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The following file is part of the

James Doyle Sell Mining Collection

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

January 27, 1964

FILE MEMORANDUM

Sacaton Basement Fault
Sacaton Prospect, Arizona

(The Sacaton "Basement fault" is the most unique feature of that deposit, and thus is here singled out for special discussion.)

(Briefly,) the Basement fault is a low-angle slightly undulating surface of slippage which was activated after initial ore formation and subsequent oxidation-enrichment. It has sliced through the entire Sacaton altered zone and shifted the hanging wall portion several miles at least, to its present position above unmineralized schist; the foot-wall block, or "root", remained stationary (and now lies somewhere covered by valley gravel.)

I think the age of the faulting is late Miocene or earliest Pliocene -- but this point will not be further elaborated. (It is, however, older than the pediment surfaces surrounding the Sacaton mountains.)

The magnitude of the fault and rocks cut by it are shown on the attached cross-section ((Attachment B).) [ ]

The Fault Plane

At the outset of exploration at Sacaton, the Basement fault was not known, nor was there any reason to suspect that such a structure existed. (D.D.H. 3), drilling in an enriched chalcocite zone, passed through about 10 feet of tight gouge and breccia of obvious post-ore origin, and then into barren gneiss at 1910 feet. (A post-ore andesite dike also was cut in this interval.) Subsequently other drill holes penetrated the fault, and its physical character, configuration and age relationship with respect to the mineralized and enriched hanging wall block are now well established.

The fault plane has been cored most extensively in the area of the east "ore body" and in the mineralized but non-commercial area just east of that. To the northeast, south, and southwest penetrations are fewer, and mostly made with the rotary drills (non-core). In the vicinity of the east "ore body" the fault is made of wavy, sheared zones of alternating basement rock and mineralized hanging wall. The shear planes and their gouge streaks are rather firm, and have been cored with little core-loss. Thickness of the fault zone is generally 5 or 10 feet, the thickest section cored being about 25 feet.

Effect on mining

The hanging wall rocks of the west "ore body", which are pervasively altered granite and monzonite porphyry, are neither brecciated nor otherwise effected by the fault movement. This is not the case in the east "ore body", however, for there the rocks have suffered intense shattering, brecciation, and granulation, which I attribute to forces set up while the fault was in progress (memo to K. Richard 7/2/63). The apparent porosity of this zone, as observed in the core, has led to the studies which Core Laboratories are now doing; their efforts being a first step to determine the possibility of leaching-in-place. This condition was described in the memo referred to and will not be repeated here. This rock condition will no doubt effect the manner in which the ground responds to block caving.

The footwall beneath the east "ore body" shows little or no crushing of the type described in the foregoing, and remains a relatively strong competent rock. I see no reason to think that haulage ways or raises cutting the footwall rocks would offer any unusual mining-stability problems. The effect of the Basement fault on raises which penetrate through it is principally a problem which must be determined at the outset of development. Post-ore, gougy faults have in some cave mines caused difficulty from swelling under static load; however, the Basement fault may be strong enough to hold without difficulty. If necessary, of course, the entire development may be placed in the hanging wall block, and held open with appropriate support.

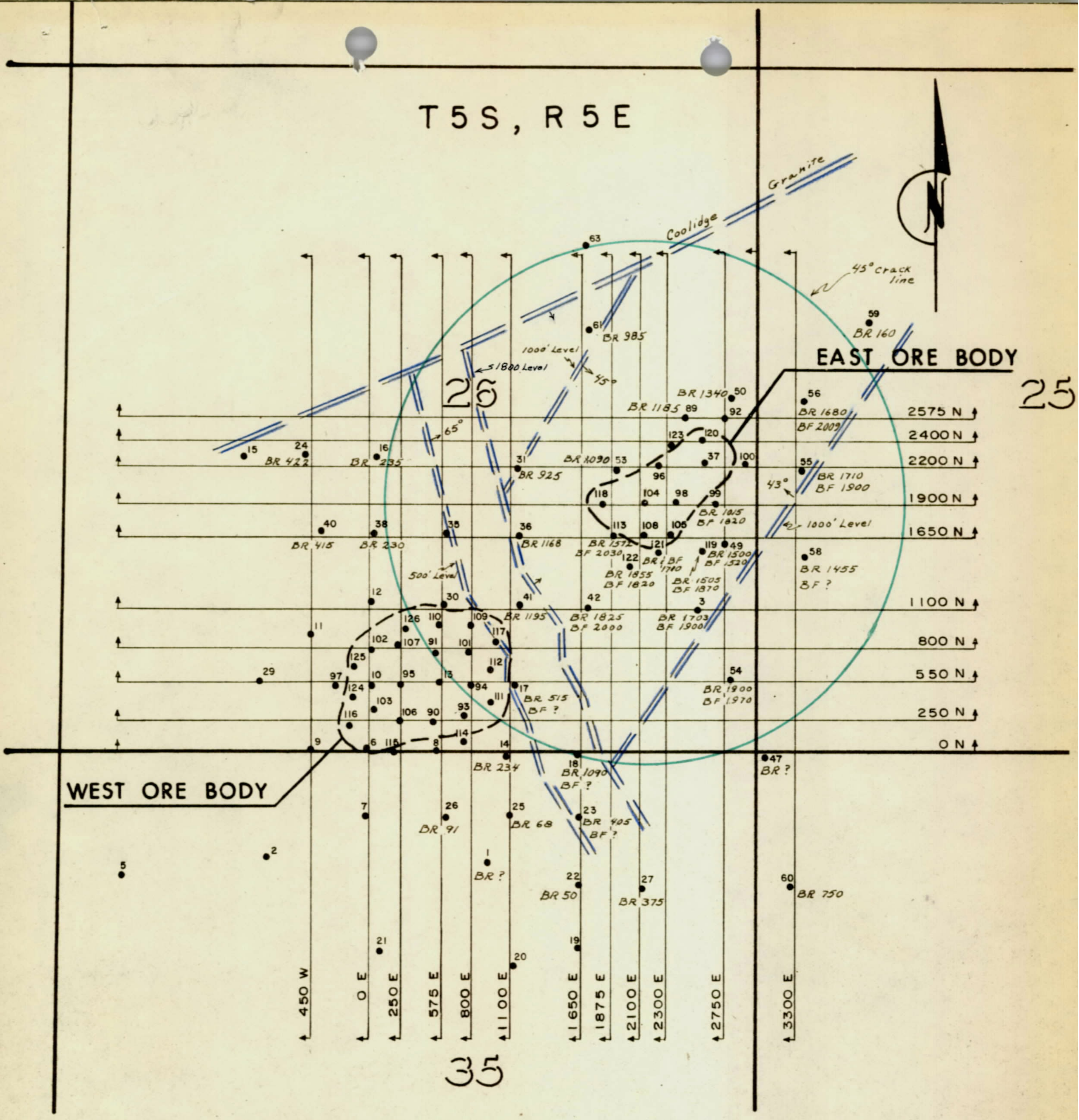
Exploration possibilities

The Sacaton porphyry copper deposit is known to be 2-1/2 miles long and one mile wide. The NE end is terminated by faults and the SE by middle Tertiary erosion, so originally the zone was of greater extent in length. The altered zone generally grades .2 to .4% Cu as chalcopyrite. The enriched blanket which covered this protore has been segmented by faults, eroded in some portions of the zone, leached in others, and cut off by the Basement fault in the vicinity of the east "ore body." It is not unreasonable to estimate that, at one time in the geologic past, a copper deposit existed which was comparable in size to Ray or Miami.

The "root" of an altered zone of this size is a target worth searching for, as has been pointed out by Mr. Courtright in earlier letters. Both leached capping with live limonite, as well as strongly enriched chalcocite bodies were cut through by the fault, and their lower portions left behind. On the basis of regional mapping and drilling, the Sacaton mountains proper as well as the covered areas between them and the Sacaton deposit have been virtually eliminated as possible "root" areas. It seems most probable that the "root" lies somewhere to the south or southwest in the Casa Grande valley, part of which is known to be underlain by rock with reasonable reach of mining exploration. More work is planned on this possibility.

John E Kinnison

J. E. KINNISON



SACATON PROJECT

Pinal County, Arizona

SCALE 1" = 1000'

October 1969

JHC

J. H. C.

JAN 22 1969

SACATON

ATTACHMENTS	SCALE
A - Arizona Index Map showing Sacaton location	
B - Plan Map showing local culture	1" = 1-1/3 miles
C - Bench tabulation sheets - Open Pit	
D - Block Cabe tabulation sheets	
E - West deposit cross sections showing open pit profile and benches (10)	1" = 200'
F - East Deposit cross-sections (4)	
G - Plan Map showing block cave and open pit layouts	1" = 200'
H - Bench plans, open pit (19) In Pocket	1" = 200'

Replace in bottom of JHC "this" Ore Res. file - Sac.

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona
January 18, 1969

Mr. J. J. Collins, Chief Geologist
American Smelting and Refining Company
120 Broadway
New York, N. Y., 10005

SACATON ORE RESERVE
PINAL COUNTY, ARIZONA

Dear Sir:

In accordance with your telephoned request, ore reserve data with plan maps and sections, supporting previously reported figures on tons and grade, were assembled and are enclosed herewith.

Final drafting is far from complete thus the maps and sections attached are in part work sheets. Further, this ore estimate was calculated at the end of September, 1968, while drilling was in progress, and is therefore an interim estimate. However, the results of subsequent drilling have not materially changed either tonnage or grade.

SUMMARY

The Sacaton area, located about midway between Phoenix and Tucson, Arizona, contains two separate porphyry copper deposits, identified as the "East" and "West" ore bodies. The copper in both occurs as chalcocite and chalcopyrite in altered granite and porphyry.

The East ore body is a gently dipping tabular mass, approximately 260' in thickness with horizontal dimensions of 600' by 1200'. It lies at an average depth of around 1500' and thus is mineable only by underground methods, such as block caving or sublevel stoping.

The West ore body, 1000' in diameter, varies in thickness from 100' to 400'. It lies at an average depth of about 400' and is amenable to open pit mining, but with a rather high stripping ration.

Results of our preliminary sulfide ore calculations (as of October 1, 1968) were as follows (tons in round figures):

	Tons	Grade	Stripping Ratio
WEST	17,000,000	0.83% Cu	w/o - 6.4/1
EAST	12,500,000	1.52% Cu	

Drilling carried out in the past few weeks has completed delineation and measurement of the West deposit. However, the East deposit will require underground development and drilling to adequately define the ore body for block-cave mine planning.

Drilling

An initial program of wide spaced drilling was conducted on the Sacaton Project between September 2, 1961 and November 1, 1962. During this period a total of 73,609 ft. of rotary and core drilling was completed in 65 holes. Significant intercepts of copper mineralization were penetrated in 11 of these holes, indicating two separate zones of bulk-low grade mineralization.

A second drilling program, initiated in February 1968 and continued to the present, has been conducted to delineate the two zones of mineralization and permit the calculation of preliminary ore reserves. A total of 45,608 ft. of rotary and core drilling has been completed in 37 holes during the second phase of drilling.

A relatively shallow, secondarily enriched blanket of ore grade copper sulfides which may be amenable to open pit mining has been defined in the West zone. This deposit has been identified as the 'West Ore Body." To the east, drilling has defined a deep deposit of enriched copper sulfides, which may be economically exploitable by underground block caving methods. This deposit has been identified as the "East Ore Body."

Preliminary ore reserves were calculated for the two deposits in September, 1968. A summary of these ore reserves and descriptions of their development follows:

West Ore Body

Open pit reserves were calculated from the assay results of 15 holes using the method of bench polygons. These holes included S-6, 8, 10, 13, 30, 90, 91, 93, 94, 95, 101, 102, 103, 106, and 107. Pit limits were determined using North-South and East-West Sections. A bench interval of 40 ft. was utilized and ultimate pit slopes were established at 33^{o} in alluvium (surface to 240 ft. depth) and 45^{o} in the underlying consolidated rocks.

Ten sections showing drill hole ore intercepts, extent of ore by bench and pit limits are attached. A surface plan showing the ultimate pit and 19 bench plans are also enclosed. A set of bench ore and waste tabulation sheets are attached. These list the data included in tonnage and grade calculations, including polygon areas and weighted averages of bench assays for each hole.

Seven holes, 2 of which have ore grade intercepts, have been drilled in the vicinity of the 'West Ore Body" since the September, 1968 ore reserve calculation. These holes are plotted on the plan maps but are not shown on sections. Bench assay values for these two recently drilled ore holes (S-109 and S-110) have been assigned from the assays of adjacent holes in the calculation of this reserve. Substitution of the actual assays from holes S-109 and S-110 would not materially alter the reserves presented herein. Other assigned bench assays listed in the tabulation sheets are the result of polygons which fall generally outside the apparent ore boundary at that particular bench elevation.

The preliminary open pit ore reserve for the 'West Ore Body" is 17,377,000 short tons averaging 0.83% Cu. The corresponding open pit waste is 111,330,000 short tons, yielding a waste/ore ratio of 6.4/1. Thirteen of the 19 benches contain ore. Waste is composed of approximately 45% unconsolidated overburden with an assumed density of 15 cu. ft/ton and 55% rock or hard congolmerate @ $12\frac{1}{2}$ cu. ft./ton.

East Ore Body

The underground reserve calculated for the "East Ore Body" is based on the assay results of 10 holes (S-37, 53, 96, 98, 99, 104, 105, 113, 118, and 120). Because the average depth of the top of ore is 1460 ft., a block cave reserve has been calculated rather than one incorporating an open pit. Drilling has established relatively thick ore columns with an average vertical extent of 260 ft.

Reserves were calculated for 14 individual blocks which are bounded laterally by vertical planes. The tops and bottoms are determined by the extend of the ore intercept in the hole or holes assigned to each block. In plan, these blocks are grouped into 4 Northeast - Southwest panels whose centerlines connect a particular series of drill holes. The "East ore body" panels are plotted on the same surface plan map which shows the ultimate pit for the 'West' Ore Body". The overall composite dimensions of the ore blocks, in plan, are 965 ft. by 640 ft. The volume of each block was determined by the product of the panel width and the Longitudinal ~~cross~~ sectional area of the block. These figures are listed on the attached panel tabulation sheets. Where two holes were assigned to a block, the grade used was the average of both holes within the vertical limits of the block. A density factor of 12.5 cu.ft./ton was assumed.

The preliminary block cave reserve for the "East Ore Body" is 12,672,000 short tons averaging 1.52% Cu.

Yours very truly,

J. H. Courtright

J. H. Courtright

W. E. Saegart

W. E. Saegart

JHC,WES:lab
cc: TASnedden
RBMeen

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

J. H. C.

MAY 28 1964

May 28, 1964

MEMORANDUM FOR MR. J. H. COURTRIGHT

Re: Sacaton Ore Reserve and Outcome Estimate Capital Cost Summary

According to our conversation today the following comparisons of Silver Bell actual costs and Mr. Desvaux's estimates for Sacaton are tabulated.

	Silver Bell Cost	Year Expended	Desvaux Estimate
General Office	$ 48,491	1953	$100,000
Assay Office	42,000	1953	-
First Aid Bldg.	22,500	1953	-
	$112,991		$100,000
Haulage Road			
Paved 21' wide 7.7 mi. E. of town	$187,274 ($ 24,321/mi.)	1953	$ 20,000/mi.

These are the only items that can be readily compared, but it appears that Mr. Desvaux did not escalate the Silver Bell cost according to the Cost Index. His figures for mine equipment and underground development are based on modern costs.

J. R. Wojcik

J. R. Wojcik

JRW:bam

Silver Bell

3-500' wells @ $8365 ---- $25,095

47,700' of 18" pipe @ $6.40 305,064

Pumps, tanks & controls 435,007

$765,166

Desvaux Estimate

2-500' wells and 4 mi. of pipe line.

$234,000

Aa-16A.3.19B

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

September 27, 1968

TO: J.H. Courtright

FROM: J.R. Wojcik

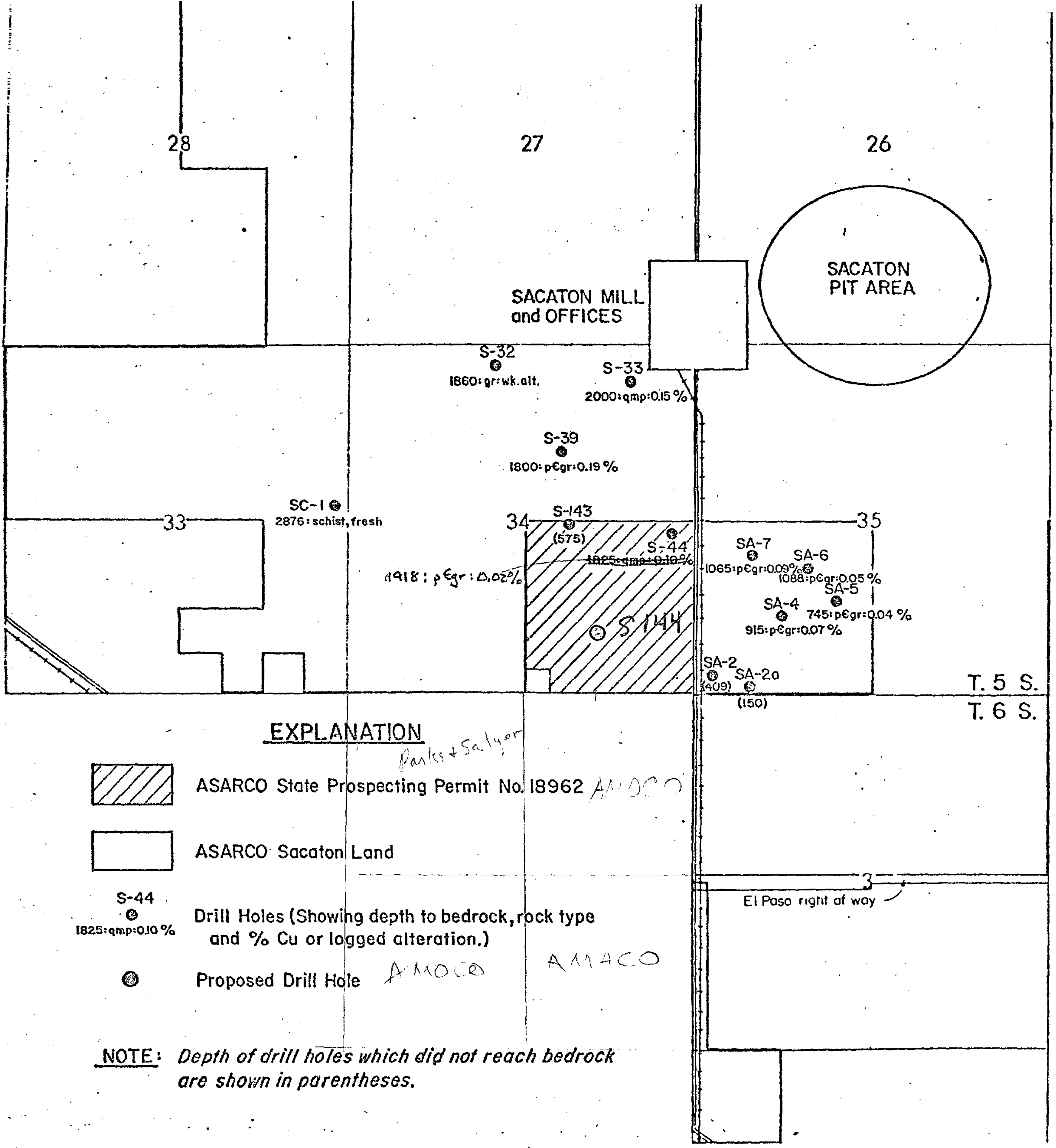
During the past two weeks, Sergei Zelenkov and myself calculated an "ore" reserve for the west "ore" body at Sacaton. An open pit was designed using 45° slopes from the bottom up to a depth of 240'. The upper 240' was planned with 33° slopes. A 60' wide road was included from the bottom of the pit to the surface. Polygons were constructed around drill holes and assays calculated for 40' benches. In total there are 19 benches, 13 containing "ore".

The calculation shows 17.37 million tons available at 0.83% Cu with an overall w/o ratio of 6.4/1. The waste is composed of approximately 45% gravels and unconsolidated overburden @ 15 cubic ft/ton and 55% rock or hard conglomerate @ 12.5 cubic ft/ton. The sections, bench plans and bench calculation sheets are in the Sacaton drawer in the drafting room.

On the east "ore" body, by constructing polygons, measuring their area and multiplying by the corresponding ore columns in drill holes 37, 96, 98 and 104 a reserve of 8,635,360 tons at 1.70% Cu (sulfide) can be calculated. Drill holes 105 and 108 will increase this tonnage substantially.

J.R. Wojcik

JRW:kzb



TO ACCOMPANY *Exploration Authorization Request*
DATED *Jan. 23, 1976*
BY *F.T. Graybeal*

LAND STATUS MAP

SACATON PROSPECTING PERMIT No. 18962

Sacaton Mining District

Pinal County, Arizona

SCALE 1" = 2000'

F.T.G. Jan. 23, 1976

MVK 2904

Hank K.

Bagdad: 0.06-0.09 oz Ag per ton of ore (Anderson, 1950)
0.02 oz Ag recovered per ton of ore

Sacaton: Underground 0.004 oz Au, 0.07 oz Ag, 0.028% Mo
Open Pit 0.003 oz Au, 0.13 oz Ag, 0.016% Mo

2-1-79

Mining Costs
from Tom Scartaccini

Sacaton dir $.45 + 50\% = .68$ (indir above 50%)

Silver Bell $.53 + 50\% = .80$

Geo Percival — 2-6-79

Sacaton Mining Cost
All Material - 1978

Direct - 0.45/ton
Indirect - 0.14/ton
Gen. Deductions - 0.32/ton
Total 0.91/ton

General Deductions include N.Y. office, SW. office, Depr. Depletion etc.

Mill costs $1.79 per ton ore. c dir. + indir.

Total Cost/ton ore - $5.13

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

May 28, 1970

TO: W. E. Saegart

FROM: A. Dalla Vista

Sacaton West Orebody
Six-inch Core Drilling
Final Report
Pinal County, Arizona

Drilling on the project area began on January 17 and was completed by the 4th of May 1970.

The prime contractor was Shelton Drilling Company of Pheonix, Arizona. However, as explained later in this report, one phase of the drilling was sub-contracted to Golden State Drilling Company of Frisco, Colorado.

No breakdowns or delays hampered the drilling program.

A total of approximately 38.2 tons of bulk sample have been produced. Of these,22 tons represented low-oxide sulphide ore (< .1% Ox-Cu) and 16.2 tons were classified as high oxide sulphides (≥ .1% Ox-Cu). In addition, about 1.3 tons of rotary cuttings, derived from what was believed to be ore grade material within the oxide zone, were collected for leaching tests.

A total of 15 holes were drilled. Ten were located about 10-15 feet from the original exploratory diamond drill holes, and five (SBM through SGM) were interspaced within the area of stronger high-oxide sulphide ore.

Two of the interspaced holes (SCM and SHM) were not cored, as originally planned.

The tabulation on the following page shows the footage rotary drilled and the footage cored in each individual hole.

Hole No.	Rotary (10") (ft)	Rotary to Core Point (8") (ft) in bedrock	Hi-Ox Suls. (ft)	Lo-ox Suls. (ft)	* Advance ft/hr
S-GM	270	34.50	106.50	103.00	1.98
S-10M	260	75.00	- -	183.50	1.97
S-13M	145	289.00	87.00	148.00	1.61
S-90M	190	219.00	99.00	- -	1.38
S-91M	105	175.00	110.00	235.00	1.40
S-93M	175	108.00	32.00	143.00	1.80
S-94M	105	270.00	96.00	148.00	1.48
S-95M	200	180.00	25.00	173.70	1.61
S-103M	260	190.00	53.20	131.00	2.00
S-111M	165	143.00	52.00	199.50	1.65
S-BM	125	246.00	100.00	- -	2.12
S-CM	165	253.00	NOT CORED		
S-DM	100	340.00	130.00	- -	2.06
S-EM	215	145.00	60.00	- -	1.70
S-FM	180	220.00	55.50	- -	1.03
S-GM	145	235.00	80.00	- -	2.10
S-HM	240	150.00	NOT CORED		
TOTALS	3,045	3,276.50	1,086.20	1,464.70	1.75

* Computed on the total time charged: Actual drilling, Mixing mud and moving.

Average weight of core was 30.0 lbs/ft.

Drilling was done in three different phases and a detailed description of the technical aspects of each is given below.

ROTARY DRILLING TO BEDROCK and CASING (10")

In the 15 holes drilled, bedrock depths ranged from 95 to 270 feet and averaged approximately 178 feet. About forty five (12 hour) shifts were required to drill the aggregate 3,045 feet through overburden.

Drilling was done with a Failing CF-15 WABCO using a 9 7/8" Tricon bit to drill a hole of sufficient diameter to set 8 5/8" OD casing.

The mud used for this phase was a mixture of quickgel, quicktrol and CC-16. Viscocity of the mixture was generally about 35-40 seconds.

Attempts to pull the casing upon completion of the holes proved unsuccessful.

Cost for this phase run $7.00 per foot of drilling. Casing not recovered was charged at the rate of $2.75 per foot.

ROTARY DRILLING TO CORE POINT IN BEDROCK (8")

When penetration using mud circulation with the Failing CF-15 proved to be far from satisfactory, this phase was subcontracted to Golden State Drilling of Frisco, Colorado. A Portadrill with a 600 cfm and a 130 psi auxiliary compressor were brought in and good penetration was achieved.

A total of 3277 feet were drilled at a cost of $4.75 per foot.

The advance averaged around 96 feet per shift (10 hours): Quickfoam was the only additive used, to help the recovery of rotary cuttings when drilling below water table.

CORE DRILLING (6")

Coring was done on a 24-hour basis with a newer model CF-15 WABCO. No difficulties were encountered with either the penetration or recovery of the core. A total of 2,551 feet of core were drilled and recovered. Actual rate of penetration varied considerably upon the rock involved, but averaged about 2 to 4 feet per hour(not including moving or mixing mud). A Longyear 7 1/2" x 10' double tube swivel type core barrel was used. The 6" x 7 7/8" diamond bits of approximately 115 carats were also purchased from Longyear. Each diamond bit averaged about 220' of coring. A salvage value of about 70% was credited for the eleven bits used.

MUD PROGRAM

A basic mixture of loloss, WOLF, $CaCl_2$, flosal, walnut flour, and condet was found to be appropriate for the specific rock types involved. The following list shows the amount of each additive per 1000 gallons of fresh water.

Loloss	50 lbs.
WOLF	50 lbs.
Flosal	50 lbs.
Walnut flour	50 lbs.
$CaCl_2$	250 lbs.
* Condet	2 gallons

* One gallon of condet was added each day to control the build up of formation solids. Viscosity was checked four times daily and kept around 45-50 seconds.

By using two large mud pits, the sand content was kept below .5%, thereby increasing rate of penetration and prolonging bit life.

DRILLING COSTS

ROTARY COST

9 7/8" hole to bedrock, 3045 feet @$7.00 per ft.	$21,315.00	
Casing 8 5/8 OD: 3045 feet @$2.75 per ft.	8,343.00	
8" hole to core point: 3276.50 ft. @ $4.75 per ft.	15,349.00	
Total Direct Cost for Rotary Drilling		$45,007.00

CORE DRILLING COST

Coring time 1575.75 hrs. @ $32.00/hr	$50,424.00	
Sub-direct cost $19.70/ft		
Bit Costs	21,843.13	
Core barrel & spare parts	3,538.85	
Total Cost	$75,805.98	
Credited to ASARCO per salvage value of bits	$14,630.00	
Total net cost of coring		$61,175.98
Total Direct Cost for Rotary Drilling brought forward		$45,007.00
GRAND TOTAL OF DRILLING, Rotary plus Coring		$106,182.98

Mud cost, overhead and other contingencies was estimated to be around 15% of the total cost.

Aggregate direct cost of core drilling including time charged, diamond bits and core barrel: $ 23.80 per foot.

Water was purchased from the Arizona Water Company at a rate of $1.00 per 1000 gallons.

CORE HANDLING and PREPARATION

From the core barrel the core was placed in metallic channels 10 feet long and very carefully washed to remove any additives and mud. After being logged, the core was placed in lined 55 gallon drums. Each drum was lined with two polyethylene bags. As an average, each drum contained approximately 20 feet of core or about 600 pounds. Number and type of sample, interval represented, and type of rock were written on the outside of the drum as well as on a tag placed on the inside. Drums were shipped, via Pacific Trucking Company to ASARCO Metallurgical Research Laboratory in El Paso.

Attached are photos showing several features of the drilling program.

A. Dalla Vista

ADV:lab

~~wlk file: Section~~

REPRINTED SECTION OF THE MARCH 1974 ISSUE

PAY DIRT

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

Number 417 | P. O. Drawer 48, Bisbee, Arizona 85603 | March 25, 1974

The two basic excavating systems used

Sacaton —
6-9-75

Hole drilled in central part of deep cc ore body to check basement fault!:

	140'	at	2.92 % Cu
+	90'	at	1.99
+	70	at	1.27
+	300'		± 2.30

No 139 — 280' @ 2.34 % Cu to -320 elev (underext level)

3-24-76 RBC —

Oper. shaft — 4 mil

Drive 1

~~develop — workings + drilling? 1~~

6 mil

X

AMERICAN SMELTING AND REFINING COMPANY
TUCSON ARIZONA

May 30, 1974

MEMORANDUM FOR RECORD:

Arizona's newest copper mine, Sacaton, came into production during March of this year, 13 years after porphyry type mineralization was recognized by ASARCO geologists in a small knoll projecting a few feet above an alluvial plain near the city of Casa Grande. The outcrop was virgin, there having been ~~being~~ no evidence of previous prospecting, although numerous pits had been dug on copper shows in the hills lying two miles to the north.

The discovery was made during the course of reconnaissance along a porphyry belt extending west-southwest from the Miami-Inspiration-Superior Districts. The area of search was narrowed by an old letter dug out of company files which reported oxide copper found in the bottom of a hole drilled for water in 1919 "Somewhere in the Vicinity of Casa Grande."

Attempts to locate the well were unsuccessful; however, the reported copper was very likely detrital, rather than "in place," as our drilling encountered copper bearing boulders in the alluvium which is several hundred feet deep in the area.

After obtaining options on the fee land and prospecting permits on the State land, an authorization of $30,000 to drill six holes was approved. The first five put down near the outcrop cut only very low values in the sulphide zone underlying the leached capping, but the sixth located one-half mile northerly found the southern edge of the chalcocite ore body that is now being mined open pit. Further prospect drilling through the gravel cover located a richer but smaller deposit at a depth of over 1500 feet, which will be mined underground by block cave. The combined operation is scheduled to produce 315,000 tons of copper metal during its 15-year life.

The initial drill program, which commenced in September 1961, was discontinued late in 1963 with the completion of extensive drill prospecting for several miles along the northeast trend of the mineralized zone. Feasibility studies did not indicate a profitable operation at that time. Four years later a much better copper market encouraged closer spaced drilling of the two "indicated" ore deposits. The total cost of exploration through to the development stage amounted to $1,170,000.

J. H. Courtright

JHC:vmh

~~Copy for~~ ~~WLK & CKM~~

GEOPHYSICAL DIVISION

3422 South 700 West
SALT LAKE CITY, UTAH 84119

RECEIVED
JUN 3 1971
S. W. U. S. EXPL. DIV.

June 1, 1971

Mr. W. L. Kurtz
Tucson Office

Dear Bill,

Thank you for your memo of May 24, 1971, commenting on my report of April 20, 1971, on "Geochemical Orientation Groundwater Study, Vicinity of Sacaton Prospect, Pinal County, Arizona".

In reply to your comments:

1. The interpretation presented in my report is of a provisional nature--mainly because of the insufficiency of the original sample coverage. When analytical data relating to more extended sample coverage becomes available, a re-interpretation, taking into account all available hydrogeologic and geologic information, will be carried out.

2. Gila is the name used (at least by the geophysicists) for the Cu prospect located some eight miles north of Casa Grande. It was drilled, I think, some years ago by Asarco?

3. The data presentation is a provisional one, using a base map previously prepared for other purposes. It has the advantage of allowing a relatively compact presentation. The final data presentation for Sacaton (and the other areas covered by our groundwater sampling program) will probably be on copies of standard topo bases.

Yours very truly,

Lloyd

L. D. JAMES

LDJ:db

cc:W.E.Saegart

Bill K.

✓ This is site of largest I.P. anomaly in world!
Was tested by geophysicists w/ one hole - Interpreted depth to top of polarizer corresponded to water table in clayey conglomerate environment. CKM should remove "Gila" from his "snake" magnetic gradient trend maps.

WES

SACATON ESTIMATE

Feb 10 1972

INTRODUCTION

by C.E. Williams

This report combines two earlier reports into a single, self-contained and updated version. The two reports that have been revised and included are "Sacaton Estimate" by Carl E. Williams, October 8, 1970 and "Sacaton Project, Capital Cost and Outcome Estimate", September 7, 1971.

The Sacaton property, which is a porphyry copper deposit, is located 75 miles northwest of Tucson and 5 miles northwest of the town of Casa Grande. (See Attachment B.) There have been 112 holes drilled in the area. Of this total 22 holes define a west orebody and 12 holes define the deeper east deposit. Both of these deposits are minable, but by different methods. The west deposit reaches a depth of about 1000 feet and has been found feasible to be mined by open pit. The deeper ore deposit to the east, which has been vertically displaced from the west deposit by faulting, reaches a depth of 2000 feet, and therefore must be mined by underground methods. A caving method has been chosen because of the apparent amenability of the ground to caving action.

The two ore bodies contain a total ore tonnage of 47,585,000 tons at an average grade of 0.95% cu.

At 9000 tons per day, or 3,186,000 tons 354 day year per year, the life of the operation will be 15 years.

The sequence of mining the two deposits will be open pit first, followed by the underground operation.

Open Pit Mining

A thickness of 80' to 100' of alluvium overlies the pit area. It is planned to strip this gravel with scrapers, either by a contractor or ASARCO personnel and equipment. To coincide with the mining schedule the gravel must be stripped at the rate of 718,000 tons per month. At this rate the underlying conglomerate will be exposed enough to allow shovel production at the end of seven months. The scraper stripping will then continue for a total of 2.7 years. The premine stripping will require two of these years. The use of the scrapers will facilitate tailings dam construction and plant site fill. The shovels will handle the premine rock and conglomerate in 1.4 years.

Due to the geometry of the pit and the depth of the ore, a great deal of premine stripping is required as well as a large waste-to-ore ratio during the early years of production.

The pit is designed with 40-foot benches, a slope of 1-1/4 to 1 in the alluvium and 1 to 1 in the rock, 8% haul roads with a width of 80 feet, and one year supply of ore exposed at all times after the pit has been put into production.

(Introduction - Cont'd)

With the exception of the gravel removal, all other mining will be done with 9 cu yd shovels and 75 ton trucks. Larger equipment was analyzed, but due to the flexibility offered by the smaller equipment the latter were chosen. A 10 cu yd front-end loader is also scheduled to offer even more flexibility and mobility when supplementing the shovels.

It is planned to transfer three 1800 P & H shovels from the Mission Unit to Sacaton and replace them with two 15 cu yd shovels. The Sacaton Unit will then reimburse the Mission Unit with a fair market price for the used shovels. Sacaton will bear the expense of moving them. It is planned that all other Sacaton mining equipment will be new. (See Attachment A for a report on the feasibility of the shovel transfer.)

Underground Mining

Further preliminary development of the east underground ore body will be necessary. This involves sinking one of the two required shafts to a depth of 1900 feet and subsequently drifting about 2000 feet to the west end of the ore body at a depth of 1700 feet. The drift would then be extended another 1000 feet longitudinally through the center of the ore body with six diamond drill stations uniformly spaced along the drift. Diamond drilling will then be done radially and at right angles to the drift to delineate the ore body. (See Attachment D.)

Bulk samples will also be gathered along the drift and from several raises for metallurgical testing. A winze will also be developed midway along the 1000 feet drift to intersect the basement fault. The estimated time for this preliminary development is 1.8 years. The remaining development is estimated to be 2.2 years for a total underground pre-production period of 4 years.

The underground operation is based on block caving methods. Pre-production will begin during the seventh year of open pit production and continue for three more years. At the end of the four year period underground production will dovetail with open pit production for about six months, and thereafter the underground ore will take over total production for five years. This concludes the life of the property.

Milling

Samples were taken from large 6" diameter diamond drill holes in the open pit area to establish the average concentrate grade and

(Introduction - Cont'd)

expected recovery for the several types of ore. This 6" core constituted the metallurgical bulk sample of the ore body.

The crushing plant is designed with a primary jaw crusher having an average rated capacity of 632 TPH. The secondary and tertiary crushing is designed for an average crushing rate of 475 TPH. A coarse ore storage is designed into the plant having a live storage of 4750 tons and a total capacity of 23,750 tons. The concentrator portion of the plant is designed for 9000 tons per day.

The initial tailing dam stripping and filling costs, as well as the plant site stripping and filling costs, are included in the pre-mine stripping costs.

Plant and Facilities

A railroad track will be constructed to the plant from the main track 2-1/4 miles to the south. A plant access road will be constructed parallel to the track.

Water will be pumped four miles from a well farm that lies east of the property in Section 28 of T-5S, R-6E. A collecting tank will be elevated adjacent to the plant.

Electricity and natural gas sources are available near the plant.

SUMMARY OF ESSENTIAL DATA

		PIT	UNDERGROUND	AVERAGES & TOTALS
1.	Ore Reserves (tons)	33,027,000	14,558,000	47,585,000
2.	Ore Grade (%Cu)	0.76	1.37	0.95
3.	Cutoff Grade (%Cu)	0.30	0.50	0.36
4.	Life of Operation (years)	10.37	4.57	14.94
5.	Mill Recovery (%) (Average)	86.1	90.00	87.3
6.	Concentrate Grade (%Cu)	30.0	30.0	30.0
7.	Ratio of Concentration	46:1	24:1	36:1
8.	Pounds of Cu Paid for Per Ton Crude Ore (Average)	12.8	24.0	16.25

9. Cu Concentrate Production

	PIT	UNDERGROUND	Tons Conct	Payable Cu Tons	Payable Cu Pounds
Average Tons Per Day	-	-	250	73	146,000
Average Tons Per Month	-	-	7,376	2,157	4,314,000
Average Tons Per Year	-	-	88,500	25,886	51,772,000
Total Tons			1,322,300	386,773	773,546,000

		PIT	UNDERGROUND	AVERAGES & TOTALS
10.	Waste Tonnage (Total Tons)			
	Gravel	27,076,500		
	Rock	131,222,500		
	Total	158,299,000	-	158,299,000
11.	Overall Stripping Ratio	4.79:1	-	-
12.	Pre-Mine Stripping	44,787,000		
13.	Remaining Stripping Ratio after Pre-Mine	3.43:1		
14.	Cu Price Used in Outcome		50¢ @ Smelter	
15.	Net Smelter Return/Ton Cu Concentrate (Average)	-	-	$229.69
16.	Rate of Return on Investment			15.2%
17.	Tons of Ore Treated			
	Per Day	-	-	9,000
	Per Month	-	-	265,500
	Per Year	-	-	3,186,000
	Total (15 Years)			47,585,000

SACATON PROJECT
CAPITAL COST SUMMARY
9,000 TONS PER DAY

ITEM	DESCRIPTION	ESTIMATED COST	TOTALS
1.	MILL AND SURFACE PLANT		
	A. G. McKee Direct Material	$ 3,435,340	
	" Labor	1,552,900	
	" Subcontract	1,573,070	
	" Indirect Costs	3,259,550	
	" Escalation	1,175,140	
	" Contingencies	660,000	
	" Fee @ 3.4%	500,000	
	" TOTAL	$ 12,156,000	
	ASARCO Direct Labor & Materials	270,370	
	" Indirect Costs	152,630	
	" Contingencies	42,000	
	" Contractor's Fees	16,000	
	" TOTAL	$ 481,000	
	ASARCO Machinery & Equipment	3,197,807	
	" Escalation @ 6.5%	207,193	
	" TOTAL	$ 3,405,000	
	TOTAL MILL AND SURFACE PLANT		$ 16,042,000
2.	WATER SUPPLY		
	Pipeline, Main and Gathering	$ 298,000	
	Booster Pumps, Electrical	57,000	
	Sand Tank	22,000	
	Wells, Pumps, Electrical, Control	138,000	
	Engineering (ASARCO)	56,000	
	Contingencies, Contractor's Fees & Overhead	196,000	
	TOTAL		767,000
3.	PIT ELECTRICAL		
	Pit Power Line	$ 43,000	
	Substation; 1500 KVA, Breaker, Etc.	25,000	
	Cable Switch Houses	29,000	
	Shovel Cable and Lot Plugs	41,000	
	Engineering (ASARCO)	8,000	
	Contingencies, Contractor's Fees & Overhead	51,000	
	TOTAL		197,000

CAPITAL COST SUMMARY (Cont'd)

ITEM	DESCRIPTION	ESTIMATED COSTS	TOTALS
7.	MINE EQUIPMENT		
	Electric Shovels (3 from Mission)	$ 550,000	
	75-Ton Trucks (20)	3,073,000	
	9" Rotary Drills (3)	721,500	
	Front-end Loader (10 cu yd)	165,000	
	Wheel Dozer (2)	180,300	
	Track Dozer (2 D-8s)	189,500	
	Truck Crane (1 4DT)	80,000	
	Secondary Drill	55,000	
	Road Grader (2 No. 16)	164,100	
	Highway Tractor & Low Bed Trailer	50,000	
	6,000 Gal. Water Trucks (2)	117,000	
	Prill Truck	30,000	
	Powder Truck	5,000	
	Pit Service Truck	4,000	
	Shovel Repair Truck	4,000	
	Welding Truck	9,000	
	Pit Man Truck	4,000	
	Portable Air Compressor (2-125 CFM)	11,000	
	Portable Light Plants (4)	20,000	
	Fork Lift Truck (12,000 lb)	25,600	
	Lub. Truck	27,000	
	Pit Pipeline	12,000	
	Powder Magazine	20,000	
	Cap Magazine	2,000	
	Prill Storage	49,000	
	Radio System	12,000	
	Contingencies	556,000	
	TOTAL		$ 6,136,000
8.	PRELIMINARY STRIPPING & SITE PREPARATION		11,582,000
9.	ARIZONA SALES & USE TAX @ 3%		383,000
	TOTAL CAPITAL COST		$ 36,868,000
	MINUS STEARNS-ROGER'S FEASIBILITY COSTS (Money spent to date, $131,000)		- 131,000
			$ 36,737,000

TABULATION OF OTHER EXPENDITURES OCCURRING DURING PRODUCTION

YEAR			
5	Five additional pit trucks		$ 768,000 ✓
7	Preliminary Underground Development		1,435,000
8	Preliminary Underground Development	$ 1,435,500	
	Underground Surface Plant	1,358,000	
	Development Equipment	1,171,000	
	Underground Development	3,604,700	
	YEAR 8 SUB-TOTAL		7,569,200
9	Underground Development		3,604,700
10	Underground Development	3,604,800	
	Production Equipment	1,950,600	
	YEAR 10 SUB-TOTAL		5,555,400
11	Underground Development	323,450	
12	Underground Development	323,450	
13	Underground Development	323,450	
14	Underground Development	323,450	
	YEAR 11-14 SUB-TOTAL		1,293,800
GRAND TOTAL			$ 20,226,100

SACATON PROJECT

SUMMARY OF ESTIMATED OUTCOME

9000 TONS PER DAY	OPEN PIT HEAD ASSAY .76% cu PER TON ORE	UNDERGROUND HEAD ASSAY 1.37% cu PER TON ORE	AVERAGE HEAD ASSAY .95% cu PER TON ORE
OPERATING COSTS			
Milling Ore	$ .75 ✓	$ .75 ✓	$ .75 ✓
Mining ore	.25 ✓	1.94 ✓	.77 ✓
Stripping waste	.85 ✓	-	.59 ✓
TOTAL DIRECT COST	$ 1.85 ✓	$ 2.69 ✓	$ 2.11 ✓
INDIRECT COSTS	.84 ✓	.84 ✓	.84 ✓
TOTAL OPERATING COSTS (Excluding production capital)	$ 2.69 ✓	$ 3.53 ✓	$ 2.95 ✓
NET SMELTER VALUE (50¢ cu)	5.02	9.46	6.38
OPERATING PROFIT PER TON	$ 2.33	$ 5.93	$ 3.43
OPERATING PROFIT PER YEAR	$7,423,000	$18,893,000	$10,928,000
CAPITAL SPENT DURING PRODUCTION	$ .02	$ 1.34	$ .42
OPERATING PROFIT PER YEAR AFTER PRODUCTION CAPITAL	$7,360,000	$14,624,000	$ 9,590,000
POUNDS COPPER PAID FOR PER TON CRUDE ORE	12.8	24.0	16.2
COST PER POUND OF PAYABLE COPPER			
OPERATING COST	$ .214	$ .141	$ .181
SMELTING & FREIGHT	.107	.107	.107
TOTAL OPERATING COST	$ .321	$ .248	$ .288

SACATON ESTIMATE

SUMMARY OF DIRECT OPERATING COSTS

FIRST YEAR OF PREMINE

Total tons = 7,546,800 (Rock only)

	Wages & Salaries	Cost Per Ton	Supplies	Cost Per Ton	Other	Cost Per Ton	Total	Cost Per Ton
Drilling								
Operation	73,365	.0097	43,771	.0058			117,136	.0155
Maintenance	17,704	.0023	24,150	.0032			41,854	.0055
Total	91,069	.0120	67,921	.0090			158,990	.0210
Blasting								
Operation	20,181	.0027	107,165	.0142			127,346	.0169
Maintenance			755	.0001			755	.0001
Total	20,181	.0027	107,920	.0143			128,101	.0170
Loading								
Operation	118,886	.0158	54,337	.0072	13,584	.0018	186,807	.0247
Maintenance	66,255	.0088	128,296	.0170			194,551	.0258
Total	185,141	.0246	182,633	.0242	13,584	.0018	381,358	.0505
Hauling								
Operation	205,576	.0272	156,973	.0208	6,792	.0009	369,341	.0489
Maintenance	153,909	.0204	146,408	.0194			300,317	.0398
Total	359,485	.0476	303,381	.0402	6,792	.0009	669,658	.0887
Roads & Dumps								
Operation	38,403	.0051	46,035	.0061			84,438	.0112
Maintenance	38,252	.0051					38,252	.0051
Total	76,655	.0102	46,035	.0061			122,690	.0163
Pit Auxiliary Services	8,301	.0011	10,566	.0014	3,773	.0005	22,640	.0030
Eng. Sampling & Assaying	--	--	--	--	--	--	--	--
Supervision	61,884	.0082					61,884	.0082
GRAND TOTAL	802,716	.1064	718,456	.0952	24,149	.0032	1,545,321	.2048

SACATON ESTIMATE

SUMMARY OF DIRECT OPERATING COSTS

10 YEAR PRODUCTION AVERAGE

Total Tons = 140,616,800

		Wages & Salaries	Cost Per Ton	Supplies	Cost Per Ton	Other	Cost Per Ton	Total	Cost Per Ton
Drilling									
Operation		1,219,678	.0087	815,577	.0058			2,035,255	.0145
Maintenance		314,271	.0022	449,974	.0032			764,245	.0054
	Total	1,533,949	.0109	1,265,551	.0090			2,799,500	.0199
Blasting									
Operation		452,688	.0032	1,996,759	.0142			2,449,447	.0174
Maintenance				14,062	.0001			14,062	.0001
	Total	452,688	.0032	2,010,821	.0143			2,463,509	.0175
Loading									
Operation		2,067,973	.0147	1,012,441	.0072	253,110	.0018	3,333,524	.0237
Maintenance		1,457,663	.0104	2,390,486	.0170			3,848,149	.0274
	Total	3,525,636	.0251	3,402,927	.0242	253,110	.0018	7,181,673	.0511
Hauling									
Operation		5,943,372	.0423	4,921,588	.0350	126,555	.0009	10,991,515	.0782
Maintenance		3,512,285	.0250	3,585,728	.0255			7,098,013	.0505
	Total	9,455,657	.0673	8,507,316	.0605	126,555	.0009	18,089,528	.1287
Roads & Dumps									
Operation		844,995	.0060	857,762	.0061			1,702,757	.0121
Maintenance		887,051	.0063					887,051	.0063
	Total	1,732,046	.0123	857,762	.0061			2,589,808	.0184
Pit Auxiliary Services		154,678	.0011	196,864	.0014	70,308	.0005	421,850	.0030
Eng. Sampling & Assaying		449,974	.0032	28,123	.0002	--	--	478,097	.0034
Supervision		1,153,058	.0082	--	--	--	--	1,153,058	.0082
GRAND TOTAL		18,457,686	.1313	16,269,364	.1157	449,973	.0032	35,177,023	.2502

SACATON PROJECT

UNDERGROUND MINING COST ESTIMATE - TONS OF ORE 14,558,000

	TOTAL COST	COST PER TON
Hoisting	$ 1,429,300	$ .098
Drawing	2,683,600	.184
Hauling (Underground & on Surface)	4,185,900	.287
Maintenance of Extraction Openings	1,735,600	.119
Ventilation and Dust Control	320,900	.022
Drainage	962,600	.066
Mine Overhead	3,004,500	.206
Handling Men and Supplies	1,633,500	.112
General Underground	1,093,900	.075
Mine Surface	1,006,400	.069
Total Direct Costs	18,056,200	1.238
Indirect Costs	7,133,400	.490
Panel Preparation (during production)	10,190,600	.700
TOTAL UNDERGROUND MINING COSTS	$35,380,200	$ 2.428

SACATON PROJECT
ESTIMATED MILLING COSTS

The Sacaton direct milling costs have been re-estimated using a revised manning table and current labor contracts. The estimated direct milling cost is $0.7378 per ton and is broken down as follows:

Item	Cost/Ton
Supervision	$ 0.0378
Operating Labor	0.1246
Maintenance Labor	0.0667
Operating Supplies	0.2073
Maintenance Supplies	0.1139
Power	0.1870
Automotive	0.0005
Total Direct Cost =	0.7378 ✓

The above cost is based on the following considerations:

1.) Southwestern Mining Department (Copper) labor rates for period 7-1-73 to 6-30-74 are used.

2.) Monthly salaries are estimated to be 15 percent above 1970 rate.

3.) Operating and maintenance supply costs are based on Silver Bell Unit average costs for period January, 1967 thru October, 1971. Reagent costs are calculated separately.

4.) Power costs are estimated from grinding test data and assume a cost of 1.1 cents per KWH.

5.) Fresh water cost has not been included. This amounts to 1.7¢ per ton ore.

SACATON PROJECT
INDIRECT COSTS PER TON ORE

ITEM	MINE	MILL	TOTAL
General Expense	$ .05	$ .04	$ .09
Vacation Allowance	.02	.01	.03
Holiday Allowances	.02	.01	.03
Pension Accruals	.02	.01	.03
Taxes	.27	.19	.46
Insurance	.04	.02	.06
Salaries	.03	.02	.05
Misc. Maintenance	.02	.01	.03
Receiving and Buying	-	-	-
Shipping and Selling	-	.02	.02
Safety and Welfare	.01	.01	.02
Medical and Surgical	.01	.01	.02
	$ .49 ✓	$ .35 ✓	$ .84 ✓

PREMINE PERIOD .06/TON MATERIAL ✓

SACATON ESTIMATE

OPEN PIT

SUMMARY

The following basic data were used as principal controls and limits under which the estimate was worked up:

(a) 40 foot benches

(b) Final slope 45° (1:1) in rock and conglomerate
Final slope 39° (1.25:1) in gravel

(c) Ore cutoff grade .3% cu

(d) Mine operating 7 days per week, 6 shovel shifts per day, 357 days per year

(e) 9000 tons per day milled, 3,186,000 tons per year, operating 354 days per year.

(f) Pre-Mine stripping to last 2 years

(g) Open pit life - 10.4 years

Calculations based on the above conditions indicated the need for the following major items of equipment:

(a) 3 - 10-cu yd shovels

(b) 3 - 9-inch rotary drills

(c) 2 - D-8 dozers

(d) 1 - 10-cu yd front-end loader

(e) 20- 75-ton trucks*

(f) 2 - rubber tired dozers

(g) 2 - road graders

Estimated costs are:

Premine stripping	$ 11,582,000 ✓
Mine equipment	6,136,000 ✓
TOTAL	$ 17,718,000 ✓

At year 5 of the open pit operation an additional 5 pit trucks will be required, for a total expenditure of $768,000. This will be charged as negative income for that year. ✓

*15-110 ton trucks would also satisfy the haulage demand and at an average savings after depreciation of about $.01/ton material. At Year 4 an additional four trucks would be required.

chk ~~#~~
5 interspaced holes

2/28/84

Harold

This is the data on the 6" core vs NX core comparison.

There were 5 interspaced 6" holes drilled.

The shaft was shut-down in Sept 1981.

RBC

S-6M	1-6	304.5 - 411.0	=	106.5'	@	.989
	7-12	411.0 - 514.0	=	103.0'	@	.460
S-10M			=	183.5	@	.49
S-13M	1-7	434.0 - 560	=	126.0	@	.835
	8-12	560.0 - 669	=	109.0	@	.744
S-103M	1-5	450.8 - ~~544.0~~ 544.0	=	93.2	@	1.33
	6-9	544.0 - ~~[illegible]~~ 635		91.0'	@	.411
S-95M				198.7'	@	.942
S-90M				99.0'	@	.85
S-91M	1-9	280 - 450	=	170'	@	1.12
	10-18	450 - 625	=	175'	@	.52
S-93M			=	175'	@	.83
S-94M	1-6	375 - 491	=	116'	@	.80
	~~6~~-13	491 - 619	=	128	@	.56
S-111M	1-5	308 - 400	=	92	@	.97
	6-13	400 - 559.5	=	159.5	@	.42
				2125.4 ✓		.757

1608.9719

2125.4'

6" core	NX	Diff
.757	1.009	.252

6" Core is 25.0% lower than NX

TABLE NUMBER 1

SACATON PROJECT
6 inch Drill Core Samples

DDH	Barrel No.	DEPTH From	DEPTH To	DEPTH Ft	6" CORE ASSAYS PERCENT Cu	6" CORE ASSAYS PERCENT NSCu(1)	6" CORE ASSAYS PERCENT Mo	D.D. CORE ASSAYS PERCENT Cu.	D.D. CORE ASSAYS PERCENT NSCu
S-6M	1	304.5	323.1	18.6	0.31	0.24	0.008	1.95	0.22
	2	323.1	336.9	13.8	0.15	0.10	0.005	0.52	0.14
	3	336.9	354.9	18.0	0.40	0.36	0.008	0.63	0.55
	4	354.9	371.9	17.0	0.12	0.04	0.008	1.17	0.25
	5	371.9	391.9	20.0	1.54	0.81	0.013	0.42	0.39
OK	6	391.9	411.0	19.1	3.02	0.65	0.016	1.19	0.27
	7	411.0	426.9	15.9	0.49	0.21	0.014	0.80	0.13
	8	426.9	445.9	19.0	0.52	0.19	0.008	0.36	0.03
	9	445.9	465.9	20.0	0.52	0.15	0.019	0.37	0.01
	10	465.9	486.0	20.1	0.49	0.10	0.010	0.40	0.01
	11	486.0	504.5	18.5	0.35	0.07	0.008	0.43	0.01
	12	504.5	514.0	9.5	0.34	0.08	0.017	0.35	TR.
S-10M	1	335.0	358.0	23.0	0.98	0.12	0.009	0.41	N.A.
	2	358.0	376.0	18.0	0.53	0.11	0.007	2.83	''
	3	376.0	396.0	20.0	0.34	0.14	0.007	1.83	''
	4	396.0	416.0	20.0	0.61	0.19	0.006	1.02	''
OK	5	416.0	435.5	19.5	0.65	0.18	0.006	0.63	''
	6	435.5	457.0	21.5	0.28	0.03	0.005	2.06	''
	7	457.0	481.0	24.0	0.35	0.16	0.008	1.41	''
	8	481.0	500.0	19.0	0.12	0.07	0.008	0.53	''
	9	500.0	518.5	18.5	0.62	0.34	0.008	0.52	''
S-13M	1	434.0	454.0	20.0	0.31	0.25	0.024	0.49	N.A
	2	454.0	473.0	19.0	0.14	0.07	0.009	0.91	N.A
	3	473.0	493.0	20.0	0.92	0.92	0.010	0.94	N.A.
	4	493.0	512.0	19.0	0.49	0.23	0.008	0.63	0.22
	5	512.0	521.0	9.0	3.85	2.60	0.007	0.54	0.24
	6	521.0	538.5	17.5	1.14	0.64	0.008	3.75	0.12
OK	7	538.5	560.0	21.5	0.66	0.30	0.010	1.63	N.A.
	8	560.0	582.4	22.4	0.61	0.13	0.008	0.50	''
	9	582.4	605.0	22.6	0.77	0.12	0.007	0.78	''
	10	605.0	625.0	20.0	0.76	0.08	0.008	0.65	''
	11	625.0	650.0	25.0	0.83	0.08	0.008	0.74	''
	12	650.0	669.0	19.0	0.76	0.05	0.008	0.57	''
S-90M	1	409.0	429.0	20.0	1.70	0.66	0.006	1.59	0.45
	2	429.0	449.0	20.0	1.08	0.56	0.006	1.43	0.43
	3	449.0	469.0	20.0	0.50	0.22	0.008	1.24	0.47
OK	4	469.0	489.0	20.0	0.36	0.17	0.006	0.47	0.17
	5	489.0	508.0	19.0	0.64	0.41	0.007	0.49	0.15

(1) Non-Sulphide Copper assays performed by hot sulphuric acid method

7.

TABLE NUMBER 1 (Cont'd)

SACATON PROJECT
6 inch Drill Core Samples

DDH	Barrel No.	DEPTH From	DEPTH To	DEPTH Ft	ASSAYS PERCENT Cu	ASSAYS PERCENT NSCu(1)	ASSAYS PERCENT Mo	D.D. CORE ASSAYS TCu.	D.D. CORE ASSAYS N.S. Cu
S-91M	1	280.0	300.0	20.0	0.92	0.27	0.017	2.01	0.18
	2	300.0	317.0	17.0	2.06	0.32	0.012	2.11	0.15
	3	317.0	337.0	20.0	1.09	0.13	0.014	1.56	0.22
	4	337.0	355.0	18.0	1.06	0.22	0.009	2.08	0.42
	5	355.0	375.0	20.0	1.07	0.62	0.023	1.42	0.20
	6	375.0	390.0	15.0	0.98	0.48	0.008	1.41	0.23
	7	390.0	410.0	20.0	0.97	0.24	0.007	0.98	0.05
	8	410.0	430.0	20.0	0.83	0.37	0.007	1.30	0.06
	9	430.0	450.0	20.0	1.20	0.34	0.006	1.31	0.06
	10	450.0	470.0	20.0	0.65	0.09	0.007	0.58	0.04
	11	470.0	490.0	20.0	0.40	0.10	0.014	0.46	0.05
	12	490.0	510.0	20.0	0.43	0.18	0.010	0.40	0.06
	13	510.0	530.0	20.0	0.40	0.13	0.007	0.40	0.05
	14	530.0	550.0	20.0	0.46	0.04	0.011	0.48	0.05
	15	550.0	570.0	20.0	0.54	0.04	0.008	0.41	0.04
	16	570.0	590.0	20.0	0.44	0.02	0.009	0.43	0.03
	17	590.0	610.0	20.0	0.53	0.05	0.007	0.82	0.04
	18	610.0	625.0	15.0	1.17	0.05	0.008	1.48	0.03
S-93M	1	283.0	303.0	20.0	0.26	0.18	0.007	1.00	0.66
	2	303.0	315.0	12.0	0.70	0.20	0.009	1.62	0.21
	3	315.0	335.0	20.0	1.12	0.17	0.008	1.50	0.04
	4	335.0	355.0	20.0	1.51	0.19	0.011	1.39	[illegible]
	5	355.0	375.0	20.0	1.77	0.26	0.010	1.63	0.07
	6	375.0	395.0	20.0	0.80	0.18	0.010	0.73	0.15
	7	395.0	415.0	20.0	0.55	0.07	0.008	0.53	[illegible]
	8	415.0	437.0	22.0	0.43	0.05	0.008	0.54	0.06
	9	437.0	458.0	21.0	0.43	0.05	0.007	0.45	0.02
S-94M	1	375.0	395.0	20.0	0.10	0.05	0.011	1.69	0.42
	2	395.0	415.0	20.0	0.55	0.27	0.010	2.59	0.28
	3	415.0	434.5	19.5	1.63	0.60	0.007	1.71	0.20
	4	434.5	454.5	20.0	1.20	0.31	0.008	0.91	0.13
	5	454.5	471.0	16.5	0.83	0.37	0.008	0.61	0.08
	6	471.0	491.0	20.0	0.58	0.26	0.008	0.58	0.05
	7	491.0	510.0	19.0	0.47	0.12	0.010	0.86	0.05
	8	510.0	530.0	20.0	0.46	0.10	0.008	0.86	0.05
	9	530.0	550.0	20.0	0.59	0.10	0.008	0.93	0.06
	10	550.0	570.0	20.0	0.56	0.03	0.008	0.85	0.05
	11	570.0	590.0	20.0	0.58	0.07	0.011	0.80	0.05
	12	590.0	610.0	20.0	0.77	0.07	0.011	0.52	0.04
	13	610.0	619.0	9.0	0.46	0.04	0.009	0.60	0.04

(1) Non Sulphide Copper assays performed by hot sulphuric acid method

TABLE NUMBER 1 (Cont'd)

SACATON PROJECT
6 inch Drill Core Samples

DDH	Barrel No.	DEPTH From	DEPTH To	DEPTH Ft	ASSAYS PERCENT Cu	ASSAYS PERCENT NSCu(1)	ASSAYS PERCENT Mo	D.D. CORE ASSAYS TCu	D.D. CORE ASSAYS N.S. Cu
S-95M	1	380.0	405.0	25.0	2.35	0.27	0.017	2.97	0.16
	2	405.0	429.0	24.0	0.69	0.27	0.018	1.04	0.04
	3	429.0	450.0	21.0	0.62	0.62	0.015	1.12	0.02
	4	450.0	474.0	24.0	0.77	0.06	0.009	0.77	0.01
	5	474.0	498.0	24.0	0.71	0.10	0.009	0.77	TR.
	6	498.0	523.0	25.0	0.77	0.11	0.009	0.93	TR.
	7	523.0	548.7	25.7	0.78	0.06	0.005	0.77	TR.
	8	548.7	578.7	30.0	0.80	0.05	0.007	0.78	0.01
S-103M	1	450.8	470.9	20.1	0.49	0.44	0.006	0.43	0.35
	2	470.9	490.0	19.1	0.66	0.57	0.008	0.79	0.30
	3	490.0	504.0	14.0	0.32	0.19	0.008	0.41	0.14
	4	504.0	524.0	20.0	4.46	4.36	0.007	0.59	0.03
	5	524.0	544.0	20.0	0.40	0.22	0.008	0.57	0.02
	6	544.0	564.0	20.0	0.52	0.17	0.034	0.62	0.06
	7	564.0	590.0	26.0	0.33	0.10	0.017	0.54	0.02
	8	590.0	610.0	20.0	0.35	0.39?	0.014	0.42	0.01
	9	610.0	635.0	25.0	0.46	0.28	0.013	0.52	0.01
S-111M	11.572 1 [illegible]	308.0	327.0	19.0	0.37	0.20	0.011	5.52	0.61
	39.[illegible] 2 66.477	327.0	350.0	23.0	1.48	0.92	0.007	2.89	1.73
	2.[illegible] 3 9.119	350.0	360.0	10.0	2.44	0.61	0.008	0.91	0.26
	.915 4 14.46	360.0	380.0	20.0	0.71	0.23	0.009	0.72	0.05
	.879 5 16.579	380.0	400.0	20.0	0.49	0.21	0.010	0.84	0.04
	.372 6 10.588	400.0	420.0	20.0	0.65	0.17	0.007	0.53	0.02
	.383 7 12.31	420.0	440.0	20.0	0.21	0.04	0.006	0.62	0.02
	.533 8 11.031	440.0	460.0	20.0	0.43	0.05	0.009	0.55	0.03
	.372 9 10.604	460.0	480.0	20.0	0.50	0.05	0.007	0.53	0.02
	.734 10 9.363	480.0	500.0	20.0	0.61	0.08	0.009	0.47	0.04
	.884 11 10.094	500.0	520.0	20.0	0.37	0.04	0.010	0.50	0.04
	.716 12 9.409	520.0	540.0	20.0	0.43	0.02	0.011	0.47	0.04
	.413 13 9.462	540.0	559.5	19.5	0.34	0.04	0.010	0.49	0.02

(1) Non Sulphide Copper assays performed by hot sulphuric acid method

Reserves by Benches — COMPARED TO MINING THROUGH FEB. '84
(Tons X Thousand)

nch	ORIG. Tons	MINED	ORIG. Grade	MINED	WASTE MINED	ORIG. Waste	Total Material		ORIGINAL LBS. Cu x1000	MINED LBS Cu x1000	*RATIO RECOVERED CU	BENCH
430					6,799.5	6,749.5	6,749.5	6,799.5				
390					14,109.7	13,909.4	13,909.4	14,109.7				
350					14,071.1	15,151.3	15,151.3	14,071.1				
310		26.9		.36	15,989.3	16,320.9	16,320.9	16,016.2	194	—		1310
270	168.0	36.3	0.86	.40	15,726.7	15,033.0	15,201.0	15,763.0	2890	290	0.12	1270
230	216.0	361.3	0.46	.38	14,684.5	13,876.3	14,092.3	15,045.8	1987	2746	1.6	1230
190		762.3		.49	12,993.3	13,041.6	13,041.6	13,755.6		7471		1190
150	592.0	1,334.7	0.64	.55	11,064.2	11,454.1	12,046.1	12,398.9	7578	14,682	2.3	1150
110	1,856.0	1,912.8	1.26	.71	9,657.6	9,218.6	11,074.6	11,570.4	46,771	27,162	0.67	1110
070	2,080.0	2,040.8	0.89	.73	8,845.4	8,050.6	10,130.6	10,886.2	37,024	29,796	0.93	1070
030	2,098.8	2,226.3	0.97	.82	7,813.5	7,133.5	9,232.3	10,039.8	40,717	36,511	1.0	1030
990	2,537.6	2,845.0	0.80	.74	6,023.9	5,829.7	8,367.3	8,868.9	40,602	42,106	1.2	990
950	2,909.2	3,093.6	0.61	.69	4,927.8	4,637.4	7,546.6	8,021.4	35,492	42,692	1.4	950
910	2,943.3	3,090.5	0.59	.71	3,971.1	3,784.1	6,727.4	7,061.6	34,731	43,885	1.5	910
870	2,350.1	3,184.0	0.81	.67	3,086.2	3,637.8	5,987.9	6,270.2	38,072	42,666	1.3	870
830	2,549.8	2,743.0	0.75	.64	2,826.4	2,737.0	5,286.8	5,569.4	38,247	35,110	1.1	830
790	2,285.6	2,496.9	0.73	.62	2,333.2	2,295.0	4,580.6	4,830.1	33,370	30,962	1.1	790
750	2,552.0	2,545.3	0.66	.56	1,720.1	1,375.2	3,927.2	4,265.4	33,686	28,507	.98	750
710	1,980.0	2,077.4	0.51	.55	1,476.5	1,345.2	3,325.2	3,553.9	20,196	22,851	1.3	710
670	1,570.3	1,585.7	0.59	.58	1,342.3	1,180.7	2,751.0	2,928.4	18,530	18,394	1.15	670
630	1,700.6	1,448.5	0.86	.63	881.3	536.2	2,236.8	2,369.8	29,250	18,251	.72	630
590	1,232.2	1,291.9	0.58	.62	593.8	519.1	1,751.3	1,885.7	14,294	16,020	1.3	590
550	802.2	1,248.7	0.85	.59	244.5	307.7	1,109.9	1,493.2				
510	341.0	969.5	1.71	.57	152.1	162.5	503.5	1121.6	470,526	455,456	1.12	THROUGH 590
470	262.4	706.2	1.16	.56	112.1	12.9	275.3	818.3				
		409.1		.44	0.3			409.4				
		43.0		.37	0			43.0				
tals & erages	33,027.0	38,519.7	0.76	.64	161,446.8	158,299.0	191,326.0	199966.5	502,010	493,052	1.14	THROUGH FEB.'84
[illegible] 590	31,621.4	35,143.2	0.74	0.65	160,937.8	157,815.9	189,437.3	196,081.0				

The overall stripping ratio is 4.79 : 1 — ORIG. RESERVE
4.19 : 1 — FEB.'84. 4.58:1 — 590

* RATIO RECOVERED CU = $\frac{\text{MINED Cu}}{0.861 \times \text{ORIG. Cu}}$

Waste remaining after premine stripping:
158,299.0 - 44,787.800 = 113,511.200

Ore remaining after premine stripping:
33,027.0 - 384.0 = 32,643.0

Remaining stripping ratio = 3.48 : 1

Harold,

Expected vs
Produced Ore
@ Sacaton
Note grade Differences
on 1110 - 1030 Benches
RBC

TABLE 3
EXPECTED ORE VS. PRODUCED ORE BY BENCH

Bench	ORE EXPECTED			ORE PRODUCED			% CHANGE FROM EXPECTED		
	M-Tons Ore	Cu Pct	Tons Cu	M-Tons Ore	Cu Pct	Tons Cu	M-Tons Ore	Cu Pct	Tons Cu
1310	0	-	0	26.9	.36	96.8			
1270	160.0	.91	1,456.0	36.3	.40	145.2	- 32.8%	- 25.6%	- 50.0%
1230	472.0	.376	1,776.0	361.3	.38	1,372.9			
1190	0	-	0	762.3	.49	3,735.3	+297.2%	- 24.1%	+201.4%
1150	528.0	.693	3,657.6	1,334.7	.55	7,289.4			
1110	1,624.0	1.343	21,806.4	1,912.8	.71	13,588.6			
1070	1,968.0	.902	17,572.0	2,070.5	.71	14,674.3	+ 11.6%	- 29.9%	- 21.9%
1030	2,224.0	.942	20,958.4	2,509.2	.76	19,008.5			
990	2,785.9	.759	21,144.5	2,619.0	.76	20,008.9			
950	2,195.0	.654	14,347.8	2,682.5	.72	19,193.7	+ 11.3%	+ 4.3%	+ 16.0%
910	1,982.2	.660	13,081.1	2,369.9	.75	17,739.9			
870	1,406.3	.720	10,129.2	1,642.1	.68	11,150.4			
830	867.4	.565	4,883.5	695.4	.62	4,278.5	- 19.8%	- 0.5%	- 20.3%
790	345.3	.694	2,397.3	276.7	.55	1,521.4			
Total	16,558.1	.806	133,389.8	19,299.6	.693	133,803.8	+ 16.6%	- 14.0%	+ 0.3%

NOTE: Totals are thru February, 1979 and based on mine records.

TABLE 2

TONS OF COPPER EXPECTED VS. ACTUAL TONS OF COPPER MINED BY BENCH

Bench	Tons Cu-Expected		Tons Cu - Actual		Difference-Bench		Cumulative Difference-Total		Expected Ore Grade
	Bench	Total	Bench	Total	Tons Cu	%	Tons Cu	%	
1310	0	0	96.8	96.8	+ 96.8	-	+ 96.8	-	-
1270	1,456.0	1,456.0	145.2	242.0	-1,310.8	-90.0	-1,214.0	-83.3	.91
1230	1,776.0	3,232.0	1,372.9	1,614.9	- 403.1	-22.7	-1,617.1	-50.0	.376
1190	0	3,232.0	3,735.3	5,350.2	+3,735.3	-	+2,118.2	+65.5	-
1150	3,657.6	6,889.6	7,289.4	12,639.6	+3,631.8	+99.3	+5,750.0	+83.5	.693
1110	21,806.4	28,696.0	13,588.6	26,228.2	-8,217.8	-37.7	-2,467.8	- 8.6	1.343
1070	17,752.0	46,448.0	14,674.3	40,902.5	-3,077.7	-17.3	-5,545.5	-11.9	.902
1030	20,958.4	67,406.4	19,008.5	59,911.0	-1,949.9	- 9.3	-7,495.4	-11.1	.942
990	21,144.5	88,550.9	20,008.9	79,919.9	-1,135.6	- 5.4	-8,631.0	- 9.7	.759
950	14,347.8	102,898.7	19,193.7	99,113.6	+4,845.9	+33.8	-3,785.1	- 3.7	.654
910	13,081.1	115,979.8	17,739.9	116,853.5	+4,658.8	+35.6	+ 873.7	+ 0.8	.660
870	10,129.2	126,109.0	11,150.4	128,003.9	+1,021.2	+10.1	+1,894.9	+ 1.5	.720
830	4,883.5	130,992.5	4,278.5	132,282.4	- 650.0	-12.4	+1,289.9	+ 1.0	.565
790	2,397.3	133,389.8	1,521.4	133,803.8	- 875.9	-36.5	+ 414.0	+ 0.3	.694

NOTE: Totals are thru February, 1979 and are based on mine records.

From: J. J. Collins

J. H. C.

To: Messrs. M. P. Barnes
W. L. Kurtz
S. A. Anzalone

NOV 26 1975

Mr. Snedden read this and did not object, only saying he would carry Barnes' report to the Tucson Office.

Mr. MacDonald read this memo and agreed with it.

J. J. Collins

RECEIVED
NOV 17 1975
S. W. U. S. EXPL. DIV.

cpy for:
JHC
RBC
FTG

New York, November 7, 1975

MEMORANDUM FOR: Mr. R. L. Hennebach

Sacaton - Ore

The Problem

Copper production has not equalled pre-mine estimates and questions have been raised as to what went wrong. Was the ore reserve calculation incorrect? Are mining operations contributing to the problem?

The Answer

Complete pre-mine drilling records have been recalculated by our Geologic Computer Group for the area that has been mined out prior to May, 1975, and the resulting average grade compares very closely with the average grade from all blast hole assays in the same area, i.e., some 3000 blast holes (both ore and waste) average 0.58% Cu and the reserve estimates by three different methods are 0.60%, 0.62% and 0.59% Cu. Thus, the overall reserve grade figures have been corroborated. Please refer to Table 3 in the attached report by Mr. Barnes entitled "Geostatistical Study."

What we do not know is the degree of sorting that has been practiced within the mined area. All the reserve maps of the ore zone reveal areas of internal waste and sub-profitable rock that should not have gone into the mill, but we do not know how much of this material was actually sent to the mill. All that we do know is that mill heads on a monthly basis, counting backward for a year from May, 1975 were: 0.82, 0.70, 0.76, 0.73, 0.64, 0.66, 0.59, 0.72, 0.66, 0.56, 0.67, 0.61, 0.61% Cu. Apparently waste was not sorted out during some months.

Explanation

The pre-mine ore reserve estimates cover a larger volume than has been mined to date hence they are not directly comparable to production data. Therefore, our Geologic Computer Group in Salt Lake City has reworked the portion of pre-mine drilling data that do coincide with the mined area. To simulate the pre-mine calculations both the polygonal and the Inverse of the Distance Squared methods were repeated, plus statistical analyses of the precision in each method. In addition, a new method, "Kriging" was applied.

All methods show a variance in the overall grade of ore of ±8% at the 95% confidence level. This is an acceptable accuracy for the overall grade of an ore reserve, but statistical analysis shows extreme variation in small volumes. A typical unit area measuring 50 x 50 x 40 feet shows a variance of ±41% for the blast hole assays.

Local variation is evident in the drill logs and is extreme at the irregular boundaries between the oxide zone, the chalcocite enrichment, and the primary sulphides. No reasonable amount of surface drilling can reveal all these irregularities. The only practical method, we think, is mine development with enough exposure of ore and waste prior to completion of a mill so that selectivity in mining can provide the desired grade to the mill. At Sacaton the mill was completed before the mine was fully developed; hence the choice of mill feed was limited and the grade apparently suffered.

<u>Recommendations</u>

Experience at Sacaton leads me to conclude that:

1. A geologist and the geologic computer group should be included in mine planning teams.

2. Two cut-off grades should be used in the mine. Most important is the grade that will generate the desired R.O.I. The second cut-off is the grade that will just pay back mine and mill operating costs. The rock between these two cut-offs should go to a <u>stockpile</u> to remain there until the profitable ore is gone.

Oxidation of Chalcocite would need to be ~~[illegible]~~ taken into account

3. Mine and mill production should be reported in pounds of contained copper to balance the too-human tendency simply to put "rock in the box." Sensitivity analysis shows that cost per ton of rock is less critical to profit than is the grade of ore milled, i.e. an extra pound of copper pays for stripping a ton of waste.

John J. Collins

Attachment

cc: TASnedden - w/att.
NVisnes "