-meters as against 130 to 400 ohm-meters to the northwest. The data is indicative of southeast dipping contact.

Pulse response is of the order of 4.0 to 8.0 milliseconds and does not vary across the resistivity contact. The data is distorted due to fence-power line effects and therefore does not yield an intelligent interpretation.

Line 40: Low resistivity (35-70 ohm-meter) is noticed on south side of the line. Resistivity to northwest ranges from 100-500 ohm-meters. The point at which a change in rock type occurs is noted at 500-700 feet southeast of the center.

Changeability is of the order of 4.0 to 13.0 milliseconds with a slight increase in response at depth. However, it is believed the values are too low to indicate presence of sulphides in economic quantities.

Line 42: Again on this line low resistivity rock occurs to the southeast and high resistivity rock to northwest. This change in physical characteristics of the rocks occurs at 600' southeast from center. To the southeast the resistivity is of the order of 40-60 ohm-meters and to the northwest 100-400 ohm-meters.

Pulse response ranges from 4.8 to 10.2 milliseconds increasing with depth, but again likely represents a high background response or at least minor (less than 1.5% by volume) sulfides at depth.

Line 44: Resistivity contact is noted 500' northwest from the center with high resistivity to northwest and low resistivity to southeast.

Average pulse response of 2.8 to 5.0 is present with a slight increase in response to 10.0 at depth. This increase could be due to a normal background response at depth with no indication of the presence of substantial amounts of sulphides.

Line 46: A reverse situation occurs on this line where high resistivity occurs to the southeast and low resistivity to the northwest. The high resistivity is related to the granite that outcrop to the southeast. The resistivity of the volcanics and sediments to the northwest is low, varying from 7 to 40 ohm-meters.

Due to the very low resistivity of the rocks, signal could not be obtained at "n's" greater than 3. thus the depth penetration is limited. Background response varies from 2.0 to as much as 15.4 milliseconds.

Line 48: A resistivity contact occurs at the center of the line. Rocks of high resistivity (100-700 ohm-meters) extend southwards from the center and the low-resistivity rocks extend nortwards from the center. Rocks to the northwest have resistivity of 16 to 90 ohm-meters.

Average chargeability background response is 6.0 milliseconds with sharp increase in response up to 27.0 milliseconds to the southeast. (Fence interference may cause the few high readings to the south).

The high pulse response is related to the high resistivity indicative of sulfides in granites.

Another gradual increase in response at depth is also noted in the low resistivity rocks to the northwest. Due to the very low resistivity of the rocks full line coverage at depth could not be obtained.

Line 50: A very clear indiaction of a lithologic contact is noted on this line at 1000' northwest from the center.

Resistivity of rocks to the southeast range from 184 to 550 ohm-meters and to the northwest from 14 to 150 ohm-meters. The high resistivity zone is associated with high pulse response of 20 to 67 milliseconds and is indicative of 2-3 percent sulphides by volume at a depth less than 400 feet from surface.

Surface pulse response to the northwest is of the order 2.6 to 6.4 milliseconds but increases slightly to more than 15.0 milliseconds at depth. This response zone may be in part due to the power line or fence that tranverse the line.

The low resistivity relates to a broad near-vertical north-south striking of alluvium or Gila Conglomerate zone and is devoid of any substantial amounts of sulphides.

Line 52: Interpretation of resistivity data of this line is a little complex as surface resistivity variation appear related to complex geology. However, in general, rocks of less than 50 ohm-meters resistivity extend northwards from C_3 and are characterized by low pulse response.

Rocks that crop out on the southeast side of the line have resistivity of 90 to 650 ohm-meters and are associated with 10.0 to 80.0 milliseconds response. The anomalous response is related to a broad zone of moderately high resistivity and

and is indicative of 3-4 percent sulphides by volume at a depth less than 200' from the surface. Line 52 does not extend far enough to cover the southern boundary of the zone.

Line 54: Resistivity varies from 190 ohm-meters to as much as 600 ohm-meters with no apparent indication of a lithologic contact.

Background pulse response of 3.0 to 10.0 is observed on the northside of the lines but increases at depth to more than 40.0 milliseconds to the southeast. This relates to a zone of anomalous responses that extends southward from C_7. This line runs parallel to the main zone of response and hence the actual true response obtained from the anomalous zone is not clear. However, there is a slight indication from the resistivity pattern that the response zone may extend to the east at depth. We feel that in order to actually define the anomaly a line at right angles to the strike of the zone should be surveyed.

Part III

INTERPRETATION

The accompanying contour maps of chargeability and resistivity provide a good picture of distribution and extent of response material, i.e. possible sulphide mineralization, and the related rock types in the area.

First look at the resistivity contour map indicates that 3 major rock types occur in the area which govern the resistivity pattern. They are granites, volcanics, and alluvium (Gila Conglomerate?).

In sections 30 and 31 the relatively high resistivity of 100 to 200 ohm-meters is related to the Tertiary-Cretaceous volcanics. The resistivity data indicates increased resistivities at a depth of 1000' plus. The resistivity of the lower layer is as much as 1000 ohm-meters-atypical of volcanic rocks.

A high resistivity anomaly of approximately 1000 ohm-meters occurs in the east 1/2 of section 32, closely associated with the BS and K mine. This anomaly is related to granite which crops out in the area. We suspect silicification (alteration) of the granite has resulted in this abnormal granite resisitivity.

Resistivities in the vicinity of the high IP response on line 52 are locally lower (100 to 200 ohm-meters) than the adjacent resistivities (200-400 ohm-meters) on line 50 and 54.

The alteration and mineralization of the granite is believed the cause of the lower resistivity here.

Extremely low resistivity of less than 100 ohm-meters in a north-south trend through sections 29, east half 31, and 32 is related to alluvium and/or Gila Conglomerate.

Effective penetration has been drastically reduced in the area of low resistivity due to loss of signal at the larger electrode spacings. This condition is most severe on line 46 where our ability to see below 1000' is nil.

The chargeability contour map provides a good picture of the distribution of the response material in the area.

High response material of 80 milliseconds occurs in section 28 and 33. The anomaly is controlled to the west by a strong north-south contact (fault). This contact feature is common to both the IP and resistivity data and represents the boundary between the mineralized intrusives on the east and the alluvium (Gila?) on the west.

The BS and K mine occurs along this contact or fault zone. The alluvium or Gila Conglomerate is marked by the area of response of less than 2.0 milliseconds on the IP contour map.

The Tertiary-Cretaceous volcanics to the west have response values of 4 to 8 milliseconds near surface. An increase in response to 15 milliseconds at depths greater than 1800' is associated with higher resistivities and could be related to rocks other than volcanics. This increase in response at depth could be due to high background response or at best minor sulfides of less than 1.5 percent by volume.

CONCLUSIONS

A zone of substantial IP response occurs on the southeast half of line 52. A calculated response of 80 to 100 milliseconds in the rock at depths from 150' to 800' is related to greater than 4 percent sulfides by volume. The zone lies parallel to our present profiles and a line across the strike of the zone is recommended to locate the northeast and southeast contacts.

Substantial response occurs off the southerly ends of lines 46, 48, 50, 52 and 54 indicating a reasonably large zone of response adjacent to the southeast of the property.

A moderately weak response of 15 milliseconds occurs at depths of about 1800' in the area of the Tertiary-Cretaceous volcanic to the west. An increase in resistivity occurs at depths greater than 1000' in this area. We suspect these interfaces indicate a bottom depth to the volcanics, and a high resistivity rock at depth that may have very minor sulfide mineralization.

RECOMMENDATIONS

A drill hole is proposed to test the response zone outlined on line 52. A minimum depth of 1000' is suggested to test the response characteristics. We strongly recommend a profile be run across the zone to better define the boundaries of the anomaly prior to locating a drill site.

Locally the anomalous response appears to be closing out to the northeast. However, further reconnaissance in the area to the north and east should be considered where geologic conditions appear more favorable.

Some consideration might be given to testing the high resistivity, moderate IP response that occurs below the Tertiary-Cretaceous volcanics in sections 30 and 31. However, there is little reason to expect more than 1 percent sulfides (related to 15 milliseconds response at depth) where a hole to 2000' would be necessary to test the physical property characteristic.

Respectfully Submitted

Avinash V. Hardas, M.S. per wgw
Geologist-Geophysical Engineer

W. Gordon Wieduwilt
Geophysicist

May 2, 1969
Tucson, Arizona

Part IV

APPENDIX I

The following personnel of CANADIAN AERO MINERAL SURVEYS LIMITED were engaged in an induced polarization - resistivity survey conducted on behalf of THE HANNA MINING COMPANY on their property in the Silver Bell District, Pima County, Arizona.

		No. of Man Days
Avinash V. Hardas Engineer, CAMS Tucson	Field Office	20 6
Gary Carpenter Helper-Tucson	Field	20
Shantikumer Toprani Helper-Tucson	Field	16
David Hunter Helper-Tucson	Field	9
Christian Dahlberg Helper-Tucson	Field	3
Robert Sigafus Helper-Tucson	Field	1
Total number of man hours		75

APPENDIX II

APPLICATION OF INDUCED POLARIZATION METHOD

The induced polarization method is basically a volume detecting technique. Effective penetration is governed by the size of target where normally a large volume of polarizable material at depth is required to give measureable response at surface. This method is relatively sensitive and is capable of detecting as little as 1% by volume of metallic sulfides. Because polarization is essentially a "particle surface" phenomenon, the induced polarization effects from a given percentage of metallic sulfides generally increases as particle size is decreased. This characteristic makes this technique especially suitable to exploration for disseminated sulfide occurrences.

Sulfides of metallic lustre produce IP effects i.e., bornite, chalcopyrite, chalcocite, pyrite, pyrrhotite, arsenopyrite, molybdenite, etc., but not sphalerite. Besides sulfides, some metallic oxides like magnetite and pyrolusite also give rise to IP effects.

Apart from sulfides and oxides, certain minerals with unsatisfied basal charged lattice, when current is applied to the ground, develope a charged double layer which acts as a leaky condenser and gives rise to IP effects. Certain of the clay mica minerals are active in this sense with montmorill-onite and vermiculite exhibiting by far the greatest response. Bentonitic tuff is exceptionally active to IP, while kaolines, chlorites, muscovites, and biotites are not generally active.

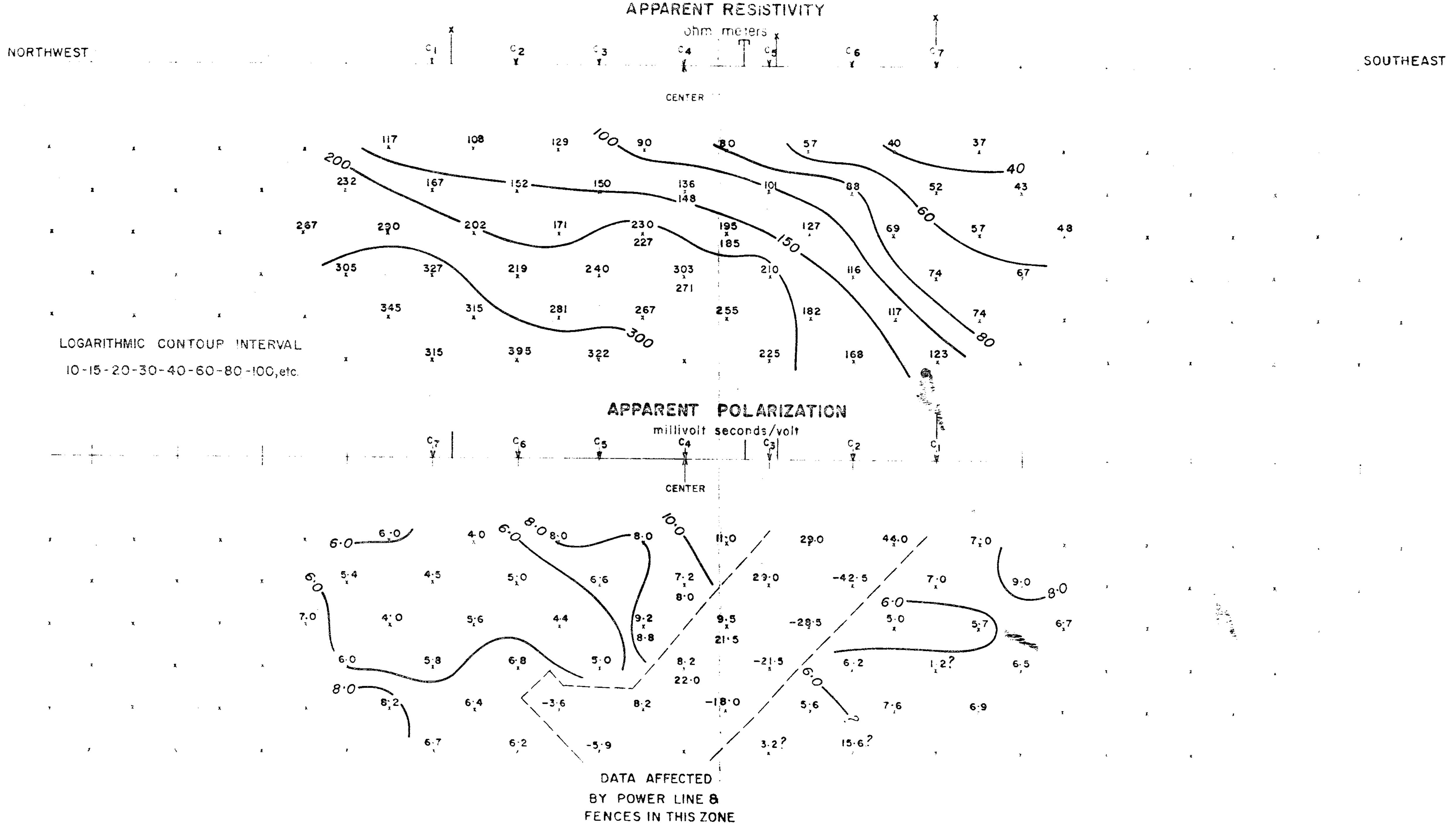
Although considerable study has taken place, this method has not yet been improved to differentiate the IP effects arising from metallic sulfides, oxides, graphite, or clay occurrences.

TIME DOMAIN INDUCED POLARIZATION AND RESISTIVITY SURVEY

BS & K PROJECT–SILVER BELL DISTRICT–PIMA COUNTY, ARIZONA

FOR

THE HANNA MINING COMPANY



DIPOLE DIPOLE ARRAY

CURRENT DIPOLE POTENTIAL DIPOLE

PLOT ˣPOINT

LINE 38

LOOKING. . . . .NE

DIPOLE
LENGTH . . . . .1000'

DATE: MAR.18,1969

LEGEND

FENCE

PIPELINE

POWERLINE

CANADIAN AERO
Mineral Surveys LTD
OTTAWA, ONT. CANADA
TUCSON, ARIZONA U.S.A.

C.A.M.S. 9618

TIME DOMAIN INDUCED POLARIZATION AND RESISTIVITY SURVEY

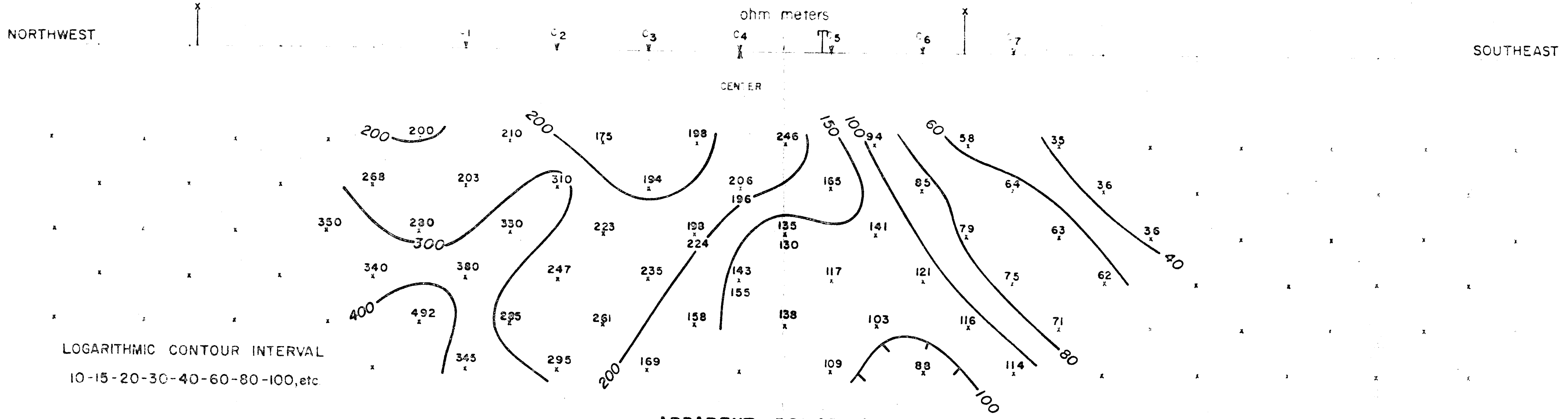
B S & K PROJECT–SILVER BELL DISTRICT–PIMA COUNTY, ARIZONA

FOR

THE HANNA MINING COMPANY

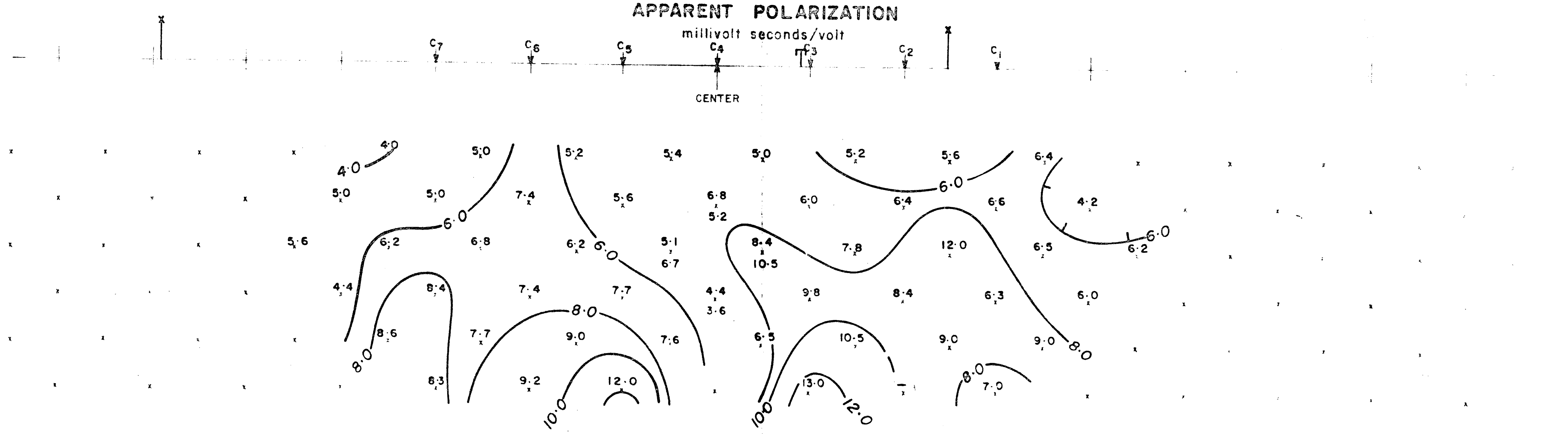
APPARENT RESISTIVITY

ohm meters



APPARENT POLARIZATION

millivolt seconds/volt



DIPOLE DIPOLE ARRAY

CURRENT DIPOLE — POTENTIAL DIPOLE

PLOT POINT

LINE 40

LOOKING . . . . . NE

DIPOLE LENGTH . . . . 1000'

DATE: MAR. 20, 1969

LEGEND

FENCE

PIPELINE

POWERLINE

CANADIAN AERO Mineral Surveys LTD

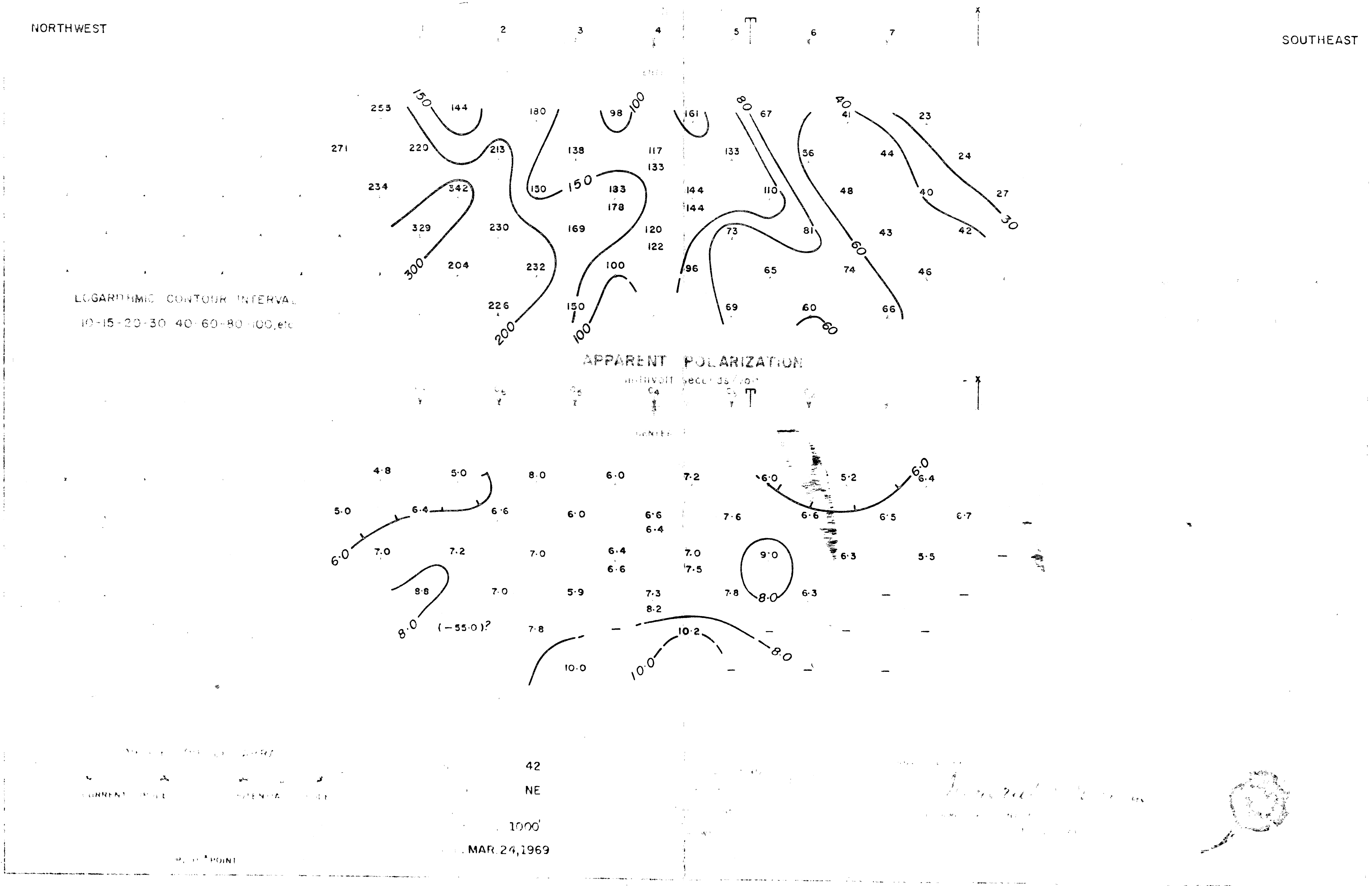
OTTAWA, ONT CANADA

TUCSON, ARIZONA U.S.A.

CAMS 9618

BS & K PROJECT–SILVER BELL DISTRICT–PIMA COUNTY, ARIZONA

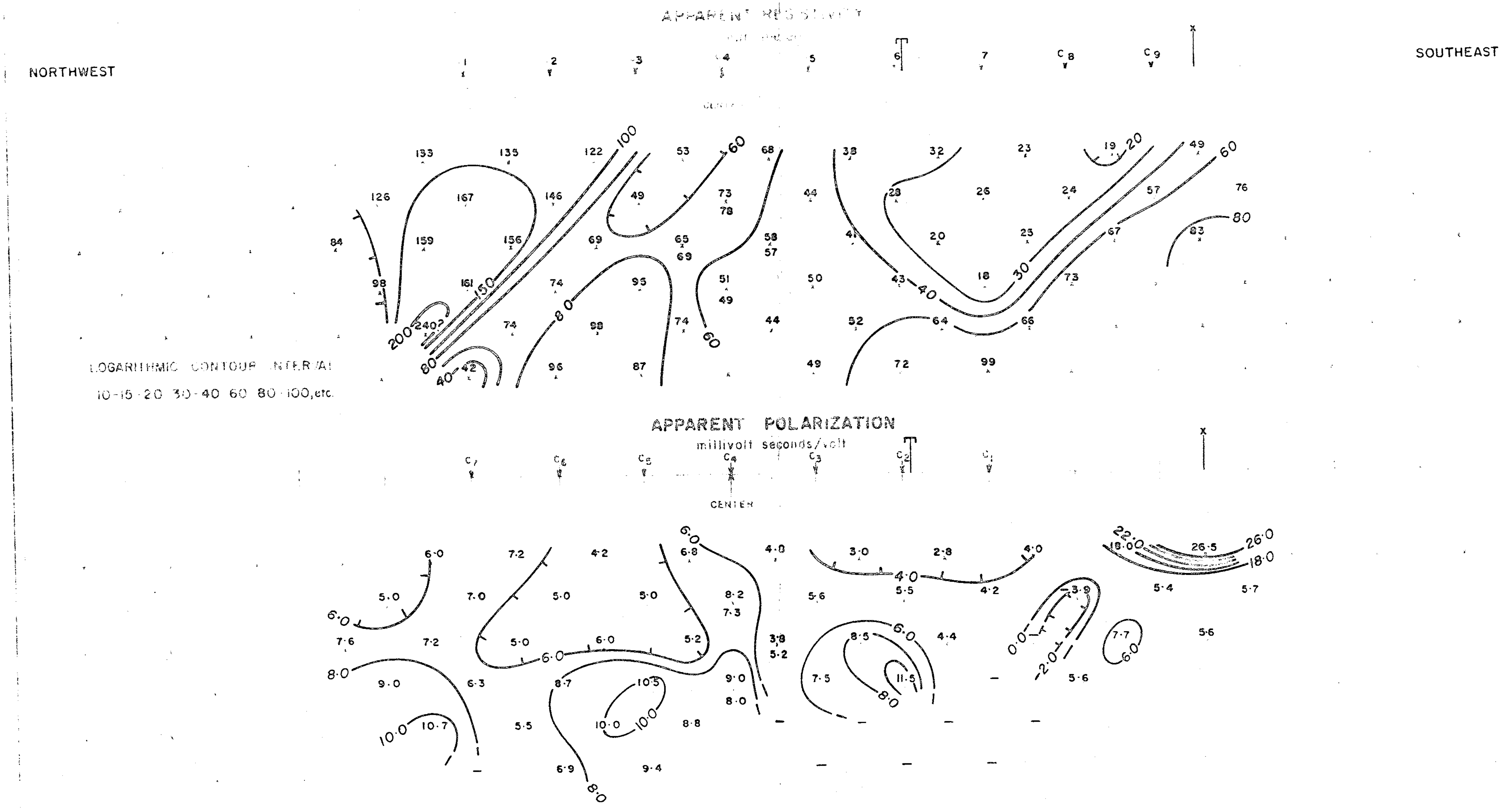
THE HANNA MINING COMPANY



TIME DOMAIN INDUCED POLARIZATION AND RESISTIVITY SURVEY

BS & K PROJECT-SILVER BELL DISTRICT-PIMA COUNTY, ARIZONA

THE HANNA MINING COMPANY



DIPOLE DIPOLE ARRAY

CURRENT POLE POTENTIAL

PLOT POINT

. 44

LOOKING . . . . . NE

DIPOLE LENGTH . . . 1000

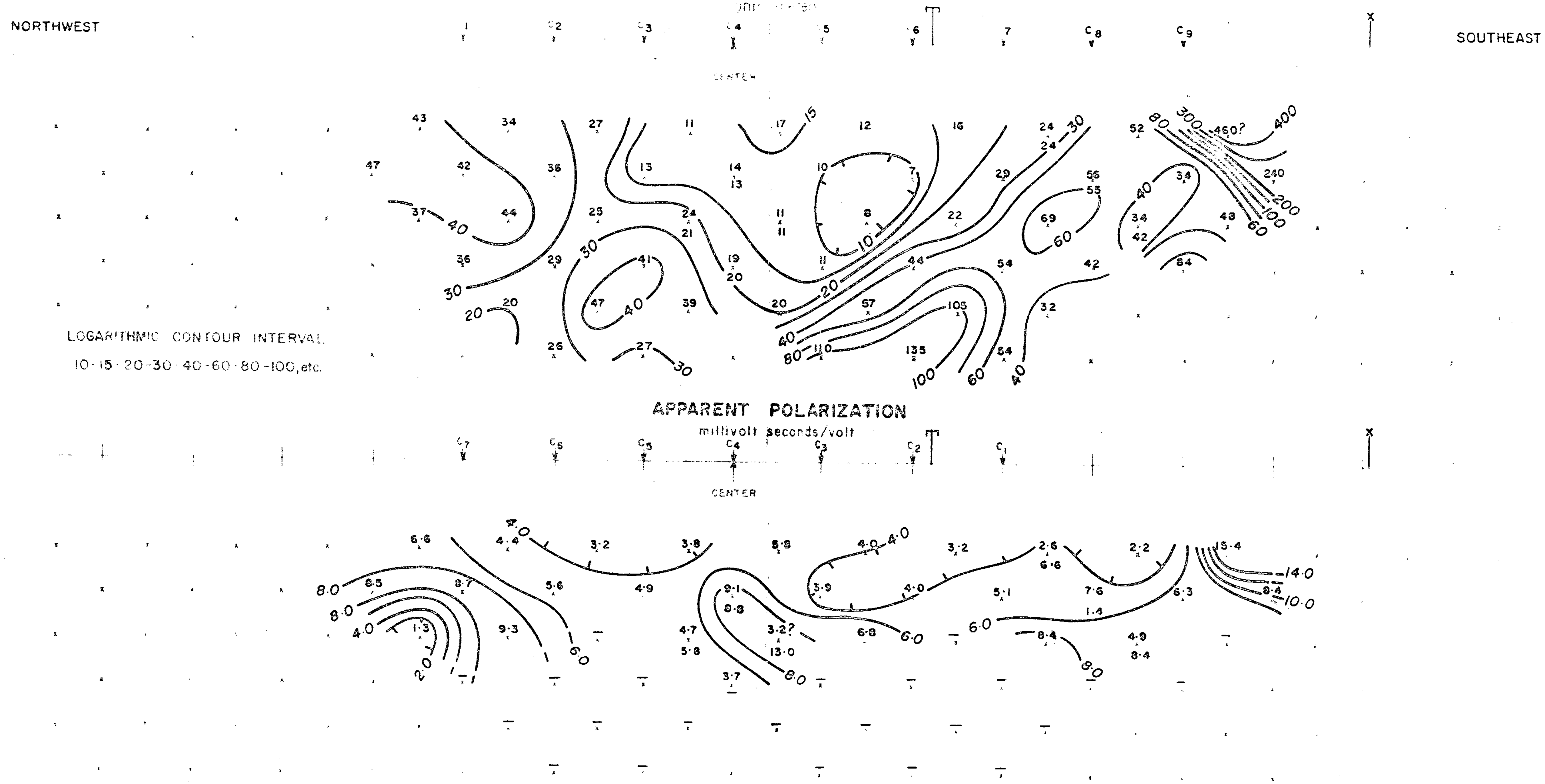
MAR.26,1969

9618

TIME DOMAIN INDUCED POLARIZATION AND RESISTIVITY SURVEY

BS & K PROJECT - SILVER BELL DISTRICT - PIMA COUNTY, ARIZONA

THE HANNA MINING COMPANY



DIPOLE DIPOLE ARRAY

CURRENT DIPOLE POTENTIAL DIPOLE

PLOT POINT

LINE 46
LOOKING . . . . . NE
DIPOLE LENGTH. . . . 1000
DATE MAR.31,1969

LEGEND
FENCE
PIPELINE
POWERLINE

CANADIAN AERO
Mineral Surveys LTD.
OTTAWA, ONT CANADA
TUCSON ARIZONA U.S.A.

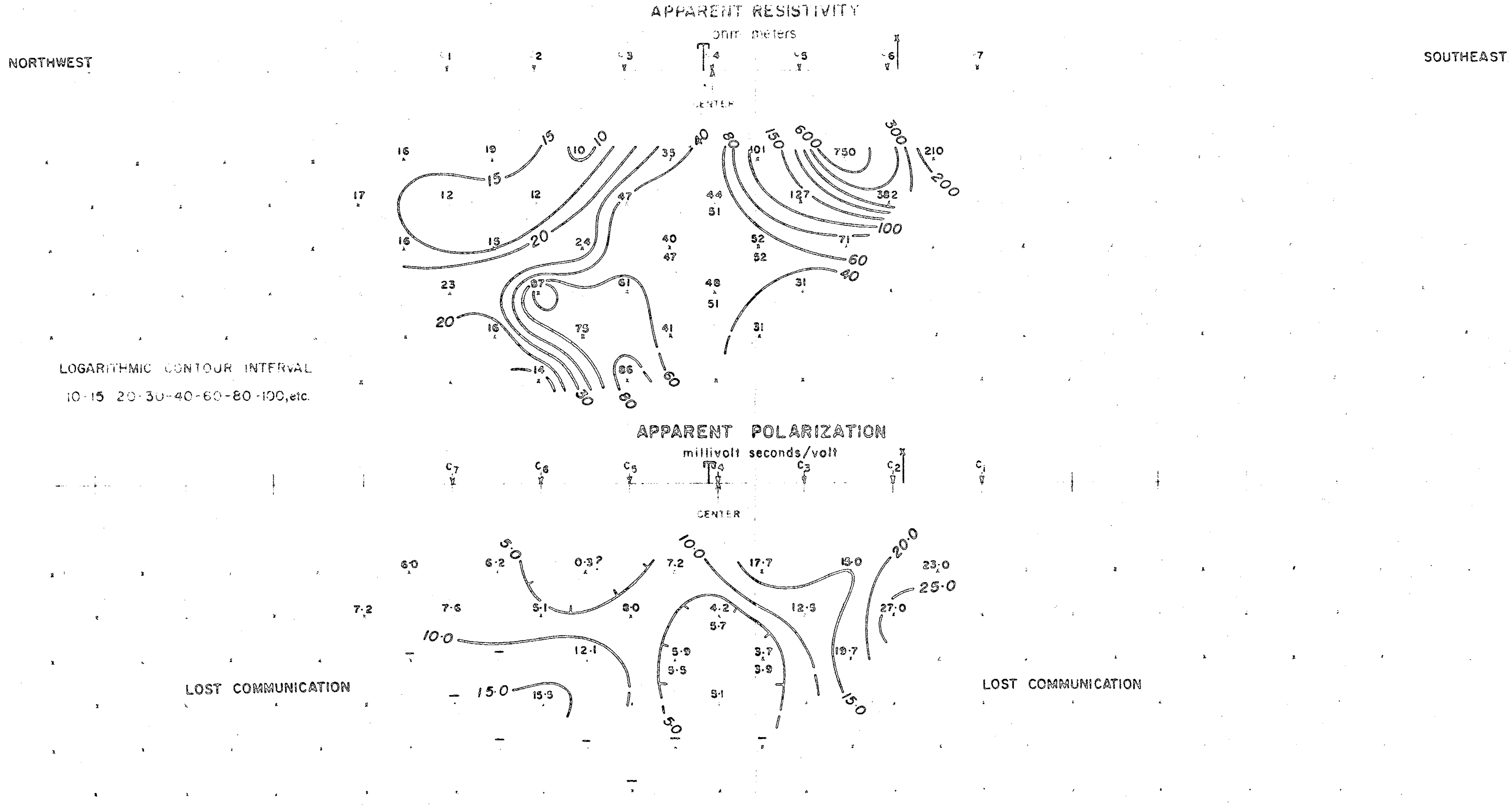
CAMS 9618

TIME DOMAIN INDUCED POLARIZATION AND RESISTIVITY SURVEY

BS & K PROJECT-SILVER BELL DISTRICT-PIMA COUNTY, ARIZONA

FOR

THE HANNA MINING COMPANY



DIPOLE DIPOLE ARRAY

CURRENT DIPOLE POTENTIAL DIPOLE

PLOT POINT

LINE 48

LOOKING . . . . . NE

DIPOLE LENGTH. . . . 1000

DATE. AP., 3, 1969

LEGEND

FENCE

PIPELINE

POWERLINE

CANADIAN AERO Mineral Surveys LTD.

OTTAWA, ONT. CANADA

TUCSON ARIZONA U.S.A.

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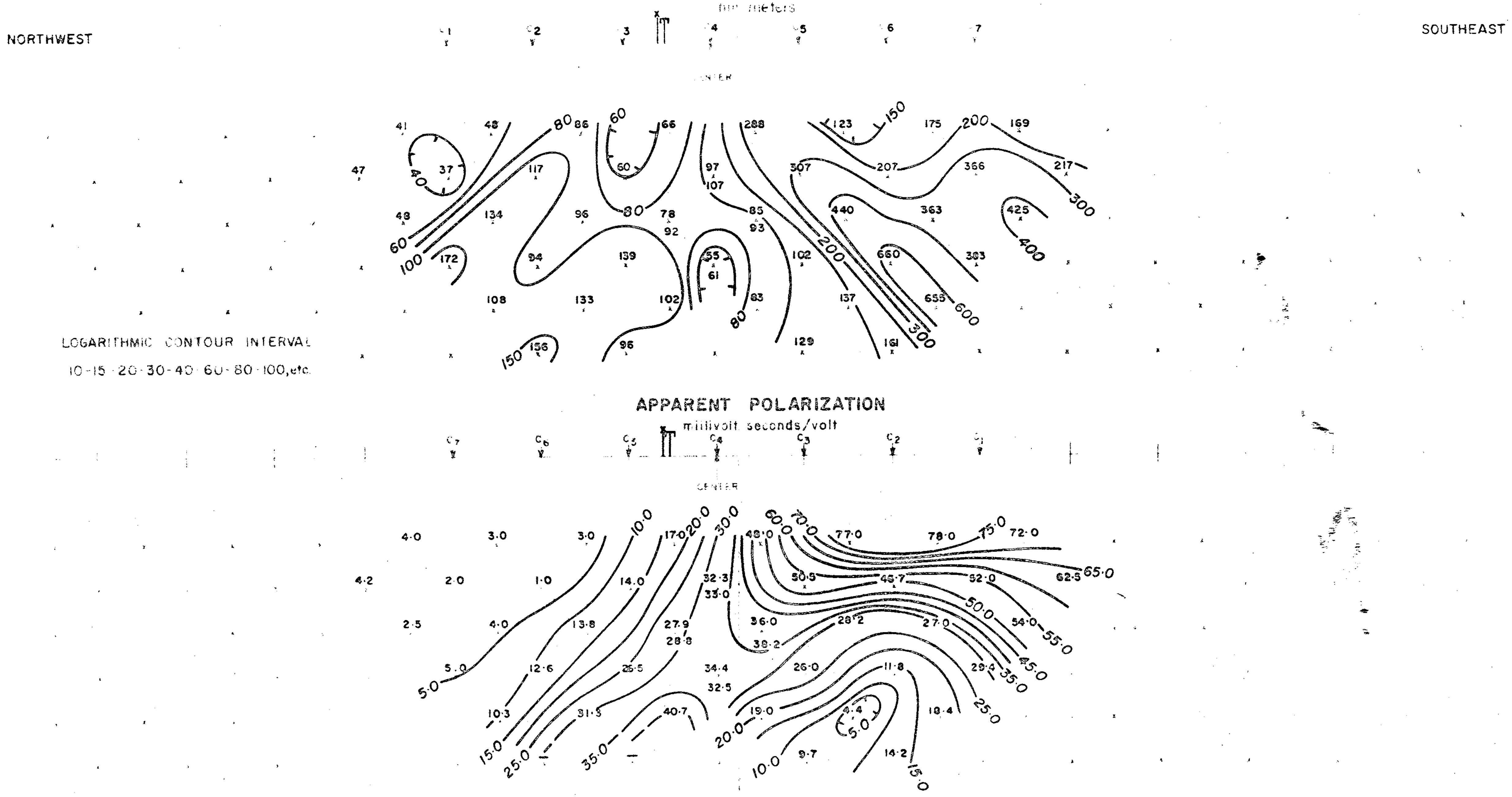
TIME DOMAIN INDUCED POLARIZATION AND RESISTIV[illegible]

BS & K PROJECT-SILVER BELL DISTRICT-PIMA COUNTY, ARIZONA

THE HANNA MINING COMPANY



DIPOLE DIPOLE ARRAY

CURRENT DIPOLE — POTENTIAL DIPOLE

PLOT × POINT

LINE 52

LOOKING. . . . .NE

DIPOLE LENGTH. . . .1000'

DATE: AP.,7, 1969

LEGEND

FENCE

PIPELINE

POWERLINE

CANADIAN AERO Mineral Surveys LTD.

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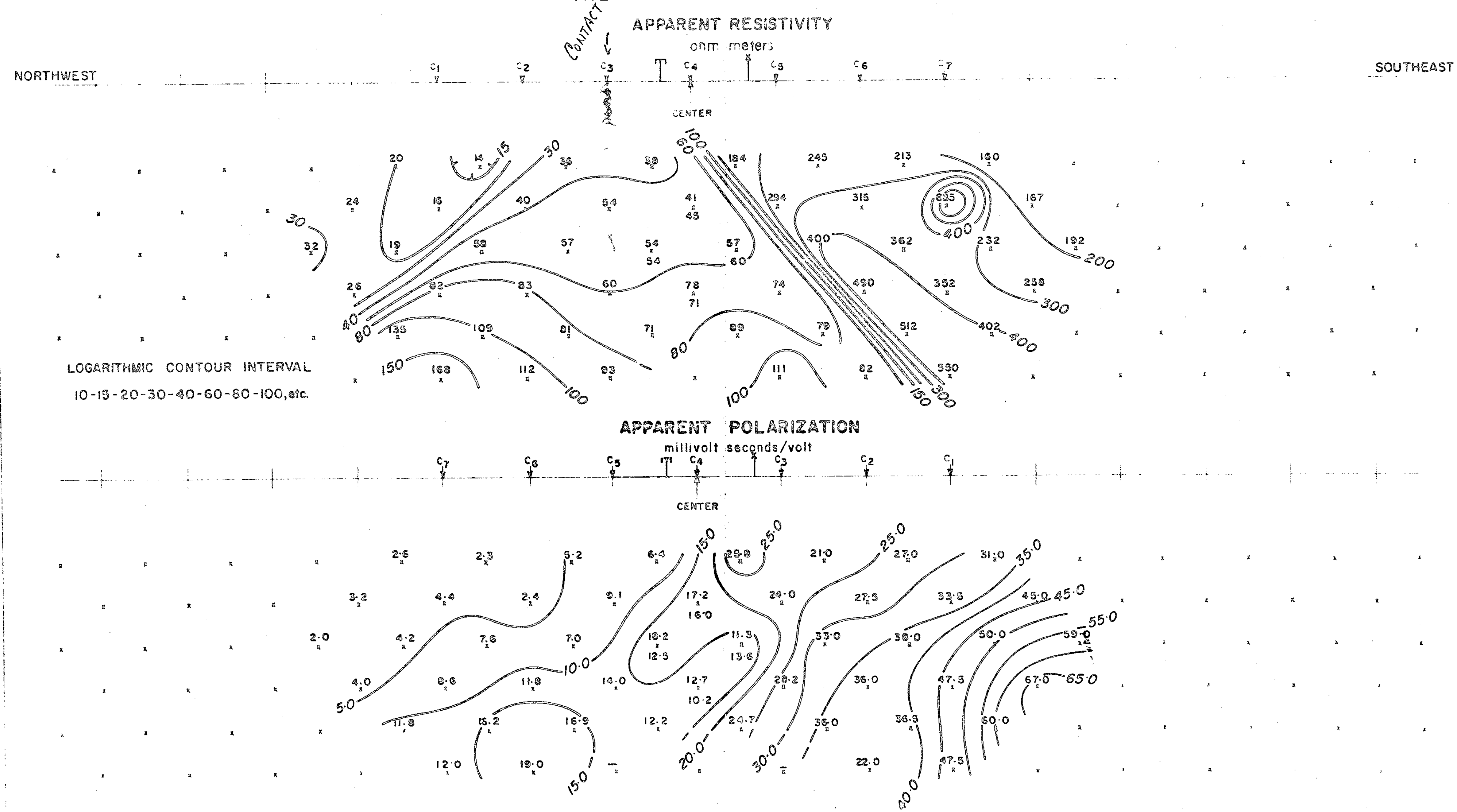
TIME DOMAIN INDUCED POLARIZATION AND RESISTIVITY SURVEY

BS & K PROJECT-SILVER BELL DISTRICT-PIMA COUNTY, ARIZONA

FOR

THE HANNA MINING COMPANY



DIPOLE DIPOLE ARRAY

CURRENT DIPOLE — na — POTENTIAL DIPOLE

PLOT POINT

LINE: 50

LOOKING: . . . . NE

DIPOLE LENGTH: . . . 1000'

DATE: AP 4, 1969

LEGEND

FENCE

PIPELINE

POWERLINE

CANADIAN AERO Mineral Surveys LTD

OTTAWA, ONT CANADA

TUCSON, ARIZONA U.S.A.

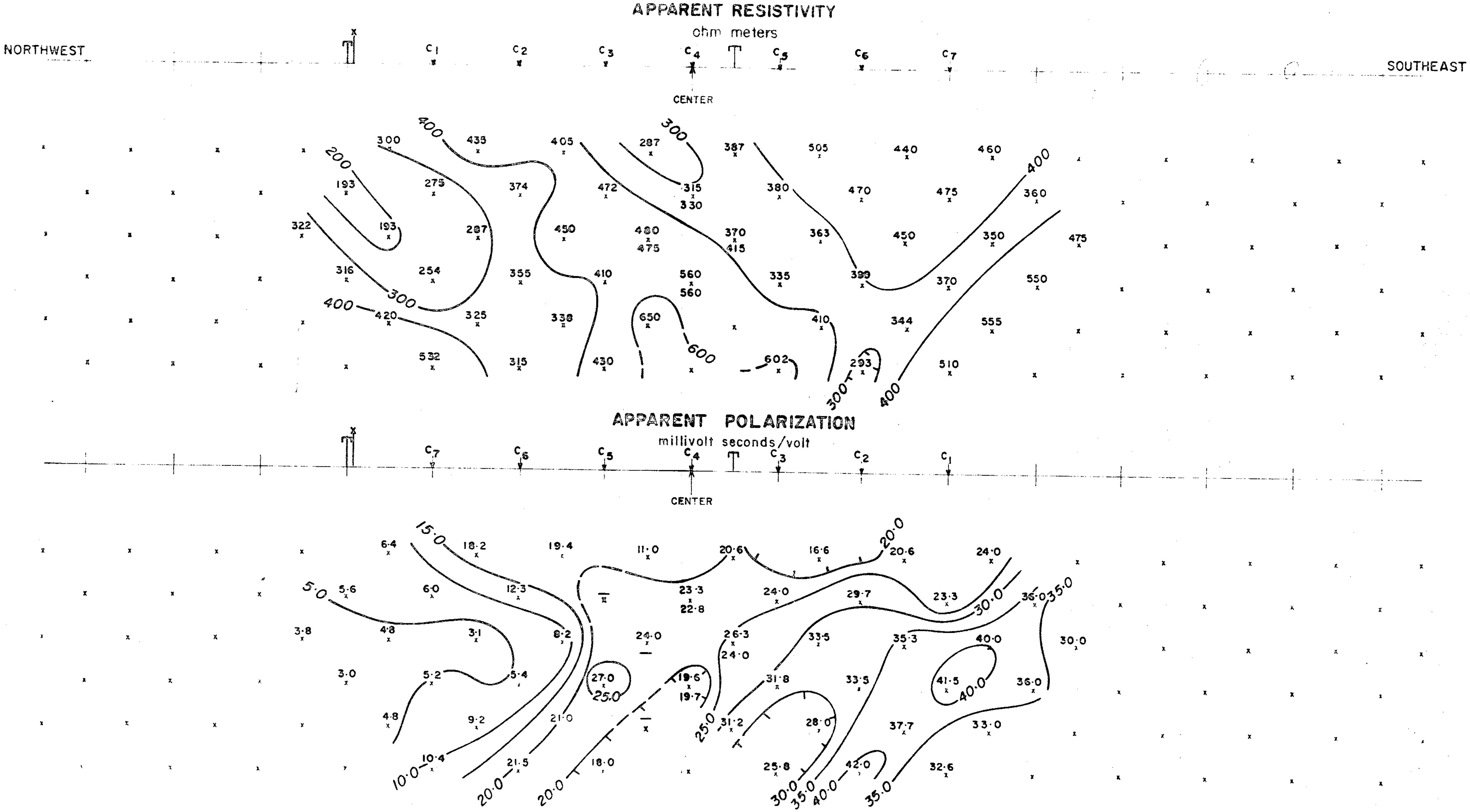
C.A.M.S 9618

TIME DOMAIN INDUCED POLARIZATION AND RESISTIVITY SURVEY

BS & K PROJECT-SILVER BELL DISTRICT-PIMA COUNTY, ARIZONA

FOR

THE HANNA MINING COMPANY



DIPOLE DIPOLE ARRAY

a na a

CURRENT DIPOLE POTENTIAL DIPOLE

PLOT X POINT

LINE: 54

LOOKING: . . . . NE

DIPOLE LENGTH: . . . 1000'

DATE: AP. 9, 1969

LEGEND

FENCE . . . .

PIPELINE . . .

POWERLINE . . . .

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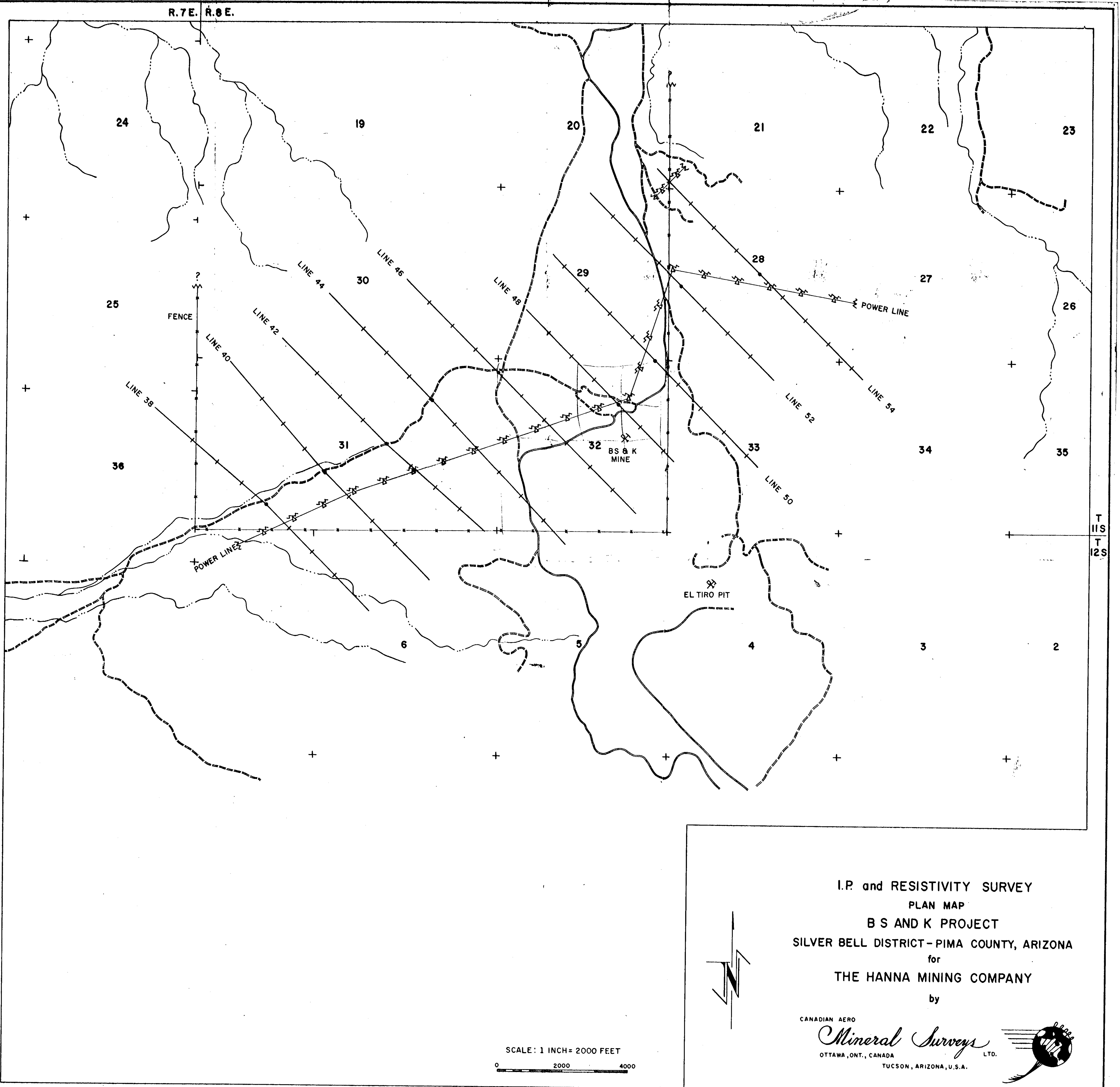
C.A.M.S. 6618



I.P. and RESISTIVITY SURVEY
PLAN MAP
B S AND K PROJECT
SILVER BELL DISTRICT - PIMA COUNTY, ARIZONA
for
THE HANNA MINING COMPANY
by
CANADIAN AERO
Mineral Surveys LTD.
OTTAWA, ONT., CANADA
TUCSON, ARIZONA, U.S.A.

C.A.M.S. 9618

I.P. and RESISTIVITY SURVEY

CHARGEABILITY CONTOUR PLAN

B S & K PROJECT

SILVER BELL DISTRICT – PIMA COUNTY, ARIZONA

for

THE HANNA MINING COMPANY

by

CANADIAN AERO Mineral Surveys LTD.
OTTAWA, ONT. CANADA
TUCSON, ARIZ. U.S.A.

C.A.M.S. 9618


R. 7 E. | R. 8 E.
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LINE 48
LINE 50
LINE 52
LINE 54
FENCE
POWER LINE
POWER LINE
B S & K MINE
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IP and RESISTIVITY SURVEY
RESISTIVITY CONTOUR PLAN
B S & K PROJECT
SILVER BELL DISTRICT – PIMA COUNTY, ARIZONA
for
THE HANNA MINING COMPANY
by
CANADIAN AERO Mineral Surveys LTD.
OTTAWA, ONT. CANADA
TUCSON, ARIZ. U.S.A.
SCALE: 1 INCH = 500 FEET
500 0 500 1000 1500
C.A.M.S. 9618

PRELIMINARY EVALUATION
FOR POSSIBLE ACQUISITION
AND OPERATION OF THE

B. S. & K. MINING COMPANY PROPERTIES

PRELIMINARY EVALUATION

FOR POSSIBLE ACQUISITION

AND

OPERATION

OF THE

B. S. & K. MINING COMPANY PROPERTIES

INTRODUCTION

THIS REPORT WAS PREPARED BY

CLYDE E. OSBORN, E.MET.
Professional Engineer

Technical Director, Natural Resources Office, Essex International, Inc.

Tucson, Arizona

This is a preliminary report submitted as a guide for management in making a decision on how to proceed with the project. Certain statements were accepted as to the tonnage and grade of ore on the stipulation that these figures would have to be confirmed by actual field work. Further, the metallurgy and the subsequent processing is based on a report that can only be used as a guide in arriving at preliminary plant and operating costs. Considerable more test work must be done on representative samples. The mine owners have indicated a willingness to allow six months to make the necessary studies to confirm:

1-The tonnage and grade ore.

2-That the ore is amenable to leaching techniques.

3-That the cost estimates will be equal to, or better than those included in this report.

Signed Clyde E Osborn
Clyde E. Osborn, E.Met.

Date March 12, 1970

INTRODUCTION

The B. S. & K copper ore body under consideration, is a blanket type deposit containing a mixture of oxide copper, chalcocite and chalcopyrite minerals. According to data supplied by Mr. Abe Kalaf of the B.S.& K Mining Company, the estimated drilled reserves are as follows:

5,000,000	tons	assaying	0.60% Cu.
7,000,000	"	"	0.15% Cu.

The waste overlying the ore blanket is estimated to be 5,200,000 tons. This material contains some oxide copper but on the whole would not average more than 0.08% Cu.

The ore is up to 50' in thickness and does not extend more than 200' below the surface, this will lend itself nicely to open cut mining methods.

Preliminary tests made by the Duval Corporation in their Tucson labs indicated an overall copper extraction of 78% from a composite of samples taken from exploration drill holes. This test is reported in a company memorandum entitled "Bacterial Leaching of B.S.& K. Exploration Composite", dated November 22, 1965, Exhibit A

In subsequent discussions with the Lab Technicians, they express the firm opinion that the minerals will respond equally as well to a sulphuric-acid ferric-sulphate leach. This will require confirmation as proposed and outlined later in this report.

A geologic description and location of the property is attached. See Exhibits

Also attached is an analysis of the ore reserves and other comments by Mr. E.Grover Heinrichs, V.P. of Heinrichs Geoexploration Company.

The B.S.& K Mine has been an operating mine in the past. As a result there are some assets which will accrue to this newer project.

Access Roads

There are good county roads into the property from Red Rock to the east and from Silver Bell to the south west. Roads have been developed on the property in conjunction with exploration drilling programs. For the most part, these roads are in good repair.

Water

One water well has been developed. It is 300 ft. deep and cased. It has delivered 150 gpm over a long period of time during past operation with only a few inches of draw down.

Power

A 3 phase 14,400 Volt power line supplies electric power to the mine. The B.S.& K. Mining Company has picked up the power at the edge of the property and extended the lines approximately 3 miles into the property where they have installed 3, 650 KVA transformers and a distribution system. This has an estimated value of $125,000.

Housing

The old camp site is in need of extensive repairs. However, the power and water distribution systems are in good repair. An excellent combination office and residence is situated on the property. Floor area of 3000 sq.ft. Modern in all respects. This building is not included in the offer but it can be bought at an appraised value, or leased.

Equipment

The principal piece of equipment available to the project is an air compressor. This is an Atlas Copco 900 c.f.m. 100 p.s.i. piston type compressor, complete with a 100 h.p. Westinghouse Motor and drive and switching equipment. All in excellent condition. Includes a receiver. This has a present value of $15,000.

Introduction #4

Other

The B.S.& K. Mining Company leases a rail side on the Southern Pacific at Red Rock, a distance of 19 miles from the mine. This siding is 400 ft. long and is equipped with a loading ramp and drop bridge. This lease cost $300 per year.

SUMMARY

SUMMARY

This preliminary study of a plan to put the "blanket" ore body of the B.S. & K. Mining Company, situated on their New York claims, into production indicates that a reasonable return on investment can be obtained if the following conditions are satisfied:

1. The tonnage and grade of ore measure up to figures represented in this report.
2. The metallurgy proves to be satisfactory.
3. The price of copper remains at 56¢/# or higher.

Conditions 1 and 2 must be checked out before any other commitments can be made. A period of six months has been allowed to carry out the necessary work.

During the six months, some engineering should be done to confirm the costs which have been estimated in this report. It is believed that the costs in this report are reasonably close and will stand up to a more detailed study. Time did not permit a detailed study of a mining method. The cost of 40¢ per ton was arrived at by a study of recent reports and by talking with some of the operators in the Tuçson area. The capital cost of the vats was factored from cost studies of a similar operation near Parker, Arizona. The writer was project engineer for the engineering company employed on this

Summary #2

job. The capital cost of the LIX and electrowinning plant was a budget estimate from Holmes and Narver, Inc., and confirmed in a subsequent conversation with engineers at Hazen Research. Operating cost for the LIX-Electrowinning process includes cost for leaching the low grade mine dumps from which an estimated 5,000,000 lbs. of copper will be extracted.

The cost for the work to confirm conditions 1 and 2 as stated above is estimated at $100,000.00.

PLAN OF OPERATION

1-Preliminary Investigation

2-Mining Operation

3-Processing (milling) Operation

Preliminary Investigation

Having spent several hours on the property with Mr. Abe Kalaf and Mr. Grover Heinrichs, Mr. Heinrichs and the writer suggest that 20 holes be drilled for the purpose of obtaining bulk samples for metallurgical testing and to further evaluate the ore body and confirm ore reserve figures. This work to be done on contract.

Six months time should be allowed for the drilling, sampling and metallurgical investigation. The drilling program will require three months. Sufficient sampling would be done during the first month to permit some metallurgical work to start.

Further, during this six months, a mining plan would be engineered. At the present time there is a question of access to the B.S.& K. Mining Company property over the most desirable and shortest road. The costs being presented in this report reflect the longer route from the mine to the proposed site.

MINING OPERATIONS

This mine will be a typical open cut mine. It will be necessary, over the five years of operation, to break, load and move approximately 17,000,000 tons of material.

A carefully planned mining program will be required because selective mining will be essential.

It will be necessary to mine approximately 500 tons per hour for 140 hours every week; 10,000 tons/day; 500 tons/hr. 20 hrs. per day.

The plan is to drill and blast, load the ore into 50 ton rock trucks with a 10 cu.yd. front-end loader.

A study of the technical literature and consulting with operators in the Tucson area provided the list of equipment described in this report.

Budgetary equipment cost figures were obtained from suppliers in Tucson and Phoenix.

It is possible that subsequent studies may prove that the rock can be ripped rather than blasted. Further, that a portable crushing plant can be used in the mine pit and conveyors be used to move the rock more economically than trucks. For the purpose of this evaluation, these alternates will not be considered.

It is contemplated that a certain amount of pre-mining work on roads, dump sites, etc., wil be necessary. Also, that it will be necessary to strip up to 1,000,000 tons (500,000 cu.yds.) of overburden to prepare for the mining operation itself. The cost for these items is included in the estimate of capital required for the project.

MILLING OPERATIONS

The copper minerals in the ore are considered to be amenable to acid leaching techniques. Although a major portion of the copper is in a sulphide form, chalcocite, the ore body is too small and the ore grade too low to justify a flotation concentrator of economic size. A test employing "bacterial" leaching by the Duval Corporation indicated an extraction of 78.35% of the copper. The same technicians are of the opinion that the ore can be leached equally as well with sulphuric acid plus ferric sulphate with a somewhat higher acid consumption.

Two types of leaching techniques were given consideration:

1-Heap leaching
2-Vat leaching.

It appears from the test mentioned above that the ore should be crushed to a small size in order to obtain the high degree of extraction that will be required to make the operation economically sound. Extraction in heaps is known to be very low; sometimes this can be accepted in face of the lower capital and operating costs. Further, in order to evaluate heap leaching, it is essential to have enough material available to conduct meaningful tests. This could be in the range of 50,000 to 100,000 tons. For these and other reasons it was decided to evaluate the project on the basis of a vat leach followed by liquid ion exchange and high current density

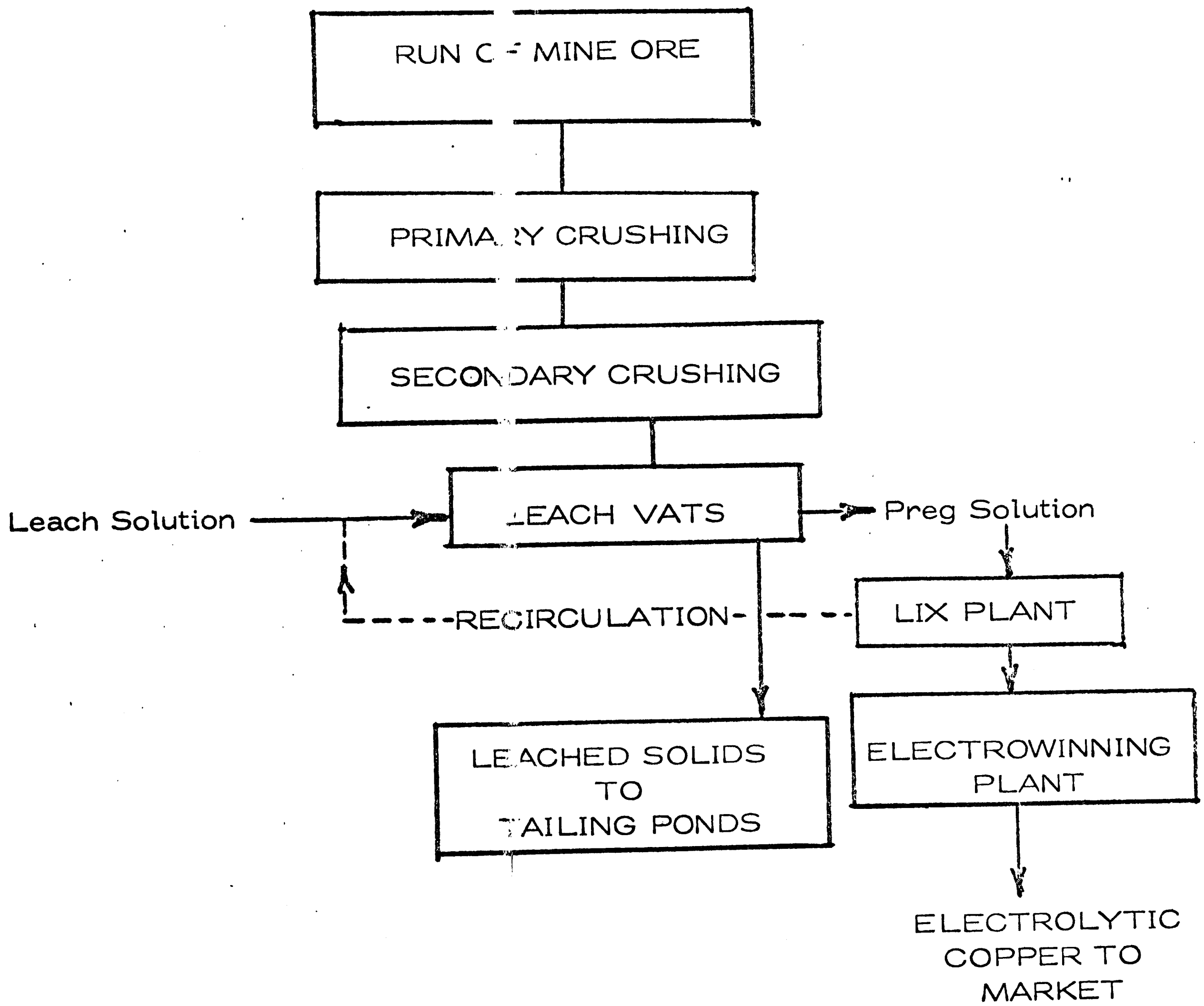
Milling Operations #2

electro winning.

In the absence of actual test data the information contained herein is based on the assumption that the project can be operated much like similar vat leaching projects presently in operation.

It has been assumed:

1. The optimum size particle is 3/8"x0
2. That 78% of the contained copper can be leached and extracted using sulphuric acid plus ferric sulphate solutions. This applies to chalcocite ore.
3. That the leaching can be accomplished in vats in an eight day cycle.
4. Upward percolation will prevent gases and slimes from blinding and/or short circuiting the ore in the vats.
5. Liquid ion exchange (LIX64) will be used to remove the copper from the leach liquor.
6. The copper will be won from the upgraded strip liquor in high current density electro-winning cells to produce an electrolytic grade copper.
7. Acid leaching of the low grade oxide copper dumps will yield approximately 25% of the copper contained therein.



SIMPLIFIED FLOW SHEET

VAT LEACHING - B.S. & K. ORE-OXIDE & CHALCOCITE.

Natural Resources Office
Essex International, Inc.
Tucson, Arizona
3/10/70

TABLE 2.—*Comparison of various leaching methods*

Method	Advantages	Disadvantages	Application
1. Heap Leaching	Minimum capital investment. Little ore preparation. Low operating cost.	Low recovery. High reagent consumption. Low grade solutions obtained.	Low grade ore bodies with marginal economics. Old leach dumps, when operation is for by-products.
2. Percolation Leaching	High grade solutions obtained. Liquid-solid separation not required. High extraction efficiencies possible.	Three-stage crushing generally necessary. Capital investment and operating cost much higher than for heap leaching.	Ore bodies which can carry the high capital investment. Ores for which high recoveries are possible at a particle size of 3/8″-3/4″. Ores which will not contain excessive amounts of fines, when crushed to the above size.
3. Agitation Leaching	Highest extraction efficiencies. Shortest leaching cycle.	Highest capital investment. Solution grade lower than in percolation leaching. Liquid-solid separation required.	Ore bodies for which percolation leaching does not result in acceptable recoveries of valuable mineral.

ESTIMATE OF COSTS

1. Capital

2. Preliminary Investigation

3. Mining

4. Process,ng

Estimates of Capital cost #2

Processing Plant

Vat Leaching

3000 tons/day. Assuming an 8 day leaching cycle - requiring 10 vats. Crushing on 2 shifts per day, 5 days per week. Optimum crush assumed to be -3/8"

Crushing Equipment

1-42"X48" Jaw crusher complete with 200 HP Motor and drive	$90,000	
1-Hopper & feeder complete	12,000	
1-El-Jay 54" Cone Crusher complete with motor and drive	55,000	
2-El-Jay 54" Fine cone Crusher complete with motor and drive	120,000	
Miscellaneous screens, conveyers, chutes etc..	150,000	
Ground storage with receeovery equipment	50,000	
Total	$507,000	
Installation (24%)	120,000	
TOTAL		$627,000

Leaching Vats with False Floors

10 vats, 80 ft.wide X120 ft.long X12 feet deep 5500 cu.yds reinforced concrete	650,000
Loading conveyors & Loading Bridge installed	420,000
Unloading equipment	210,000
Misc. pumps, pipes, valves, tanks & sumps installed	250,000
	$1, 530,000.00

ESTIMATES OF CAPITAL COST

Mine Equipment

The estimates are based on a mining rate of approximately 10,000 tons of rock per day.

1-10cu.yd. rubber tired front-end loaders	$120,000
3- Trucks 50 tons International rock truck	180,000
1-Drill-Reich #650	80,000
1-Road Maintainer	15,000
1-D8 Caterpillar and Dozer	85,000
1-Water Sprinkler Truck	10,000
1-Grease truck	7,500
1-Bulk Powder Truck	6,000
1-General Service Truck (fuel svc)	4,000
2-Pickups	5,000
1-Portable Light Plant	3,500
Total	$516,000

There is a 900 cfm Atlas Capco Air Compressor and receiver complete on the property

Pre-mining costs

Roads and dump sites	$ 50,000	
Waste Stripping 1,000,000 tons @ 30¢/ton	300,000	
	350,000	
Total estimated Mine		$866,000

Estimates of Capital cost #2

Processing Plant

Vat Leaching

3000 tons/day. Assuming an 8 day leaching cycle - requiring 10 vats. Crushing on 2 shifts per day, 5 days per week. Optimum crush assumed to be -3/8"

Crushing Equipment

1-42"X48" Jaw crusher complete with 200 HP Motor and drive	$90,000	
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Ground storage with receovery equipment	50,000	
Total	$507,000	
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Leaching Vats with False Floors

10 vats, 80 ft.wide X120 ft.long X12 feet deep 5500 cu.yds reinforced concrete	650,000
Loading conveyors & Loading Bridge installed	420,000
Unloading equipment	210,000
Misc. pumps, pipes, valves, tanks & sumps installed	250,000
	$1, 530,000.00

Estimates of Capital Cost #3

Processing Plant

LIX Electro-winning 1250 gpm-3gpl Cu. to produce 36,000 to 40,000 lbs. of copper per day

Factored estimate	$2,250,000
(source:H.S.McGarr of Holmes & Narver Inc.)	
Total Processing	$4,407,000
Total Mine & Processing	$5,273,000
Add 10% contingency	527,000
TOTAL	$ 5,800,000
Add preliminary investigation	100,000
Add initial payment to B.S.& K.Mining Co.	200,000
Total Investment Capital	$ 6,100, 000

Pre-Production and Working Capital

Pre-production costs which will require additional capital, amounts to	1,550,000
Working capital	350,000
Grand Total	$ 8,000, 000

*See notes on following page.

NOTES ON CAPITAL REQUIREMENTS

In this type of operation there is considerable pre-production expense before any income can be realized.

Mining operations must start as soon as possible in order to open the ore body for production. A certain amount of this can be capitalized as provided under Capital Cost Estimates. In this projection we are assuming that mining operations will begin one year after the agreements have been finalized and the preliminary investigations have satisfied the conditions. In the absence of an engineered mining plan, it is assumed that 1,000,000 tons of waste will be stripped and 800,000 tons of low grade oxide will be mined and placed on dumps during first six months of the second year. Then approximately 80 days will lapse until any cash is realized from production. During this time another 800,000 tons of material will be stripped and/or mined, of which at least 300,000 tons of ore should be ore for processing in the vat leach. This 300,000 tons of ore will carry crushing costs. Approximately 240,000 tons of this ore will carry vat leaching costs and some LIX-Electrowinning costs.

The capital required for this pre-production expense is estimated at $1,550,000.

ESTIMATED COST FOR PRELIMINARY INVESTIGATION

1. Drilling and Bulk Sampling

   20 holes @ 250' each, 5,000 ft.

   Estimated cost at $10.00/ft. $50,000.00

2. Metallurgical Testing

   At least 4 months at $10,000.00

   per month $40,000.00

This budget should allow for a reasonable amount of consulting service from a firm such as A.H.Ross & Associates.

3. Engineering services for a mining plan $10,000.00

TOTAL $ 100,000.00

ESTIMATE OF OPERATING COSTS

Mining(10,000 tons/day)

1. Waste Stripping 5,200,000 tons @ 30¢/ton	$ 1,560,000	
2. Mining, hauling and dumping low grade copper oxide material 7,000,000 tons @ 40¢/ton	2,800,000	
3. Mining and delivering ore to crushing plant 5,000,000 tons @ 46¢/ton	2,000,000	
TOTAL		$6,300,000

Processing

Crushing Ore for Vat Leach (3000 tons/day)

	¢/ton	
Labor	15	
Supplies	6	
Maintenance	12	
Total	33	
5,000,000 tons @ 33¢/ton		$ 1,650,000

Leaching Vat Operation (3000 tons/day)

	¢/ton	
Labor	13	
Maintenance & Operating Supplies	6	
Ferric Sulphate	8	
Sulphuric Acid	44	
Total	71	
5,000,000 tons @ 71¢/ton		$ 3,550,000

LIX-Electrowinning

	¢/#Cu.	
Power	1	
Labor	3	
Organic Solvent	2	
Miscellaneous	1	
Total	7	
51,800,000 lbs. Cu @ 7¢/#		$ 3,634,000

Overhead & Supervision

0.5¢/# Cu.	$ 259,000

Estimate of Operating Cost #2

Recap of Operating Costs

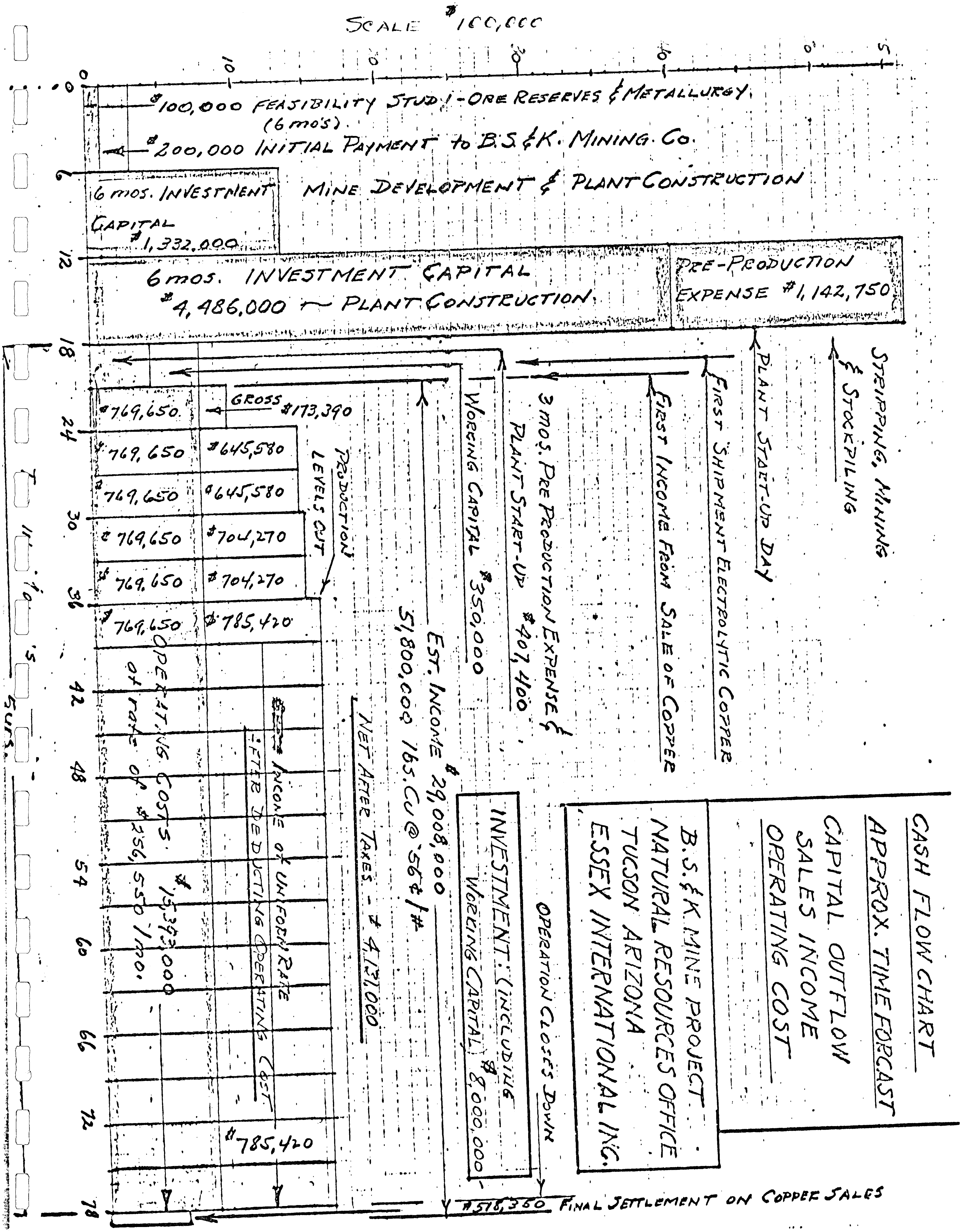
Mining	$6,300,000
Crushing	1,650,000
Leach Vats	3,550,000
LIX-Electrowinning	3,634,000
Overhead & Supervision	259,000
TOTAL	$15,393,000

Note: The cost of acid for leaching and recovering copper from the low grade oxide dumps is included in the above costs. Dump leaching will be done for the most part with acid contained in bleed streams.

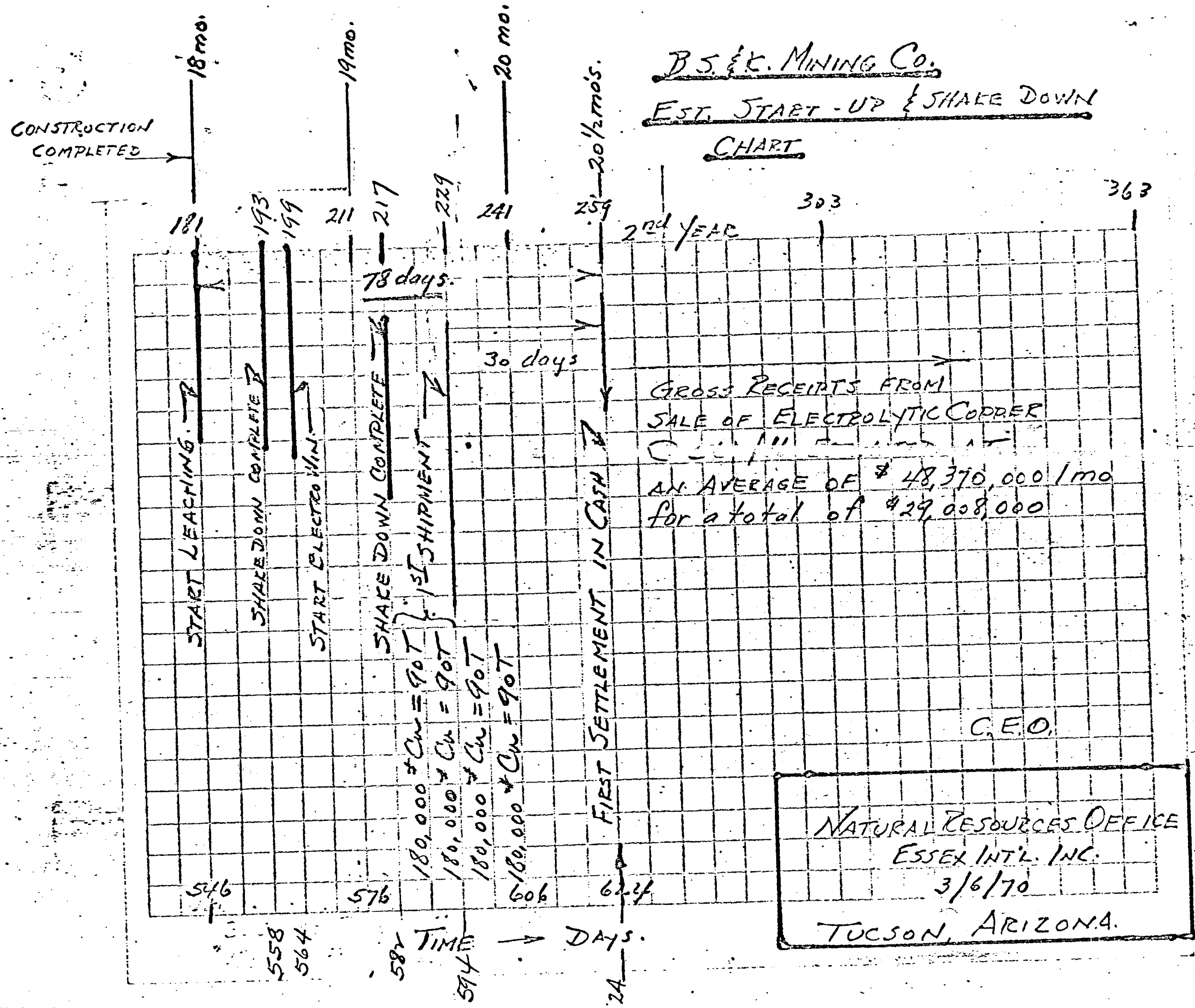
CASH FLOW

Cash Flow

Time did not permit a close detailed study of the cash flow. The time factors for capital outflow, income from sales, start up of operations, operating costs, etc., are shown on the following charts.


SCALE $100,000
0
10
20
30
40
50
$100,000 FEASIBILITY STUDY - ORE RESERVES & METALLURGY (6 mo's)
$200,000 INITIAL PAYMENT to B.S.&K. MINING Co.
6
6 mos. INVESTMENT CAPITAL $1,332,000
MINE DEVELOPMENT & PLANT CONSTRUCTION
12
6 mos. INVESTMENT CAPITAL $4,486,000 - PLANT CONSTRUCTION
PRE-PRODUCTION EXPENSE $1,142,750
18
STRIPPING, MINING & STOCKPILING
PLANT START-UP DAY
FIRST SHIPMENT ELECTROLYTIC COPPER
FIRST INCOME FROM SALE OF COPPER
3 mos. PRE PRODUCTION EXPENSE & PLANT START-UP $407,400
WORKING CAPITAL $350,000
EST. INCOME $29,008,000
51,800,000 lbs. Cu @ 56¢/#
NET AFTER TAXES - $4,131,000
PRODUCTION LEVELS OUT
GROSS $173,390
$769,650
24
$769,650
$645,580
$769,650
$645,580
30
$769,650
$704,270
$769,650
$704,270
36
$769,650
$785,420
42
48
54
60
66
72
78
OPERATING COSTS $15,393,000 at rate of $256,550/mo.
INCOME at UNIFORM RATE AFTER DEDUCTING OPERATING COST
$785,420
$518,350 FINAL SETTLEMENT ON COPPER SALES
INVESTMENT: (INCLUDING WORKING CAPITAL) $8,000,000
OPERATION CLOSES DOWN
CASH FLOW CHART
APPROX. TIME FORCAST
CAPITAL OUTFLOW
SALES INCOME
OPERATING COST
B.S.&K. MINE PROJECT
NATURAL RESOURCES OFFICE
TUCSON ARIZONA
ESSEX INTERNATIONAL INC.
5 yrs.
5
10
11

B.S. & K. MINING CO.
EST. START-UP & SHAKE DOWN
CHART



NATURAL RESOURCES OFFICE
ESSEX INT'L. INC.
3/6/70
TUCSON, ARIZONA.

STATEMENT

OF

INCOME AND EARNING

B.S. & K. MINING COMPANY PROJECT

STATEMENT OF INCOME & EARNING

Income

Sale of 51,800,000 lbs. of copper @ 56¢/#		$29,008,000

Cost of Production

Mining	$6,300,000	
Crushing	1,650,000	
Leach Vat Operation	3,550,000	
LIX-Electrowinning operation	3,634,000	
Overhead & Supervisior	259,000	
Total Production Cost		$15,393,000

Capital Cost

Mining Equipment	$ 516,000	
Pre-Mining Cost	350,000	
Crushing Equipment installed	627,000	
Leaching Vats installed	1, 530,000	
LIX-Electrowinning plant installed	2, 250,000	
Total	$5, 273,000	
Add 10% Contingency	527,000	
Total	$5, 800,000	
Add preliminary investigation	100,000	
Add initial pay't B.S.& K.Mine.Co.	200,000	
Total Capital Cost		6,100,000
Total Investment plus Operating Cost		$21,493,000
Gross Profit before provision for payment of interest on Investment and before provisions for Depletion.		$ 7,515,000
Provision for interest on Investment		1,970,000
Net income before provision for taxes on Income		$ 5,545,000
Provision for Federal Income Tax		1,414,200
Net Income		$ 4,130,800

Statement of Income & Earning #2

Annual estimated taxable income $1,109,000

Estimate of Federal Income Taxes

Gross annual sales	$5,800,000
Base for applying depletion	$5,800,000
15% -percentage depletion	870,000
50% Taxable Income (maximum)	554,500
Taxable income before depletion	1,109,000
Less percentage depletion allowed	554,500
Taxable income	$ 554,500

U.S. Federal Income Tax

Normal 30% on $554,500	$ 166,350
Surtax 22% on $554,500 -25,000	116,490
Total tax	$ 282,840

Net income as a percent of sales

$$\frac{4,130,800}{29,008,000} = 14.24\%$$

File

Please make 2 Xerox copies T. Jancic

G. A. Girard November 22, 1965

V. F. Hollister All Concerned

Bacterial Leaching of B.S. & K. Exploration Composite

H_2SO_4

Abstract

Total copper extraction from the B.S. & K. exploration composite gave a recovery of 77%. Recovery from acid-soluble copper fraction was about 85%; from chalcocite copper was 82%; and about 51% from chalcopyrite.

There is no requirement for purchased acid. An initial acidification would, however, increase the initial copper production.

Microscopic examination of chalcopyrite particles revealed substructures of that mineral species within a network of more soluble chalcocite. The new surface exposure generated by the dissolution of chalcocite is believed to be responsible for the increased extraction from chalcopyrite.

Background

It was requested by the Exploration Department that a bacterial leach test be run on specified drill hole samples from the B.S. & K. location to determine the acid consumption of the ore, the response of the ore to copper recovery by bacteria and to evaluate this response in terms of our laboratory experience with other ores. The test head consisted of specified intervals from drill holes 300, 301, 302, 303 and 305.

Purpose

It was the purpose of this experiment to determine in a bacterial leach process:

1. The total recovery of copper over a four week leach period,
2. The copper recovery from each of the mineral species,
3. The total acid consumed and the purchased acid requirement.

Procedure

The composite sample was crushed to pass 10 mesh, the minus 100 mesh fraction rejected. Four test charges were split out from the minus 10 plus 100 mesh sample fraction. An additional sample was removed for chemical analyses, the results of which appear in Table I.

One hundred grams of sample were charged into each of four percolation columns. One hundred ml of sterile barren leach solution were added to each of the columns. The columns were each inoculated with 25 ml of an 8 day culture of *T. ferrooxidans* having a population of 1×10^8 bacteria per ml. The percolators had an adjusted air flow of 50 to 75 ml per minute. Each week the solutions were drained from the columns and submitted for copper, iron and pH analyses. The copper present in the sterile barren leach solution was deducted from the total copper recovered.

The ore in columns 1 and 2 were controlled at pH 2.0 by daily adjustment with sulfuric acid.

The ore in columns 3 and 4 was initially fed a solution of pH 2.0. The solutions fed the columns the second and subsequent weeks matched the pH of the effluent solutions of the previous week.

Upon termination of the test the columns were drained and rinsed with 20 ml of pH 2.0 distilled water. The combined solutions were submitted for analysis. The columns were then rinsed with two 25 ml portions of barren leach solution followed by a wash with 100 ml of one normal H_2SO_4 for five minutes. The columns were then rinsed with 20 ml of pH 2.0 distilled water. The combined solution, submitted for analysis, represented desorbed and dissolved ferric and cupric ion. The columns were then given a final rinse with 100 ml of pH 2.0 distilled water to determine the quantity of copper and ferric iron remaining after strong acid treatment. This solution was submitted for analysis.

The ore was removed from the columns, dried and submitted for chemical analysis. The test data are presented below.

TABLE I
B.S. & K. LEACH TEST

SAMPLE NO.		1	2		3	4
Assayed Head,	% A-S Cu	------	------	.174	------	------
" "	% cc Cu	------	------	.269	------	------
" "	% cpy Cu	------	------	.091	------	------
" "	% Total Cu	------	------	.535	------	------
Calculated Head,	% Cu	.510	.555		.481	.520
Residue,	% A-S Cu	.022	.026		.023	.028
"	% cc Cu	.048	.048		.045	.050
"	% cpy Cu	.046	.046		.046	.043
"	% Total Cu	.116	.120		.114	.121
Copper Extracted	(#/Ton)	7.90	8.71		7.33	7.92
$\#H_2SO_4$ Consumed Per Ton Feed*	Total	17.0	18.1		-	-
	Cu	12.2	13.4		11.3	12.2
	Gangue	4.8	4.7		-	-
Distribution %Cu	Filtrate	77.5	78.5		76.2	76.1
	Residue	22.5	21.5		23.8	23.9
$\#H_2SO_4$ Consumed Per #Cu	Recovered	2.2	2.1		-	-
	Available	1.7	1.6		-	-
% A-S Cu Recovered		87.5	85.1		86.8	84.0
% cc Cu "		82.2	82.2		83.3	81.5
% cpy Cu "		49.5	49.5		49.5	52.7

* The acid consumed by columns 1 and 2 (duplicate) was based upon actual acid addition to maintain a pH of 2.0.

Data and Results:

extraction 78.35%

Copper Extraction

Slightly more copper was extracted from the columns maintained at pH 2.0 (columns 1 & 2) than from the columns with no pH control (columns 3 & 4). An average of 78% copper was extracted from the columns maintained at pH 2.0 while only 76% was the average copper extracted from the columns with no pH control (see Figure I, Table II). With the exception of the first week's extraction, the weekly extraction is identical and indicates that a benefit could be derived from an initial dump acidification.

Acid Consumption by Columns One and Two

The total acid consumed by the pH controlled columns is given in Table I. Table III representing the average of pH controlled columns gives a breakdown of the acid consumed or produced by the ore and solution each week and cumulatively. Table III and Figures II and III gives specific information on the consumption of acid by oxidized ferrous ion and the net acid consumption by the ore. The total acid consumption is the actual acid necessary to maintain the system at pH 2.0. This acid was consumed by the ore and ferrous ion oxidation in solution:

$$2FeSO_4 + H_2SO_4 + 1/2\ O_2 \xrightarrow{\text{Bacteria}} Fe_2(SO_4)_3 + H_2O$$

The total acid consumption by ore and solution and the acid consumption by oxidized ferrous iron are illustrated in Figure III on a weekly and cumulative basis. The net acid consumption by the ore is shown in Figure IV. It will be noted that, after seventeen days, more acid was being produced by the ore than was being consumed by the ore and solution.

Acid Consumption by Columns Three and Four

The acid consumption data by the columns with no pH control is presented in Table IV and illustrated in Figures IV and V. The acid consumption is based on the actual ferric hydroxide formation:

$$Fe_2(SO_4) + 6H_2O \xrightarrow{\text{pH } 2.7} 2Fe(OH)_3 + 3H_2SO_4$$

The system is kept below pH 2.7 by the production of acid through ferric precipitation. A rise in the pH of the leach liquor throughout the test represents, also, an acid consumption. These two factors are combined in Table IV and Figure IV under the heading of "total acid consumption from $Fe(OH)_3$ formation." The acid consumption by oxidized ferrous in solution is deducted from Figure IV to give the net acid consumption represented by Figure V.

Comparison of Acid Consumption Data From Each Column Pair

It will be noted from Figure V that the weekly acid consumption approaches zero asymtotically with time. The pH of this system seems to stabilize at 2.35 over the four week test. In Figure III there is a net acid production. It follows, then, that gangue consumption must be greater in the pH controlled system since acid producing constituents in the ore have been released.

The pH-controlled columns give a true picture of the acid consumption necessary to maintain a pH of 2.0. The columns with no pH control were designed to indicate any need for purchased acid. Other than the benefit to be derived from initial dump acidification, it is concluded that no additional acid need be purchased.

The Structural Nature of Chalcopyrite Used in This Test

It was noted from the results of chemical analysis of the chalcopyrite mineral species that the recovery was 12 to 25 times greater than most reported results both in our laboratory tests and in the literature. Previous work with Battle Mountain ore yielded, in certain cases, results similar to those in this test with chalcopyrite. These prior results were regarded with suspicion and attributed to inconsistancies in the analytical procedure. Since a recovery of about 51% occurred with this mineral species, it was decided to resubmit the samples for analysis. The samples were leached for seven minutes and twenty minutes with cyanide in two separate tests to determine whether all the chalcocite had been removed prior to the analysis for chalcopyrite. These test results were in agreement with each other and those first reported.

It was then thought that this chalcopyrite must be structurally different from other specimens of the same mineral species. A polished section was made of the sink fraction of a heavy media separation. A microscopic examination of the chalcopyrite particles at 450 power revealed intimate intergrowths of chalcocite replacing chalcopyrite.

A 48 mesh particle of chalcopyrite, for example, exhibited a subparticle size of 400 mesh in many cases. Since chalcocite is readily leached in a bacterial system the surface exposure of chalcopyrite is greatly increased as leaching proceeds.

Based on our laboratory experience with unaltered chalcopyrite from the bottom of the Esperanza pit the recovery from B.S. & K. chalcopyrite is 1040% greater.

A microprojection of a typical chalcopyrite particle, illustrated in Figure VI, was traced on thin translucent paper.

Conclusions

1. Copper extraction over the four week test period gave a total recovery of about 77%. Recovery of copper from chalcopyrite about 51% and was 10.4 times greater than expected. Microscopic examination revealed the special nature of this mineral species.

2. The data indicate that purchased acid is not necessary except for the benefit to be derived from initial dump acidification.

Respectfully submitted,

George A. Girard

George A. Girard
Process Chemist

GAG:jfg

cc: R. W. Livingston (2)
R. W. Flagg
C. H. Curtis
G. E. Atwood
J. E. Frost
E. K. Drechsel
R. R. Nelson
D. J. Bourne

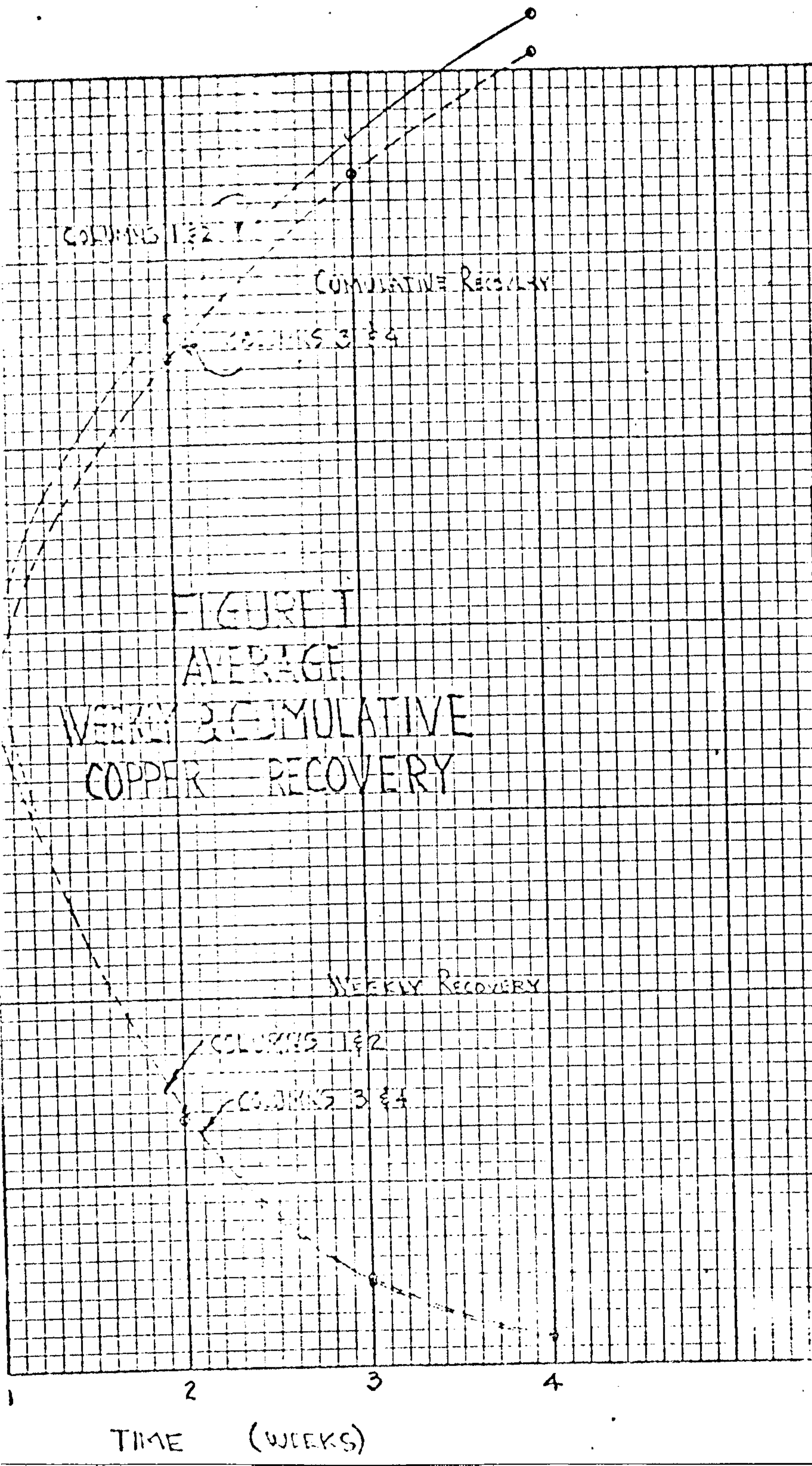


FIGURE I
AVERAGE
WEEKLY & CUMULATIVE
COPPER RECOVERY

TABLE II

% RECOVERY

WEEK No.	COLUMN 1		COLUMN 2		AVERAGE CUMULATIVE RECOVERY 1 & 2	COLUMN 3		COLUMN 4		AVERAGE CUMULATIVE RECOVERY 3 & 4
	WEEKLY	CUMUL.	WEEKLY	CUMUL.		WEEKLY	CUMUL.	WEEKLY	CUMUL.	
1	47.4	47.4	44.0	44.0	43.7	41.2	41.2	30.2	[illegible]	[illegible]
2	19.0	61.5	17.9	62.0	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
3	9.6	71.1	10.0	72.0	71.6	9.8	69.6	9.8	69.4	69.5
4	6.4	77.5	6.5	78.5	78.0	6.6	76.2	6.7	76.1	76.2

AVERAGE ACID CONSUMPTION OF COLUMNS 1 & 2

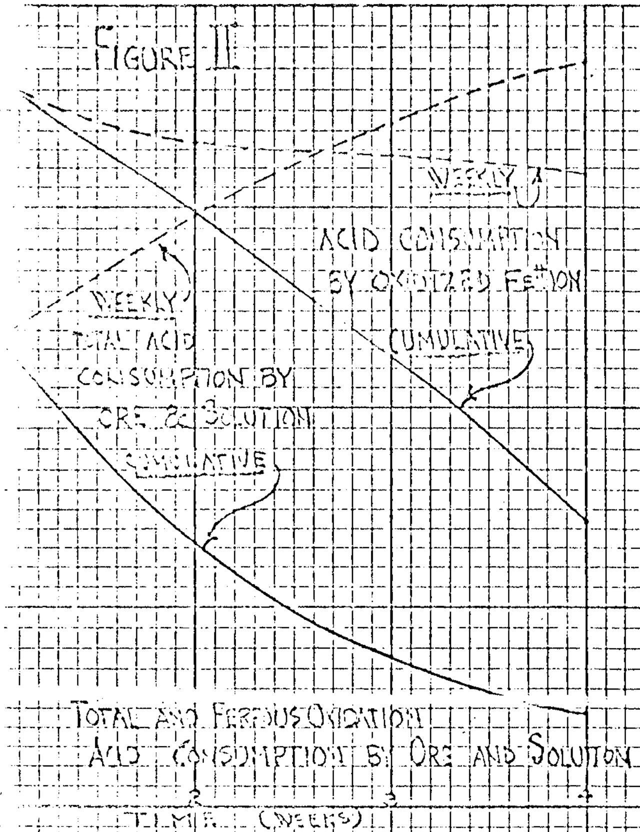


FIGURE II
TOTAL AND FERROUS OXIDATION ACID CONSUMPTION BY ORE AND SOLUTION

TABLE III

WEEK NO.	TOTAL ACID CONSUMPTION BY ORE & SOLUT. (#/TON)		ACID CONSUMPTION BY OXIDIZED Fe++ ION (#/TON)		NET ACID CONSUM. AND PRODUCTION BY ORE (#/TON)	
	WEEKLY	CUMUL.	WEEKLY	CUMUL.	WEEKLY	CUMUL.
1	-8.17	-8.17	-1.66	-1.66	-6.51	-6.51
2	-5.71	-13.38	-3.12	-5.02	-1.79	-8.30
3	-2.78	-16.16	-3.61	-8.69	.83	-7.47
4	-1.40	-17.56	-4.17	-12.25	2.77	-4.70

NOTE:
THE TOTAL ACID CONSUMPTION IS THE ACTUAL ACID ADDED TO MAINTAIN A pH OF 2.0.

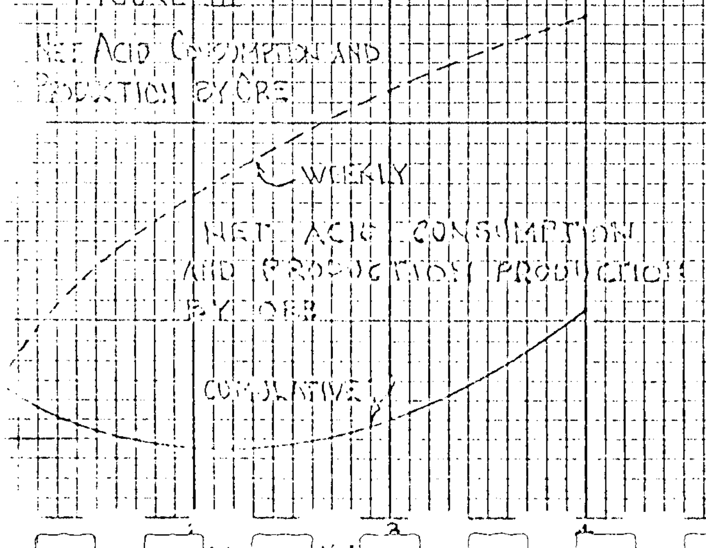
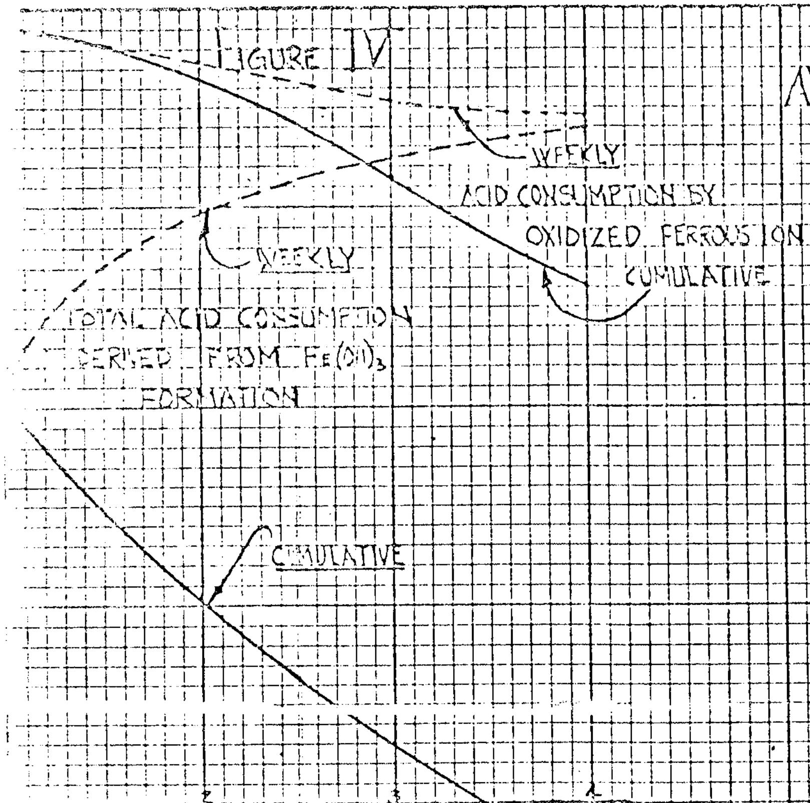
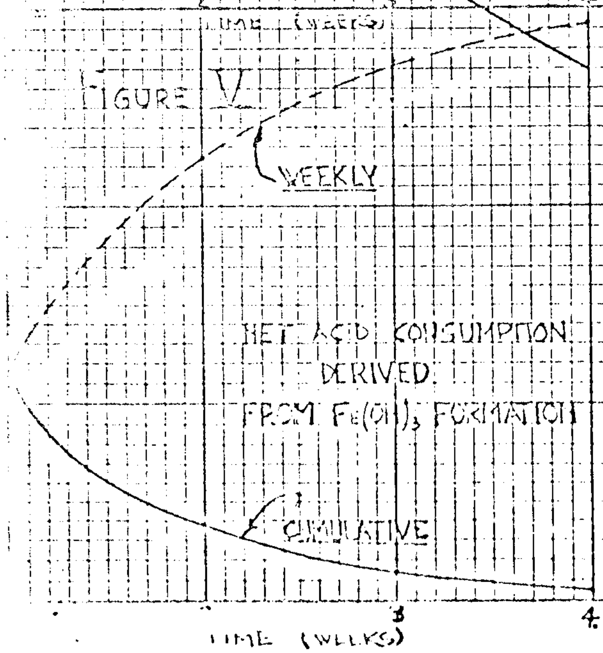


FIGURE III
NET ACID CONSUMPTION AND PRODUCTION BY ORE





AVERAGE ACID CONSUMPTION DERIVED FROM FERRIC HYDROXIDE FORMATION IN COLUMNS 3 & 4

TABLE IV

WEEK NO.	TOTAL ACID CONSUMPTION FROM $Fe(OH)_3$ FORMATION (#/TON)		ACID CONSUMPTION BY OXIDIZED FERROUS ION (#/TON)		NET ACID CONSUMPTION FROM $Fe(OH)_3$ FORMATION (#/TON)	
	WEEKLY	CUMUL.	WEEKLY	CUMUL.	WEEKLY	CUMUL.
1	-9.70	-9.70	-0.53	-0.53	-9.23	-9.23
2	-5.06	-14.76	-1.28	-1.81	-3.78	-13.01
3	-3.74	-18.50	-2.43	-4.24	-1.31	-14.32
4	-3.09	-21.59	-2.70	-6.94	-0.39	-14.71

NOTE: THE TOTAL ACID CONSUMPTION DERIVED FROM $Fe(OH)_3$ FORMATION ALSO INCLUDES ACID CONSUMED THRU A SYSTEMIC pH RISE.

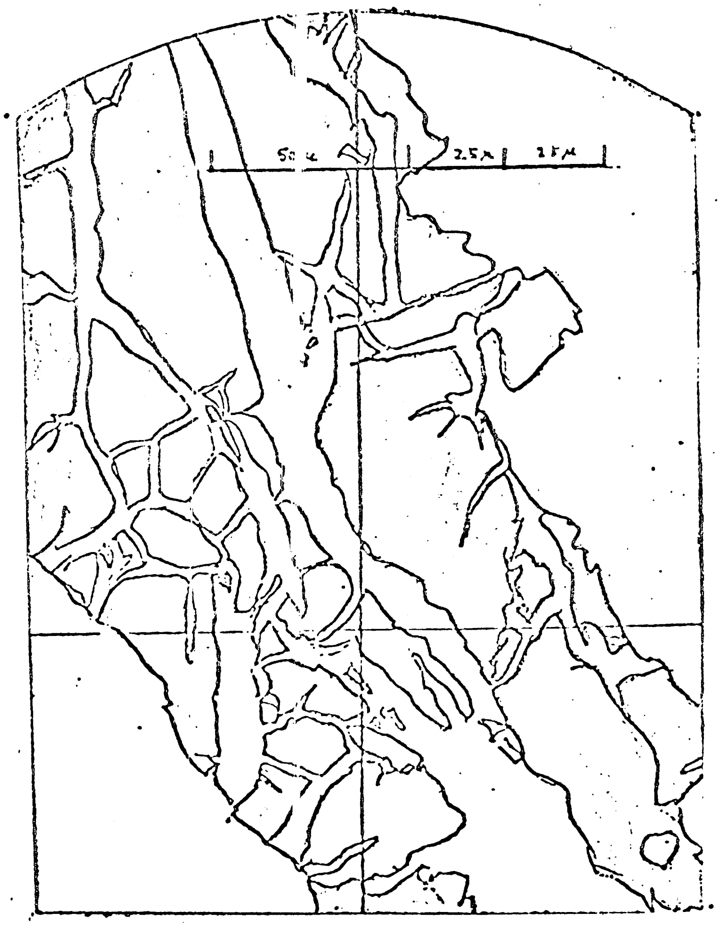


FIGURE VI

MICROPROJECTION TRACING (530X)

OUTLINE SUMMARY

ECONOMIC GEOLOGY, ORE RESERVES
AND
PROPERTY STATUS

of
THE B. S. AND K. MINING COMPANY

for
ESSEX INTERNATIONAL INCORPORATED

MARCH 1970

by
HEINRICHS GEOEXPLORATION COMPANY
P. O. Box 5671 Tucson, Arizona

ARIZONA

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TABLE OF CONTENTS

INTRODUCTION

This summary outline has been prepared for Essex International Inc., and represents part of a report entitled "Preliminary Evaluation for Possible Acquisition and Operation of the B. S. and K. Mining Company Properties", by Clyde Osborn, Technical Director of the Natural Resource Office of Essex International Inc.

No field geological study of the B. S. & K. property has been conducted. However the generalized geology of the area is well published and known.

The B. S. & K. Mining Co. property consists primarily of two major mineralized areas, the New York Claim Group and the Atlas Mine.

NEW YORK CLAIM GROUP

These claims are located in the immediate area of the quarter corner of Sections 28/33, T. 11 S., R. 8 E. and consisting of the following lode mining claims, New York, Nevada No. 2, Georgetown and fractional lode claims NSB 9, 10, 11, 12, 13 and 14 and comprising a total of approximately 58.25 acres.

The mineralized area of Silverbell Mine of A. S. & R. to the immediate south is a chalcocite blanket 100 to 200 feet thick lying under a leached capping which is approximately 100 ft. thick. The New York area of B. S. & K appears to be an extension of the El Tiro pit to the north. Available drilling information partly substantiates this assumption. (See accompanying copies of drill logs of Duval Corp.)

The leached capping of oxidized copper mineralization that overlays the chalcocite blanket carries values up to 0.15% to 0.2% copper over a considerable portion of the New York area. The chalcocite blanket varies in thickness from 30 feet to 100 feet and appears to be increasing in grade and thickness to the south and west.

Inferred Ore Reserves

Duval Corporation reportedly calculated 60,000,000 pounds of inferred copper reserves in the New York Claim Group and this figure is reasonable based on the following data:

1. Average thickness of chalcocite blanket - 40.0 ft. (See drill logs and assays of Duval Corp.)
2. Calculated tonnage of area 8,441,666 of which 75% may be mineable = 5,627,777 tons.
3. Assume average grade 0.6% Cu.
4. Solution = 67,533,324 pounds of copper.

In order to obtain sufficiently absolute actual tonnage and grade figures, a comprehensive evaluation program of twenty drill holes, drilled to a depth of 250 ft. each, is recommended. In addition, three drill holes, each 1,500 ft. deep to test the possible downward extensions of the chalcocite blanket is also recommended.

The Induced Polarization geophysical results by GEOEX and Canadian Aero suggests no cut-off of sulfides at depth in this area and therefore some deep drilling is justified and certainly should be programmed.

Total cost of the shallow drilling and sampling program will be approximately $50,000. Total cost of the deep drilling program of three holes, 1,500 feet deep would be approximately $60,000.

Legal Status

The New York Claim Group is completely surrounded on three sides by A. S. & R. and if possible a right-of-way should be negotiated with A. S. & R. as shown on the photo mosaic as Route 1. Route 2 is currently the only available access to the New York Group from the proposed leach and dump areas and from a mining cost standpoint the less desirable route.

Litigation between B. S. & K. Mining Co. and A. S. & R has been in the courts for many years and it is recommended that a thorough investigation be conducted by a lawyer, into the findings of the court prior to consummation of an agreement.

Favorable access negotiations with A. S. & R. is a definite possibility as indicated by the favorable solution of past problems of a similar nature and also because of the Arizona condemnation statute regarding rights-of-way for mining purposes.

THE ATLAS MINE

Located in the SE quarter of the NE quarter of Sec. 32, T. 11 S., R. 8 E., the Atlas Mine has had ore production history dating back to 1900. Production included argentiferous lead, zinc and copper sulfides and carbonates occuring in veins and pods in the paleozoic sediments. The ore occurrence has been intermittent and difficult to follow. However, when an ore pod was encountered the values reportedly ran as high as 45% zinc and 4% to 5% copper, with some recoverable values in silver.

No factual information is available to compute ore reserves in this area, however, good exploration possibilities do exist in the area based on I. P. anomalism on some work done by McPhar Geophysics Ltd. of Canada in 1960.

Some drilling should be done eventually to fully develop the economic mineral potential of this area.

Legal Status

The Atlas Mine area is on patented lode mining claims and the balance of the contiguous unpatented claims are reportedly in good condition from a legal standpoint. However, a thorough check of the Pima County Courthouse records should be made prior to consummation of an agreement.

Respectfully submitted,
Heinrichs GEOEXploration Company

E. Grover Heinrichs, Vice President

Approved:

Walter E. Heinrichs, Jr., President

BS#K 300

ASSAY— GEOLOGY COMPOSITE LOG

Sheet of 1-1

BEARING	Coörd. N	HOLE No 300-R
INCL. VERT.	E	COLLAR ELEV.
BS#K 300		DEPTH 400'

DESCRIPTION	Geol	Asy	INT.	CR	% Cu	% As Cu	% Mo	% As Mo	% EQIV	SAMPLE CMC	REC'Y %	GROUPING — RMKS.
182'-323':			182-190		+1000	.14	+300	.045		1	69.6	
K: Dacite porphyry made of rounded quartz eyes and pinkish feldspars in gray siliceous matrix.			190-200		+1000	.15	165	.016		2	82.8	
Ex Alt.: Thin flat qtz veining w/ rims of secondary orthoclase. Moderate argillization along steep joints			200-210		1000	.10	114	.011		3	91.6	
Min. Chalcopyrite & moly, in grains along qtz veins. Minor diss. sulfides			210-220		+1000	.11	238	.024		4	80.4	
			220-230		+1000	.14	245	.024		5	79.2	
			230-240		+1000	.12	171	.017		6	42.3	
			240-250		+1000	.34	+300	.042		7	46.1	
			250-260		+1000	.20	193	.019		8	23.1	
			260-270		+1000	.14	205	.020		9	14.1	
			270-280		+1000	.16	101	.010		10	21.2	
			280-290		+1000	.17	214	.021		11	25.6	
			290-300		+1000	.24	174	.017		12	54.1	
			300-310		+1000	.31	139	.014		13	21.6	
[illegible]3-340: strongly altered monzonite(?). Abund chlorite after biotite. Some fine diss. sulfides. Very poor core recov.			310-320		+1000	.16	+300	.034		14	50.0	
			320-330		+1000	.20	277	.028		15	18.4	
			330-340		+1000	.28	41	.004		16	44.5	
[illegible]10-400: Dacite porphyry as 182-323', but chlorite is present. At 348' about 1" Chalcopyrite vein			340-350		+1000	.91	+300	.032		17	71.3	
			350-360		+1000	.37	+300	.044		18	61.3	
			360-370		+1000	.21	282	.028		19	92.1	
			370-380		+1000	.18	272	.027		20	94.0	
			380-390		+1000	.15	178	.018		21	80.5	
			390-400		+1000	.20	256	.026		22	95.2	
						.227		.024			56.8	AVERAGE REC'Y
						.318						Equiv. Cu-Mo

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ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 3

PROJECT Atlas Min, BS & K Mining Co., BS & K No 3 Claim

Coörd. : N	BEARING vert	HOLE Nº 302	
E	INCL. vert	COLLAR ELEV.	
START 8-6-65 COMPL. 8-8-65	Nc core size	DEPTH 182 feet	

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz	oz	% EQUIV.	GROUPING—RMKS.
0-90 Dacite Porphyry, Brown fine grained, soft, porphyritic, highly fractured, numerous qtz veinlets, Arg. Alteration, Original Sulfide was ± 3%, entirely in fractures, Core very badly broken, No green or blue copper minerals visible, though entire section is oxide.			0-10	56	.08	.002						
			10-20	70	.08	.002						
			20-30	68	.09	.00[illegible]						
			30-40	76	.14	.001						average oxide is
			40-50	56	.14	.008						.13 Cu .005 Mo
			50-60	90	.16	.00[illegible]						for 90 feet
			60-70	[illegible]	.15	.009						
			70-80	85	.16	.006						
			80-90	87	.17	.009						
90-95 Dacite Porphyry, Gray Mixed oxide and sulfide. Soft, porphyritic, [illegible] [illegible] [illegible] qtz veinlets, [illegible] as oxide [illegible] sulfide.			90-95	93	.94	.025						oxide / Mixed
			95-[illegible]	86	.79	.01[illegible]						Supergene Sulfide
			[illegible]		.51	.01[illegible]						
			[illegible]	[illegible]	.60	.004						Average grade of Mixed and supergene is 35 feet of
[illegible]-125 Dacite Porphyry Supergene zone, Gray, soft [illegible]			110-[illegible]	8[illegible]	.24	.00[illegible]						
			115-[illegible]	88	.12	.011						.47 Cu
			120-125	83	.11	.012						.013 Mo

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ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 2

PROJECT Atlas Mine, BS&K Mining Co., New York Claim

Coörd. : N	BEARING Vert	HOLE Nº 301
E	INCL. Vert	COLLAR ELEV.
START 8-9-65 COMPL. 8-10-65		DEPTH 206

DESCRIPTION

0-100' Dacite Porphyry - Brown - oxide zone
Soft, strong Argillic Alt., Porphyritic, fine grained. Original Sulfide was ± 4%, 90% in fractures, 10% diss. Trace sericite lining fractures. Qtz veinlets rare. Some green Cu stain visible.

100-110' Dacite Porphyry Mixed Oxide and Sulfide
Brown, Soft, fine grained, Porphyritic, Wk sericite and strong Arg. Alt. Copper occurs as Malachite and chalcocite, Core badly shattered.

110-125' Dacite Porphyry Supergene Sulfide zone
Dark gray, soft, weak Sericite and strong Arg. Alt. Copper occurs as chalcocite with weak pyrite, mostly in seams. Core badly broken.

Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz	oz	% EQUIV	GROUPING — RMKS.
		0-10	20	.12	.006						0'-30' .16 Cu .09 Mo
		10-20	50	.20	.012						
		20-30	67	.17	.010						
		30-40	80	.18	.009						30'-50' .17 Cu .011 Mo
		40-50	85	.16	.014						
		50-60	80	.11	.016						50-80 .12 Cu .016 Mo
		60-70	86	.12	.016						
		70-80	90	.13	.017						
		80-90	87	.24	.010						80'-100' .17 Cu .009 Mo
		90-100	90	.10	.009						
		100-105	90	.15	.013						100'-125' /83 Cu .018 Mo
		105-110	85	1.31	.040						
		110-115	86	1.74	.019						
		115-120	82	.49	.008						
		120-125	87	.45	.013						

LOGGED BY VRH DATE 8-10-65

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ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 2 of 2

PROJECT ____________________

Coörd. : N	BEARING	HOLE Nº [illegible]
E	INCL.	COLLAR ELEV.
START COMPL.		DEPTH

DESCRIPTION

125-170 Dacite Porphyry

Hypogene zone, Gray, Soft, porphyritic. Copper occurs in seams and veinlets as crushed chalcopyrite, with pyrite. Numerous qtz. veinlets. Much pyrite in seams is crushed and [illegible]. Total sulfide is [illegible]. Sulfide is 70% in seams, [illegible] as diss. [illegible] on seams.

170-182 Dacite P[illegible], Gray

Hypogene Zone, Soft, [illegible]. Copper occurs [illegible] chalcopyrite with pyrite. Total sulfide [illegible] is 3%. Su[illegible] seams and [illegible] 20% as diss. [illegible] veinlets. Pyrite is [illegible] crushed [illegible] seams, with Molybdenite visible in quartz veinlets. Largest veinlet is 2" qtz-pyrite-chalcopyrite at 177'. This veinlet also has traces of chalcocite.

Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz	oz	% EQUIV.	GROUPING—RMKS.
		[illegible]	82	.18	.01[illegible]						Average Hypogene grade is .18 Cu .015 Mo
		[illegible]	[illegible]	.14	.0[illegible]						
			[illegible]	.12	.011						
		[illegible]	76	.11	.0[illegible]						
		[illegible]	70	.11	.010						
		[illegible]	75	.12	.015						
		[illegible]	80	.16	.016						
		[illegible]	[illegible]	.12	.00[illegible]						
		[illegible]	[illegible]	.15	.031						
		[illegible]	75	.30	.01[illegible]						
		17[illegible]-182	7[illegible]	.32	.017						

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ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 2 of 2

PROJECT Atlas Mine

Coörd. : N	BEARING	HOLE Nº 300
E	INCL.	COLLAR ELEV.
START COMPL.		DEPTH 206

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz	oz	% EQUIV.	GROUPING — RMKS.
125-206 Dacite Porphyry **Supergene zone as above**			125 130	80	1.21	.006						125'-150' .74 .009 Mo
			130 135	90	.70	.008						
			135 140	85	.65	.009						
			140 145	70	.39	.015						
			145 150	65	.76	.009						
			150 155	56	.47	.013						150'-175' .49 Cu .027 Mo
			155 160	72	.38	.043						
			160 165	80	.36	.024						
			165 170	85	.51	.040						
			170 175	86	.72	.014						
			175 180	85	.50	.010						175'-200' .69 Cu .018 Mo
			180 185	85	.57	.052						
			185 190	80	.88	.011						
			190 195	90	1.12	.013						
			195 200	92	.40	.004						
			200 206	85	.19	.006						

End of hole

Average Mo for Hole is 0.0158 Mo.
Average from 105 to 200' is .72% Cu
.018 Mo

VEH DATE 8-10-65

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ASSAY—GEOLOGY COMPOSITE DRILL LOG

PROJECT Atlas Mine, BS&K Mining Co., BS&K

Coörd. : N	BEARING Vert	HOLE Nº	302
E	INCL. Vert	COLLAR ELEV.	
START 8-11-65 COMPL. 8-12-65		DEPTH	151

DESCRIPTION

0-68 Dacite Porphyry
Oxide zone, Brown, fine grained, porphyritic, strong Argillic Alt. with trace Sericite. Core badly fractured, Original Sulfide content was est. 5%. Some Malachite visible 20' to 68'

68-70 fault zone. Core crushed

70-90 Quartz Monzonite Porphyry
Brown, Oxide zone, Granitic, holocrystalline texture. Weak Argillic Alteration, with No sericite. Fracturing very strong at 70' gradually weakens down the hole. Mineralization fades with fracturing.

90-95 Quartz Monzonite Por.
Mixed Oxide and Sulfide, Copper occurs as chalcocite, with limonite.

95-131 Quartz Monzonite Por.
Supergene zone fracturing weakens with depth. Some Moly visible in rare quartz veinlets.

Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz	oz	% EQUIV.	GROUPING—RMKS.
		0-10	45	.07	.009						
		10-20	50	.06	.012						
		20-30	62	.09	.006						
		30-40	70	.07	.004						
		40-50	70	.07	.009						
		50-60	65	.10	.009						
		60-70	50	.19	.007						
		70-80	55	.11	.012						
		80-90	60	.20	.010						
		90-95	70	.66	.010						
		95-100	85	.66	.016						
		100-105	85	.61	.005						
		105-110	90	.44	.013						
		110-115	95	57	.006						
		115-120	97	54	.004						
		120-125	90	.32	.005						

VFH 8-12-65

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

PROJECT Atlas Mine, Drill Hole for Nevada No 2 Claim

Coörd. : N		BEARING Vert	HOLE Nº 303
E		INCL. Vert	COLLAR ELEV.
START 8-19-65	COMPL. 8-20-65		DEPTH 109

DESCRIPTION

0-15 Dacite Porphyry.
Brown. Oxide zone, soft, Porphyritic, fine grained. Original sulfide content was 4%. Strong Arg. Alt.; tr ser.

15-25 Quartz Monzonite
Brown, oxide zone, soft Holocrystalline, Est. 2% Original sulfide. Wk Arg. Alteration.

25-60 Quartz Monzonite
Gray. Supergene zone Holocrystalline. Est 2% Sulfide. Chalcocite occurs on fractrues. Weak Arg. Alt. No sericite.

60-109 Quartz Monzonite
Gray as above. Hypogene zone. Est 2% sulfide, in fractures. Trace of chalcopyrite.

Geol.	Assay	Int.	C/R	% Cu	% Mo	%	%	oz.	oz.	% Equiv.	Grouping—Rmks.
		0-10	40	.02	.021						
		10-20	70	.02	.017						
		20-25	80	.04	.009						
		25-30	90	.09	.001						
		30-40	78	.28	.001						
		40-50	80	.29	.009						
		50-60	90	.16	.008						Average from 30 to 50 is .28% Cu, .005 Mo.
		60-70	90	.13	.015						
		70-80	90	.11	.003						
		80-90	90	.12	.003						
		90-100	90	.07	.007						
		100-109	95	.001							

LOGGED BY VFH DATE 8-20-65

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet __ of __

PROJECT Atlas Mine, BS&K Mining Co., Hole for BS&K No. 1

Coörd. : N	BEARING Vertical	HOLE Nº 306
E	INCL. Vertical	COLLAR ELEV.
START 8-21-65 COMPL. 8-22-65		DEPTH 100

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz	oz	% EQUIV.	GROUPING — RMKS.
0-30 Dacite Porphyry Brown, fine grained, soft. Original sulfide now entirely oxidized. Original sulfide was 3% of rock. It occurred entirely in fractures. Arg AH., No Sericite			0 10	30	.04	.007						
			10 20	45	.05	.004						Average from 0'-30' .06 Cu, .027 Mo
			20 30	55	.10	.070						
30-50 Dacite Porphyry Mixed Oxide and sulfide zone Fe Ox and Pyrite, with traces of chalcocite. Rock as above.			30 40	75	.20	.030						Average from 30' to 70' is .29% Cu, .015 Mo.
			40 50	90	.34	.009						
			50 60	90	.38	.009						
50-60 Dacite Porphyry Supergene zone Chalcoite coatings on pyrite. Rock as above.			60 70	85	.23	.011						
			70 80	90	.21	.016						
60-100 Dacite Porphyry Hypogene zone. No chalcocite. Rock as above.			80 90	90	.16	.002						
			90 100	85	.10	.006						

LOGGED BY VFH DATE 8-23-65

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 1

PROJECT B. S. & K.

Coord. : N	BEARING	HOLE Nº RDH 309
E	INCL. Vertical	COLLAR ELEV.
START 22 Mar.66 COMPL. 22 Mar 66'		DEPTH 110'

DESCRIPTION				C/H	% Cu	% Mo	%	%	%	%	% EQUIV	GROUPING—RMKS.
SILVERBELL DACITE - argillically altered and siliceous, medium hardness but very abrasive, qtz. sulphide strgrs. w/ very weakly developed sericite near strgrs., thin capping: lim. after py. w/ secondary hm.derived from py.			10		N.S.							
			20		.07	Nil						
			30		.08	Tr.						
			40		.08	Tr.						
			50		.07	.001						
Sulphide zone @52'			60		.44	Tr.						
			70		.40	Nil						
			80		.44	.001						
			92		.38	.001						
			100		.29	.001						
			110		.22	Nil						

LOGGED BY BLW DATE 22 Mar. 66'

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

PROJECT Atlas

Coörd. : N	BEARING Vert	HOLE Nº 302
E	INCL.	COLLAR ELEV.
START COMPL.		DEPTH 151

DESCRIPTION	Geol.	Assay INT.	C/R	% Cu	% Mo	%	%	oz.	oz.	% EQUIV.	GROUPING — RMKS.
135-151 Quartz Monzonite Hypogene zone, Holoxline, Weak argillic Alt. Pyrite and trace chalcopyrite sparingly occur in rare fractures. Estimated total sulfide is 2%, occuring entirely as fracture filling.		125-130	90	.43	.009						
		130-135	92	.40	.009						
		135-140	95	.21	.005						
		140-145	90	.20	.004						
		145-151	95	.21	.003						
			END OF HOLE								

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 1

PROJECT B. S. & K.

Coord. N E	BEARING	HOLE Nº RDH 310
	INCL. Vertical	COLLAR ELEV.
START 22 Mar. 66 COMPL. 22 Mar. 66'		DEPTH 140'

DESCRIPTION	GEOL	FOOTAGE	WT.	C/H	% Cu	% Mo	%	%	oz	oz	% EQUIV	GROUPING — RMKS.
QUARTZ MONZONITE PORPHYRY												
Argillically altered, "average" hardness.												
Sulphide zone @55'		55			N.S.							
		60			.82	.004						
Some dampness 60 - 70'		70			.82	.009						
		80			.65	.037						
Moisture increase @ 89'		90			.30	.010						
Increase qtz. @ 90½'		100			.26	.009						
		110			.29	.010						
		120			.32	.023						
		130			.26	.011						
		140			.23	.006						

LOGGED BY BLW DATE 22 Mar. 66'

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 1

PROJECT B. S. & K.

Coörd. : N	BEARING	HOLE Nº RDH 311
E	INCL. Vertical	COLLAR ELEV.
START 23 Mar.66' COMPL. 23 Mar.66'		DEPTH 150'

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz	oz	% EQUIV	GROUPING — RMKS.
SILVERBELL DACITE - argillically altered and siliceous, medium hardness but very abrasive, qtz. sulphide strgrs. w/very weakly developed sericite near strgrs., thin capping: lim. after py. w/secondary hm. derived from py. Sulphide zone @ 48½'			50		N.S.							
Mixed oxide & sulphide @ 62'			60		.73	.002						
			70		.20	.001						
All sulphide @ 80'			80		.26	Tr.						
			90		.23	.033						Fines from dust Collector bin (90-100') Cu .38%, Mo. 004%
			100		.22	.002						
			110		.23	.005						Minor Cave @ 100-110
			120		.23	.002						
			130		.22	.003						
			140		.22	.003						
Some dampness @ 150'			150		.28	.002						

LOGGED BY BLW DATE 23 Mar. 66'

PRELIMINARY FEASIBILITY

EVALUATION

B. S. & K. MINING CO.

March 27, 1970

PRELIMINARY FEASIBILITY EVALUATION

B. S. & K. MINE

INTRODUCTION

The B. S. & K. Mine was evaluated by Messrs. C.Osborn and Grover Heinrichs in a report forwarded to Corporate Management, March 19, 1970.

The favorable features of this mining proposal which encouraged further consideration for possible acquisition were as follows:

1-Immediate development to provide Essex producer priced copper at a rate of 5,000 tons per year over a five year period commencing within three years.

2-Revenue to support a long term exploration program.

3-Early development of mining and processing know-how within Essex which can serve as a base for expanded production.

4-Acquisition of property with deep reserve sulfide copper potential.

5-Rights to a property that could be used to negotiate a long term relationship with a primary producer.

The initial proposal for joint development of the property was outlined by Mr. Abe Kalaf, one of the principal owners, in a meeting

In Tucson on March 17, 1970. A copy of the verbatim transcript of Mr. Kalaf's notes is in Exhibit "A". Mr. Kalaf set a March 31, 1970 deadline for Essex to reach a decision to proceed with a six (6) month option.

The mineralization of the deposit and the property configuration immediately raised questions as to property being a successful mining venture. Also, the limited drilling data made the ore reserve estimates speculative. The mining plan, the metallurgical processing and the costing all needed refinement or confirmation prior to an option commitment.

This study was undertaken to assemble sufficient engineering evaluation to enable a decision on an opinion. A decision to proceed would result in an in-depth study during the six (6) month option period requiring drilling and testing to define ore grade, minerology, stripping and mining plans, metallurgical processing and costing.

RESULTS OF INVESTIGATION

The accompanying documents in the Exhibits are reports from consultants which provide interpretations of the available data. The specialists used were selected on the basis of technical competence and practical operational experience. Reports included are as follows:

Exhibit B - Legal opinions from the firm Verity & Smith who specialize in mining law in Arizona. Mr. Smith's letter deals with the following issues:

1. A preliminary search to establish B.S.& K. property rights.

2. An opinion regarding the extent and cost for litigation to gain egress rights across ASARCO property.

3. A recommended provision to be included in an option agreement, protecting Essex in the event that a full title search does not give B.S.& K. full title.

Exhibit C - A report from Dr. Kent Perry, Economic Geologist at Hazen Research, Inc., covering two aspects of the previous geological interpretations by Heinrichs Geoexploration.

<u>Results of Investigation contd</u>.

1. Re-evaluation of ore reserves in the chalcocite blanket.

2. An opinion of the deep ore reserve potential based on structure.

<u>Exhibit D</u> - A report by James Kinnison, a geologist, formerly with ASARCO, considered to be an expert in this mining district. This is an opinion as to the deep reserve potential of the property.

<u>Exhibit E</u> - A report by J.W.Still, mining consultant on the limitations of mining the irregular and narrow dimensions of the property. Mr. Still included an estimate of ore reserves in the blanket based upon mining concepts.

Exhibit E also includes an estimate from the M.M. Sundt Construction Co., regarding earth moving costs which was used to derive a stripping and mining cost.

<u>Exhibit F</u> - A report from Hazen Research Inc., evaluating the metallurgical process proposed for extraction from the chalcocite and the refining to electrowon cathodes. This report also covers a re-estimation of the capital and operations costs.

Results of Investigation contd.

A letter from Mountain States Mineral Enterprises, Inc., also is included which supports a number of capital estimates.

Economic Evaluations

Exhibit G includes all cost evaluation data. Included in this exhibit are the following:

Table 1 - A summary of mining and metallurgical criteria. This tabulation shows two options; (1) processing only the chalcocite ore and (2) processing the chalcocite and low grade oxide ore.

Table 11 a - Capital costs for Option A (processing chalcocite ore only).

Table 11 b - Capital costs for Option B (processing both chalcocite and oxide ores).

Table 111 a - Operating costs for Option A.

Table 111 b - Operating costs for Option B.

Table 1V a - Economic evaluation for Option A.

Table 1V b - Economic evaluation for Option B.

DISCUSSIONS & CONCLUSIONS

The economic evaluations have been based on two options.

Option A - Vat leaching only 5,000,000 tons of chalcocite ore averaging 0.6% copper.

Option B - Vat leaching the 5,000,000 tons of chalcocite ore and heap leaching 7,000,000 tons of oxide ore averaging 0.12% copper.

This approach was taken because of the lack of minerological data on the so called oxide ore. Option A, therefore, is a conservative approach which would indicate assured success of the project if pay-out could be achieved. It is probable that the low-grade oxide ore can be leached although the success of the operation can only be assessed following more extensive testing. Option B, should be used in evaluating the economics of the project for it is based upon the most realistic assumptions and criteria.

The key factor in the project is the mining costs, thus, considerable attention has been given to costs for stripping overburden and mining the ore. The initial report used 30¢/ton for waste stripping and 40¢/ton for mining. This study uses an average of 45¢ for Option A which was intended to give this plan a more favorable opportunity for success.

Discussions and conclusions contd.

Option B was based on 50¢/ton average which is probably a more realistic cost.

A summary of the data presented in Exhibits G is as follows:

Option A

Total Income	$25,143,750	
" Capital Cost		$6,977,114
" Operating Cost		18,100,000
	Total Cost	$ 25,877,114
Earnings before Interest and taxes (7 years)		66,636

(no pay-out and cash flow computation)

Option B

Total Income	$28,608,000	
" Capital Cost		6,977,114
" Operating Cost		20,506,000
	Total Cost	$27,483,114
Earnings before interest and taxes (7 years)		$ 1,124,886

The economic evaluations have been based on supplying copper at producer cathode price (55-7/8¢)

Discussions and Conclusions contd.

Thus, no credit has been taken for corporate savings resulting from the market differential of approximately 20¢/lb. In Option B, the corporate saving would be over $10,000,000 over the seven (7) year investment period.

Another factor to be weighed in a decision is the residual interest retained by Essex in a property with strategic location in an area of deep sulfide ore potential. Three professional opinions have been obtained indicating a mixed reaction. The opinion by Mr. John E. Kinnison which concludes

> "Therefore, I cannot recommend investment in the subject property, if such investment is predicated on deep exploration potential."

cannot be ignored. Mr. Kinnison was retained because of his specific knowledge of this area. However, there is evidence that the area does have potential which could only be established by a drilling program.

The location of the deposit also has strategic value in relationship to the ASARCO operation in the Tiro Pit. It is highly probable that the B.S.& K. property ultimately must be mined.

Recommendations:

In view of the fact that the property cannot sustain itself

Recommendations contd.

under the conditions of the Kalaf proposal, it is not recommended that Essex proceed with the option as stated. Recognizing (1) that the property could provide copper at producer price, thus offering corporate savings of over $10,000,000 over a seven (7) year period, and (2) that the property ownership residual is of long term intrinsic value, it is recommended that negotiation with Kalaf proceed along this line:

1. Essex will take a six (6) month option with no initial payment to Kalaf.

2. During the six (6) months, Essex will expend a minimum of $50,000 in drilling at least three (3) deep holes to (a) assist evaluation of shallow reserves and (b) ascertain potential deep ore reserves.

3. Other terms of the option agreement would be defined by further economic evaluation.

4. If an option is exercised and production facilities are installed, Essex will receive all copper at producer price with a maximum of 60¢/lb.

Exhibit A

B S & K JOINT VENTURE PROPOSAL

The following is a verbatim transcription from notes made by Mr. Abe Kalaf in a meeting in Tucson, Arizona, March 17, 1970:

"1. Suitable Joint Venture Partner

2. Contract - 5 yr. for 50M # cu. with floor of 50¢ and maximum of 65¢ # - First call on all cu. at mkt price, thereafter

3. Grant a 6 mo. option for consideration of $100,000.00 and an expenditure of $150,000.00 for drilling purposes only under Essex and BS & K supervision. (18 mo.) - ($250,000.00)

5. At end of option period Essex furnishes 100% of preproduction expenditures and 60% of capital cost for plant facilities - BS & K furnishes balance of 40%.

6. Essex recaptures 100% of preproduction cost and 11% of the 60% capital expenditures. Balance of capital expenditures (49%) recaptured by Essex via depreciation.

7. Above conditions entitle Essex to 49% interest in BS & K Mining property and BS & K retains 51% in mining property and management of operations."

3/27/70
CLARIFICATION OF ITEM 3
60M PAYABLE AT SIGNING.
150M EXPENSE FOR 6 MO. DRILLING & METALLURGY
6 MO — DECISION —
BAL #250M MO. PAYMENT OVER 30 MO
IF IN PRODUCTION @ END OF 3 YRS ALL #500M IS RECAPTURED FROM PROFITS.

Note: In this meeting the only concessions offered by Kalaf were:

1. He recognized the need for one party determining the drilling program, therefore, he concurred with a pre-drilling plan using a consulting referee with Essex's supervision of work.

2. He offered management rights to Essex.

3. He indicated that if he could not raise the 40% capital, he would relinquish the 51% control and take a smaller percent.

Exhibit B

VERITY & SMITH
ATTORNEYS AT LAW
SUITE 902 TRANSAMERICA BUILDING
177 NORTH CHURCH AVENUE
TUCSON, ARIZONA 85701

VICTOR H. VERITY
LEO N. SMITH
JAMES E. MUELLER
JOHN C. LACY
KENNETH L. ALLEN

AREA CODE 602
TELEPHONE 622-7445

March 27, 1970

Mr. Howard Lanier
Plant Manager
Essex International, Inc.
c/o 302 Ramada Inn
404 North Freeway
Tucson, Arizona

Re: B. S. & K. Property

Dear Mr. Lanier:

I am writing this letter to summarize briefly several different matters affecting the B. S. & K. Mining Company property in the Silver Bell Mining District, Pima County, Arizona.

Limited Record Title Examination

This office has conducted a limited record title examination on the seven unpatented claims extending between the Asarco claims in Sections 28 and 33, T. 10 S., R. 8 E., Pima County, Arizona. This preliminary examination covered the New York, Nevada No. 2, Georgetown, B. S. & K. Nos. 3, 4, 5 and 15X claims. Boundaries of the claims, as between B. S. & K. and Asarco, were adjudicated in recent litigation.

The entire undivided title to the latter four claims is vested in B. S. & K. Mining Company. With respect to the Nevada No. 2 claim, record title to an undivided one half interest is vested in B. S. & K. Mining Company. With respect to the remaining undivided one half interest in the Nevada No. 2 and the entire undivided interest in the New York and Georgetown, record title is vested in one G. E. Baker, subject to a "Conditional Sales Contract" whereunder Baker agreed to sell all his right, title and interest in and to the three claims to B. S. & K. Mining Company (a copy of which is attached). This Contract called for payment in full on or before March 15, 1960 and I would assume that B. S. & K. Mining Company performed. However, our search indicates that no deed from Baker to B. S. & K. has been recorded.

Arizona Condemnation Statutes

You also asked that I summarize briefly the Arizona Statutes pertaining to condemnation of rights of way for mining purposes. A.R.S. § 12-1111 provides as follows:

> Subject to the provisions of this title, the right of eminent domain may be exercised by the state, a county, city, town, village, or political subdivision, or by a person, for the following uses:
>
> ***
>
> 14. Roads, tunnels, ditches, flumes, pipes and dumping places for working mines, and outlets, natural or otherwise, for the flow, deposit or conduct of tailings or refuse matter from mines . . .
>
> ***

The prerequisites for condemning land for the purposes set forth above, are (1) that the use to which the property is to be applied is a use authorized by law, and (2) that the taking is necessary to such use.

In addition to the general provisions relating to exercise of the power of eminent domain by a private citizen (or company), A.R.S. § 12-1201 through 12-1203 provides a manner in which "private ways of necessity" can be acquired. A private way of necessity is defined by A.R.S. § 12-1201 as a:

> Right of way on, over, across, or through the land of another for means of ingress and egress, and the construction and maintenance thereon of roads, overhead transmission lines, pole lines, power lines, canals, ditches, flumes, shafts, tunnels, pipelines, drains . . . and tramways, including, but not limited to, aerial tramways and industrial railroads for mining, milling, lumbering, agricultural, domestic or sanitary purposes.

The method of acquiring such a private way of necessity is by condemnation and the party bringing the action is entitled to "take lands of another, sufficient in area for the construction and maintenance of the private way of necessity." A.R.S. § 12-120 specifically provides that "private ways of necessity" may cross "mines or mining claims."

An important factor with respect to condemnation proceedings that must be kept in mind is that such proceedings can be time-consuming and costly, especially in the case of a private person condemning a private way of necessity. The private individual is not entitled to possession of the property to be condemned until <u>after</u> completion of the condemnation proceeding and payment of the compensation to the party whose property is being taken. Litigation can be extensive if all routes of appeal are exhausted before an award becomes final.

Title Examination Provisions

Because of time limitations, it has been suggested that perhaps Essex and B. S. & K. could enter into an agreement which would be contingent upon verification of title by Essex prior to payment of the initial consideration. The following is one type of clause that might be used for this purpose:

> Upon execution of this Agreement, Essex shall place in escrow with ________________________________ (hereinafter called "Escrow Agent") the sum of $______ __________ together with an executed copy of this Agreement. Essex shall have a period of forty-five (45) days from the effective date of this Agreement to make an examination of Optionor's title. Optionor agrees to deliver to Essex for inspection all abstracts of title and other papers and instruments pertaining to the title to the Subject Premises which Optionor may have. If Essex shall determine that Optionor's title is satisfactory, it shall notify the Escrow Agent to pay the money held in escrow to Optionor. If Essex shall determine from its title examination that Optionor's title is not satisfactory, it shall so notify Optionor, specifying the defects, and Optionors shall have a period of thirty (30) days thereafter to cure such defects. If such defects are not cured to the satisfaction of Essex within the said 30-day period, Essex may, at its option, notify the Escrow Agent to return to Essex all money held in escrow, or Essex may waive the defects and notify the Escrow agent to pay such money to Optionor. If Essex shall notify the Escrow Agent to return such money to Essex, Essex shall execute and deliver to Optionor a quitclaim deed conveying to Optionor all of Essex' rights hereunder. If the money is paid to Optionor, it shall constitute the initial consideration as recited in Section ___ of this Agreement.

Tax Treatment - Exploration Expenses

As I mentioned to you over the phone, we have not been involved in extensive analysis of the tax consequences resulting from elections to either expense or capitalized exploration expenditures. Since depletion, in the event of production, can be taken on only the greater of cost basis or percentage basis, it is usually advantageous to expense rather than to capitalize exploration expenditures. There are limitations upon expensing of exploration expenditures and there are elections available as to whether to currently deduct expenses or to defer deductions. The decision as to whether exploration expenses are deducted or capitalized will of course depend upon the accounting practices involved and upon the total tax picture of a particular exploration company. I assume that your accounting department would be in a position to advise as to the treatment heretofore used by Essex. If you would want us to give a more detailed examination, please let me know.

Time Period - Title Examination

It appears that it would not take more than a couple of weeks to run the check on the title on the unpatented claims. With respect to the patented claims, I would guess that a title company would possibly be able to furnish a report within the same time period.

If you have any questions concerning the above, please let me know.

Very truly yours,

Leo N. Smith

LNS:vs
Enclosures

1518 522

CONDITIONAL SALES CONTRACT

I, the undersigned, G.E. BAKER, conditional vendor, in consideration of the payment of One Thousand and no/100 Dollars hereby sell all my right, title and interest in and to the following described mining claims unto B.S.&K. Mining Co., and B.S. & K. Mining Co. hereby purchases from the conditional vendor those certain mining claims, described as: "New York", recorded in Book 77, page 540, Pima County, Arizona; "Georgetown", recorded in Book 750, page 301, Pima County, Arizona; and "Nevada #2", all in the Silver Bell Mining District of said County and State, according to the following terms and conditions:

1. Title to the aforedescribed mining claims shall remain in the vendor until the purchase price is paid in full.

2. The full purchase price shall be paid as follows: $250.00 upon the execution of this agreement and $750.00 on or before March 15, 1960.

3. In event of default by the conditional vendee, B.S. & K. Mining co., it is understood that the conditional vendor shall be entitled only to the payments made hereunder before default as liquidated damages, and to no other damages.

B.S. & K. MINING CO.
Conditional Vendee

By George Kabel
Title Vice-Pres.

CONDITIONAL VENDOR

March 27, 1970

Dear Howard:

I neglected to attach a copy of the enclosed Conditional Sales Contract to the letter delivered to you this morning.

LNS

Exhibit C

HAZEN RESEARCH, INC.

4601 INDIANA STREET
GOLDEN, COLORADO • 80401
TELEPHONE 303/279-4547

March 25, 1970

Mr. Howard Lanier, Manager
Essex International Inc.
Three Rivers, Michigan 49093

Re: HRI Project #722

Dear Howard:

The attached letter report covers our investigation of the metallurgical aspect of treating copper ores from the New York claims, all in accordance with our letter agreement dated March 20, 1970.

This report is written confirmation of the information given to you verbally, by telephone, on March 24, 1970.

Yours very truly,

Ralph H. Light

Ralph H. Light

RHL/ph

Xerox copy mailed to Howard Lanier, Ramada Inn, 404 N. Freeway, Tucson, Arizona 85705, via air mail, special delivery on March 25, 1970.

PRELIMINARY METALLURGICAL ASSESSMENT OF PROCESSING COPPER ORES FROM NEW YORK CLAIMS

March 25, 1970

CONCLUSIONS

We have evaluated the extractive metallurgy associated with leaching and recovery of copper from the B. S. and K. oxide and sulfide ores. We feel that the vat leaching of the high grade ore (0.60% Cu) and heap leaching of the low grade oxide ore offers the best economic approach. Our cost calculations indicate that the capital, depreciation, and operating costs for the chosen process will be $0.38 per pound of copper recovered. We have not included any mining costs (capital or operating), head office administration costs, interest on money, escalation of labor or materials costs, inventory costs, or working capital, property or option costs.

The capital and operating costs presented in the milling operations report given to us by you are considered low. Capital costs did not include heap leaching costs; operating costs were most optimistic and extraction from various minerals was high in one case and low in the other.

If the selling price for the copper produced is $0.56 to $0.60 per pound of copper, the metallurgical costs will consume 2/3rds of the selling price. The opportunity for a large profit does not, therefore, appear to be present.

Hazen Research, Inc.

SUMMARY OF OUR STUDIES

	Estimated Capital costs Total $	Estimated operating costs Total $ for 5 years	Total costs $ per pound copper recovered
Hydrometallurgical plant & operations plus heap leaching	$14,100,000	$11,453,500	$0.509
Heap leaching	6,000,000	8,595,000	0.392
Vat leaching & heap leaching	7,640,000	9,953,500	0.381

Pounds copper recovered (total):

Hydrometallurgical plant plus heap leaching	-	50,200,000
Heap leaching	-	37,200,000
Vat leaching and heap leaching	-	46,200,000

EXTRACTIVE METALLURGY ON B. S. AND K. ORE

A review of extractive metallurgy applicable to the B. S. and K ore, an oxide sulfide ore, indicates only three possibilities for processing. We have eliminated processes such as pressure leaching with ammonia, acid pressure leaching, etc., as the ore grade is low. These latter processes are usually applicable to concentrates of one type or another. Roasting has been eliminated because of probable high costs for such a low grade ore.

We have, therefore, considered three types of processing. These are as follows:

1. Vat leaching the higher grade ore and heap leaching the lower oxide ore.
2. Heap leaching of all ores.

3. Hydrometallurgical treatment of the high grade ore by grinding, leaching, filtration, etc., and heap leaching of the low grade oxide ore.

To develop any new processing technique would require a large test program and could yield negative results. The metallurgical assumptions made in our studies for this ore are considered realistic.

$$\text{Pounds copper produced per year} = \frac{46{,}200{,}000}{5} = \underline{\underline{9{,}240{,}000}}$$

METALLURGICAL DATA

Assumptions:

1. Ore grade of vat leaching ore is 0.60% Cu total, made up as follows:

Copper A-S[1/]	=	0.196%
Copper as chalcocite	=	0.302%
Copper as chalcopyrite	=	0.102%
Total		0.600%

2. Total tons of 1. -- 5,000,000
3. Ore grade of heap leach material all as oxides -- 0.15% copper
4. Total tons of 3. -- 7,000,000
5. Percent recovery from chalcocite = 70% (at 25°C using acid + ferric sulfate, AIME Trans. Vol. 106)

   Percent recovery from copper A-S = 85%

   Percent recovery from chalcopyrite = 0%

   Heap leaching of oxides -- 40% over 90 days considering low grade and no test work.

1/ A-S = Acid Soluble

PRODUCTION

Vat Leaching

Tons per day	= $\approx$ 3,000	
Life	= 5 years	
Pounds of copper from chalcocite	= 0.70 x 5,000,000 x 6.04 =	21,140,000
Pounds of copper from A-S copper	= 0.85 x 5,000,000 x 3.92 =	16,660,000
	Total	37,800,000

Heap Leaching

0.40 x 3.0 x 7,000,000		8,400,000
	Total recoverable pounds	46,200,000

Capital Costs -- Vat and Heap Leaching

Item	Dollars Vat Leaching	Heap Leaching	Total
Crushing	707,000	-	707,000
Leaching vats	1,660,000	-	1,660,000
Processing plant (LIX-electrowinning)	2,500,000	-	2,500,000
Site preparation	50,000	-	50,000
Roads	-	150,000	150,000
Ponds (main ponds)	-	40,000	40,000
Preparation heap leaching piles	-	50,000	50,000
Tailings pond	150,000	-	150,000
Collection ponds, ditches	-	10,000	10,000
Mechanical equipment	-	40,000	40,000
Piping	Incl. above	200,000	200,000
Electrical	Incl. above	30,000	30,000
Water system	25,000	5,000	30,000
Equipment - lab, office, shops	30,000	-	30,000
Buildings - for mill	200,000	-	200,000
Buildings - office, laboratory shops, warehouse	150,000	-	150,000
Engineering & construction			600,000
Preproduction test work			50,000
Total			$6,647,000
Contingency 15%			993,000
Grand total			$7,640,000

No allowance in above for escalation of labor, interest on money, starting inventories, or working capital.

$ Per pound copper recovered $\dfrac{7,640,000}{46,200,000}$ = $ 0.165

Hazen Research, Inc.

Operating Costs -- Vat and Heap Leaching

Item	Dollars Per Year			\$/lb Copper Recovered
	Vat Leaching	Heap Leaching	Total	
Supervision			\$ 190,000	\$0.0205
Labor			387,800	0.0420
Analytical			45,000	0.0050
Maintenance	140,000	10,800	150,800	0.0163
Reagents	760,000	126,000	886,000	0.0959
Power	111,800	17,200	129,000	0.0140
Water	52,500	11,000	63,500	0.0069
Heat			12,000	0.0013
Operating supplies	50,000	14,600	54,600	0.0059
Miscellaneous			72,000	0.0078
Total			\$1,990,700	\$0.2156

No allowance in above for escalation of labor or materials.

Exhibit D

HAZEN RESEARCH, INC.

4601 INDIANA STREET
GOLDEN, COLORADO • 80401
TELEPHONE 303/279-4547

March 25, 1970

Mr. Howard Lanier, Manager
Essex International, Inc.
Three Rivers, Michigan 49093

Re: HRI Project No. 722

Dear Mr. Lanier:

This letter report is a preliminary evaluation and interpretation of prior geologic work on the New York claims of the B. S. and K. Mining Company, located in Sections 28 and 33, T. 11 S., R. 8 E., Pima County, Arizona. This property is located in the Silver Bell Mining District.

CONCLUSIONS

1. Inferred ore reserves, based on presently available rather sketchly drilling data, are 5,450,000 tons of minable ore with an average grade of 0.68% Cu.

2. Overburden computed using 60^{o} pit slopes with a 20 foot wide safety berm every 50 feet of vertical depth is 14,100,000 tons, giving a stripping ratio of 2.6:1.

3. Of this overburden an estimated 8,100,000 tons with average grade of approximately 0.12% Cu might be selectively mined for dump leaching.

4. Ore reserves in the supergene chalcocite blanket of the New York area could be adequately defined with approximately 16 proposed shallow drill holes.

5. Geologic data presently available suggests excellent potential for deep exploration. However, the three deep 1500-foot holes proposed in the Heinrichs report of March, 1970 would be warranted only if a cooperative agreement can be reached with

American Smelting & Refining Company to consolidate the New York area with surrounding A. S. & R. property, unless your objective is a high grade target rather than open pit ore.

ORE RESERVES

No field geologic work was done by Hazen Research, Inc. Our evaluation is based on a very limited amount of drilling data given in a Heinrichs Geoexploration Company report of March, 1970, and our work assumes the validity of core logs and assays given. Particularly questionable is the data for three A. S. & R. drill holes. At this stage ore reserves are strictly inferred.

The ore reserve estimates were made using a Numerical Surface Techniques software package on an IBM-1130 computer system.

Plate 1 is an isopach of overburden thickness. This plate also shows the boundary of the New York claims, the location of present drill holes (including three A. S. & R. drill holes which we arbitrarily numbered ASR1, ASR2, and ASR3), and locations of 16 proposed shallow drill holes that could be used to more closely define reserves.

The ore perimeter outline shown on Plate 1 is based on 60° pit slopes with 20 foot wide safety berms every 50 feet of vertical depth. These pit slopes combined with the highly irregular property boundary result in only 61% of the projected ore area being minable.

Plate 2 (clear blue overlay) is an isopach of inferred ore thickness using a 0.30% Cu grade cutoff.

Plate 3 (clear red overlay) is an isograd surface of ore grade within the ore zone defined by using 0.30% Cu cutoff.

Calculations gave the following results:

Inferred minable ore:	5,450,000 tons of 0.68% Cu
Overburden:	14,100,000 tons
Stripping ratio:	2.6:1

Dump leach (included in above overburden figure, might possibly be selectively stripped): 8,100,000 tons of 0.12% Cu.

DEEP EXPLORATION POTENTIAL

The Silver Bell district is related to a first class intersection 1/ of four major tectonic lineaments related to ore deposits:

1. Texas Lineament (E-W).
2. Utah-Arizona belt (N-S).
3. Morenci belt (NE-SW).
4. Southwest Arizona belt (NW-SE).

In addition, core logs from the B. S. and K. property indicate promising types of mineralization; a highly fractured porphyritic host rock, strong argillic alteration, quartz veinlets, and sulfide mineralization both within the quartz veinlets and disseminated in the host rock.

Deep exploration would be warranted only if agreement can be reached with A. S. & R. to consolidate the New York area with the surrounding A. S. & R. property for a joint venture.

PROPOSED DRILLING

We were requested to propose drilling locations. We feel that the shallow supergene chalcocite zone of the New York area could be adequately defined with a minimum of 16 drill holes, as indicated on Plate 1.

If deep drilling is done, we suggest that the following three proposed holes be extended to a depth of 1,500 feet:

PDH 250 feet NW of ASR1.
PDH 480 feet NE of DH300.
PDH 150 feet NE of DH309.

Very truly yours,

J. Kent Perry

J. Kent Perry
Senior Research Engineer

JKP/ph

1/ Mayo, E. B., 1958, Lineament Tectonics and Some Ore Deposits of the Southwest: Mining Engineering, November, p. 1169-1175.

Exhibit E

901 S. Campbell Ave. • P. O. Box 2164 • Tucson, Arizona 85702 • (602) 792-2800

MOUNTAIN STATES MINERAL ENTERPRISES, Inc.

March 25, 1970

Re: No. 70-004

Mr. Clyde E. Osborn, E. Met.
Essex International, Inc.
5315 East Broadway
Tucson, Arizona

Dear Sir:

Re: BS & K Estimate

As per your request of March 23, I have reviewed your estimate for the BS & K development to the extent possible in two days. In accordance with our discussion of March 23, I have estimated the electrical costs for the crushing and screening plant.

An estimate has also been prepared for an alternate with 18' deep leaching vats having the same holding capacity as the 12' deep vats covered by your estimate. Less concrete will be required and the cost for this alternate will be approximately $100,000 less than for the 12' deep vats.

Quotations were obtained for conveyors and screens for the crushing plant. The cost of this equipment totals $57,800, chutes and supports not included. It therefore appears that the $150,000 allowed for miscellaneous equipment in the crushing plant is adequate. The conveyor lengths were calculated on level site basis. It is very possible that by making use of sloping ground the conveyor lengths and horsepower required can be reduced considerably. It is assumed that due to the remote location of the plant, no dust collection will be required, but that a few water sprays will be installed to cut down the dust.

$45,000 has been added to the crushing plant estimate to cover electrical work.

As agreed during our discussion this morning, I have marked the suggested changes to the estimate in pencil on the extra copies of the estimate sheets that you gave me.

It is my feeling that the estimate is realistic for an economically designed plant with the crushing facilities placed in the open using "contractors' type" semi-portable crushers, screens and conveyors.

If we can be of further service, please do not hesitate to call.

Very truly yours,

T. Skagestad

T. Skagestad, P. Eng.

TS:el

Exhibit F

BS&K

HEINRICHS GEO-EX

DEEP EXPLORATION APPRAISAL

by

John E. Kinnison

Chief Geologist

March 26, 1970

Geo-Comp Exploration, Inc.
1706 W. Grant Road
Tucson, Arizona 85705
(602) 623-5448

GEO-COMP EXPLORATION, INC.

GEO-COMP EXPLORATION, INC.
SUBSIDIARY OF G.F.I. COMPUTER INDUSTRIES, INC.
1706 WEST GRANT ROAD • TUCSON, ARIZONA 85705 • Tel. 602/623-5448

EXPLORATION FOR NATURAL RESOURCES

March 26, 1970

Mr. Grover Heinrichs
Heinrichs Geoexploration Company
808 W. Grant Road
Tucson, Arizona 85705

Dear Mr. Heinrichs:

On the 24th of this month, you inquired if I was familiar with the Silver Bell Mining District, and in particular, with the area between the El Tiro Pit operated by American Smelting and Refining Company, and land held in the vicinity of the Atlas Mine by the BS&K Company. As I then informed you, I have had considerable acquaintance with the Silver Bell district in general, gained while employed by ASARCO. This company, as you are aware, has accumulated much knowledge concerning the geology and mineralization of this area, gradually collected during exploration and operation of the Asarco mines. Although I am not ethically free to divulge details of ore occurrence on Asarco ground, I have drawn freely on my general experience in the district. Also, as I informed you, I had not previously been into the specific area of BS&K ground which is now under consideration, and thus made a brief field inspection to establish certain geologic features in the subject area.

You informed me that a client of yours was considering the acquisition of BS&K claims north of El Tiro, and that this client desired to have an independent geologic evaluation of the exploration possibilities within these claims, with special emphasis on the possibilities to be found at considerable depth beneath the surface.

I herewith present my findings and conclusions.

SUMMARY AND RECOMMENDATIONS

The "corridor" of BS&K claims, as shown on the attached sketch, is without question a part of the same structural block as the area of the thin chalcocite blanket drilled by Asarco near the Old Silver Bell Camp. The character of the leached capping is similar in both areas, and the presence of a chalcocite blanket beneath the BS&K claims is thus not surprising. The total sulphide content, estimated from the leached capping, is less than 3%.

The I.P. anomaly shown by Hanna's work lies to the north of the BS&K claims, and does not appear to be reflected by the outcrops on those claims.

Our Mr. Fink has briefly examined the Hanna I.P. data, and in his opinion the psuedo section plot of this data suggests that the strong anomaly is associated with sulphides at relatively shallow depths, and that the sulphide content may decrease with increasing depth. He also notes that the strongest response is on ASARCO's ground to the north of the BS&K "corridor", and that the area of strongest response is probably limited in areal extent. However, I.P. data indicate that the rock surrounding the I.P. anomaly also contains sulphides, but of lesser concentration. This conforms to conditions I observed in the field.

In any event, it is questionable whether I.P. methods could respond to the only ore-grade target at the probable depth involved, say, 1500 feet or greater.

I see no chance for ore at depth within the dacite porphyry, below the near surface chalcocite blanket.

There is a remote possibility the Mescal limestone may underlie the dacite at a depth of 1500 feet. If this is actually the case, the limestone might have responded, within the limits of pervasive mineralization, as a favorable host rock. Limestones near the El Tiro pit average a little under 1% copper. The Mescal limestone in this area (if present at all) would not exceed 150 feet in thickness. Such a thin host rock, at depths of 1500 feet or more, would have to be of considerably higher grade to have intrinsic mining value.

I must emphasize that the geologic hazards are extremely great. The only hope of improved mineralization at depth lies with the Mescal limestone, and its mere presence is only a theoretical possibility. Therefore, I cannot recommend investment in the subject property, if such investment is predicated on deep exploration possibilities.

FIELD EXAMINATION

Tuesday, March 24, 1970, I made a brief inspection of the southern corridor of claims held by BS&K to determine the nature and extent of rock and alteration. This corridor is, according to claim maps furnished me by Grover Heinrichs, shown to be encircled by ground held by the American Smelting and Refining Company.

I first revisted an area on ASARCO ground to the south, near the old Silver Bell campsite, with which I was already familiar. In this area I am aware that ASARCO has drilled out a medium grade - about 0.6% Cu - chalcocite blanket about 50 feet beneath the level of the old Silver Bell camp. The rock is here made of dacite porphyry, which in this area is known to have a sill-like configuration, and which on the basis of a few deep drill holes on ASARCO ground is apparently about 1500 feet thick. The leached capping over the chalcocite zone on ASARCO's ground is strongly altered, but with relatively little indication (by the color and nature of the limonite present) that chalcocite is in fact beneath this capping. The primary mineralization, consisting of chalcopyrite, is relatively low, however, usually grading about 0.1 to 0.2% copper.

The dacite extends northerly to the BS&K ground, in their isolated corridor, and is there similarly altered. Progressing eastward across this corridor however, the rock alteration and mineralization fades and then terminates. The attached sketch shows the approximate eastern limits of strong mineralization with indication of the presence of sulphides prior to oxidation and leaching. Although there are a few scattered fissure zones which have been mineralized, easterly of the contact which I have drawn, these are isolated from the main zone of alteration. Thus, it appears probable that the chalcocite zone will not extend much further east than the line which I have shown as the margin of strong mineralization. Any possibility of ore occurrence at depth must be considered with respect to the limits of pervasive mineralization as exposed at the surface.

I noted a single breccia pipe with a larger quantity of oxidized sulphide cavities than the surrounding dacite. The limonite filling these cavities is indicative of derivation from copper sulphides, and may well represent a higher copper content than that of the surrounding dacite porphyry. Unfortunately, the breccia is too small to be of commercial interest.

DISCUSSION

ASARCO had, during the time I was employed by them, done interspaced drilling to substantiate values from old churn drill holes sunk prior to 1920, in the vicinity of the old Silver Bell camp. In this area, a relatively low grade chalcocite blanket, fairly thin, lies approximately 50 feet beneath the surface of the old Silver Bell campsite. The mineralization occurs in dacite porphyry. The leached capping over this zone of chalcocite on ASARCO's ground is not, in itself, diagnostic of the existence of this chalcocite blanket, although strong pervasive rock alteration and evidence of former sulphides is present in the leached capping. The narrow corridor of BS&K claims now in question lies on an extension of the structural block containing this dacite porphyry, and the leached capping is similar to that near the old Silver Bell camp. It is not surprising, then, that drill holes on the BS&K ground show low to medium grade chalcocite at a shallow distance below the surface.

The following points are pertinent to consideration of possible deep exploration targets.

1. The dacite porphyry, as known through mining and drilling in the Silver Bell area, is one of the less favorable host rocks for disseminated mineralization. The grade of primary copper, as chalcopyrite, is

generally less than 0.2% copper, and is frequently less than 0.1%. Drill holes which have penetrated considerable distance into some of the dacite porphyry bodies do not show any tendency to increase in grade with increasing depth. Further, I should point out that, in the general case of porphyry copper deposits, an increase in grade with depth can be found in a very few deposits only. The vast majority may show vertical variations in grade, but rarely is there a significant increase or decrease in the amount of copper when explored vertically. I see no reason to expect an improvement with depth of copper content in the dacite on the BS&K claims.

2. The dacite porphyry in the vicinity of the old Silver Bell camp, and probably also on the BS&K claims, is evidently a low-dipping sill, rather than a plug with a great vertical depth. A few deep ASARCO drill holes near the El Tiro pit suggest that this sill is over 1500 feet thick.

The general habit of the dacite porphyry at Silver Bell is to intrude along bedding planes of the sedimentary rocks, and thus split the sedimentary section.

On Jesuit Peak, west of the old Silver Bell camp, the dacite porphyry appears to have intruded beneath the Cambrian Bolsa quartzite. Throughout much of southern Arizona, the Bolsa quartzite rests on the pre-Cambrian Apache group sedimentary series, of which one member - the Mescal limestone - is a favorable host for copper mineralization in many districts. In the Silver Bell area, the Apache group has been partly eroded prior to deposition of the Bolsa quartzite. Thus, across the valley to the northwest the full section of Apache group is present, while about 5 miles north of the Atlas mine a much thinner section is present, and in the Waterman mountains south of Silver Bell the Apache group is absent. There is a remote chance, therefore, that on the BS&K ground the Apache group and Mescal limestone might have been present, and thus been split from the Bolsa quartzite by the intervening dacite porphyry sill. If the Mescal limestone was preserved from pre-Bolsa quartzite erosion, and is actually present beneath the sill, there is a possibility that it might be mineralized with grades approaching 1% Cu. If it is assumed that the boundary of mineralization as seen at the surface may be projected vertically downward, then the potential for mineralized limestone at depth would occupy that portion of the BS&K corridor to the west of the limits of mineralization as shown on the attached sketch.

Even if the Mescal limestone is present beneath the sill, and is mineralized by chalcopyrite approaching the average of the mineralization in limestone at El Tiro - about 1% - its value, other than as a negotiating lever with ASARCO, is doubtful. It is my opinion that expenditure of option money, to say nothing of the cost of drilling, is an unwise investment in view of the geologic uncertainties listed above, combined with the questionable value of deep mineralization in the Mescal limestone.

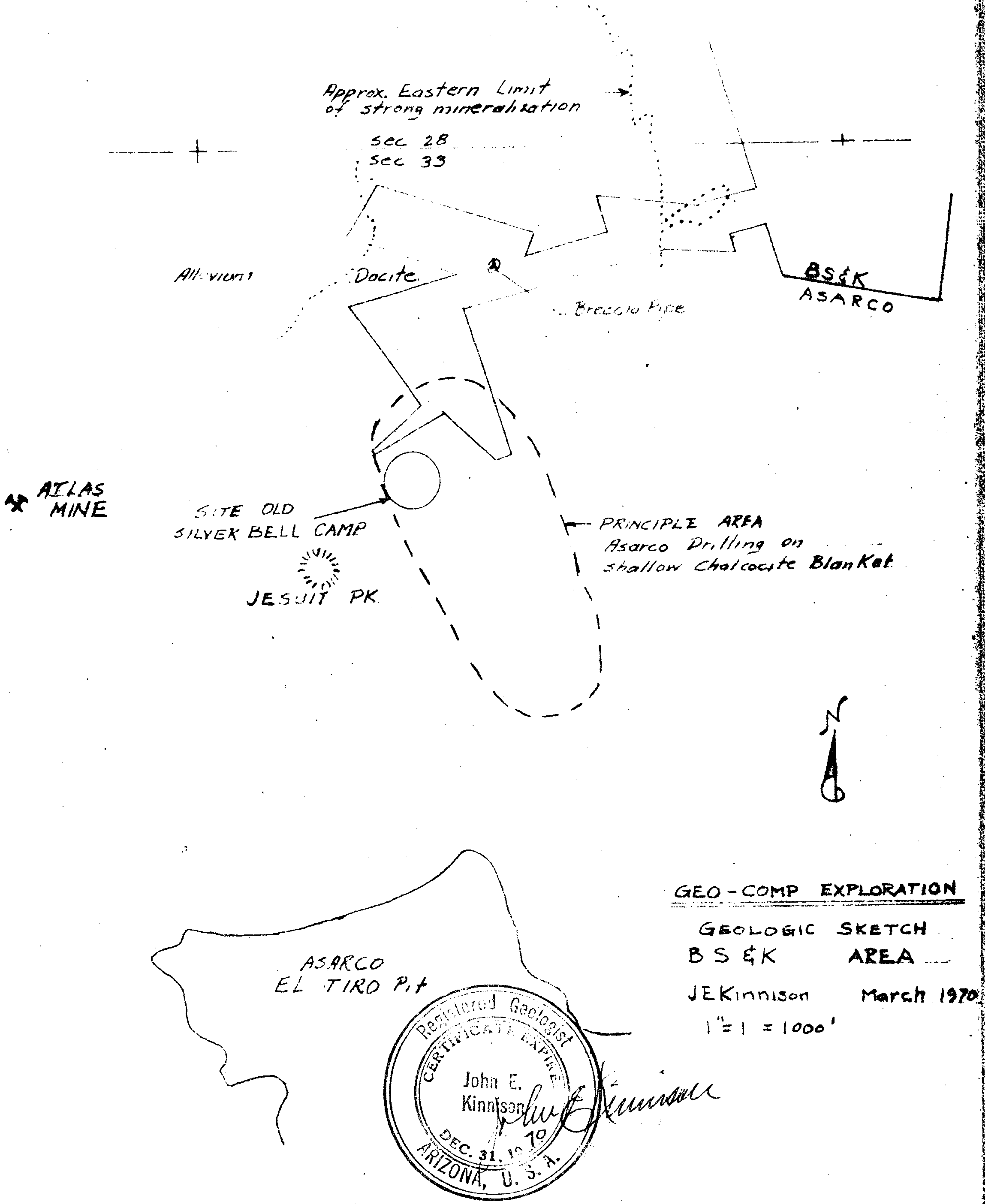
I trust that this letter will satisfy the purposes of your client.

Yours very truly,

Registered Geologist
CERTIFICATE
John E.
Kinnison
DEC. 31, 1970
ARIZONA U. S. A.

John E. Kinnison
Chief Geologist
Geo-Comp Exploration, Inc.
Registered Geologist No. 4822
State of Arizona

JEK:cc
Attachments



Approx. Eastern Limit
of strong mineralization
sec 28
sec 33
Alluvium
Dacite
Breccia Pipe
BS&K
ASARCO
ATLAS
MINE
SITE OLD
SILVER BELL CAMP
PRINCIPLE AREA
Asarco Drilling on
shallow Chalcocite BlanKet
JESUIT PK.
N
ASARCO
EL TIRO Pit
GEO-COMP EXPLORATION
GEOLOGIC SKETCH
B S & K AREA
JEKinnison March 1970
1" = 1 = 1000'
Registered Geologist
Certificate Expires
John E.
Kinnison
Dec. 31, 1970
Arizona, U.S.A.

KINNISON, JOHN E.

Vice President and Chief Geologist, Geo-Comp Exploration, Inc.

Tucson, Arizona

Registered Geologist, State of Arizona

Education:

M.S. in Geology, University of Arizona, 1958.

B.S. in Mining Engineering, University of Arizona, 1953.

Experience:

May, 1969 - Present: Chief Geologist, Geo-Comp Exploration. Appointed Vice President in November, 1969. My duties with Geo-Comp have been of four catagories:

a. I directed geologic mapping of certain properties held by General Earth Minerals, an in-house corporation, and formulated plans for exploration by drilling.

b. I have worked with my staff geologists, in the office and in the field, both as a director of a technical development program designed to raise overall expertise of the staff in mining and geological matters, and also as associates in the examinations of specific mineral properties. This mutual association of the past months has materially aided in welding together an expert staff who are able to work together and independently, as the situation demands.

c. I have examined mineral properties in the western U.S. and Canada, for our clients, and made recommendations thereon. Many of these examinations have required initial feasibility studies leading to an estimated economic outcome.

d. I have begun the initial steps leading to programs for regional exploration, as a forerunner of impending mineral exploration funds.

1957 - April, 1969: Geologist, Southwestern Exploration Department, American Smelting and Refining Company. Work included a detailed review of drill hole data at the Mission Mine and on the San Xavier Indian Reservation. From this review, aided by petrographic studies, a clarification was made of rocks and their alteration products into correlative units, and preparation of cross-sections and plan maps which formed the basis for ore reserve calculation. A small exploration shaft was sunk and several thousand feet of drifts, raises, and diamond drill holes were cut prior to developing the open pit which now exists. Had major responsibility for this work and reporting thereon in a final report. Subsequently mapped in detail the Twin Buttes district, and four other porphyry copper districts. Geologic reconnaissance which was conducted led to the discovery of three previously unknown porphyry copper deposits, of which two are now under development. During 1960, supervised a small drilling program and later was charged with all field supervison in Casa Grande area, accounting for about 90 drill holes and an expenditure of more than one million dollars. Four rotary and four diamond drills were operated three shifts. Headed a staff consisting of 17 junior geologists and samplers and shared with the drilling engineer the responsibility of directing all activities pertaining to the quality of the sample, including control of mud chemistry, cementing, bit types, and when necessary, drilling details such as rotary speed, weight on the bit, etc. Following 1964, work consisted of special assignments of short tenure and wide diversity, ranging from prospect evaluations to detailed mineralogical distribution studies with the aid of expert consultants. Made commodity studies and price forecasts for mercury and uranium, and initiated exploration programs for the same. Beginning in 1966, ASARCO greatly expanded its staff, and duties included orientation and training of these new geologists. In April 1968, testimony was given for ASARCO in the dual capacity of witness and expert witness in the U.S. Court of Claims case #443-65 ASARCO, Plaintiff vs. United States of America, defendant; a test case concerning income tax loss deduction of exploration monies. Preparation for this case consumed three months during 1967-1968.

1956: Civil Engineer, City of Tucson, City Engineering Department. Office studies related to city improvement projects. Work consisted largely of surveying calculations in "trouble spots" where surveys, public and private, did not close together; secondarily, writing legal land descriptions.

1955 - 1956: Geologist, U.S. Atomic Energy Commission, Globe, Arizona. Examination and mapping of small uranium mines and prospects, and general field reconnaissance. Logged drill core from Bureau of Mines drills on contract to A.E.C. In collaboration with R. Schwartz, prepared ore reserves of the Sierra Ancha district according to A.E.C. specifications, and recommended drill hole targets.

1954 - 1955: Geologist, Cyprus Mines Corporation. Examination of the Pima Mine near Tucson. Mapped underground levels, and the adjoining Daise Mine. Logged all old drill core as well as new drilling during option period. Surface churn drills and underground diamond drills. Compiled geologic maps and cross-sections for final report.

1952 - 1954: Geology teaching fellowship, University of Arizona, three semesters.

Societies:

Society of Economic Geologists

Society of Mining Engineers, A.I.M.E.

Past Chairman, Arizona Section, Mining Geology Division

Arizona Geological Society

Past Secretary

Publications and Papers:

"The Mission Copper Deposit, Arizona"; in the Wilson volume, Geology of the Porphyry Copper Deposits, Southwestern North America, edited by S.R. Titley and C.L. Hicks, pp 281-287, 1966.

"Probable Origin of the Mission Copper Deposit"; Soc. of Min. Eng. of A.I.M.E., Annual Meeting, 1963.

"Probable Origin of Mission Copper Deposit, Arizona"; A.I.M.E. pre-print No. 63133, 1963.

"Geology of the Mission Copper Deposit, Arizona"; A.I.M.E. Arizona section, Mining Geol. Division, Annual Meeting, 1961.

"Chaotic Breccias in the Tucson Mountains"; Geol. Soc. Am. Cordilleran section, Annual Meeting, 1959.

"Structure of the Saginaw Area, Tucson Mountains, Arizona"; Arizona Geol. Soc. Digest No. 2, pp 146-51, 1959.

"Chaotic Breccias in the Tucson Mountains, Arizona"; Abs., Bull., Geol. Soc. Am., Vol. 70, p 1727, 1959.

"Chaotic Breccias in the Tucson Mountains, Arizona"; Guide Book for Field Trips No. 2, Ariz. Geol. Soc., in collaboration with J.H. Courtright, 1959.

"Chaotic Breccias in the Tucson Mountains, Arizona"; Ariz. Geol. Soc. Digest No. 2, pp 49-57, 1959.

"The Lower Cretaceous Age of the Amole Arkose, Tucson Mountains, Arizona"; Abs. Bull., Geol. Soc. Am., Vol. 65, p 1235. In collaboration with D.L. Bryant, 1954.

"Alteration Features of Porphyry Copper Deposits"; Member of panel discussion, Soc. of Min. Eng. of A.I.M.E., Annual meeting, 1963.

PROJECT
ANALYSIS
CONTROL
EVALUATION
THE TOTAL CONCEPT

M. M. SUNDT CONSTRUCTION CO.

GENERAL CONTRACTORS

P. O. BOX 2592 · 440 SOUTH PARK AVENUE · TUCSON, ARIZONA 85702 · AREA CODE 602 623-7531

March 26, 1970

Heinricks Geoexploration Company
808 West Grant Road
Tucson, Arizona

Attention: Mr. Grover Heinricks

Re: Essex International, Inc.
B. S. & K. Project

Gentlemen:

This will serve to confirm our telephone conversation of today in regard to our budget estimate for mining the above mentioned property.

Our estimate which is based on the presently known general features is as follows:

Mine and haul a distance of approximately 1.5 miles at a rate of 4,000 cubic yards per day @ $1.24/bank cubic yard.

Mine and haul a distance of approximately 1.0 mile at a rate of 4,000 cubic yards per day @ $0.99 per cubic yard.

If the above rate of mining should be doubled to 8,000 cubic yards per day deduct $0.05 per cubic yard.

The above budget estimate is based on the following:

1. Mining a total 15,000,000 tons of material with a conversion factor of 12.5 cubic feet per ton.

2. We propose to control drilling and blasting methods to produce the required fragmentation, however, we do not include costs for secondary blasting.

3. We do not assume the responsibility for determining ore or waste.

4. Costs have not been included for abnormal selective mining.

We hope the above will be of help in your present planning. If we can be of further service, please do not hesitate to call on us.

Very truly yours,

M. M. SUNDT CONSTRUCTION CO.

M. A. Hustad

M. A. Hustad
Assistant Vice President and
Manager, Mining Department
Heavy Engineering Division

MAH:lh

J. W. STILL
Consulting Mining Engineer
5213 N. ORACLE RD. 602 887-5341
TUCSON, ARIZONA 85704

March 24, 1970

Mr. Howard Lanier, Manager
Copper Processing Division
Essex International, Inc.
Three Rivers, Michigan

Re: New York Area - B.S. & K. Property

Dear Sir:

I have read over the material you left with me on the above noted project and have drawn the following conclusions:

1. It is probable that there are about 6,000,000 tons of open pit mineable ore in the claim group. Just what the average grade is is impossible to estimate based on the data available. The indicated stripping ratio is 3W : 1 Ore.

2. It would be impossible for the one 10-yard front end loader and the three trucks listed in the Osborne mine equipment list to mine at the rate of 12,000 t/d, this consisting of 3000 ton day of leach ore (which I understand is your target) plus 9000 t/d of waste.

3. It is questionable whether a 12,000 t/d operation would be economically feasible in the restricted pit area available. A close study on this question would determine what might be a maximum tonnage rate.

4. While I have used the Osborne-Heinrich data in the detail that follows, obviously the adverse factors listed above would certainly increase the capital estimated for mine equipment and would probably increase the mining costs they have used.

In view of the above, the basic question on proceeding with this project is just what copper cost would be acceptable to your company.

General Detail:

From the maps by Heinrichs and assuming that the ore is overlain by 100' - 120' of waste - and that the ore will average about 95' thick - the irregular area (containing holes #309, 301, 300) - with no back slope rights from AS&R - contains a little over 6,000,000 mineable tons by open pitting. This figure was derived from four sections and is an extremely rough figure.

The 5,000,000 tons of .60% copper mentioned in the Osborne report is a Heinrich figure - with the grade admittedly a somewhat rough estimate.

On the financial end - assuming .6% mined head and a 78% recover (9.36 lbs Cu recoverable per ton) - and further assuming a 3:1 waste:ore stripping ratio, the financial outfall would approximate the following; with 5,000,000 tons mineable:

Item	$'s	cost/t	cost/lb Cu
Mining: waste 3 x 30¢)			
ore 1 x 40¢)	$ 6,500,000	$1.30	13.0¢
Plant costs	$ 9,093,000*	1.82	19.4¢
		$3.12	33.3¢
Capital Amortization	$ 8,070,000**	$1.61	17.2¢
Indicated Op & Capital Amortization	$23,663,000	$4.73	50.5¢
Sales***	$26,208,000	$5.24	56.0¢
Margin to carry Income Taxes, Profit & Property Payments	$ 2,545,000	$0.509	5.4¢

Notes: *From page headed "STATEMENT OF INCOME & EARNINGS - Osborne report

** " " " " " " " " " " , and consists of capital cost plus "provision for interest."

*** 5,000,000 tons x 9.36 = 46,800,000 total lbs Cu recovered x 56¢ = $26,208,000.

I gather from the Osborne report that 3000 t/d of ore will be treated. On a 3:1 stripping ratio, this would mean a mining program of 12,000 t/d.

With the restricted working area, it is probable that a 12,000 t/d operation would be handicapped for working room and not very efficient. This coupled with the small apparent financial margin makes the project not very attractive.

The basic foundation of any mining operation is the ore reserve - and the figure of 5,000,000 tons @ .6% copper is far from a firm figure. Additional drilling to more accurately determine the potential grade and a detailed study on the total potentially mineable tonnage might improve the picture.

J. W. Still

Registered Professional Engineer (Mining)
CERTIFICATE EXPIRES
JACK W. STILL
DEC. 31, 1970
ARIZONA, U.S.A.

J. W. Still - Mining Engineer
Tucson, Arizona - 3/24/70

JWS:h

Exhibit G

TABLE 1

MINING AND METALLURGICAL CRITERIA

Option A - Processing only Chalcocite Ore.

Ore Reserve Estimate

	Max.	Min.	Accepted
Total tons to mine	20,000,000	19,550,000	20,000,000
Stripping ratio	3:1	2.6:1	3:1
Overburden	15,000,000	14,100,000	15,000,000
Ore	5,450,000	5,000,000	5,000,000
Grade of Ore	0.68	0.60	0.60
Pounds copper contained in ore	74,120,000	60,000,000	60,000,000

Recovery and Purification

Crushing to	3/8" Nominal	
(Vat leaching with sulfuric acid and ferric sulfate solutions)		
Acid consumption		35 lbs/ton of ore
Ferric Sulfate		5 lbs/ton of ore
Recovery by LIX-64 (solvent extraction) followed by electrowinning purification		
Pounds of estimated recoverable copper based on accepted contained copper (75% of 60,000,000)		45,000,000

Metal Value

Based on 55 7/8¢/lb. copper	$25,143,750.00

TABLE 1 contd.

Option B - Processing Chalcocite and Oxide ore.

Ore Reserve Estimate

	Max.	Min.	Accepted
Total tons to mine	20,000,000	19,550,000	20,000,000
chalcocite ore	5,450,000	5,000,000	5,000, 000
oxide ore	(14,100,000	7,000,000	7,000, 000
overburden	(	8, 000,000	8,000, 000
Stripping ratio	3:1	0.67:1	0.67:1
Grade of Ore:			
chalcocite	0.68	0.60	0.60
oxide *	0.12	0.15	0.15
Total pounds copper contained	74,120,000	81,000,000	81,000,000

Recovery and Purification

Chalcocite vat leached w/acid, Ferric sulfate

Oxide ore heap leached w/acid

Acid consumption	35 lbs/ton of ore
Ferric sulfate consumption	5 lbs/ton of ore

Pounds of estimated recoverable copper based on accepted contained copper:

Chalcocite ore (75% of 60,000,000)	45,000,000
Oxide ore (29.5% of 21,000,000)	6,200,000
Total	51,200,000 lbs.

TABLE 1 contd.

Metal Value

Based on 55 7/8¢/lb. copper $28,608,000

* 0.12% Cu is associated with 14,100,000 tons and the

0.15% Cu is associated with 7,000,000 tons of oxide ore.

TABLE 11 A

Recap Capital Costs
Option A - Vat-leach Chalcocite only

Mining Equipment	$872,000
Processing Plant and Equipment	
a. Crushing	$642,000
b. Leaching Vats	1, 430,000
c. LIX-Electrowinning	2, 250,000
Total	$4, 322,000
Site Preparation Ponds, Roads, Etc.	575,000
Maintenance Shop & Laboratory	100,000
Engineering Fee	210,200
	$5,979, 200
10% Contingency	597,920
Investigation costs and property payments to B.S.& K.	399,994
Total Investment	$6,977,114
Pre-Production and working Capital	2,575,000
TOTAL	$ 9,552,114

TABLE 11 B

ESTIMATES OF CAPITAL COST

Option ~~B~~-Vat-Heap Leach

Mine Equipment

2-8 Cu.Yd. rubber tired front-end Loaders @ $76,000 ea.	$152,000	
7-35 ton rock trucks @ $72,000 ea.	504,000	
1-Drill - Reich #650	80,000	
1-Road Maintainer	15,000	
1-D8 Caterpillar and Dozer	85,000	
1-Water Sprinkler Truck	10,000	
1-Grease Truck	7,500	
1-Bulk Powder Truck	6,000	
1-General Service Truck (fuel svc)	4,000	
2-Pickup Trucks	5,000	
1-Portable Light Plant	3,500	
Total		$872,000

Processing Plant and Equipment

Vat Leaching

3000 tons/day. Assuming an 8 day leaching cycle-requiring 10 vats. Crushing on 2 shifts per day, 5 days per week. Optimum crush assumed to be -3/8".

Estimates of Capital Cost contd.

Crushing Equipment

1-42"x48" Jaw Crusher complete with 200 HP motor and drive	$90,000	
1-Hopper and Feeder complete	12,000	
1-El-Jay 54" Cone Crusher complete with motor and drive	55,000	
2-E1-Jay 54" Fine Cone Crusher complete with motor and drive	120,000	
Miscellaneous screens, conveyors, chutes, etc.	150,000	
Ground storage with recovery equipment	50,000	
Electrical work	45,000	
Total	$ 522,000	
Installation (24%)	120,000	
Total		$642,000

Leaching Vats with False Floors

10 vats, 62 ft.wide X 100 ft.long X 18 ft. deep, 4800 Cu.Yds. reinforced concrete	550,000	
Loading conveyors and Loading bridge installed	420,000	
Unloading equipment	210,000	
Miscellaneous pumps, pipes, valves, tanks and sumps installed, including electrical	250,000	
Total		$1,430,000

Estimates of Capital Cost contd.

LIX-Electrowinning 1250 gpm

3 gpl Cu. to produce 36,000 to 40,000 lbs. of copper per day. Factored Estimate (including Engineering) (Source: H.S.McGarr of Holmes & Narver, Inc.)	$2,250,000	
Total Processing		$4,322,000
Total Mine and Processing		5,194,000
Site preparation; roads, Main, collection and tailings ponds; preparation heap leach piles	450,000	
Control Laboratory	25,000	
Maintenance Shop and equipment	100,000	
Engineering Fee (10% of 2,102,000)	210,200	
Total		5,979,200
Add 10% contingency	597,920	
Add initial investigating and property payments to B.S.& K.Mining Co.	399,994	
Total Investment Capital		$6,977,114

Pre-Production and Working Capital

Pre-production costs which will require additional capital amounts to	$2,050,000	
Working capital (45 days)	525,000	
Grand Total		$9,552,114

TABLE 111 A

Option A - Vat-Leach - Chalcocite only.

RECAP OF OPERATING COSTS

1. Mining	$9,000,000	
2. Crushing	1,650,000	
3. Leach Vats	3,300,000	
4. LIX-Electrowinning	3,600,000	
5. Overhead and Supervision	550,000	
TOTAL		$18,100,000

TABLE 111 B

OPTION B - VAT-HEAP LEACH

ESTIMATE OF OPERATING COSTS.

Mining Costs

1. Mine 5,000,000 tons of chalcocite ore @ 50¢/ton	$2,500,000	
2. Mine 7,000,000 tons of oxide material @ 50¢/ton	3,500,000	
3. Mine 8,000,000 tons of waste material @ 50¢/ton	4,000,000	
Total		$10,000,000

Processing Costs

1. Crushing Ore for Vat leach (3000 t/d)	¢/ton	
Labor	18	
Power & Operating supplies	7	
Maintenance supplies & Part.	8	
	33	
Total 5,000,000 tons x 33¢ =		1,650,000
2. Leaching Vat Operation (3000 t/d)	¢/ton	
Labor	13	
Maintenance & Operating supplies	6	
Ferric sulfate	8	
Sulfuric acid	39	
	66	
Total 5,000,000 tons x 66¢ =		3,300,000

Option B-Vat-Heap Leach contd.

3. Heap Leach Acid, etc. $910,000

4. LIX-Electrowinning

	¢/lb Cu
Power	1
Labor	3
Reagents	4
Total	8

51,200,000 lbs. Cu @ 8¢ 4,096,000

Overhead and Supervision 550,000

RECAP OPERATING COSTS

1. Mining	$10,000,000
2. Crushing	1,650,000
3. Leach Vats	3,300,000
4. Heap Leach	910,000
5. LIX-Elec-winning	4,096,000
6. Overhead & Supervision	550,000
TOTAL	$20,506,000

TABLE 1V A

ECONOMIC EVALUATION-OPTION-A

Not Computed

VAT-HEAP LEACH OPTION B

TABLE 1V B

COST AND INCOME STATEMENT PER YEAR

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Costs							
Initial payment to B.S.&K.	$100,000						
Investigating pay't to B.S.& K.	150,000						
Eng. Svc.for Crushing plant and Vat leach (60%)	210,000						
Eng. for LIX plant & Electro-winning (60%)	250,000						
Mine equip. purchase (60% of equipment costs)	304,800	218,400					
Plant construction costs (60% of plant consts)	557,600	2,230,560					
Stripping Operations	134,000	337,500					
Property payments to B.S.& K.	49,998	99,996					
Preproduction Expenses		889,600	688,900				
Working Capital			525,000				
Operating Expenses			3,075,000	3,691,000	3,691,000	3,691,000	3,691,00[illegible]
Cleanup							106,80[illegible]
TOTAL COSTS	$1,756,438	$3,776,056	$4,288,900	$3,691,000	$3,691,000	$3,691,000	$3,797,80[illegible]
Income							
Sale of Copper			$4,328,760	$5,834,970	$5,834,970	$5,834,970	$6,806,16[illegible]
TOTAL INCOME			$4,328,760	$5,834,970	$5,834,970	$5,834,970	$6,806,16[illegible]
Gross Profit	(1,756,438)	(3,776,056)	+ 39,860	$2,143,970	$2,143,970	$2,143,970	$3,008,36[illegible]

TABLE 1V B

COMPILATION OF CASH FLOW TO ESSEX

VAT - HEAP LEACH OPTION B

Description	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Costs	$1,756,438	$3,776,056	$4,288,900	$3,691,000	$3,691,000	$3,691,000	$3,797,800
Income			4,328,760	5,834,970	5,834,970	5,834,970	6,806,160
Gross Profit	(1,756,438)	(3,776,056)	39,860	2,143,970	2,143,970	2,143,970	3,008,360
Interest @ 11%	(193,208)	(609,456)	(743,740)	(743,740)	(585,029)	(419,504)	(300,856)
Depreciation (Essex Capital)	(598,028)	(598,028)	(598,028)	(598,028)	(598,028)	(598,028)	(598,028)
Depletion @ 15% gross				(875,245)	(875,245)	(875,245)	(1,020,924)
or 50% net-Least amt.	-	-	(19,930)	-	-	-	-
	(2,547,674)	(4,983,550)	(1,321,840)	(75,044)	(77,172)	251,192	1,008,552
Taxes at 50%	(1,273,851)	(2,491,775)	(660,920)	(37,522)	(38,586)	(125,596)	(504,276)
+ Depreciation	598,028	598,028	598,028	598,028	598,028	598,028	598,028
+ Depletion	-	-	19,930	875,245	875,245	875,245	1,020,942
Cash Flow	(675,809)	(1,893,746)	(42,961)	1,435,751	1,511,860	1,598,870	1,525,027
Capital Recovery	-	-	-	1,435,751	1,511,860	87,879	-
-Depreciation for Capital Rec.	-	-	-	-	-	598,028	598,029
Essex Profits (49%)	-	-	-	-	-	447,351	747,356
Gross Cash Flow to Essex	-	-	-	-	-	-	-
Capital Recovery	-	-	-	1,435,751	1,511,860	~~1,045,379~~ 1,133,258	1,345,027
Net Cash Flow to Essex	-	-	-	-	-	-	-

Note: The assumption is made that Essex continues to use positive cash flows to pay off capital.

TABLE 1V ~~A~~ B

STATEMENT OF INCOME AND EARNINGS

Vat-Heap Leach – Option B

Income

Sale of 51,200,000 lbs. of copper		
@ 55-7/8¢ lb.		$28,608,000
Cost of Production	$20,506,000	
Total Capital Costs	6,977,114	
Total	$27,483,114	
Gross earnings before interest, taxes, depletion and depreciation		$ 1,124,886

NOTES ON CASH FLOW
Option B

Total Interest Paid	$3,586,533	
" Capital Invested	6,761,280	2,050,000 PRE PROD. 500,000 - WORK CAP. 4,[illegible]280 - PLANT INV.
" " Recovered	~~5,396,253~~ 5,425,896	
" " Not Recovered	~~$1,365,027~~ 1,335,394	

PRELIMINARY EVALUATION

FOR POSSIBLE ACQUISITION

AND

OPERATION

OF THE

B. S. & K. MINING COMPANY PROPERTIES

INTRODUCTION

THIS REPORT WAS PREPARED BY

CLYDE E. OSBORN, E.MET.
Professional Engineer

Technical Director, Natural Resources Office, Essex International, Inc.

Tucson, Arizona

This is a preliminary report submitted as a guide for management in making a decision on how to proceed with the project. Certain statements were accepted as to the tonnage and grade of ore on the stipulation that these figures would have to be confirmed by actual field work. Further, the metallurgy and the subsequent processing is based on a report that can only be used as a guide in arriving at preliminary plant and operating costs. Considerable more test work must be done on representative samples. The mine owners have indicated a willingness to allow six months to make the necessary studies to confirm:

1–The tonnage and grade ore.

2–That the ore is amenable to leaching techniques.

3–That the cost estimates will be equal to, or better than those included in this report.

Signed Clyde E Osborn
Clyde E. Osborn, E.Met.

Date March 12, 1970

INTRODUCTION

The B. S. & K copper ore body under consideration, is a blanket type deposit containing a mixture of oxide copper, chalcocite and chalcopyrite minerals. According to data supplied by Mr. Abe Kalaf of the B.S.& K Mining Company, the estimated drilled reserves are as follows:

5,000,000	tons	assaying	0.60% Cu.
7,000,000	"	"	0.15% Cu.

The waste overlying the ore blanket is estimated to be 5,200,000 tons. This material contains some oxide copper but on the whole would not average more than 0.08% Cu.

The ore is up to 50' in thickness and does not extend more than 200' below the surface, this will lend itself nicely to open cut mining methods.

Preliminary tests made by the Duval Corporation in their Tucson labs indicated an overall copper extraction of 78% from a composite of samples taken from exploration drill holes. This test is reported in a company memorandum entitled "Bacterial Leaching of B.S.& K. Exploration Composite", dated November 22, 1965, Exhibit A

In subsequent discussions with the Lab Technicians, they express the firm opinion that the minerals will respond equally as well to a sulphuric-acid ferric-sulphate leach. This will require confirmation as proposed and outlined later in this report.

A geologic description and location of the property is attached. See Exhibits

Also attached is an analysis of the ore reserves and other comments by Mr. E.Grover Heinrichs, V.P. of Heinrichs Geoexploration Company.

The B.S.& K Mine has been an operating mine in the past. As a result there are some assets which will accrue to this newer project.

Access Roads

There are good county roads into the property from Red Rock to the east and from Silver Bell to the south west. Roads have been developed on the property in conjunction with exploration drilling programs. For the most part, these roads are in good repair.

Water

One water well has been developed. It is 300 ft. deep and cased. It has delivered 150 gpm over a long period of time during past operation with only a few inches of draw down.

Power

A 3 phase 14,400 Volt power line supplies electric power to the mine. The B.S.& K. Mining Company has picked up the power at the edge of the property and extended the lines approximately 3 miles into the property where they have installed 3, 650 KVA transformers and a distribution system. This has an estimated value of $125,000.

Housing

The old camp site is in need of extensive repairs. However, the power and water distribution systems are in good repair. An excellent combination office and residence is situated on the property. Floor area of 3000 sq.ft. Modern in all respects. This building is not included in the offer but it can be bought at an appraised value, or leased.

Equipment

The principal piece of equipment available to the project is an air compressor. This is an Atlas Copco 900 c.f.m. 100 p.s.i. piston type compressor, complete with a 100 h.p. Westinghouse Motor and drive and switching equipment. All in excellent condition. Includes a receiver. This has a present value of $15,000.

Introduction #4

Other

The B.S.& K. Mining Company leases a rail side on the Southern Pacific at Red Rock, a distance of 19 miles from the mine. This siding is 400 ft. long and is equipped with a loading ramp and drop bridge. This lease cost $300 per year.

SUMMARY

SUMMARY

This preliminary study of a plan to put the "blanket" ore body of the B.S. & K. Mining Company, situated on their New York claims, into production indicates that a reasonable return on investment can be obtained if the following conditions are satisfied:

1. The tonnage and grade of ore measure up to figures represented in this report.
2. The metallurgy proves to be satisfactory.
3. The price of copper remains at 56¢/# or higher.

Conditions 1 and 2 must be checked out before any other commitments can be made. A period of six months has been allowed to carry out the necessary work.

During the six months, some engineering should be done to confirm the costs which have been estimated in this report. It is believed that the costs in this report are reasonably close and will stand up to a more detailed study. Time did not permit a detailed study of a mining method. The cost of 40¢ per ton was arrived at by a study of recent reports and by talking with some of the operators in the Tucson area. The capital cost of the vats was factored from cost studies of a similar operation near Parker, Arizona. The writer was project engineer for the engineering company employed on this

LENGTH OF HAULAGE.

Summary #2

job. The capital cost of the LIX and electrowinning plant was a budget estimate from Holmes and Narver, Inc., and confirmed in a subsequent conversation with engineers at Hazen Research. Operating cost for the LIX-Electrowinning process includes cost for leaching the low grade mine dumps from which an estimated 5,000,000 lbs. of copper will be extracted.

The cost for the work to confirm conditions 1 and 2 as stated above is estimated at $100,000.00.

PLAN OF OPERATION

1-Preliminary Investigation

2-Mining Operation

3-Processing (milling) Operation

Preliminary Investigation

Having spent several hours on the property with Mr. Abe Kalaf and Mr. Grover Heinrichs, Mr. Heinrichs and the writer suggest that 20 holes be drilled for the purpose of obtaining bulk samples for metallurgical testing and to further evaluate the ore body and confirm ore reserve figures. This work to be done on contract.

Six months time should be allowed for the drilling, sampling and metallurgical investigation. The drilling program will require three months. Sufficient sampling would be done during the first month to permit some metallurgical work to start.

Further, during this six months, a mining plan would be engineered. At the present time there is a question of access to the B.S.& K. Mining Company property over the most desirable andshortest road. The costs being presented in this report reflect the longer route from the mine tothe proposed site.

MINING OPERATIONS

This mine will be a typical open cut mine. It will be necessary, over the five years of operation, to break, load and move approximately 17,000,000 tons of material.

A carefully planned mining program will be required because selective mining will be essential.

It will be necessary to mine approximately 500 tons per hour for 140 hours every week; 10,000 tons/day; 500 tons/hr. 20 hrs. per day.

The plan is to drill and blast, load the ore into 50 ton rock trucks with a 10 cu.yd. front-end loader.

A study of the technical literature and consulting with operators in the Tucson area provided the list of equipment described in this report.

Budgetary equipment cost figures were obtained from suppliers in Tucson and Phoenix.

It is possible that subsequent studies may prove that the rock can be ripped rather than blasted. Further, that a portable crushing plant can be used in the mine pit and conveyors be used to move the rock more economically than trucks. For the purpose of this evaluation, these alternates will not be considered.

It is contemplated that a certain amount of pre-mining work on roads, dump sites, etc., will be necessary. Also, that it will be necessary to strip up to 1,000,000 tons (500,000 cu.yds.) of overburden to prepare for the mining operation itself. The cost for these items is included in the estimate of capital required for the project.

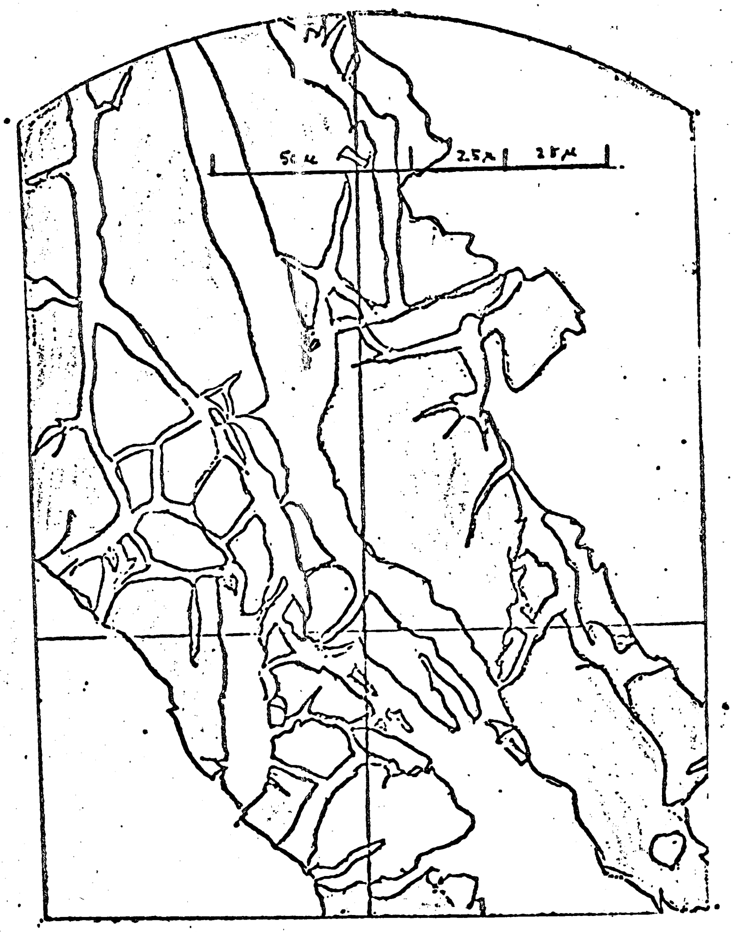


FIGURE VI

MICROPROJECTION TRACING (530X)

OUTLINE SUMMARY

ECONOMIC GEOLOGY, ORE RESERVES AND PROPERTY STATUS of THE B. S. AND K. MINING COMPANY

for
ESSEX INTERNATIONAL INCORPORATED

MARCH 1970

by
HEINRICHS GEOEXPLORATION COMPANY
P. O. Box 5671 Tucson, Arizona

ARIZONA

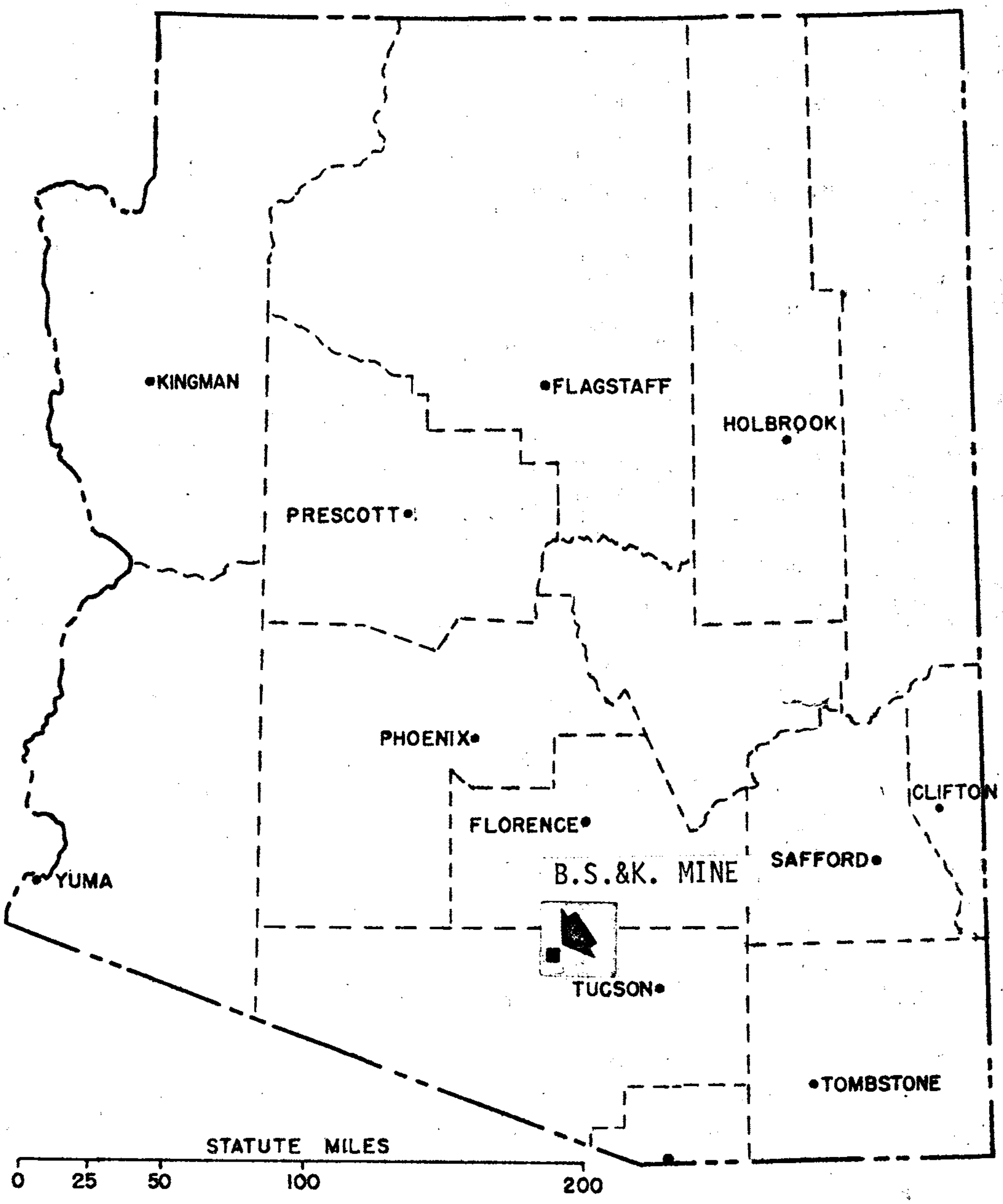


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TABLE OF CONTENTS

INTRODUCTION

This summary outline has been prepared for Essex International Inc., and represents part of a report entitled "Preliminary Evaluation for Possible Acquisition and Operation of the B. S. and K. Mining Company Properties", by Clyde Osborn, Technical Director of the Natural Resource Office of Essex International Inc.

No field geological study of the B. S. & K. property has been conducted. However the generalized geology of the area is well published and known.

The B. S. & K. Mining Co. property consists primarily of two major mineralized areas, the New York Claim Group and the Atlas Mine.

NEW YORK CLAIM GROUP

These claims are located in the immediate area of the quarter corner of Sections 28/33, T. 11 S., R. 8 E. and consisting of the following lode mining claims, New York, Nevada No. 2, Georgetown and fractional lode claims NSB 9, 10, 11, 12, 13 and 14 and comprising a total of approximately 58.25 acres.

The mineralized area of Silverbell Mine of A. S. & R. to the immediate south is a chalcocite blanket 100 to 200 feet thick lying under a leached capping which is approximately 100 ft. thick. The New York area of B. S. & K appears to be an extension of the El Tiro pit to the north. Available drilling information partly substantiates this assumption. (See accompanying copies of drill logs of Duval Corp.)

The leached capping of oxidized copper mineralization that overlays the chalcocite blanket carries values up to 0.15% to 0.2% copper over a considerable portion of the New York area. The chalcocite blanket varies in thickness from 30 feet to 100 feet and appears to be increasing in grade and thickness to the south and west.

Inferred Ore Reserves

Duval Corporation reportedly calculated 60,000,000 pounds of inferred copper reserves in the New York Claim Group and this figure is reasonable based on the following data:

1. Average thickness of chalcocite blanket - 40.0 ft. (See drill logs and assays of Duval Corp.)
2. Calculated tonnage of area 8,441,666 of which 75% may be mineable = 5,627,777 tons.
3. Assume average grade 0.6% Cu.
4. Solution = 67,533,324 pounds of copper.

In order to obtain sufficiently absolute actual tonnage and grade figures, a comprehensive evaluation program of twenty drill holes, drilled to a depth of 250 ft. each, is recommended. In addition, three drill holes, each 1,500 ft. deep to test the possible downward extensions of the chalcocite blanket is also recommended.

The Induced Polarization geophysical results by GEOEX and Canadian Aero suggests no cut-off of sulfides at depth in this area and therefore some deep drilling is justified and certainly should be programmed.

Total cost of the shallow drilling and sampling program will be approximately $50,000. Total cost of the deep drilling program of three holes, 1,500 feet deep would be approximately $60,000.

Legal Status

The New York Claim Group is completely surrounded on three sides by A. S. & R. and if possible a right-of-way should be negotiated with A. S. & R. as shown on the photo mosaic as Route 1. Route 2 is currently the only available access to the New York Group from the proposed leach and dump areas and from a mining cost standpoint the less desirable route.

Litigation between B. S. & K. Mining Co. and A. S. & R has been in the courts for many years and it is recommended that a thorough investigation be conducted by a lawyer, into the findings of the court prior to consummation of an agreement.

Favorable access negotiations with A. S. & R. is a definite possibility as indicated by the favorable solution of past problems of a similar nature and also because of the Arizona condemnation statute regarding rights-of-way for mining purposes.

THE ATLAS MINE

Located in the SE quarter of the NE quarter of Sec. 32, T. 11 S., R. 8 E., the Atlas Mine has had ore production history dating back to 1900. Production included argentiferous lead, zinc and copper sulfides and carbonates occuring in veins and pods in the paleozoic sediments. The ore occurrence has been intermittent and difficult to follow. However, when an ore pod was encountered the values reportedly ran as high as 45% zinc and 4% to 5% copper, with some recoverable values in silver.

No factual information is available to compute ore reserves in this area, however, good exploration possibilities do exist in the area based on I. P. anomalism on some work done by McPhar Geophysics Ltd. of Canada in 1960.

Some drilling should be done eventually to fully develop the economic mineral potential of this area.

Legal Status

The Atlas Mine area is on patented lode mining claims and the balance of the contiguous unpatented claims are reportedly in good condition from a legal standpoint. However, a thorough check of the Pima County Courthouse records should be made prior to consummation of an agreement.

Respectfully submitted,
Heinrichs GEOEXploration Company

E. Grover Heinrichs, Vice President

Approved:

Walter E. Heinrichs, Jr., President

BSZK 300

ASSAY— GEOLOGY COMPOSITE LOG

Sheet of 1-1

BEARING	Coörd. :: N	HOLE Nº 300-R
INCL. VERT.	E	COLLAR ELEV.
BSZK #3		DEPTH 400'

DESCRIPTION

182'-323':
Rock: Dacite porphyry made of rounded quartz eyes and pinkish feldspars in gray siliceous matrix.
Ex Alt.: Thin flat qtz veining w/ rims of secondary orthoclase. Moderate argillization along steep joints
Min. Chalcopyrite & moly in grains along qtz veins. Minor diss. sulfides

323-340: strongly altered monzonite(?). Abund chlorite after biotite. Some fine diss. sulfides. Very poor core recov.

340-400: Dacite porphyry as 182-323', but chlorite is present At 348' about 1" Chalcopyrite vein

Geol	Asy	INT	C.R	% Cu	% As Cu	% Mo	% As Mo	% EQIV	SAMPLE CMC	REC'Y %	GROUPING—RMKS.
		182-190		+1000	.14	+300	.045		1	69.6	
		190-200		+1000	.15	165	.016		2	82.8	
		200-210		1000	.10	114	.011		3	91.6	
		210-220		+1000	.11	238	.024		4	80.4	
		220-230		+1000	.14	245	.024		5	79.2	
		230-240		+1000	.12	171	.017		6	42.3	
		240-250		+1000	.34	+300	.042		7	46.1	
		250-260		+1000	.20	193	.019		8	23.1	
		260-270		+1000	.14	205	.020		9	14.1	
		270-280		+1000	.16	101	.010		10	21.2	
		280-290		+1000	.17	214	.021		11	25.6	
		290-300		+1000	.24	174	.017		12	54.1	
		300-310		+1000	.31	139	.014		13	21.6	
		310-320		+1000	.16	+300	.034		14	50.0	
		320-330		+1000	.20	277	.028		15	18.4	
		330-340		+1000	.28	41	.004		16	44.5	
		340-350		+1000	.91	+300	.032		17	71.3	
		350-360		+1000	.37	+300	.044		18	61.3	
		360-370		+1000	.21	282	.028		19	92.1	
		370-380		+1000	.18	272	.027		20	94.0	
		380-390		+1000	.15	178	.018		21	80.5	
		390-400		+1000	.20	256	.026		22	95.2	
					.227		.024			56.8	AVERAGE REC'Y
					.318						Equiv. Cu-MO

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 3

PROJECT Atlas Mine, BS&K Mining Co., BS&K No 3 Claim

Coörd. : N	BEARING Vert	HOLE Nº 300
E	INCL. Vert	COLLAR ELEV.
START 8-6-65 COMPL. 8-8-65	NC core size	DEPTH 182 feet

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz.	oz.	% EQUIV.	GROUPING — RMKS.
0-90 Dacite Porphyry, Brown fine grained, soft, porphyritic, highly fractured, numerous qtz veinlets, Arg. Alteration, Original Sulfide was ± 3%, entirely in fractures, Core very badly broken, No green or blue copper minerals visible, though entire section is oxide.			0-10	56	.08	.002						
			10-20	70	.08	.002						
			20-30	68	.09	.00[illegible]						
			30-40	76	.14	.001						average oxide is .13 Cu .005 Mo for 90 feet
			40-50	86	.14	.008						
			50-60	90	.16	.00[illegible]						
			60-70	86	.15	.009						
			70-80	85	.16	.006						
			80-90	87	.17	.009						oxide
90-95 Dacite Porphyry, Gray Mixed oxide and sulfide. Soft, porphyritic, strong arg. Alt., numerous qtz veinlets. Copper occurs as oxide and chalcocite.			90-95	90	.94	.025						Mixed
			95-100	86	.79	.015						Supergene Sulfide
			100-105	83	.51	.010						
			105-110	8[illegible]	.60	.009						Average grade of Mixed and supergene is 35 feet of
			110-115	87	.24	.008						
95-12[illegible] [illegible] Dacite Porphyry Supergene [illegible]. Gray, soft Copper occurs as chalcocite. Total sulfide is ± 3%			115-120	80	.12	.011						.47 Cu
			120-125	82	.11	.023						.013 Mo

LOGGED BY V. F. H DATE 8-8-65

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

PROJECT

Coörd. : N	BEARING	HOLE No. [illegible]
E	INCL.	COLLAR ELEV.
START COMPL.		DEPTH

DESCRIPTION

125-170 Dacite Porphyry

Hypogene zone, Gray, soft, porphyritic. Copper occurs in seams and veinlets as crushed chalcopyrite, with pyrite. Numerous qtz. veinlets. Much pyrite in seams is crushed and black. Total sulfide is [illegible]. Sulfide is 90% in seams, 10% in diss. wk Ser. on seams.

170-182 Dacite Porphyry, Gray

Hypogene zone, Soft, [illegible] porphyritic. Copper occurs [illegible] chalcopyrite with pyrite. Total sulfide is 3%. Su[illegible] 80% in seams and gouges and 20% as diss. M[illegible] veinlets. Pyrite is [illegible] crushed black. Wk sericite on seams, with Molybdenite visible in quartz veinlets. Largest veinlet is 2" qtz-pyrite - chalcopyrite at 177'. This veinlet also has traces of chalcocite.

Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz.	oz.	% EQUTV.
		[illegible]	82	.18	.010					
		[illegible]	86	.14	.028					
		[illegible]	[illegible]	.12	.011					
		[illegible]	76	.11	.011					
		[illegible]	70	.11	.010					
		[illegible]	75	.12	.015					
		[illegible]	80	.16	.016					
		[illegible]	85	.12	.008					
		[illegible]	[illegible]	.15	.031					
		170 [illegible]	76	.30	.011					
		175 182	7[illegible]	.162	.017					

GROUPING—RMKS.

Average Hypogene grade is .18 Cu .015 Mo

LOGGED BY V. [illegible] DATE 8-8-6[illegible]

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 2

PROJECT Atlas Mine, BS&K Mining Co., New York Claim

Coörd. : N		BEARING Vert	HOLE Nº 301
E		INCL. Vert	COLLAR ELEV.
START 8-9-65	COMPL. 8-10-65		DEPTH 206

0-100' Dacite Porphyry - Brown - oxide zone
Soft, strong Argillic Alt., Porphyritic, fine grained. Original Sulfide was ± 4%, 90% in fractures, 10% diss. Trace sericite lining fractures. Qtz veinlets rare. Some green Cu stain visible.

100-110' Dacite Porphyry Mixed Oxide and Sulfide
Brown, Soft, fine grained, Porphyritic, Wk sericite and strong Arg. Alt. Copper occurs as Malachite and chalcocite, Core badly shattered.

110-125' Dacite Porphyry Supergene Sulfide zone
Dark gray, soft, weak Sericite and strong Arg. Alt. Copper occurs as chalcocite with weak pyrite, mostly in seams. Core badly broken.

Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz.	oz.	% EQUIV.	GROUPING — RMKS.
		0-10	20	.12	.006						0'-30' .16 Cu .09 Mo
		10-20	50	.20	.012						
		20-30	67	.17	.010						
		30-40	80	.18	.009						30'-50' .17 Cu .011 Mo
		40-50	85	.16	.014						
		50-60	80	.11	.016						50-80 .12 Cu .016 Mo
		60-70	86	.12	.016						
		70-80	90	.13	.017						
		80-90	87	.24	.010						80'-100' .17 Cu .009 Mo
		90-100	90	.10	.009						
		100-105	90	.15	.013						100'-125' /83 Cu .018 Mo
		105-110	85	1.31	.040						
		110-115	86	1.74	.019						
		115-120	82	.49	.008						
		120-125	87	.45	.013						

LOGGED BY VKH DATE 8-10-65

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

PROJECT Atlas Mine

Coörd. : N	BEARING	HOLE No 300
E	INCL.	COLLAR ELEV.
START COMPL.		DEPTH 206

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz.	oz.	% EQUIV.	GROUPING — RMKS.
125-206 Dacite Porphyry Supergene zone as above			125 130	80	1.21	.006						125'-150'
			130 135	90	.70	.008						.74 .009 Mo
			135 140	85	.65	.009						
			140 145	70	.39	.015						
			145 150	65	.76	.009						
			150 155	56	.47	.013						150'-175'
			155 160	72	.38	.043						.49 Cu .027 Mo
			160 165	80	.36	.024						
			165 170	85	.51	.040						
			170 175	86	.72	.014						
			175 180	85	.50	.010						175'-200'
			180 185	85	.57	.052						.69 Cu .018 Mo
			185 190	80	.88	.011						
			190 195	90	1.12	.013						
			195 200	92	.40	.004						
			200 206	85	.19	.006						

End of hole

Average Mo for Hole is 0.0158 Mo.
Average from 105 to 200' is .72% Cu
.018 Mo

LOGGED BY VFH DATE 8-10-65

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 2

PROJECT Atlas Mine, BS&K Mining Co., BS&K

Coörd. : N	BEARING Vert	HOLE Nº 302
E	INCL. Vert	COLLAR ELEV.
START 8-11-65 COMPL. 8-12-65		DEPTH 151

DESCRIPTION

0-68 Dacite Porphyry
Oxide zone, Brown, fine grained, porphyritic, strong Argillic Alt. with trace Sericite. Core badly fractured, Original Sulfide content was est. 5%. Some Malachite visible 20' to 68'

68-70 fault zone. Core crushed

70-90 Quartz Monzonite Porphyry
Brown, Oxide zone, Granitic, holocrystalline texture. Weak Argillic Alteration, with No sericite. Fracturing very strong at 70' gradually weakens down the hole. Mineralization fades with fracturing.

90-95 Quartz Monzonite Por.
Mixed Oxide and Sulfide, Copper occurs as chalcocite, with limonite.

95-135 Quartz Monzonite Por. Supergene zone fracturing weakens with depth. Some Moly visible in rare quartz veinlets.

Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz.	oz.	% EQUIV.	GROUPING — RMKS.
		0-10	45	.07	.009						
		10-20	50	.06	.012						
		20-30	62	.09	.006						
		30-40	70	.07	.004						
		40-50	70	.07	.009						
		50-60	65	.10	.009						
		60-70	50	.19	.007						
		70-80	55	.11	.012						
		80-90	60	.20	.010						
		90-95	70	.66	.010						
		95-100	85	.66	.016						
		100-105	85	.61	.005						
		105-110	90	.44	.013						
		110-115	95	57	.006						
		115-120	97	54	.004						
		120-125	90	.32	.005						

LOGGED BY VFH DATE 8-12-65

FORM # 355
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ASSAY—GEOLOGY COMPOSITE DRILL LOG

PROJECT Atlas

Coörd. : N	BEARING Vert	HOLE Nº 302
E	INCL.	COLLAR ELEV.
START COMPL.		DEPTH 151

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz	oz	% EQUIV.	GROUPING — RMKS.
135-151 Quartz Monzonite Hypogene zone, Holoxline, Weak argillic Alt. Pyrite and trace chalcopyrite sparingly occur in rare fractures. Estimated total sulfide is 2%, occuring entirely as fracture filling.			125 130	90	.43	.009						
			130 135	92	.40	.009						
			135 140	95	.21	.005						
			140 145	90	.20	.004						
			145 151	95	.21	.003						
				END OF HOLE								

LOGGED BY_____________ DATE_____________

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 1

PROJECT Atlas Mine, Drill Hole for ~~Nevada~~ Nor Claim

Coörd. : N	BEARING Vert	HOLE Nº 303
E	INCL. Vert	COLLAR ELEV.
START 8-19-65 COMPL. 8-20-65		DEPTH 109

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz.	oz.	% EQUIV.	GROUPING — RMKS.
0-15 Dacite Porphyry. Brown, Oxide zone, soft, Porphyritic, fine grained. Original sulfide content was 4%. Strong Arg. Alt.; tr ser.			0-10	40	.02	.021						
			10-20	70	.02	.017						
			20-25	80	.04	.009						
15-25 Quartz Monzonite Brown, oxide zone, soft Holocrystalline, Est. 2% Original sulfide. Wk Arg. Alteration.			25-30	90	.09	.001						
			30-40	78	.28	.001						
			40-50	80	.29	.009						
25-60 Quartz Monzonite Gray. Supergene zone Holocrystalline. Est 2% Sulfide. Chalcocite occurs on fractrues. Weak Arg. Alt. No sericite.			50-60	90	.16	.008						Average from 30 to 50 is .28% Cu, .005 Mo.
			60-70	90	.13	.015						
60-109 Quartz Monzonite Gray as above. Hypogene zone. Est 2% sulfide, in fractures. Trace of chalcopyrite.			70-80	98	.11	.003						
			80-90	90	.12	.003						
			90-100	90	.07	.007						
			100-109	95	.001							

LOGGED BY VFH DATE 8-20-65

ASSAY-GEOLOGY COMPOSITE DRILL LOG

FORM # 355
REVISED 10-62

Sheet ___ of ___

PROJECT Atlas Mine, BS&K Mining Co., Hole for BS&K No. 1

Coörd.: N	BEARING Vertical	HOLE Nº 306
E	INCL. Vertical	COLLAR ELEV.
START 8-21-65 COMPL. 8-22-65		DEPTH 100

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz.	oz.	% EQUIV.	GROUPING — RMKS.
0-30 Dacite Porphyry Brown, fine grained, soft. Original sulfide now entirely oxidized. Original sulfide was 3% of rock. It occurred entirely in fractures. Arg AH., No Sericite			0-10	30	.04	.007						
			10-20	45	.05	.004						Average from 0'-30' .06 Cu, .027 Mo
			20-30	55	.10	.070						
30-50 Dacite Porphyry Mixed Oxide and sulfide zone Fe Ox and Pyrite, with traces of chalcocite. Rock as above.			30-40	75	.20	.030						Average from 30' to 70' is .29% Cu, .015 Mo.
			40-50	90	.34	.009						
			50-60	90	.38	.009						
			60-70	85	.23	.011						
50-60 Dacite Porphyry Supergene zone Chalcoite coatings on pyrite. Rock as above.			70-80	90	.21	.016						
60-100 Dacite Porphyry Hypogene zone. No chalcocite. Rock as above.			80-90	90	.16	.002						
			90-100	85	.10	.006						

LOGGED BY VFH DATE 8-23-65

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 1

PROJECT B. S. & K.

Coörd. : N	BEARING	HOLE Nº RDH 309
E	INCL. Vertical	COLLAR ELEV.
START 22 Mar.66 COMPL. 22 Mar 66'		DEPTH 110'

DESCRIPTION	Geol.	Assay	INT.	C/H	% Cu	% Mo	%	%	oz	oz	% EQUIV	GROUPING — RMKS.
SILVERBELL DACITE - argillically altered and siliceous, medium hardness but very abrasive, qtz. sulphide strgrs. w/ very weakly developed sericite near strgrs., thin capping: lim. after py. w/ secondary hm.derived from py.			10		N.S.							
			20		.07	Nil						
			30		.08	Tr.						
			40		.08	Tr.						
Sulphide zone @52'			50		.07	.001						
			60		.44	Tr.						
			70		.40	Nil						
			80		.44	.001						
			92		.38	.001						
			100		.29	.001						
			110		.22	Nil						

LOGGED BY BLW DATE 22 Mar. 66'

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 1

PROJECT B. S. & K.

Coörd. : N	BEARING	HOLE Nº RDH 310
E	INCL. Vertical	COLLAR ELEV.
START 22 Mar. 66 COMPL. 22 Mar. 66'		DEPTH 140'

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz	oz	% EQUIV.	GROUPING — RMKS.
QUARTZ MONZONITE PORPHYRY												
Argillically altered, "average" hardness.												
Sulphide zone @55'			55		N.S.							
			60		.82	.004						
Some dampness 60 - 70'			70		.82	.009						
			80		.65	.037						
Moisture increase @ 89'			90		.30	.010						
Increase qtz. @ 90½'			100		.26	.009						
			110		.29	.010						
			120		.32	.023						
			130		.26	.011						
			140		.23	.006						

LOGGED BY BLW DATE 22 Mar. 66'

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 1

PROJECT B. S. & K.

Coörd. : N	BEARING	HOLE Nº RDH 311
E	INCL. Vertical	COLLAR ELEV.
START 23 Mar.66' COMPL. 23 Mar.66'		DEPTH 150'

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz.	oz.	% EQUTV.	GROUPING — RMKS.
SILVERBELL DACITE - argillically altered and siliceous, medium hardness but very abrasive, qtz. sulphide strgrs. w/very weakly developed sericite near strgrs., thin capping: lim. after py. w/secondary hm. derived from py. Sulphide zone @ 48½'			50		N.S.							
Mixed oxide & sulphide @ 62'			60		.73	.002						
			70		.20	.001						
All sulphide @ 80'			80		.26	Tr.						
			90		.23	.033						Fines from dust Collector bin (90-100') Cu .38%, Mo. 004%
			100		.22	.002						
			110		.23	.005						Minor Cave @ 100-110'
			120		.23	.002						
			130		.22	.003						
			140		.22	.003						
Some dampness @ 150'			150		.28	.002						

LOGGED BY BLW DATE 23 Mar. 66'

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 1

PROJECT B. S. & K.

Coörd. N	BEARING	HOLE Nº RDH 310
E	INCL. Vertical	COLLAR ELEV.
START 22 Mar.66 COMPL. 22 Mar. 66'		DEPTH 140'

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz	oz	% EQUIV	GROUPING—RMKS.
QUARTZ MONZONITE PORPHYRY												
Argillically altered, "average" hardness.												
Sulphide zone @55'			55		N.S.							
			60		.82	.004						
Some dampness 60 - 70'			70		.82	.009						
			80		.65	.037						
Moisture increase @ 89'			90		.30	.010						
Increase qtz. @ 90½'			100		.26	.009						
			110		.29	.010						
			120		.32	.023						
			130		.26	.011						
			140		.23	.006						

LOGGED BY BLW DATE 22 Mar. 66'

FORM # 355
REVISED 10-62

ASSAY—GEOLOGY COMPOSITE DRILL LOG

Sheet 1 of 1

PROJECT B. S. & K.

Coörd. : N	BEARING	HOLE Nº RDH 311
E	INCL. Vertical	COLLAR ELEV.
START 23 Mar.66' COMPL. 23 Mar.66'		DEPTH 150'

DESCRIPTION	Geol.	Assay	INT.	C/R	% Cu	% Mo	%	%	oz.	oz.	% EQUIV.	GROUPING—RMKS.
SILVERBELL DACITE - argillically altered and siliceous, medium hardness but very abrasive, qtz. sulphide strgrs. w/very weakly developed sericite near strgrs., thin capping: lim. after py. w/secondary hm. derived from py. Sulphide zone @ 48½'			50		N.S.							
Mixed oxide & sulphide @ 62'			60		.73	.002						
			70		.20	.001						
All sulphide @ 80'			80		.26	Tr.						
			90		.23	.033						Fines from dust Collector bin
			100		.22	.002						(90-100') Cu .38%, Mo. 004%
			110		.23	.005						Minor Cave @ 100-110'
			120		.23	.002						
			130		.22	.003						
			140		.22	.003						
Some dampness @ 150'			150		.28	.002						

LOGGED BY BLW DATE 23 Mar. 66'

B. S. & K. MINE PROJECT

Notes PRELIMINARY EVALUATION

Under Summary

For the purpose of this evaluation, the recoverable copper is derived from the following sources:

A- Chalcocite Blanket

5,000,000 tons @ 12#/ton = 60,000,000 #
78% Rec. - 46,800,000 lbs.

B- Oxide Capping

7,000,000 tons @ 3#/ton = 21,000,000 #
25% Rec. - 5,000,000 lbs.

C-Overburden

5,200,000 tons @ 1.6#/ton = 8,320,000 lbs.
No treatment planned

Total recoverable copper 51,800,000 lbs.

30
29
28
27
31
32
33
34
B. S. & K.
B S & K NEW YORK AREA
T. 10 S.
A. S. R.
EL TIRO PIT
IMPERIAL PIT
6
5
4
3
T. 11 S.
R. 8 E.
WEAK STRONG I. P. ANOMALISM (GEOEX, CANADIAN AERO & MCPHAR)
B. S. & K. BOUNDARY
HEINRICHS GEOEXPLORATION COMPANY
POST OFFICE BOX 5671, TUCSON, ARIZONA, 85703
Phone: 602/623-0578 Cable: GEOEX, Tucson
vancouver sydney
geophysical engineers
B.S. & K. MINING CO.
ATLAS MINE & VICINITY
PIMA COUNTY, ARIZONA
OWNERSHIP
FOR
ESSEX INTERNATIONAL INC.
SCALE 1" = 1000'
DRAWN BY EGH & JCD
DATE MARCH 1970

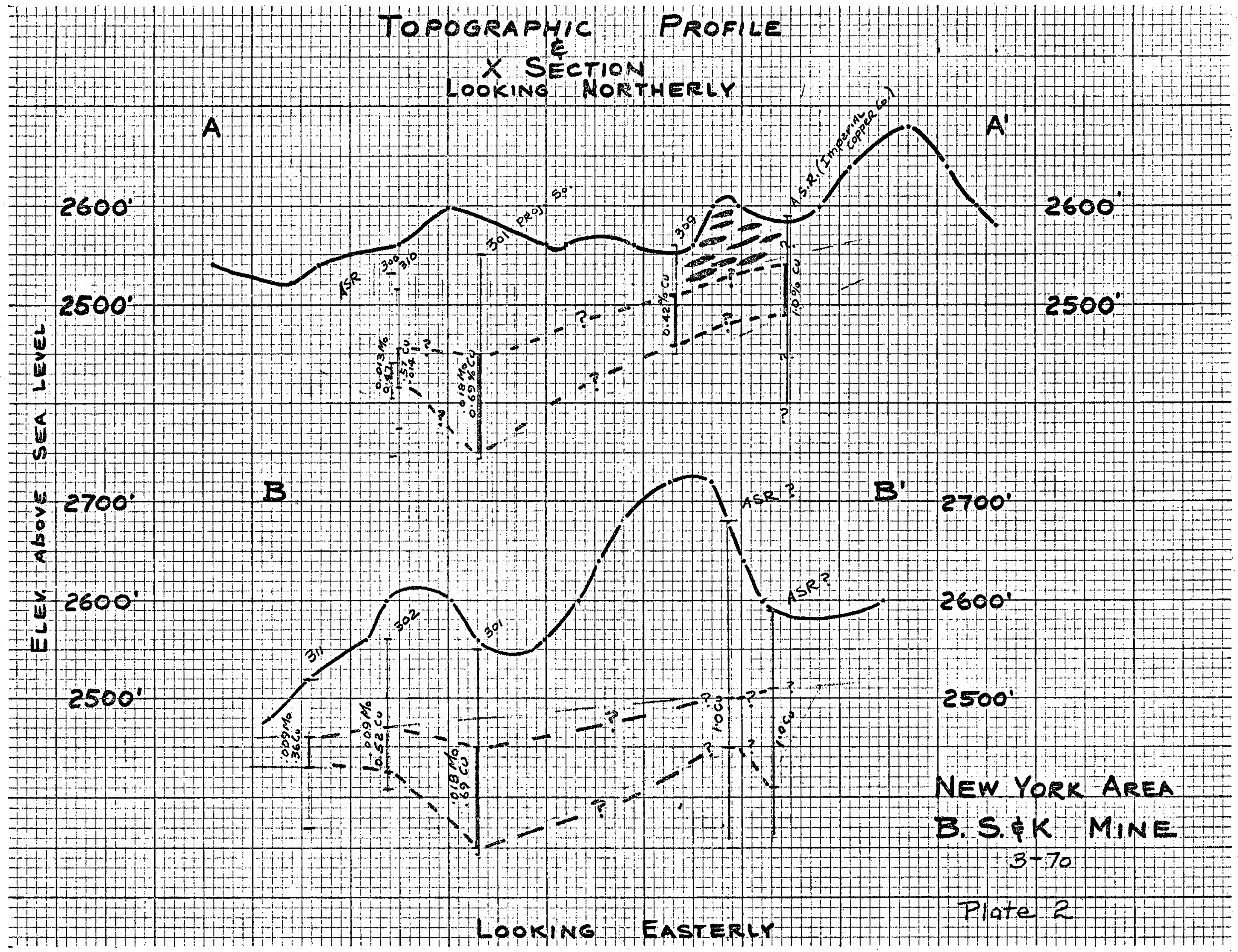
HEINRICHS
GEOEXPLORATION COMPANY
POST OFFICE BOX 5671, TUCSON, ARIZONA, 85703
Phone: 602/623-0578 Cable: GEOEX, Tucson
vancouver
sydney
geophysical engineers
511 - 70

TOPOGRAPHIC MAP
B. S. & K. MINING CO. AREA
PIMA COUNTY, ARIZONA

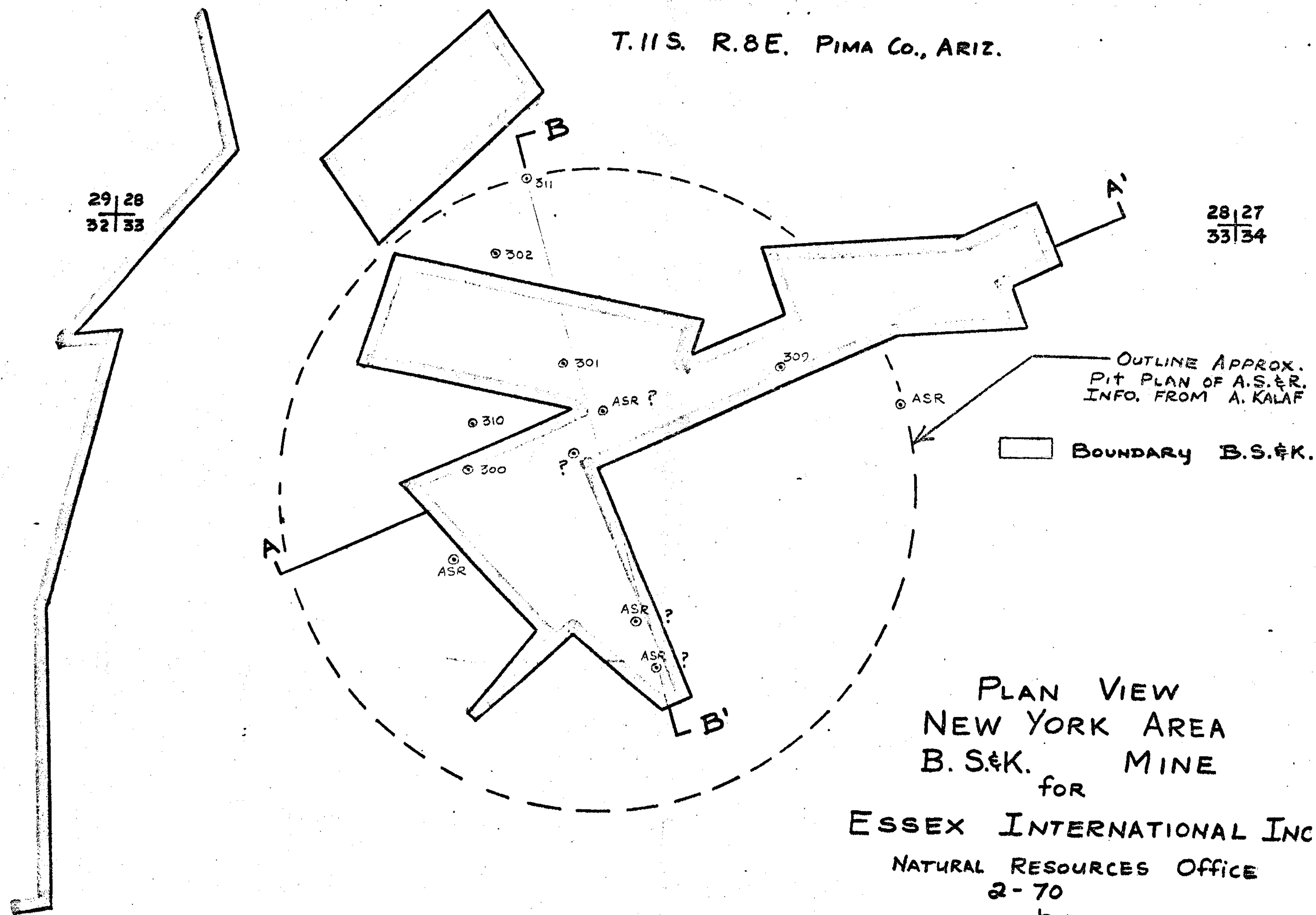
FOR

ESSEX INTERNATIONAL. INC.

SCALE: 1' = 1000' | DRAWN BY: EGH-JD | DATE: MAR 1970



TOPOGRAPHIC PROFILE
&
X SECTION
LOOKING NORTHERLY
A
A'
2600'
2500'
ELEV. ABOVE SEA LEVEL
ASR
300
310
301 PROJ. So.
309
A.S.R. (Imperial Copper Co.)
0.013 Mo
0.42
.57 Cu
.014
.018 Mo
0.69 % Cu
0.42 % Cu
1.0 % Cu
B
B'
2700'
311
302
301
ASR ?
.009 Mo
.36 Cu
.009 Mo
0.52 Cu
.018 Mo
.69 Cu
1.0 Cu
NEW YORK AREA
B.S.&K MINE
3-70
Plate 2
LOOKING EASTERLY



Plan View
New York Area
B. S.&K. Mine
for
Essex International Inc.
Natural Resources Office
2-70
by
Heinrichs Geoexploration Co.

Plate 1


PROPOSED DUMP & LEACH AREA
ROUTE 2
ROUTE 1
NEW YORK CLAIM GROUP
ATLAS MINE
2/3 B.S.&K.
1/3 Steinfeld
EL TIRO PIT
IMPERIAL PIT
HEINRICHS
GEOEXPLORATION COMPANY
POST OFFICE BOX 5671, TUCSON, ARIZONA, 85703
Phone: 602/623-0578
Cable: GEOEX, Tucson
vancouver
sydney
geophysical engineers
AERIAL PHOTO MOSAIC
of
B. S. & K. MINING CO. PROPERTIES
T 10 & 11 S, R 8 E
PIMA COUNTY, ARIZONA
for
ESSEX INTERNATIONAL INC.
Scale: 1" = 1000' 13 March 1970

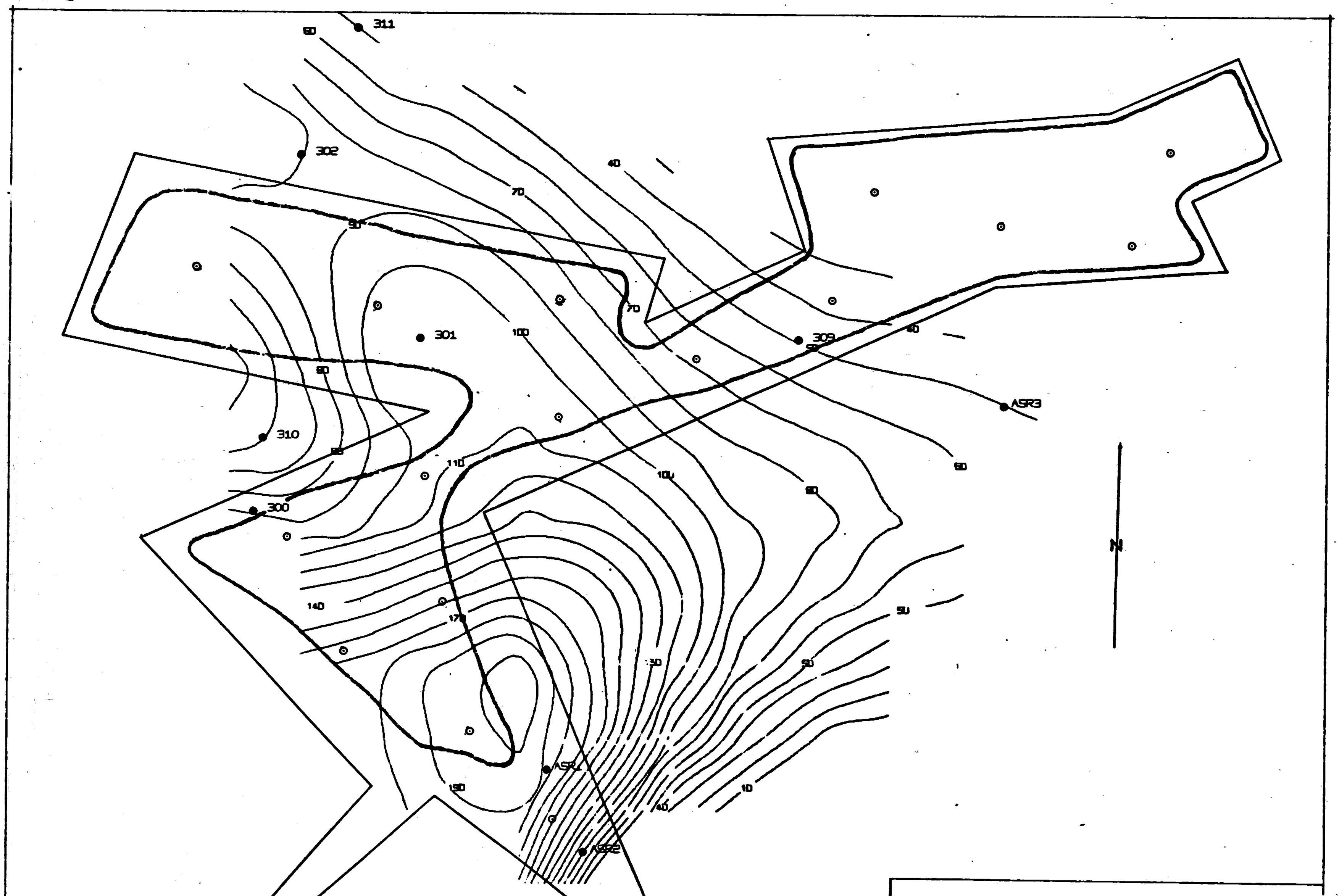


PLATE 1

ISOPACH OF OVERBURDEN, NEW YORK CLAIMS,

Sec. 22 & 33, T.11 S., R.8 E., Pima Co., Arizona.

Scale 1"= 200'

Contour Interval = 10 Feet.

● Drill Hole, ⊙ Proposed Drillhole.

〜 Ore Perimeter

HRI Proj. 722

ISOPACH OF OVERBURDEN THICKNESS. NEW YORK CLAIMS. CI= 10 SCALE 1 IN = 200

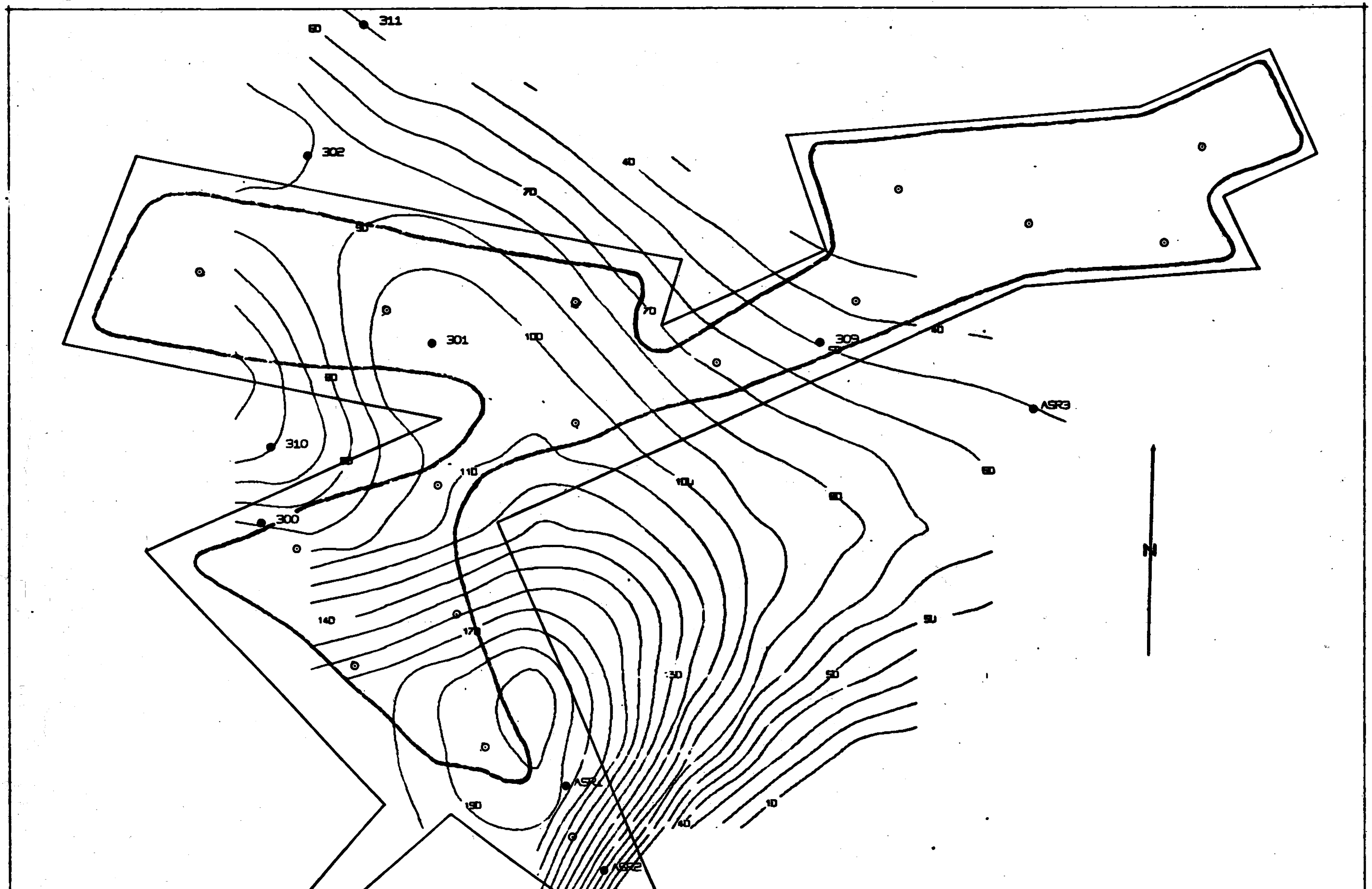


PLATE 1

ISOPACH OF OVERBURDEN, NEW YORK CLAIMS,

Sec. 22 & 33, T.11 S., R.8 E., Pima Co., Arizona.

Scale 1" = 200'

Contour Interval = 10 Feet.

● Drill Hole, ⊙ Proposed Drillhole.

~ Ore Perimeter

HRI Proj. 722

ISOPACH OF OVERBURDEN THICKNESS. NEW YORK CLAIMS. CI= 10 SCALE 1 IN = 200

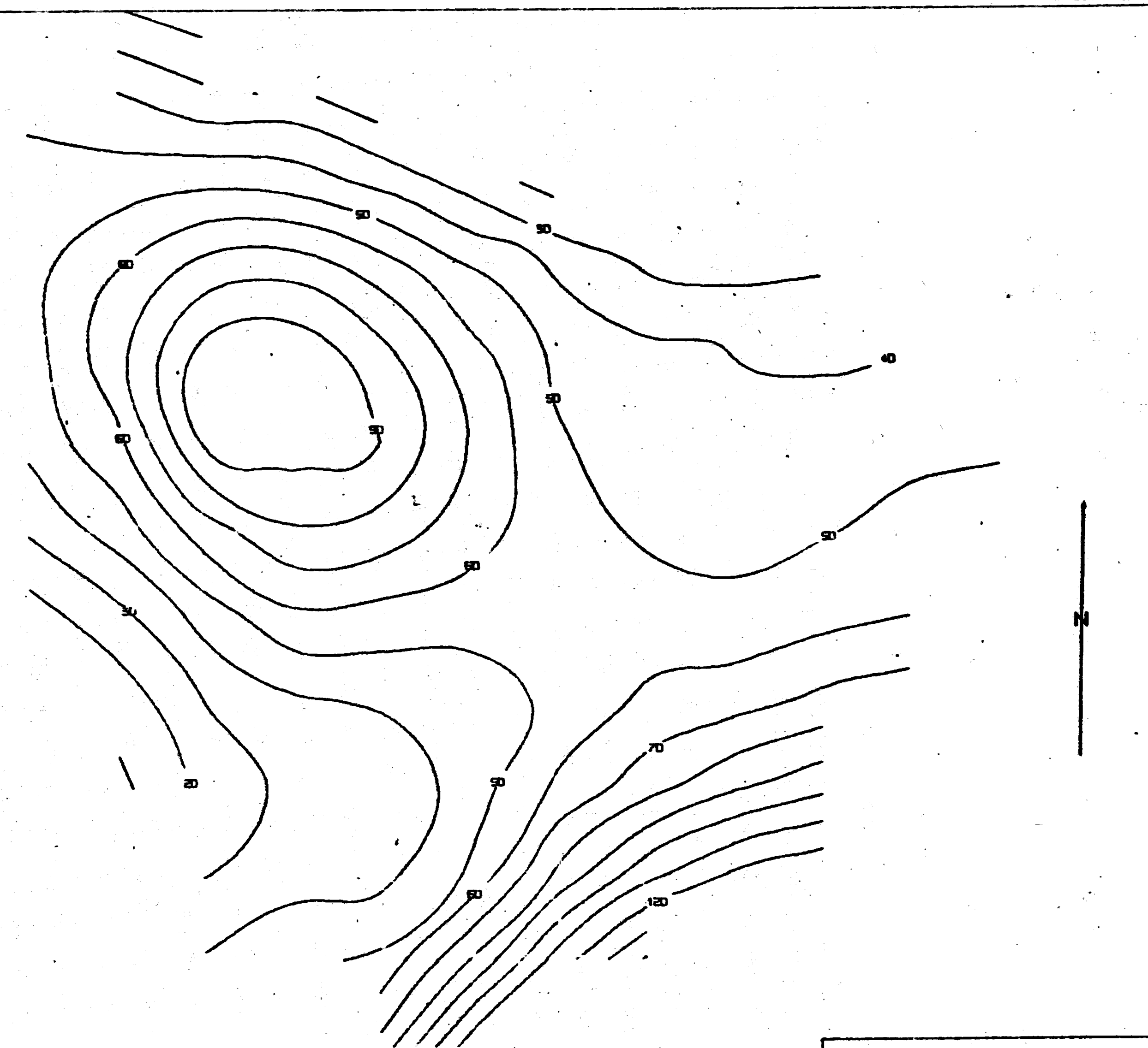


PLATE 2

ISOPACH OF INFERRED ORE, NEW YORK CLAIMS,

Sec. 28 & 33, T.11S., R.8E., Pima Co. Arizona.

Scale 1" = 200 Feet.

Contour Interval = 10 Feet.

HRI Proj. 722

PLATE 3

ISOGRAD OF ORE GRADE, NEW YORK CLAIMS,

Sec. 28 & 33, T. 11 S., R. 8 E., Pima Co., Arizona

Scale 1" = 200 Ft.

Contour Interval = 5/100 % Cu, Cutoff = 0.30% Cu

HRI Proj. 722

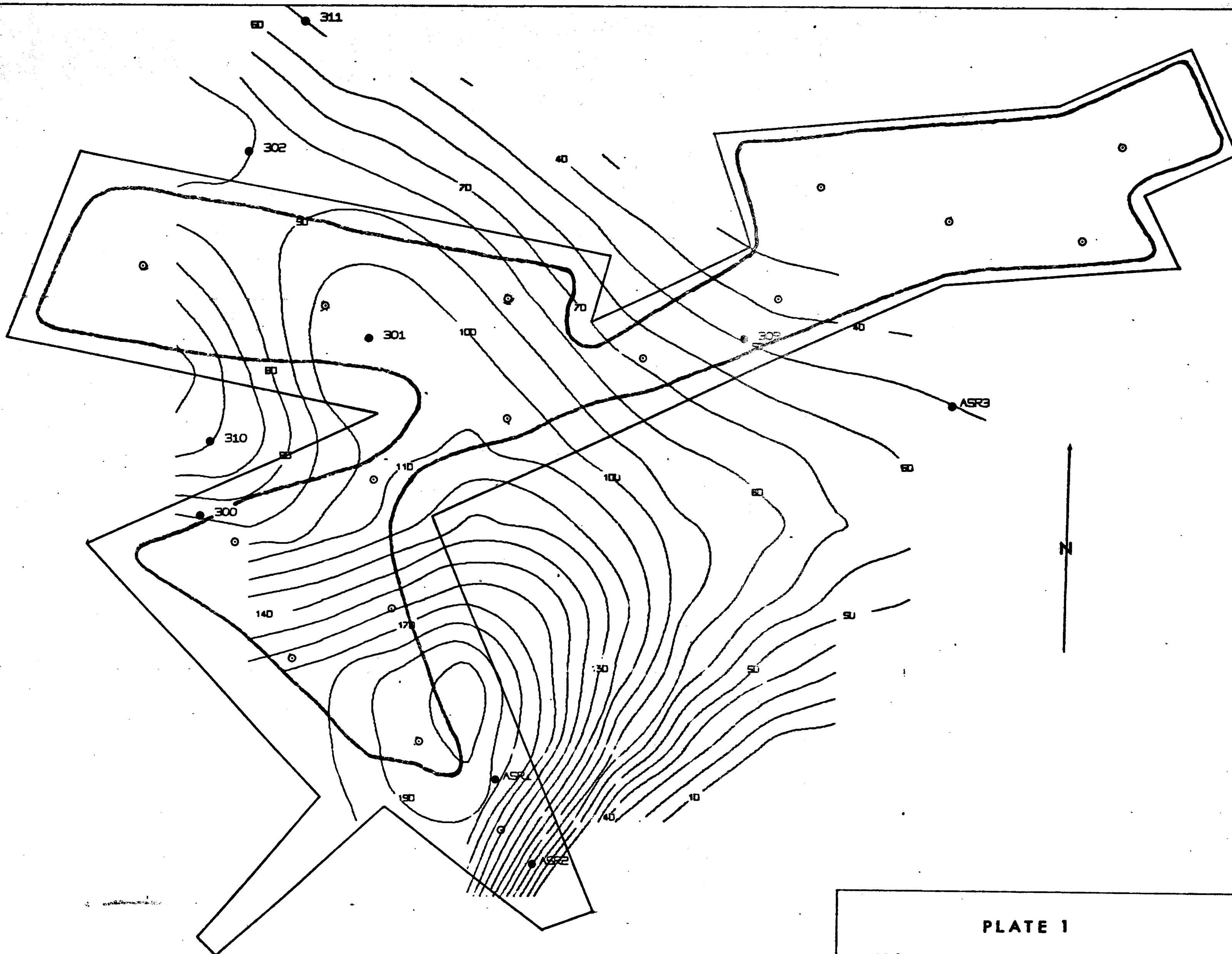


PLATE 1

ISOPACH OF OVERBURDEN, NEW YORK CLAIMS,

Sec. 22 & 33, T.11 S., R.8 E., Pima Co., Arizona.

Scale 1" = 200'

Contour Interval = 10 Feet.

● Drill Hole, ⊙ Proposed Drillhole.

~ Ore Perimeter

HRI Proj. 722

ISOPACH OF OVERBURDEN THICKNESS. NEW YORK CLAIMS. CI= 10 SCALE 1 IN = 200

Frank H. Buchella, Jr.
1802 West Grant Road, Suite 110 · Tucson, Arizona 85705 · 602/623-9007

October 13, 1980

Mr. A. M. Kalaf, President
B. S. & K. Mining Company
P.O. Box 50325
Tucson, Arizona 85703

Dear Mr. Kalaf:

Enclosed is a summary of the latest work performed on the B. S. & K. Property with the value and costs projected as of October 1, 1980. All estimates and calculations were escalated to that date using government indices for labor and materials. Market quotations as of that date were used for determining the value of the minerals on the property.

The highest value for the property is 115 million dollars using the sulfide flotation case, but the price of copper required to break-even is 96 cents per pound of copper which detracts from the value. The best case is for crushing the sulfide and leach material using a portable crusher and dump leaching the material. This case has a total value of 75 million dollars and a break-even cost of 84 cents per pound of copper. This case is a toss-up against processing the sulfide ore through ASARCO's Silver Bell Mill if the moly is recovered, which it has not for at least the last five years. This is primarily due to the high haulage cost of transporting the ore to the mill.

A joint venture with ASARCO would provide the best costs for crushing and dump leaching because of the higher volume per unit. The break-even cost for the joint venture would be approximately 70 cents per pound of copper and the value for B. S. & K.'s portion would be 88 million dollars.

Based on the update of the feasibility study for the B. S. & K. Property, I would recommend development of the property by open pit mining, crushing of the sulfide and leach ore and processing by solvent-extraction-electrowinning either for B. S. & K. alone or in a joint venture with ASARCO. If your intention is to sell the deposit, then your asking price should be 8.8 million dollars for the open pit ore and 11 million for all of the property.

If there is an interest in developing the project either alone or in a joint venture with ASARCO, then a detailed study and cash flow should be conducted on the portable crushing and dump leaching case before a final decision is made. Areas to be analyzed is more detail are the metallurgy and the portable crushing plant.

In addition to the analysis on the crushing and dump leaching, a drilling program should be developed for the Atlas Mine Area to explore for new limestone replacement orebodies.

Thank you for the opportunity of updating the information on the B. S. & K. Property.

Sincerely,

F. H. Buchella, Jr.
Mining Consultant

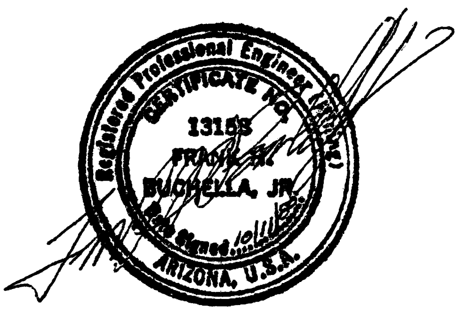
B. S. & K. Project

Summary

October 1980

Frank H. Buchella, Jr.

Mining Consultant



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INTRODUCTION

Location: B. S. & K. properties comprise 4,000 acres including patented lode claims, unpatented lode claims and state leases. Included in this area are the B. S. & K. chalcocite blanket and the Atlas copper-zinc limestone replacement deposit. The property is located in the Silver Bell Mining District, Pima County, Arizona, approximately 45 miles west-northwest of Tucson, Arizona. From the town of Silver Bell, a county maintained dirt road provides access to within 1/4 to 1/2 mile of the Atlas and B. S. & K. areas respectively. The Silver Bell Unit of American Smelting and Refining Company (ASARCO) has two open pit copper mines in the district. The northernmost one is 6,000 feet south of the B. S. & K. deposit.

History: The B. S. & K. Mining Company operated the Atlas Mine from 1953 to 1964 producing zinc and copper from a small replacement orebody. Drilling on the adjacent secondary enrichment deposit began in 1955 by ASARCO. During the period from 1955 to 1978, 150 drill holes were completed on and adjacent to the B. S. & K. claims by various companies to delineate the secondary enrichment copper deposit, explore for additional reserves and condemnation of plant and dump site areas. Numerous geological studies have been conducted in the district with five studies pertainning mainly to the Atlas Mine and the B. S. & K. chalcocite blanket. The most comprehensive study was compiled by C. A. Oakley in 1977.

Work to Date: In 1973, the consulting firm of Pincock, Allen and Holt prepared a feasibility study for B. S. & K. based on the data developed from Duval, B. S. & K. and ASARCO drilling.

During 1977 and 1978, Minerals Exploration Company conducted a complete and comprehensive study on the B.S. & K. property. In addition to geologic studies, this work included additional drilling on the chalcocite deposit, exploration drilling for new ore bodies, condemnation drilling, a pilot mine, geophysical surveys, ore reserves and mine designs, metallurgical testing and a complete feasibility study and financial analysis.

Cases: Case studies and financial analysis were conducted on B. S. & K. ore with sulfide flotation; B. S. & K. ore with heap leach, solvent extraction and electrowinning; and an ASARCO-Minerals Exploration joint venture.

The purpose of this report is to summarize all the previous work that has been compiled on the B. S. & K. property and update the financial analysis to present day costs. In addition to the updating of the three case studies previously prepared, several new cases are analyzed.

GEOLOGY AND ORE RESERVES

Geology: Porphyry copper mineralization with associated limestone replacement copper-zinc mineralization occurs two miles north of ASARCO's El Tiro Pit at the northern end of the Silver Bell mineralization-alteration trend. The porphyry copper mineralization is in the form of a chalcocite blanket found in porphyrotic quartz monzonite and quartz porphyry. The limestone replacement ore was mined at the Atlas Mine. Ore was found in garnetized Permian (?) limestones dissected by porphyritic quartz monzonite dikes.

Drilling: The chalcocite mineralization on the B. S. & K. property has been drilled out on approximately 200-foot centers.

It is known to extend on ASARCO ground both to the south and east. The chalcocite mineralization on the B. S. & K. claims is well delineated with the only remaining areas of ore potential on adjacent ASARCO ground. No additional Atlas-like mineralization was found by deep exploratory and condemnation drilling west of the porphyry blanket, although there is potential that such ore might be located in the general vicinity of the Atlas Mine. A large scale close pattern drilling program might develop more limestone replacement orebodies.

Pilot Mine: A pilot mine was developed to provide "fresh" bulk samples for metallurgical testing and confirm assays and interpretation used in modeling the deposit. An area central to the deposit was selected for the development of the pilot mine. A decline was driven at -11° for 324 feet to reach the lower portion of the secondary enrichment zone. Horizontal drifts were driven for a total of 460 feet. Three raises were driven on locations of percussion drill holes. A total of 179 feet of raise work was necessary to reach through the sulfide zone and into the oxide zone. A total of 3,821 tons of material was mined, sampled and stockpiled.

The drifting intersected three polygonal areas of influence. Weighted averages of polygons versus drift equaled a grade of 0.39% for the polygons, while the actual muck ran 0.44% for a 13% increase. Comparison between the drill holes and the raises was 0.51% versus 0.51% or 100%.

The increases in grade in the pilot mine are a result of small scale selective mining methods which are expected to compensate for dilution in larger scale open pit mining. Horizontal and vertical continuity of the ore in the pilot mine was good. The ore continuity and gradational contacts indicate that dilution with barren material should not be a significant problem, and in fact, the method of calculating composite bench assays has yielded conservative figures.

Pit Design: A bench height and reference elevation study was performed to maximize the reserves and determine the sensitivity of the deposit to various mining bench heights and reference elevations. Bench heights from 15 to 40 feet and elevations varied on five foot intervals for the studies. The studies indicated that the deposit was not sensitive to the reference elevation and only the secondary enrichment zone was sensitive to the bench height variations. The best case was to use lower bench heights and use selective mining methods. Twenty-five foot bench heights were used primarily on practical equipment selection.

Level maps were prepared for each twenty-five foot bench which included the oxide-secondary and secondary-primary contacts and polygons around each drill hole with a maximum 200-foot radius of influence.

A break-even stripping ratio analysis was run to establish the economic limits of the pit. The pit limit was expanded in small increments until the revenue generated from ore grade material equaled the cost of stripping, mining and processing. The analysis was applied to 15 N24W x-sections and seven N66E x-sections. The pit limits so defined in the sections were transformed to a plan view for contouring. During contouring, consideration was given to working room, haul road access and conditions to permit an efficient mine operations. The pit designs are controlled primarily by the constraints of the property boundaries. Pit designs were overlaid on the polygon level maps for final reserve tabulations.

Drill hole information and assays are also available for the area immediately surrounding the B. S. & K. property. The ground controlled by ASARCO has the same type and grade of mineralization as observed at B. S. & K. Information is adequate for three-fourths of this area to delineate the limits of mineralization. The remaining area is not drilled, but has favorable surface exposure. Reserves have been calculated for the entire geologic deposit with reasonable projections into undrilled areas.

Reserves: The ore reserves of the B.S. & K. property and the immediate area surrounding the property have been determined through an extensive drilling program and well established methods of reserve calculations. The resulting figures are representative of the deposit.

	Flotation with Heap Leach	Heap Leach Only	ASARCO Joint Venture
Flotation Reserve - Tons	9,065,600		24,000,000
Cutoff Grade - Percent	0.33		0.30
Total Copper - Percent	0.59		0.56
Molybdenum - Percent	0.014		0.014
Silver - Ounces/Ton	0.02		0.02
Copper Equivalent - Pct.	0.63		0.60
Leach Reserve	6,104,700	15,548,900	25,000,000
Cutoff Grade - Percent	0.17	0.13	0.12
Total Copper - Percent	0.26	0.40	0.23
Total Reserve - Tons	15,170,300	15,548,900	49,000,000

Total Copper - Percent	0.48	0.40	0.39
Copper Contained - Tons	69,400	62,200	191,100
Recovery - Percent			
Flotation	80		80
Leach	60	60	60
Copper Recovered - Tons	52,300	37,300	114,700
Waste Rock - Tons	15,476,500	8,249,000	29,000,000
ASARCO (waste) - Tons	2,603,400	1,603,400	
Total Waste	18,079,900	9,309,9000	29,000,000

ANCILLARY AND ENVIRONMENTAL CONSIDERATIONS

Access: Realignment and improvement of 6.5 miles of road from Silver Bell, Arizona, to the property will be required.

Power: Trico Electric Supply services the area of the B. S. & K. site via a 14.4 KVA line which passes through the plant site. Peak demand for a 4,000 ton per day concentrator will be approximately 4,100 KW which would require upgrading the present line or generating your own power. Power demand for the solvent extraction and electrowinning plant would be 2,000 KW, which could be provided with the existing line. Portions of the Trico line and portions of a Tucson Power Company line will have to be relocated.

Telephone service would be brought into the project from Silver Bell.

Hydrology: Hydrological considerations involve the diversion of surface runoff from the pit, removal of groundwater seepage from the pit, and a groundwater supply for domestic and processing use.

Earth dams constructed from premined stripping materials would be used for surface runoff control. A sump pump would be used for controlling seepage into the pit during mining in the bottom 125 feet of the pit. Water supply for domestic and metallurgical processes is available at the site by pumping from wells. Hydrologic testing would be required for efficient well sizing and development.

Permits: Several environmental studies and monitoring programs would be required in order to obtain the permits necessary to develop the project.

A one year monitoring program for any pollutants, which exceed standards, and meteorology data would be required to develop an air model of pollutants from the mine, heap leach and process. This data would be used to obtain two air permits.

A National Pollutant Discharge Elimination System (NPDES) permit is required whenever aqueous effluents are discharged into natural waters. Permits would be required for heap leaching for dams, channel changes and possible overflow. A groundwater monitoring program should be developed and continued throughout the life of the project to guard against polluting the groundwater.

Arizona has a native plant act which protects a majority of the native plants in the area. Major native plants in areas to be disturbed must be purchased and moved before work begins.

Permits will be required from the state for drilling the culinary and process water wells and from the county for sewage and solid waste disposal.

Sideline Agreement: On May 1, 1973, an agreement was reached between B. S. & K. and ASARCO which allows either party to mine their ore and strip material on the adjacent ground. Under the agreement, either party may extend its backslope onto the adjacent or surrounding properties. Access across and use of adjacent property is also allowed. Waste material is to be removed and dumped on the operator's property. Material identified as ore or leach is to be delivered to owner's storage sites without cost to the owner. A maximum three mile haul is specified for ore or leach material. All work is to be performed in a careful manner with proper notice and access to records insured. The terms of the agreement are transferable with property rights and effective for 30 years.

CASE I: SULFIDE FLOTATION

Description: Case I of the B. S. & K. feasibility study considers the recovery of sulfide copper mineralization through a conventional milling and flotation process. Molybdenum and silver are recovered as by-products of this process. Revenue is also generated through recovery of oxide copper mineralization and low-grade secondary copper sulfides by heap leaching and solvent extraction-electrowinning. The deposit will be mined by open pit methods employing front-end loaders and truck haulage.

Mill processes and mining methods were selected after evaluation of several options. Three different mill sizes and four mining rates were considered in order to optimize production rates. Consideration was also given to leasing mine equipment versus ownership.

Financial analysis is performed on the recommended case of a 4,000 ton per day mill with owner operated equipment giving a 6.2 year production life.

Financial Analysis:

Production

Flotation Cu	$85,579,264	@	$1.01/lb	=	$ 86,435,056
SX-EW Cu	12,697,800	@	1.01/lb	=	12,824,778
Total Cu	98,148,700				$ 99,130,178
Mo (lb)	1,269,184	@	10.31/lb	=	13,085,287
Ag (oz)	145,050	@	21.45/oz	=	3,111,314
					$ 16,196,601
Total Value					$115,456,935

Capital Expense

Capital Cost		$28,384,200
Reproduction Mine		3,690,200
Reproduction G & A		1,079,500
Total		$33,154,400
Less Salvage	-	732,800
Less Mo & Ag Credit	-	16,196,600
TOTAL		$16,225,000

Capital Cost/lb $= \frac{\$16,225,000}{98,148,700} =$ $0.165

Operation Cost/lb = 0.797

Total Cost to Produce $0.962/lb Cu

Following are the basic production criteria:

Operating Days per Year

Mill	365
Mine	250

Mine Production

Tons per Year	4,380,000

Flotation Plant

Tons per Year	1,462,194
Assay - Percent Total Copper (Cu)	0.59
Recovery - Percent Total Copper (Cu)	80
Pounds Copper Recovered per Year (Cu)	13,803,107
Assay - Percent Molybdenum (Mo)	0.014
Recovery - Percent Molybdenum (Mo)	50
Pounds Molybdenum Produced per Year (Mo)	204,707
Assay - Ounces Silver per Ton (Ag)	0.02
Recovery - Percent Silver (Ag)	80
Ounces Silver Recovered per Year (Ag)	23,395
Operation Life - Years	6.2

Heap Leach

Tons per Year	872,100
Assay - Percent Total Copper (Cu)	0.26
Recovery - Percent Total Copper (Cu)	40
Pounds Copper Recovered per Year (Cu)	1,813,968
Operation Life - Years	7

Mining: The B. S. & K. deposit will be developed by open pit mining methods using 15 YD^3 front-end loaders and 50 ton trucks as the principle mining equipment.

Preproduction stripping is scheduled for 55 weeks, during which time 6.05 million tons of waste and leach material shall be moved. This advanced stripping will establish a more consistent 2:1 stripping ratio throughout the mine life.

Dump areas were selected west and southwest of the pit. The leach dump will overlie a natural drainage basin with primarily hard rock bottom. The waste dump is located due west of the pit area over level valley fill. Both dumps are sequenced to coincide with the pit reserves.

The mine production is scheduled on a 15, 8 hour shifts per week, 50 weeks per year basis. A shift productivity of 5,480 tons (ore, leach and waste) must be maintained in order to provide an annual feed of 1.46 million tons of ore to the mill along with 2.92 million tons of waste and leach material. Mine production will continue for 6.2 years, at which time over 27 million tons of material will have been moved.

The open pit operation will utilize one 15 YD^3 front-end loader and three 50 ton haul trucks as the principle equipment for both waste removal and mining of the copper ore. Benches 25 feet high will be drilled and charged with ANFO providing the necessary fragmentation to facilitate loading. The need for secondary breakage should be insignificant. Haulage distances going one way to B. S. & K.'s leach, waste and ore facilities average about 3,000 feet, while the distance to ASARCO's leach and ore dump is approximately 3 miles one way. Pit ramps are relatively short in duration and designed at a maximum grade of 8%. Maintenance of the mining equipment will be in a shop facility located at the plant site. A mine pump and pit sumps are recommended for dewatering any wet mine areas.

Equipment selection involved the joint consideration of material productivity requirements, haul distances and profiles and common industrial practices in similar mining operations. A computerized haulage system model which evaluated haul trucks within a 50 to 100 ton capacity range was used to determine the recommended fleet of 3 operating 50 ton trucks. A backup unit of one is adequate based upon a mechanical availability of 75%. Wheel loaders were selected because of the fine match of machine life to mine life, allowing for favorable utilization and depreciation. Other considerations were the lower initial capital cost, extra machine mobility and the wide industrial acceptance of front-end loaders in similar mine material. Various combinations of loader size and mine scheduling operating one 15 YD^3 size loader 3 shifts per day proved most ecomonical. Although the loader will be operated at 80% of its productive capability of 7,325 tons per shift, the high utilization of 15 out of 21 potential work shifts will require purchase of a standby unit.

Primary blast hole drilling will be performed by one rotary rubber-tire truck drill. Over 135 holes spaced on an 18 x 18 x 30 foot pattern will be bulk loaded with ANFO and shot each week. This will require only seven 9 hour shifts of drilling per week and will not require the purchase of a standby drill. Bulk loading and dewatering of blast holes will be performed by an outside contractor.

The major support equipment includes (1) track dozer w/ripper, (1) motorgrader, (1) water truck, and (1) rough terrain service crane.

<u>Processing:</u> Recovery of copper mineralization involves both flotation and leaching processes. The primary process is a conventional 4,000 ton per day sulfide flotation concentrator. Selection of the concentrator size was made after a preliminary cash flow analysis on 4,000, 3,000 and 2,000 ton per day sizes. Molybdenum and silver are also recovered as by-products of the flotation and smelter processing. In addition to the concentrator the oxide ma-

terial and the lower grade secondary sulfide will be placed for heap leaching with sulfuric acid solutions. Copper is then recovered through solvent extraction and electrowinning.

The run of the mine ore is fed through a grizzly to a 42" x 48" jaw crusher. Minus 5 inch material is conveyed to a 12,000 ton coarse ore pile. Coarse ore is fed to closed circuit secondary-tertiary crushers that reduce the product to minus 3/8 inch which is conveyed to a 2,000 ton fine ore storage.

The fine crushed ore is fed to a 14 foot x 15 foot ball mill at a controlled rate of 174 dry tons per hour. The grinding mill operates in closed circuit with two 26 inch cyclones and has a designed circulating load of 400 percent. Target grinding density is 70 percent solids and cyclone feed density is 58 percent solids. Reagents are fed to the cyclone underflow.

The cyclone overflow which is at a nominal 65 mesh grind is fed to the copper rougher flotation circuit consisting of ten 300 cubic foot cells. The rougher concentrate produced is fed to a 15 inch cyclone with the underflow reporting to a 6 foot x 12 foot concentrate regrind ball mill. The mill operates in closed circuit with the same 15 inch cyclone. The overflow of the cyclone reports to a rougher concentrate thickener. Underflow from the thickener at 35 percent solids comprises the feed to the first copper cleaner. The rejects from the cleaner stage are run through six scavenger stages. The scavenger concentrate is returned to the regrind mill and the scavenger tails are combined with the copper rougher tailings for thickening prior to discard. The cleaner concentrate is given a second stage of cleaning before thickening for feed to the molybdenum recovery circuit.

The thickened concentrate (60 percent solids) is fed to a series of three 4 foot x 4 foot conditioners where modifers, depressants, and frothers are added. Design retention time in the conditioners is approximately two hours.

The conditioned concentrate slurry is run through six #15 cells to produce a moly rougher concentrate. The resultant concentrate is given four seccessive stages of cleaning to produce a final concentrate containing 54 percent molybdenum. The concentrate is then thickened, filtered, dried and packaged for shipment.

The rejected concentrate slurry from the molybdenum rougher circuit is subsequently thickened and filtered for shipment to the smelter. The grade of the concentrate produced is in excess of 30 percent copper.

Uncrushed leach ore will be hauled directly from mine to dump. At a rate of 3,488 tons per day, five days per week, a 104,640 ton dump will be built in six weeks. The leaching solution, 1,000 gallons per minute, will be distributed in eight-inch mains, three-inch headers, three-quarter-inch branches, and one-quarter-inch distributing hoses. Expected makeup water is 80 gallons per minute. The dump size will be 150 feet by 490 feet by 25 feet deep.

A solvent extraction circuit will be used to handle 1,000 gallons per minute of pregnant dump leach solution. The feed solution to solvent extraction contains 0.72 grams of copper per liter, and the raffinate has 0.10 grams per liter. Thus, 0.62 grams per liter net is recovered in solvent extraction. Approximately 30 gallons per minute of electrolyte will be circulated to the tank house to recover 50 cathodes daily, each weighing 149 pounds.

<u>Costs:</u> Mine capital costs are based on current preliminary

budget quotations from manufacturers and dealers. Mobile equipment costs include freight charges to minesite and erection costs when applicable. Capital costs for the maintenance shop (5,130 sq. ft.), mine warehousing (1,710 sq. ft.) and the changehouse (1,000 sq. ft.) include construction material and labor and all related tooling and equipment. Mobile equipment, buildings and facilities have all been costed new.

Mine operating costs are a function of an hourly operating cost per unit applied over the scheduled operating hours. The equipment operating costs reflect parts and supplies and all maintenance and operating wages. Hourly wages include a mine payroll burden of 40 percent.

Using actual costs for completed beneficiation facilities, Mountain States Mineral Enterprises has developed a method of estimating, in a preliminary manner, installed capital cost. For a 4,000 ton per day copper flotation concentrator treating an ore containing 0.59 percent total copper, the installed capital cost estimate is $16.8 million.

The molybdenum recovery equipment was factored to a total installed cost. This cost was found to be $1.0 million.

Combining the above installations gives an estimated preliminary installed capital cost for the copper-molybdenum facility of $17.8 million.

Capital Costs

Ancillary	$1,365,500
Environmental	127,000
Mine Equipment	
Initial	4,011,900

Replacement - 4th Year	$ 50,800
Replacement - 5th Year	232,400
Reproduction	3,690,200
Mill	
Engineering and Design	1,143,000
Flotation	17,780,000
SX-EW	3,674,100
Total	$28,384,700

Operating Costs

Mine		
Production	$ 2,634,600	$0.160
Mill		
Flotation	3,639,800	0.264
Smelter	4,376,000	0.318
SW-EW	832,500	0.311
Administration and Supervision	1,143,000	0.070
TOTAL COST	$12,625,900	$0.797

Personnel:		
	Mine	29
	Mill	29
	Leach	15
	G & A	17
	Total	90

CASE II: HEAP LEACH, SOLVENT EXTRACTION, ELECTROWINNING

Description: Case II for the B. S. & K. feasibility study considers the recovery of copper through a heap leach-solvent extraction-electrowinning process. All secondary sulfide and oxide mineralization will be placed in heaps for leaching by sulfuric acid solutions. Recovery of copper through the solvent extraction-electrowinning system will result in a directly saleable product. Mining is designed for open pit methods employing front-end loaders and truck haulage.

Financial Analysis:

Production
Leach Cu 49,756,480 lbs @ $1.01/lb = $50,254,044

Capital Expense
Capital Cost $12,019,500

Capital Cost/lb	$= \frac{\$12,019,500}{49,756,480}$	= $0.242
Operating Cost		= 0.760
Total Cost to Produce		= $1.002

Following are the basic production criteria:

Operating Days per Year	
Mill	365
Mine	250
Mine Production	
Tons per Year	3,660,000
Heap Leach -- SX-EW	
Tons per Year	2,221,250
Assay - Percent Total Copper	0.40
Recovery - Percent Total Copper	40
Pounds Copper Recovered per Year	7,108,000
Operation Life	7

Mining: Development of the B. S. & K. enrichment deposit involves the design, development and operation of an open pit mine. The mining of oxide and secondary sulfide copper mineralization re- sults in the development of a shallow open pit which is slightly smaller than that in Case I. Production rates are not limited by mill feed requirements and more optimum use of front-end loaders and truck haulage is possible. Consideration was given to purchasing new and rebuilt equipment, leasing equipment, or contract mining. Production for owner operated equipment was established at 8,885 tons per day going to the heaps with three months allowed for preproduction development. Dump areas for Case II are located the same as Case I.

The mine operation is scheduled to work 10, 8 hour shifts per week, 50 weeks per year. Shift productivity is estimated at 7,325 tons. With an overall estimated waste to leach ratio of 0.60:1, the mine will produce over 2.38 million tons of leach ore and 1.28 million tons of waste annually. Mine production will continue for about 6.8 years, allowing for the removal of over 24.8 million tons of total material.

The mining method parallels Case I in principle design and basic content. The only exception, with regard to mine capital, is the utilization of three 85 ton haul trucks with the one 15 YD^3 loader. Larger haul units are required to match the increased shift productivity.

The purchase of used mining equipment has been considered as an option for the leach operation. Scheduled operating hours for much of the mining equipment is less than half the expected machine life. The condition of this equipment and subsequent capital expenditure is based upon the required project use and the production dependency of each unit. Most of the used equipment is low hour and/or reconditioned, and will therefore have operating and availability figures similar to that of new equipment.

Used mine equipment includes 3 haul trucks, a track dozer, motorgrader, blast hole drill and water truck. The only service vehicle which merits purchasing used is the rough terrain crane. The remainder of the mine and general service vehicles are to be purchased new.

Processing: The selected heap area site is in a natural basin with a rocky bottom. The leaching pad surface will be prepared by clearing and grubbing the vegetation, followed by moderate contouring and compaction with mechanized equipment, making use of whatever material is available on the site. A pregnant solution dam will be prepared at the narrow part of the lower basin, keyed onto the rock bottom with concrete is possible. If not, an earth and clay dam lined with Hypalon (or similar material) will be constructed. Monitoring wells will be drilled below the dam to permit routine checks for leaks during operations.

The uncrushed ore will be loaded in the mine and trucked directly to the leach site. Each truck will be dumped on a previously leveled surface in a continuous series of windrows about five feet or more deep. When each five foot layer has been completed, it will be cross ripped to a depth of about seven feet with a tractor. Five, 5 foot layers totalling 25 feet will constitute a lift. After leaching is completed in 60 to 90 days, a new lift will be placed on top of the old one.

Leaching solution at 2,500 gallons per minute, coming from the solvent extraction plant raffinate storage, will be pumped to the leaching area. At a rate of 8,885 tons per day, five days per week, each heap of 266,550 tons will be built in six weeks and will have a nominal leaching cycle of 12 weeks. It will measure about

150 feet by 1,250 feet by 25 feet deep. The makeup water is estimated at 200 gallons per minute.

Solutions from the leach dump are fed through a solvent extraction plant at 2,300 gallons per minute. This is combined with 2,300 gallons per minute of organic so that the aqueous to organic ratio is maintained at a 1:1 ratio. The loaded organic is fed to the two stages of solvent stripping at a rate of 2,300 gallons per minute. Pregnant electrolyte from the solvent stripper is fed to electrowinning.

Costs: Mining capital and operating costs are based on the same criteria as Case I. Leaching, solvent extraction and electrowinning costs are based on 1979 leaching tests conducted by Mountain States Research and Development.

Capital Costs

Ancillary	$ 854,500
Environmental	127,000
Mine	
Equipment (used option)	2,326,600
Replacement Equipment - 4th Year	50,800
Preproduction Operation - 3 mo	530,200
Haul Road Construction	436,900
Plant	
Engineering & Design	698,500
SX-EW Construction	6,995,000
TOTAL	$12,019,500

Operating Costs

	Annual	$/lb Cu

Mine	$2,102,000	$0.296
Plant	2,155,700	0.303
General & Administration	1,143,000	0.161
TOTAL	$5,400,700	$0.760

Personnel:	Mine	29
	Leach	22
	G & A	12
	Total	63

CASE III: VAT LEACH, SOLVENT EXTRACTION, ELECTROWINNING

Description: Case III for the B. S. & K. feasibility study considers the recovery of copper through a heap leach-solvent extraction-electrowinning process where the mine run material is crushed to minus ½ inch before being placed on the dump. All secondary sulfide and oxide mineralization will be placed in heaps for leaching by sulfuric acid solutions. Recovery of copper through the solvent extraction-electrowinning system will result in a directly saleable product. Mining is designed for open pit methods employing front-end loaders and truck haulage.

Financial Analysis:

Production
Leach Cu = 74,634,720 lbs @ $1.01/lb = $75,381,067

Capital Expense
Capital Cost $14,067,600

$$\text{Capital Cost/lb} = \frac{\$14,067,600}{74,634,720} = \$0.188$$

Operating Cost = 0.653

Total Cost to Produce = $0.841

Following are the basic production criteria:

<u>Operating Days per Year</u>	
Mill	365
Mine	250
<u>Mine Production</u>	
Tons per Year	3,660,000
<u>Heap Leach -- SX-EW</u>	
Tons per Year	2,221,250
Assay - Percent Total Copper	0.40
Recovery - Percent Total Copper	60
Pounds Copper Recovered per Year	10,662,000
Operation Life	7

<u>Mining:</u> The mining methods and equipment are the same as in Case II, except for addition of crushing.

Mine run material will be hauled to the leach area and dumped into a portable three stage crusher. The crushing unit will reduce the material to minus ½ inch. The crushing unit will be moved in a manner so that the leach dump can be built on a continuing basis with feed from the crushing plant

<u>Processing:</u> The heap leach preparation and leaching, solvent extraction and electrowinning will be the same as presented in Case II.

<u>Costs:</u> Mining and processing costs were derived in the same manner as in Case I and II. Crushing costs were based on a quotation from Rexnard for capital and industry costs for operating.

<u>Capital Costs</u>

Ancillary	$ 854,500

Environmental	127,000
Mine	
Equipment (used option)	2,326,600
Replacement Equipment - 4th Year	50,800
Preproduction Operation - 3 mo	530,200
Haul Road Construction	436,900
Crushing Plant (portable)	2,048,100
Plant	
Engineering & Design	698,500
SX - EW Construction	6,995,000
TOTAL	$14,067,600

Operating Costs

	Annual	$/lb Cu
Mine	$3,373,700	$0.317
Plant	2,443,700	0.229
General & Administration	1,143,000	0.107
TOTAL	$6,960,400	$0.653

Personnel:		
	Mine	41
	Leach	22
	G & A	12
	Total	75

ASARCO - B. S. & K. JOINT VENTURE

This case of the B. S. & K. feasibility study considers the possibility of a joint venture between the B. S. & K. Mining Co. and ASARCO. The recovery of copper is considered for both processes of conventional flotation with heap leach of low grade and two all heap leach designs. Reserves have been estimated based upon available drill data and projections into undrilled areas. No pit designs were actually made because of the lack of detailed data on adjoining ASARCO ground. Estimates as to the mineable reserves were made based upon previous pit design results. Thirty-eight percent of the mineable reserves are on B. S. & K. ground. Production rates for both the flotation and heap leach alternatives were twice those used in Case I through III respectively.

Mining methods for the joint venture will be the same as for previous cases. The production rates for each option will double from those previously used in Case I through III. Additional production units were added to the capital cost as necessary. Support equipment was unchanged and the shop and other facilities were increased according to the increase in equipment. No allowance has been made for replacement of equipment as it is not believed significant for the scope of this option. New equipment costs were used throughout.

Process description for both options are essentially the same as Case I through III. The production rates have been doubled to accommodate the larger reserves. Capital and operating costs were scaled up from reports by Mountain States Research where applicable.

Sulfide Flotation:

Production

Flotation Cu	215,040,000 lbs @ $1.01/lb =	$217,190,400
SX-EW Cu	46,000,000 lbs @ $1.01/lb =	46,460,000
Total		$263,650,400

Mo (lb)	3,360,000 lbs @ $10.31/lb =	$ 34,641,600
Ag (oz)	384,000 ozs @ 21.45/oz =	8,236,400
		42,878,400
Total Value		$306,528,800

Capital Expense

Capital Cost	$ 48,499,900
Preproduction Mine	7,380,400
Preproduction G & A	2,159,000
Total	$ 58,039,300
Less Salvage	- 1,465,600
Less Mo & Ag Credit	- 42,878,400
Total	$ 13,695,300

Capital Cost/lb = $\frac{\$13,695,300}{261,040,000}$ = $0.052

Operating Cost/lb = 0.774

Total Cost to Produce = $0.826

Heap Leach, Solvent Extraction, Electrowinning:

Production

Leach Cu 152,880,000 lbs @ $1.01/lb = $154,408,800

Capital Expense

Capital Cost $ 24,789,000

Capital Cost/lb = $\frac{\$24,789,000}{152,880,000}$ = $0.162

Operating Cost = 0.665

Total Cost to Produce = $0.827

Vat Leach, Solvent Extraction, Electrowinning:

Production

Leach Cu 229,320,000 lbs @ $1.01/lb = $231,613,200

Capital Expense

Capital Cost		$28,885,100	
Capital Cost/lb	=	$\frac{\$28,885,100}{229,320,000}$	= $0.126
Operating Cost			= 0.568
Total Cost to Produce			= $0.694

Following are the basic production criteria:

	Flotation	Heap Leach	Vat Leach
Production Rate (Tons/Day)	8,000 Mill 6,086 Leach	17,770	17,770
Production Period (Days/Year)	365	250	250
Percent Copper	0.56 Mill 0.23 Leach	0.39	0.39
Percent Recovery	80 Mill 40 Leach	40	60
Annual Production (Pounds)	29,050,000	13,861,000	20,791,000
Production Life	8.2 Mill 11 Leach	11	11

Capital Costs (M Tons)

	Flotation	Heap Leach	Vat Leach
Ancillary			
Access	$ 571.5	$ 571.5	$ 571.5
Power	1,605.3	1,605.3	1,605.3
Communitations	50.8	50.8	50.8
Water	317.5	78.7	78.7
Environmental	254.0	254.0	254.0

Engineering & Design	$ 1,778.0	$ 1,117.8	$ 1,117.8
Mine			
Equipment & Support	6,828.8	7,120.9	11,217.0
Plant			
Flotation (8,000 T/D)	30,099.0		
SW-EW (2,300 gpm)	6,995.0		
SW-EW (4,600 gpm)		13,990.0	13,990.0
TOTAL	$48,499.9	$24,789.0	$28,885.1

Operating Costs (M Tons)

	Flotation	Heap Leach	Vat Leach
Mine			
Annual Cost	$ 5,340.4	$3,656.3	$ 6,237.4
Plant			
Flotation	5,228.3		
SX-EW	2,155.7	4,038.6	4,038.6
Smelter	7,713.9		
Supervision & Administration			
	2,032.0	1,524.0	1,524.0
Totals			
Annual Cost	$22,470.3	$9,218.9	$11,800.0
Cost per Pound	$ 0.774	$ 0.665	$ 0.568

Personnel

	Flotation	Heap Leach	Vat Leach
Mine	58	58	73
Mill	43	--	--
Leach	23	33	33

G & A	25	18	18
Total	149	109	124

Silver Bell Mill Option: This case is the same as Case I, except that no mill would be built. All millable sulfide ore will be hauled and processed through the Silver Bell Mill.

Mining methods and costs are the same as Case I except for the additional capital and haulage cost to transport the ore to the mill.

Processing methods and costs are the same as Case I except for a custom milling fee being added to the mill cost and the elimination of capital for the mill.

Total Value (Case I) = $115,456,935

Capital Expense

Capital Cost		$ 9,671,900
Preproduction Mine		3,690,200
Preproduction B & A		1,079,500
Total		$14,441,600
Less Salvage	-	732,800
Less Mo & Ag Credits	-	16,196,601
Total	-	$ 2,487,801

Capital Cost/lb $= \frac{\$2,487,801}{98,148,700} = \$\langle 0.025 \rangle$

Operating Cost/lb $= 0.884$

Total Cost to Produce $= \$\ 0.859$

Capital Costs

Ancillary $ 854,500

Environmental		$ 127,000
Mine Equipment		
	Initial	4,011,900
	Replacement - 4th Year	50,800
	Replacement - 5th Year	232,400
	Preproduction	3,690,200
	Haulage (ore)	372,000
Mill		
	Engineering & Design	400,000
	SX-EW	3,674,100
TOTAL		$13,362,100

Operating Costs

		Annual	Cost/lb
Mine			
	Production	$ 2,634,600	$0.160
	Haulage (ore)	823,400	0.052
Mill			
	Flotation	3,639,800	0.264
	Smelting	4,376,000	0.318
	SX-EW	832,500	0.311
Administration and Supervision		1,143,000	0.070
Custom Milling Cost		550,000	0.035
Total Cost		$13,999,300	$0.884

Frank H. Buchella, Jr.
1802 West Grant Road, Suite 110 • Tucson, Arizona 85705 • 602/623-9007

October 21, 1980

Mr. A. B. Kalaf, President
B. S. & K. Mining Company
P.O. Box 50325
Tucson, Arizona 85703

Dear Mr. Kalaf:

Enclosed are cash flows for Case III (crushing, heap leach, solvent extraction and electrowinning), Silver Bell Mill Option (sulfide flotation-all metals recovered), Silver Bell Mill Option (sulfide flotation-no leaching), and ASARCO-B. S. & K. Joint Venture (crushing, heap leach, solvent extraction and electrowinning).

The best case for you would be the joint venture-heap leach which would net you 22 million dollars over an eleven year operation. The Silver Bell Mill Option with no leaching is a toss-up with heap leaching your portion alone. Leaching with the Silver Bell Mill Option is marginal.

In summary, your best option is a joint venture-heap leach with ASARCO, second would be to sell the property to ASARCO, third would be to develop the property yourself by heap leaching and fourth would be to mine sulfides only and put the ore through the Silver Bell Mill.

Sincerely,

F. H. Buchella, Jr.
Mining Consultant

Registered Professional Engineer (Mining) CERTIFICATE NO. 13158 FRANK H. BUCHELLA, JR. Date Signed ARIZONA, U.S.A.

CASE III - CASH FLOW

CRUSHING, HEAP LEACH, SOLVENT EXTRACTION & ELECTROWINNING

	1981	1982	1983	1984	1985	1986	1987	1988	1989	TOTAL
INCOME										
COPPER @ $1.01/LB			10,768.7	10,768.7	10,768.7	10,768.7	10,768.7	10,768.7	10,768.7	75,381.0
COST										
ANCILLARY	427.0	427.5								854.5
ENVIRONMENTAL	127.0	—								127.0
MINE CAPITAL	—	5,341.8				50.8				5,392.6
MINE OPERATING	—	—	3,373.7	3,373.7	3,373.7	3,373.7	3,373.7	3,373.7	3,373.7	23,615.9
PLANT CAPITAL	—	6,995.0								6,995.0
PLANT OPERATING	—	—	2,443.7	2,443.7	2,443.7	2,443.7	2,443.7	2,443.7	2,443.7	17,105.9
ENG. & DESIGN	698.5	—								698.5
GEN. & ADMINISTRATIVE	—	572.0	1,143.0	1,143.0	1,143.0	1,143.0	1,143.0	1,143.0	1,143.0	8,573.0
BEFORE TAX INCOME OR EXPENSE	<1,252.5>	<999.5>	3,808.3	3,808.3	3,808.3	3,757.5	3,808.3	3,808.3	3,808.3	24,355.4
DEPRECIABLE CAPITAL	—	12,336.8								12,336.8
BEFORE TAX CASH FLOW	<1,252.5>	<13,336.3>	3,808.3	3,808.3	3,808.3	3,757.5	3,808.3	3,808.3	3,808.3	12,018.6
TAXES										
DEPRECIATION			3,524.8	2,517.7	1,798.4	1,284.5	917.5	655.4	468.1	11,166.4
BALANCE			283.5	1,290.6	2,009.9	2,473.0	2,890.8	3,152.9	3,340.2	15,440.9
DEPLETION			141.8	645.3	1,004.9	1,236.5	1,445.4	1,576.4	1,670.1	7,720.4
TAXABLE INCOME	<1,252.5>	<999.5>	141.7	645.3	1,005.0	1,236.5	1,445.4	1,576.5	1,670.1	5,468.5
INCOME TAX	<601.2>	<479.8>	68.0	309.7	482.4	593.5	693.8	756.7	801.6	2,624.7
INVESTMENT TAX CREDIT	—	1,233.7								1,233.7
NET TAX	<601.2>	<1,713.6>	68.0	309.7	482.4	593.5	693.8	756.7	801.6	1,391.0
BEFORE TAX CASH FLOW	<1,252.5>	<13,336.3>	3,808.3	3,808.3	3,808.3	3,757.5	3,808.3	3,808.3	3,808.3	12,018.6
NET TAX	<601.2>	<1,713.6>	68.0	309.7	482.4	593.5	693.8	756.7	801.6	1,391.0
WORKING CAPITAL			-1,740.1						+1,740.1	—
SALVAGE									1,170.4	1,170.4
			PAYOUT 4 YEARS - 1 MONTHS							
NET CASH FLOW	<657.3>	<11,622.7>	2,000.3	3,498.6	3,325.9	3,164.0	3,114.5	3,051.6	5,917.2	11,798.0

CERTIFICATE 13158 FRANK H. BUCHELLA, JR. Date Signed ARIZONA, U.S.A.

CASH FLOW
SILVER BELL MILL OPTION
SULFIDE FLOTATION (ALL METALS RECOVERED)

	1981	1982	1983	1984	1985	1986	1987	1988	1989	TOTAL
INCOME										
COPPER @ $1.01/LB			18,385.7	18,385.7	18,385.7	18,385.7	18,385.7	18,385.7	5,142.6	115,456.8
COST										
ANCILLARY	427.0	427.5								854.5
ENVIRONMENTAL	127.0	—								127.0
MINE CAPITAL	—	8,074.1				50.8	232.4			8,357.3
MINE OPERATING	—	—	3,458.0	3,458.0	3,458.0	3,458.0	3,458.0	3,458.0	1,122.8	21,870.8
PLANT CAPITAL	—	3,674.1								3,674.1
PLANT OPERATING	—	—	9,398.3	9,398.3	9,398.3	9,398.3	9,398.3	9,398.3	2,260.1	58,649.9
ENG. & DESIGN	400.0	—								400.0
GEN. & ADMINISTRATIVE	—	572.0	1,143.0	1,143.0	1,143.0	1,143.0	1,143.0	1,143.0	572.0	8,002.0
BEFORE TAX INCOME OR EXPENSE	<954.0>	<999.5>	4,386.4	4,386.4	4,386.4	4,335.6	4,154.0	4,386.4	1,187.7	25,269.4
DEPRECIABLE CAPITAL	—	11,748.2								11,748.2
BEFORE TAX CASH FLOW	<954.0>	<12,747.7>	4,386.4	4,386.4	4,386.4	4,335.6	4,154.0	4,386.4	1,187.7	13,521.2
TAXES										
DEPRECIATION			3,356.6	2,397.6	1,712.6	1,223.3	873.7	624.1	445.8	10,633.7
BALANCE			1,029.8	1,988.8	2,673.8	3,112.3	3,280.3	3,762.3	741.9	16,589.2
DEPLETION			514.9	994.4	1,336.9	1,556.1	1,640.2	1,881.1	371.0	8,294.6
TAXABLE INCOME	<954.0>	<999.5>	514.9	994.4	1,336.9	1,556.2	1,640.1	1,881.2	370.9	6,341.0
INCOME TAX	<457.9>	<479.8>	247.2	477.3	641.7	747.0	787.2	903.0	178.0	3,043.7
INVESTMENT TAX CREDIT	—	1,174.8								1,174.8
NET TAX	<457.9>	<1,654.6>	247.2	477.3	641.7	747.0	787.2	903.0	178.0	1,868.9
BEFORE TAX CASH FLOW	<954.0>	<12,747.7>	4,386.4	4,386.4	4,386.4	4,335.6	4,154.0	4,386.4	1,187.7	13,521.2
NET TAX	<457.9>	<1,654.6>	247.2	477.3	641.7	747.0	787.2	903.0	178.0	1,868.9
WORKING CAPITAL			− 3,499.8						+ 3,499.8	—
SALVAGE									1,114.5	1,114.5
NET CASH FLOW	<496.1>	<11,093.1>	639.4	3,909.1	3,744.7	3,588.6	3,366.8	3,483.4	5,624.0	12,766.8

PAYOUT 3 YEARS – 11 MONTHS

Registered Professional Engineer
CERTIFICATE NO. 13158
FRANK W. BUCHELLA, JR.
Date Signed
ARIZONA, U.S.A.

CASH FLOW
SILVER BELL MILL OPTION
SULFIDE FLOTATION (NO LEACHING)

	1981	1982	1983	1984	1985	1986	1987	1988	1989	TOTAL
INCOME										
COPPER @ $1.01/LB			16,553.6	16,553.6	16,553.6	16,553.6	16,553.6	16,553.6	3,310.5	102,632.1
COST										
ANCILLARY	746.5	—								746.5
ENVIRONMENTAL	60.0	—								60.0
MINE CAPITAL	—	8,074.1				50.8	232.4			8,357.3
MINE OPERATING	—	—	3,458.0	3,458.0	3,458.0	3,458.0	3,458.0	3,458.0	691.6	21,439.6
PLANT CAPITAL	—	—								—
PLANT OPERATING	—	—	8,553.1	8,553.1	8,553.1	8,553.1	8,553.1	8,553.1	1,710.6	53,029.2
ENG. & DESIGN	—	—	—	—	—	—	—	—	—	—
GEN. & ADMINISTRATIVE	—	923.3	923.3	923.3	923.3	923.3	923.3	923.3	184.7	6,647.8
BEFORE TAX INCOME OR EXPENSE	<806.5>	<923.3>	3,619.2	3,619.2	3,619.2	3,568.4	3,386.8	3,619.2	723.6	20,425.8
DEPRECIABLE CAPITAL		8,074.1								8,074.1
BEFORE TAX CASH FLOW	<806.5>	<8,997.4>	3,619.2	3,619.2	3,619.2	3,568.4	3,386.8	3,619.2	723.6	12,351.7
TAXES										
DEPRECIATION			2,691.4	1,794.2	1,196.2	797.4	531.6	354.4	—	7,365.2
BALANCE			927.8	1,825.0	2,423.0	2,771.0	2,855.2	3,264.8	723.6	14,790.4
DEPLETION			463.9	912.5	1,211.5	1,385.5	1,427.6	1,632.4	361.8	7,395.2
TAXABLE INCOME	<806.5>	<923.3>	463.9	912.5	1,211.5	1,385.5	1,427.6	1,632.4	361.8	5,665.4
INCOME TAX	<387.1>	<443.2>	222.7	438.0	581.5	665.0	685.2	783.6	173.7	2,719.4
INVESTMENT TAX CREDIT		807.4								807.4
NET TAX	<387.1>	<1,250.6>	222.7	438.0	581.5	665.0	685.2	783.6	173.7	1,912.0
BEFORE TAX CASH FLOW	<806.5>	<8,997.4>	3,619.2	3,619.2	3,619.2	3,568.4	3,386.8	3,619.2	723.6	12,351.7
NET TAX	<387.1>	<1,250.6>	222.7	438.0	581.5	665.0	685.2	783.6	173.7	1,912.0
WORKING CAPITAL			-3,233.6						+3,233.6	—
SALVAGE									708.9	708.9
			PAYOUT - 3 YEARS & 8 MONTHS							
NET CASH FLOW	<419.4>	<7,746.8>	162.9	3,181.2	3,037.7	2,903.4	2,701.6	2,835.6	4,492.4	17,148.6

13158
FRANK H. BUCHELLA, JR.
ARIZONA, U.S.A.

CASH FLOW
ASARCO - B.S. & K. JOINT VENTURE
CRUSHING, HEAP LEACH, SOLVENT EXTRACTION & ELECTROWINNING

B.S. & K = 38%

	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	TOTAL
INCOME			21,055.7	21,055.7	21,055.7	21,055.7	21,055.7	21,055.7	21,055.7	21,055.7	21,055.7	21,055.7	21,055.7	231,612.7
COPPER @ $1.01/LB														
COST														
ANCILLARY	1,153.0	1,153.3												2,306.3
ENVIRONMENTAL	254.0	—												254.0
MINE CAPITAL	—	11,217.0				101.4	464.8			101.4				11,884.6
MINE OPERATING	—	—	6,237.4	6,237.4	6,237.4	6,237.4	6,237.4	6,237.4	6,237.4	6,237.4	6,237.4	6,237.4	6,237.4	68,611.4
PLANT CAPITAL	—	13,990.0												13,990.0
PLANT OPERATING	—	—	4,038.6	4,038.6	4,038.6	4,038.6	4,038.6	4,038.6	4,038.6	4,038.6	4,038.6	4,038.6	4,038.6	44,424.6
ENG. & DESIGN	1,117.8	—												1,117.8
GEN. & ADMINISTRATIVE	—	762.0	1,524.0	1,524.0	1,524.0	1,524.0	1,524.0	1,524.0	1,524.0	1,524.0	1,524.0	1,524.0	1,524.0	17,526.0
BEFORE TAX INCOME OR EXPENSE	<2,524.8>	<1,915.3>	9,255.7	9,255.7	9,255.7	9,154.3	8,790.9	9,255.7	9,255.7	9,154.3	9,255.7	9,255.7	9,255.7	96,705.0
DEPRECIABLE CAPITAL	—	25,207.0												25,207.0
BEFORE TAX CASH FLOW	<2,524.8>	<27,122.3>	9,255.7	9,255.7	9,255.7	9,154.3	8,790.9	9,255.7	9,255.7	9,154.3	9,255.7	9,255.7	9,255.7	71,498.0
TAXES														
DEPRECIATION			4,583.1	3,749.8	3,068.0	2,510.2	2,053.8	1,680.4	1,374.9	1,124.9	920.3	753.0	616.1	22,434.5
BALANCE			4,672.6	5,505.9	6,187.7	6,644.1	6,737.1	7,575.3	7,880.8	8,029.4	8,335.4	8,502.7	8,639.6	78,710.6
DEPLETION			2,336.3	2,753.0	3,093.8	3,322.0	3,368.6	3,787.6	3,940.4	4,014.7	4,167.7	4,251.4	4,319.8	39,355.3
TAXABLE INCOME	<2,524.8>	<1,915.3>	2,336.3	2,752.9	3,093.9	3,322.1	3,368.5	3,787.7	3,940.4	4,014.7	4,167.7	4,251.3	4,319.8	34,915.2
INCOME TAX	<1,211.9>	<919.3>	1,121.4	1,321.4	1,485.0	1,594.6	1,616.9	1,818.1	1,891.4	1,927.1	2,000.5	2,040.6	2,073.5	16,759.3
INVESTMENT TAX CREDIT		2,520.7												2,520.7
NET TAX	<1,211.9>	<3,440.0>	1,121.4	1,321.4	1,485.0	1,594.6	1,616.9	1,818.1	1,891.4	1,927.1	2,000.5	2,040.6	2,073.5	14,238.6
BEFORE TAX CASH FLOW	<2,524.8>	<27,122.3>	9,255.7	9,255.7	9,255.7	9,154.3	8,790.9	9,255.7	9,255.7	9,154.3	9,255.7	9,255.7	9,255.7	71,498.0
NET TAX	<1,211.9>	<3,440.0>	1,121.4	1,321.4	1,485.0	1,594.6	1,616.9	1,818.1	1,891.4	1,927.1	2,000.5	2,040.6	2,073.5	14,238.6
WORKING CAPITAL			- 2,950.0										+ 2,950.0	—
SALVAGE													1,315.5	1,315.5
NET CASH FLOW	<1,312.9>	<23,682.3>	5,184.3	7,934.3	7,770.7	7,559.7	7,174.0	7,437.6	7,364.3	7,227.2	7,255.2	7,215.1	11,447.7	58,574.9

PAYOUT = 3 YRS & 7 MONTHS