CONTACT INFORMATION
Mining Records Curator
Arizona Geological Survey
416 W. Congress St., Suite 100
Tucson, Arizona 85701
602-771-1601
http://www.azgs.az.gov
inquiries@azgs.az.gov

(1)

U. S. COPPER CORPORATION

An Economic and Geologic Report

John C. Lawrence

December 5, 1967

Mackay, Idaho

CONTENTS

Page

Abstract. 1

Introduction 2

Geology 2

General. 2

Brazer Limestone 2

Tactite 2

Iron dike 3

Granite porphyry 3

Ore deposits 4

Pit assays 4

Summary of drilling program 7

Ore estimate of 14-hole area 9

Proposed drilling program 11

Production. 12

Operating costs & profit of vat leaching 14

Daily operating cost 14

Daily operating profit 14

Mining costs 15

Crushing costs 15

Plant & ore control costs 16

Miscellaneous personnel costs 16

Fixed overhead costs 16

A B S T R A C T

Out of the 20 holes drilled by New Idria, the average weighted assay for the 11 holes completed to 200 feet is 0.27% copper, and that for the 9 holes not completed to 200 feet is 0.49% copper. The lower grade zones have less clay and are more easily drilled. Ore estimates of tonnage and grade are based on the drill holes and are probably lower than they should be.

Approximately 5,000,000 tons of 0.33% copper were drilled in the program of 20 holes. Within a 14-hole area the probable ore is:

Category	Tons	Percentage Copper
Vat ore	1,180,000	0.85%
Blend ore	311,000	0.39%
Heap leach ore	1,084,000	0.17%
Waste	751,000	0.10%

By mixing the blend ore with vat ore, 1,491,000 tons of 0.75% copper are available. The stripping ratio of blended vat ore (0.75%) to heap leach ore and waste is 1:1.25.

The proposed drilling program has three purposes. First, the area already drilled should be drilled on 100-foot centers to at least 500 feet to categorize the reserves as proven. Second, the completed drilling program has explored less than one half the area of mineralization. Two completely unexplored areas are proposed for drilling. Third, the program should determine the depth of oxide mineralization.

An 80-85% recovery on 0.75% copper will yield 6,600 pounds of contained copper per day. The cost per pound of copper is $0.28, it is sold at 0.38%, and the profit margin is 35%.

INTRODUCTION

This report is a summary of the New Idria drilling program, and an evaluation of the economics of the U. S. Copper Corporation mine near Mackay, Idaho. Only new information is presented and for more detail the reader is referred to the U. S. Geological Survey open file report by F. W. Farewell and R. P. Full, of 1944.

GEOLOGY

General.

The mine is in the southern end of a crescent shaped contact metamorphic zone of tactite. The convex side of the crescent faces east and is bounded by the Brazer Limestone of Mississippi age. Silicified granite porphyry and some tactite are on the west within the concave side of the crescent. Granite porphyry is in dikes and in larger zones throughout the skarn and is directly in contact with the Brazer Limestone in several places. The higher grade copper mineralization is confined to the tactite, but much of the granite porphyry is mineralized.

Brazer Limestone.

Unaltered Brazer Limestone consists of dark-gray massive limestone with some thin sandstone lentils. The unit is abundantly fossiliferous in thin beds and contains crinoids and tetracorals. Near tactite or granite porphyry the limestone is altered to coarsely crystalline white marble.

Tactite.

Two distinct types of tactite are recognized. The first is a light- to medium-brown rock comprised of garnet (grossularite and andradite) and diopside. Magnetite is present as the matrix surrounding brecia fragments of tactite in pipe-like structures. It is also present in thin dikes and isolated pods. Much of the magnetite has oxidized to form hematite. Epidote, scheelite, actinolite and scapolite are present in minor amounts. Copper oxides are primarily blue, brown, black and dark-red varieties of chrysacolla; others include malachite, azurite and possibly neodosite. Sulfides are primarily chalcopyrite and bornite. Chalcocite is present in thin zones near the base of deeply leached zones at the surface, but it is not abundant.

The first type of tactite is generally overlain by 20 to 100 feet of deeply leached tactite extremely high in kaolinite. The leach zone is generally marked by green jarosite, red hematite, and brown limonite-stained clay and tactite. Several of the tactite zones appear leached to depths of over 200 feet. Most of the tactite drilled within the pit area contained 5 to 25% clay. However, drill hole NI 6 penetrated 55 feet of leached tactite cap, and bottomed in hard clay-free tactite. This was the only hole that penetrated unleached tactite, and the tactite was the highest grade drilled. Similar grade tactite in other areas can be expected only by deeper drilling.

A second type of tactite is a highly silicified form that may be silicified granite porphyry and not tactite. It is comprised of more than 85% quartz and is usually light gray and massive. The form is arbitrarily distinguished from silicified granite porphyry on the basis of its lack of feldspar grains. The silicified tactite is generally low in clay, and copper mineralization is as pervasive as it is in the first type of tactite.

Iron Dike

An iron dike crops out on the west side of the pit area. The trend of this dike does not coincide with the crescent shape of the main metamorphic zone. It trends north-northwest (see map) behind the crescent-shaped zone thus isolating a block of silicified granite porphyry on the concave side of the crescent.

The iron dike is magnetite and hematite. Copper in the iron is probably microscopic chalcopyrite; exposed surfaces of the iron show malachite straining, and some have extensive boxwork resulting from the leaching of sulfide minerals. The dike is not homogeneous, and it contains large inculsions of tactite and granite porphyry (see map).

Granite Porphyry

Granite porphyry is on the concave side of the crescent shaped skarn zone, within the skarn, and in many places on the outside of the skarn in contact with the Brazer Limestone. West of the mine area, granite porphyry grades into massive more resistant granite.

The granite porphyry is light gray and contains phenocrysts of plagioclase, orthoclase, and rounded quartz grains in a groundmass of feldspar and quartz. It shows various degrees of silicification and is generally low in clay. Minor amounts of chloritized biotite, hornblende, and magnetite are present.

Much of the granite porphyry has some degree of copper mineralization. Although the mineralization is usually of a lower grade than the tactite mineralization, it is more uniform. West of the crescent shaped tactite zone, the granite porphyry crops out over a large area. Silicified granite porphyry float that contains disseminated chalcopyrite and bornite is in this area, but only one small outcrop of this rock is known. Drill hole NI 20 penetrated 200 feet of essentially barren granite porphyry within the crescent, and the source of the mineralized float is problematic.

In the skarn zone mineralized granite porphyry was penetrated in drill holes NI 10, 11, 12, and 16. The copper mineralization varies from 0.02 to 0.51%; several zones averaged 0.20% copper. Much of the copper is present as fine-grained chalcopyrite and bornite with fine haloes of chrysacolla.

O R E D E P O S I T S

Pit Assays.

Several groups of assays are presented here to give the reader an idea of the grade and extent of mineralization in the pit area.

The Cleveland Cliffs Iron Ore Company of Cleveland, Ohio, core-drilled the iron dike, and the following assays are composites for the eight holes they drilled:

Drill Holes	Iron	Phosphorous	Silica	Sulfur	Copper
2.3	57.04%	0.076%	10.04%	0.018%	0.26%
4.5	60.12%	0.082%	6.14%	0.029%	0.15%
6.7	57.95%	0.060%	7.61%	0.006%	1.23%
8.	59.47%	0.052%	6.90%	0.012%	0.66%
9.	55.94%	0.077%	8.10%	0.010%	0.19%

John Ortman, the geologist in charge of the drilling, has told me that he estimated approximately 3,000,000 tons of ore that averaged close to 60% iron. A larger tonnage of lower grade ore is undoubtedly present.

The Joe Hill drift is on the southern end of the property, and it is open for approximately two hundred feet. The drift has two headings (see map) and represents a large area of unexplored country. Quaternary alluvium covers the area directly above the drift, and there is no evidence of mineralization at the surface.

J. R. Simplot Company personnel took 20- and 30-foot wall samples (see map) from the drift in September, 1966 and submitted them to the Union Assay Office, Inc. The samples were not from the entire length of the drifts, but they appear representative:

Sample	Length of Sample	Copper
3446	20'	1.058
3447	20'	0.415
3448	20'	0.768
3449	20'	1.707
3450	30'	1.058
	Total length sampled= 110'	Average weighted= 1.00%

The Bear Creek Mining Company sampled the full length of the Joe Hill drift in September, 1967, and their assay work was done by the Montana Laboratory Company of Philipsburg, Montana. Although I do not have the exact location or length of each sample, an unweighted average of 0.997% copper for their samples compares very closely with the weighted average of 1.00% copper obtained from the J. R. Simplot Company samples. The Bear Creek results for the Joe Hill drift are:

Sample	Total Copper	Au	Ag
92360	0.58	Trace	0.10
92361	1.04	Trace	0.65
92362	1.06	0.018	0.60
92363	1.23	Trace	0.55
92364	1.11	Trace	0.10
92365	0.70	Trace	0.30
92366	0.68	Trace	0.25
92367	0.94	Trace	0.05
92368	1.28	Trace	0.20
92369	1.35	Trace	0.70
	Average unweighted assay= 0.997%	Trace	Average unweighted assay= 0.345

J. R. Simplot Company personnel cut channel samples from the Davis tunnel (see map) and from dozer cuts in the pit area, and their samples were assayed by the Union Assay Office, Inc. The assays and weighted average from the Davis tunnel are as follows:

Sample	Length of Sample	Copper
3451	35'	0.396%
3452	40'	0.283
3453	40'	0.170
3454	35'	0.126
Total length	= 150'	0.24%= Average weighted assay

The assays and weighted average of the J. R. Simplot Company assays for the pit area are:

Sample	Sample Length	Copper
3401	75'	0.056
3402	20'	0.170
3403	40'	0.756
3404	70'	0.088
3405	100'	0.422

Sample(cont.)	Sample Length (cont.)	Copper (cont.)
3406	50'	0.189
3407	30'	1.026
3408	40'	1.423
3409	25'	0.044
3410	40'	0.037
3411	40'	0.038
3412	40'	0.390
3413	50'	1.209
3414	25'	1.285
3415	90'	0.113
3416	50'	0.264
3417	45'	0.107
3418	50'	0.182
3419	55'	0.560
3420	30'	0.497
3421	45'	1.253
3422	80'	0.762
3423	35'	0.289
3424	30'	0.075
3425	45'	0.737
3426	65'	0.957
3427	45'	0.737
3428	75'	0.283
3429	70'	0.088
3430	80'	0.289
3431	35'	0.239
3432	35'	0.044
3433	45'	0.793
3434	45'	0.913
3444	35'	0.245
3445	30'	1.769
3455	20'	0.610
3456	35'	1.134
3457	10'	0.010
Total =	1825'	Average weighted assay= 0.492% Copper

I have assayed the U. S. Copper cuttings from blast holes (BH) and exploration holes (EH and DH) drilled with an Ingersol-Rand drillmaster and the sludge samples from holes drilled with a Joy 12b diamond core drill (CH). These holes are located on the map and the assays are as follows:

Sample	Interval	Copper
BH1	0-20'	0.57%
BH2	0-20'	0.58
BH3	0-20'	0.85
BH4	0-20'	2.12
BH5	0-20'	0.44
BH6	0-20'	1.19
BH7	0-20'	0.68
BH8	0-20'	0.49
BH9	0-20'	0.38
BH10	0-20'	0.30
EH-1	0-20'	0.44
EH-2	0-20'	0.35
	240' Total	Average Weighted= 0.70%
CH-1b	10-15'	1.40%
CH-1b	15-20'	0.95
CH-1b	20-25'	0.58
CH-1b	25-30'	0.86
CH-1b	30-40 '	0.45
	30' Total	Average Weighted=0.78%
DH-2b	0-10'	0.75
DH-2b	10-20'	0.37
DH-2b	20-30'	0.62
DH-2b	30-40'	0.58
DH-2b	40-50'	0.67
	50' Total	Average Weighted=0.60%

Summary of Drilling Program.

The New Idria Mining and Chemical Company started drilling on September 6, 1967. A Chicago Pneumatic Reichdrill equipped with a Mission low-pressure down-the-hole hammer was used. The samples were returned from the 6 3/4" hole by air and were caught in a burlap sack. Each five foot sample was split with a Denver riffle; splits were given to the New Idria Mining and Chemical Company, the U. S. Copper Corporation, and Bear Creek Mining.

New Idria had one set of their splits assayed for total and leachable copper by Metallurgical Laboratories, Inc. of San Francisco, California. Bear Creek had the first three holes assayed for total copper, silver, and gold, and the remaining holes for total copper by the Montana Laboratory Company of Philipsburg, Montana. The assay figures of New Idria were generally higher than those of Bear Creek on the first three holes, and thereafter the New Idria figures were generally lower. Discrepancies between the assayers were in some cases over 1.00% copper and commonly were about 0.30% copper. I chose several of the U. S. Copper samples and assayed them. My assay figures checked within 0.01 to 0.10% copper of the Bear Creek assay figures; consequently, I have used the Bear Creek figures throughout this report.

New Idria's original plan was to drill twelve 200-foot holes to test a block of 10,000,000 tons. High clay associated with the leach cap and some deeply leached tactite, and old underground workings prevented many of the holes from going to the full 200 foot depth. The program tested a block of approximately 5,000,000 tons, half the original planned tonnage.

The average weighted (for footage) assay for the 11 holes completed to 200 feet is 0.27% copper, and that for the 9 holes not completed to 200 feet is 0.49% copper. The average depth of the 9 holes not completed is 86 feet. It is obvious that the better ore is in areas that are hard to drill. Because the reserve figures are based on the drill figures, a large bias is introduced , and the calculated ore tonnages and grades are significantly lower than they should be. The average weighted assay of the 9 holes not completed (0.49%) is the same as the weighted average of the channel samples (0.492%) taken by the J. R. Simplot Company personnel from the pit area.

One possible other source of error in the drilling should be mentioned . The most common oxide mineral chrysacolla, is a brittle silicate that shatters more easily than the gangue. Several dust samples were taken by Bear Creek and the following are the results:

Hole	Interval	Regular Sample	Dust Sample
N13	55-60	0.46% copper	0.60% copper
N16	80-85	4.00% copper	3.47% copper
N19	215-220	0.14% copper	0.19% copper

The data is insufficient to make any accurate conclusion regarding dust loss.

Considering all 20 holes, the approximate area drilled is 370,000 sq. ft. The average area of influence per hole is 18,500 square feet, and in ore that is as variable in grade as that drilled, the area of influence is too large to categorize any of the reserves as proven. The average depth of the 20 holes is 150'. Using these figures and a factor of 11 cubic feet per ton, the total tonnage tested is slightly over 5,000,000. Assuming an average area of influence of 18,500 square feet, the average weighted assay of the 5,000,000 tons is 0.33% copper. This block would contain 33,000,000 pounds of copper.

The results of the 20 drill holes are:

Sample	Interval	Copper	Weighted Assay (cu x interval)	Au.	Ag.
NI 1	200'	0.12%	24.0	0.002 oz/ton	0.07 oz/ton
NI 2	110'	0.47	51.7	0.006	0.14
NI 3	200'	0.26	52.0	0.019	0.165
NI 4	75'	0.38	28.5		
NI 5	65'	0.14	9.1		
NI 6	135'	1.47	198.0		
NI 7	50'	0.02	0.75		
NI 8	17'	0.09	1.49		
NI 9	91'	0.18	1.60		
NI 10	200'	0.52	104.0		
NI 11	200'	0.27	54.0		
NI 12	200'	0.19	37.0		
NI 13	200'	0.50	100.0		
NI 14	90'	0.54	48.6		
NI 15	200'	0.36	71.0		
NI 16	200'	0.26	52.0		
NI 17	200'	0.04	8.0		
NI 18	140'	0.25	35.0		
NI 19	225'	0.32	72.0		
NI 20	200'	0.24	48.8		
	Average footage= 150'	Average weighted assay= 0.33%	Total weighted assay= 997.5		

Ore Estimate of 14 Hole Area.

An area covered by 14 drill holes, numbers NI 2, 3, 4, 6, 10, 11, 12, 13, 14, 15, 16, 18, 19, and 20, is used for the ore estimate because values in the area are higher. Polygons around each hole represent the area of influence of the hole. The sides of the polygons intersect at right angles to one half the distance between that hole and each surrounding hole. The total area of influence is 216,850 sq. ft., and the average is about 15,500 sq. ft. Ore within such a large area of influence must be classified as probable and not proven.

The rock in each hole is subdivided into four categories. The first category is vat ore, and 0.60% copper is used as a lower cut off. The second category is "blend" ore, and the lower cut off is 0.35% copper. The third category is heap-leach ore, and 0.10% copper is used as a lower cut off. Waste, the fourth category, constitutes anything below 0.10% copper. Footage is not broke in less than 10 foot intervals and in most cases in not less than 20 foot intervals, because smaller intervals can not be mined selectively.

In dividing the footage. into any one category, some intervals were included that were below the lower cut off percentage, provided the total interval would average above the lower cut off. The following is the breakdown of the drill footage in each of the 14 holes:

Hole	Total Footage	Footage	Copper	Category
2	110'	85'	0.62%	vat
		25'	0.29%	heap
3	200'	55'	0.60%	vat
		90'	<0.10%	waste
4	75'	10'	0.60%	vat
		55'	0.30%	blend
		10'	0.14%	heap
6	135'	110'	1.82%	vat
		25'	0.18%	heap
10	200'	145'	0.68%	vat
		55'	0.18%	heap
11	200'	25'	0.67%	vat
		10'	0.41%	blend
		165'	0.19%	heap
12	200'	35'	0.61%	vat
		70'	0.15%	heap
		95'	<0.10%	waste
13	200'	115'	0.74%	vat
		50'	0.17%	heap
		35'	<0.1 %	waste
14	90'	65'	0.65%	vat
		15'	0.43%	blend
		10'	<0.1 %	waste
15	200'	45'	1.05%	vat
		50'	0.36%	blend
		35'	0.11%	heap
		70'	<0.10%	waste
16	200'	45'	0.62%	vat
		10'	0.37%	blend
		105'	0.17%	heap
		40'	<0.1 %	waste
18	140'	35'	0.65%	vat
		65'	0.16%	heap
		40'	<0.1 %	waste
19	225'	40'	1.47%	vat
		45'	0.17%	heap
		140'	0.01%	waste
20	200'	15'	0.68%	vat
Average total Footage	169'	55'	0.40%	blend
		100'	0.14%	heap
		30'	0.65%	waste

The tons in each category are based on the footage drilled times the area of influence divided by a factor of 11 cubic feet per ton. The average weighed assay for each category is equal to the total of the weighed assays (tons x assay) divided by the total tons.

The breakdown is as follows:

Hole	Area of Influence	Tons in area of Influence	VATS %	VATS Tons	BLEND %	BLEND Tons	HEAP %	HEAP Tons	WASTE
2	18,800	188,000	0.62	145,000	-----	------	0.29%	43,000	------
3	10.800	196,000	0.60	54,000	0.37	54,000	-----	------	88,000
4	15,200	104,000	0.60	14,000	0.38	76,000	0.14	14,000	------
6	13,800	169,000	1.82	138,000	----	------	0.18	31,000	------
10	16,800	306,000	0.68	222,000	----	------	0.18	84,000	------
11	11,100	202,000	0.66	25,000	0.41	10,000	0.19	167,000	------
12	14.800	269,000	0.61	47,000	----	------	0.15	94,000	128,000
13	16,300	296,000	0.74	170,000	----	------	0.17	74,000	51,000
14	16,300	133,000	0.85	96,000	0.43	22,000	----	------	15,000
15	8,500	155,000	1.05	35,000	0.36	39,000	0.11	27,000	54,000
16	23,400	425,000	0.62	96,000	0.37	21,000	0.17	224,000	84,000
18	15,500	197,000	0.65	49,000	-----	------	0.16	92,000	56,000
19	17,800	364,000	1.47	-65,000	-----	------	0.17	73,000	227,000
20	17,700	322,000	0.68	89,000	0.40	89,000	0.14	161,000	48,000
	216,850	3,326,000		1,180,000		311,000		1,084,000	751,000
	Average Area of Influence- 15,500	Total Tons Drilled 0.40% Copper		Tons of Vat ore 0.85% Copper		Tons of Blend Ore 0.39% Copper		Tons of Heap ore 0.17% Copper	Tons of Waste <0.10% Copper

The summary of the probable ore in the 14 hole area is:

Category	Tons		% Copper	
Vat ore	1,180,000	1,491,000	0.85%	0.75%
Blend ore	311,000		0.39%	
Heap leach ore	1,084,000		0.17%	
Waste	751,000		<0.10%	
Total	3,326,000		0.40%	

The stripping ratio of blended vat ore (0.75% copper) to heap leach ore and waste is 1:1.25.

P R O P O S E D D R I L L I N G P R O G R A M

The proposed program has a threefold purpose. First, the area that has already been drilled by New Idria should be drilled on 100-foot centers and to a depth of at least 500 feet. Holes should be placed between the New Idria holes so that all of the drill information can be utilized.

Second, the New Idria drilling program has drilled less than one-half of the area of mineralization. The largest unexplored area is between 6400 and 5600 West and 2800 and 3500 North, a surface area of 560,000 square feet. Mineralization is present throughout the area. Drilling should start in the vicinity of the Joe Hill level and should proceed outwards on 100 foot centers; a 500 foot depth should be the objective. Another area for exploration is west of the crescent shaped skarn zone. This area is between 3500 and 4600 North and 6000 to 5300 West, west of the area drilled. The size and grade of the mineralized area between 3900 and 4100 North and 5600 and 5400 West should be explored.

Third, the proposed drilling would delineate the depth of oxide mineralization. Sulfide mineralization would require different facilities for benefication. It would also be potentially glamorous, because the silver and gold values could be readily recovered. The ore is unusually high in silver and gold compared with other copper deposits. A 25-foot interval in NI 1 averaged 0.137 ounces gold per ton or $4.80; a 70-foot interval in NI 3 averaged 0.0517 ounces per ton gold or $1.80.

P R O D U C T I O N

An idealized flow sheet for vat production is shown on the next page. Production rates are determined by the leach vat cycle time which is seven days:

32 hours	Unload and recharge vat (shorter for small vats.)
72 hours	Maintain 50 gpl H_2SO_4
40 hours	Allow H_2SO_4 to drop to 8 gpl H_2SO_4, copper should be 20-35 gpl.
24 hours	Run to precipitation plant and flush.
7 days	

On a seven day cycle, 3,750 tons of ore are leached every week, or 550 tons each day. A recovery rate of 80-85% on 0.75% copper would yield 12 pounds of copper per ton of crude ore. The daily production would be 6,600 pounds of contained copper. At this production level on a 330-day year, approximately 180,000 tons of crude ore would be processed. Based on the probable ore reserves that have been drilled, the mine life would be over eight years.

A heap leach program will be initiated in the spring of 1968 to handle the heap leach ore and some of the rock categorized as waste. The cost of moving the rock has already been deducted from the vat leaching profit, and this will be a low cost method of increasing production rates.

Mine-550 tpd of 0.75% blended vat ore.
770 tpd heap leach ore and waste.

18" Grizzly

30"x16" Apron Feeder

54'x36' Conveyor

3'x10' Screen — -2" rock → Wet Scrubber

+ 2" rock

18'x38' Western -Austin Jaw Crusher

-2"½ +10 mesh

Screen

+40 mesh

-10 mesh

30"x50' Conveyor

Hydraulic Cyclone

Surge Stockpile

-40 mesh

Waste

20"x8' Apron Feeder

76'x30" Conveyor

4' Telsmith Standard Core Crusher

20'x30" Conveyor

4'x5' Screen

36" Traylor Gyratory

-3/8" rock

50'x30" Stacking Conveyor with Denver Sampler

Surge

Apron Feeder

Conveyor

-3/8" rock

3 small vats,-250 ton capacity each

2 large vats.— 1500 ton capacity each

Copper sulfate
Solution

4 celled gravity launder and scrap iron

cement copper 80-85% Cu

pan driers

bonded warehouse

OPERATING COSTS & PROFITS OF VAT LEACHING

Daily Operation Cost.

BASIS---- (1) 7 Day Week (2) 550 tpd of 0.75% (15 pounds) copper ore per ton. (4) 80-85% recovery (12 pounds per ton),

COST

	TOTAL COST	PER TON	PER POUND
Mining (includes mining heap ore & waste on 1:1.25 stripping basis.)	$ 360.00	$ 0.73	$ 0.061
Crushing	110.00	0.20	0.017
Plant & Ore Control	251.00	0.46	0.038
Miscellaneous Personnel	157.00	0.28	0.023
Fixed Overhead	57.00	0.10	0.008
Acid (1:4.5 ratio. Acid cost $30.00/ton on hill)	445.50	0.81	0.067
Scrap (1:1.8 ratio. Scrap cost $55.00/ton on hill)	330.00	0.60	0.050
Railroad & Trucking concentrates ($0.015/pound)	99.00	0.18	0.015
	$ 1809.50	$ 3.36	$ 0.279

Daily Operating Profit.

BASIS-----(1) 7 Day Week (2) 550 tpd of 0.75% copper (3) 80-85% recovery (12 pounds per ton) (4) market price $ 0.38/ pound.

Total Gross Income	Total Net Profit	Profit /Ton	Profit/Pound
$ 2508.00	$ 660.00	$ 1.20	$ 0.10

Percentage Profit= 35%

Mining Costs.

BASIS-----(1) 7 Day Week (2) 1:1.25 (blended ore:heap ore and waste) stripping ratio (3) 1250 tpd moved

Quarry Superintendent	$ 35.00
Shovel Operator ($3.00/hr.)	25.00
2 Euclid Drivers ($3.00/hr.)	50.00
80-D Shovel ($5.00/hr)	40.00
D-9G Cat ($20.00/hr. 2hrs.)	40.00
IR Drillmaster (400' drilled/week @ 200'/day)	20.00
Powder	40.00
2 Euclid 12 TDT & TDT ($30.00/day each)	60.00
No. 12 Motor Grader ($3.00/hr.)	25.00
Grader Operator	25.00
	$ 360.00/day

Mining cost per day = $360.00
Mining cost per ton rock + $0.29
Mining cost per ton ore = $0.73

Crushing Costs.

BASIS-----(1) 5 8-hour shifts, (2) 770 Tons Crushed/day

HD-16 (6 hrs. @ $ 5.00/hr)	$ 30.00
Crusher Superintendent	35.00
2 Crusher Operators ($3.00/hr. each)	50.00
2 Cone Crushers and 1 Jaw Crusher, Conveyors, Feeders, Lights, etc.	40.00
	$ 155.00/day

Cost per day on 5 day basis = $155.00
Cost per day on 7 day basis = $110.00
Cost per ton + $ 0.20

Blyth and Company, Inc.
2100 Russ Building
San Francisco, California 94104

Attention: Mr. Marvin Small

Gentlemen:

I hand you herewith an outline report on the copper mining venture of the U. S. Copper Corporation near Mackay, Idaho. This is made at your request and results from study of data provided by that corporation and from observations made on a visit to the property from May 1 to 4, last.

You understood on making the request that my conclusions must necessarily be made on the basis of the somewhat incomplete engineering data available supported by simple impression and mining intuition.

Your questions before I went to Idaho concerned advisability of making advances largely for further drilling and other exploration. As a result of the visit I feel that enough is known for the time being concerning the ore potentials, and also concerning tons and grade of a quantity of ore that seems sufficient to justify further pursuit of the current program of U. S. Copper aimed at bringing a steady production operation into being. Accordingly I recommend that any money advanced be devoted for the present to the needs of that program, rather than to further expansion.

My economics are based wholly on cop er production. Chance exists that the average gold and silver content of the copper ore may be on the order of $2 per ton. Recovery of some part of this, or, say, about $1 per ton, may be possible in the future, but at present it is not.

2801 Oak Knoll
Berkeley 94705
May 27, 1968

Philip R. Bradley

Mining Venture of U. S. Copper Corporation, Mackay, Idaho

OUTLINE REPORT 5-24-68

General A production operation not yet paying and not yet mechanically complete but having high chance of early success is now under way at the mine. Some further refinement of procedure in ore handling and in leaching is needed as well as certain additions of equipment before the operation can reach a consistent level of profitability. To do so is the aim of the program now in effect. This program should be complete by fall.

The mining is being conducted by open pit in an area that in part was drilled last year. Twenty holes were put down vertically, 11 reaching target depth of 200 feet. Analysis of results by U.S. Copper indicates presence of a mineable block of ground containing 3.3 million tons. Of this, 2.6 million would go to treatment and yield at least 20 million pounds copper. The pit has since been extended and is producing good ore from an area not drilled. It seems highly probable that at least 50 million pounds of recoverable copper is present in about 8 million tons. No question exists concerning further extensions of mineralized zone: it is massive, well defined by underground workings of an earlier day, and dimensionally offers chance for tonnages on the order of 50 to 100 million. Question exists, however, in respect to grade and economics of working any part beyond, say, the first 10 million tons, though chance is fair that most beyond this will be more or less similar to it. In any event the ore now present is sufficient to justify continuation of the current program and also expansion as fast as operating surpluses or other moneys permit.

Those who would invest in this mine might plan to treat the orebody itself as the commodity eventually to be sold, after delineation and evaluation by extensive drilling and other work, rather than treating the contained copper as the commodity and the business opportunity as lying in extracting and selling the metal. My own feeling is that any new money put into the venture should go into support of the present program directed toward establishing a paying operation.

To do so would resolve for this mine those uncertainties that now have the greatest relative importance, being those having to do with the techniques and economics of mining, and recovering, the copper of an orebody about which sufficient seems known for the time being.

Whether the business future of the mine after establishing a paying steady-gait state of operation lies in expanded operation or lies in outright sale, either would require some campaign of delineation and evaluation by drilling or otherwise of the ore available for such future. This work could begin at any time after it is established that a steady and paying operation has come into being.

It should be borne in mind that the true economic value of an orebody, whether a value established for purposes of sale or of future operations, depends on evaluation of the operating aspects and economics as well as of the tons and metal content of the orebody. Thus to continue the present operating program contributes to either purpose.

Present operations Present mining consists essentially of power shovel operation in a good-sized but still shallow open pit located virtually in the outcrops of an extensive mineralized zone standing more or less vertically.

As noted above, this zone dimensionally offers opportunity for many millions of tons of low grade copper ore; laterally, it has been opened by old underground workings for a length of some 4 thousand feet, with width at the open pit of more than 700 feet; underground mining operations of the past in this zone show a vertical range of more than 1,600 feet.

At the horizon now being mined more than 90% of the copper is in minerals that are acid soluble, and production of copper essentially requires only that the ore be crushed and leached. Heap leaching of crushed run-of-mine ore was attempted two years ago but failed because of mechanical problems not properly foreseen and solved. Leaching in vats is now going on. This is applied to the higher grade portions of the ore. Lower grade ore now mined is being set aside for heap leaching that will be resumed when proper preparation has been made, taking advantage of lessons taught by the earlier failure. Precipitation uses standard shredded can material, and a high grade cement product is shipped.

Expected gait of operation at present is for leaching in vats a daily average of 550 tons. It is probable that capacity of even the vats as now installed will be increased by improvements needed and planned in the ore dressing section of the plant. In addition, more vats are under construction. It is possible that the steady gait operation that should be attained this fall will see treatment of 1,000 tons daily in vats and about 800 in heaps.

At the present time all mine product of grade better than 8 lb. copper per ton ore is slated for vats; lower than 2 lb. rock is held to be waste; the 2 to 8 lb. middle range is destined for heap leaching.

Efficiency of copper recovery in vats is high, but their use is more costly; heap leaching is low in cost but also is low in efficiency. Ore of 15 lb. content treated in vats should yield 12 to 13 lb. Average grade of heap leach ore is expected to be about 3-1/2 lb. Slightly more than 2 lb. should be recovered. For my own purposes I have made use of the figure of 8 lb. as average recovery from all ore.

It is thus probable that total average daily copper production treating the 1,800 tons mentioned above will come to 15,000 lbs. At this gait operating costs should not exceed $0.25 per pound copper produced. As time passes and plant operating efficiencies are increased this cost may be reduced to 23¢ or even less.

Certain deficiencies and difficulties that contribute to unnecessary operating expense now exist in both mine and plant, but seem curable by addition of equipment together with some rearrangement, all of which is touched on below.

Orebody   Drilling and other exploration work done to date is not so extensive or complete that any large tonnage of ore of some certain, unquestionable grade has been blocked out, and the drilling program undertaken last year and on which ore estimates must now largely depend was neither well planned or well executed; nevertheless, mine production results are running ahead of grades so far indicated by exploration. Thus error in exploration results to date apparently is on the safe side.

A fairly safe interpretation made for present purposes from data available at this time seems to indicate almost certain presence of not less than 50 million pounds of copper in an ore that can be mined and treated at the cost levels required for profit. A strong chance exists that an additional 250 million pounds of copper can be shown either by production mining or by advance drilling to be present, and some chance exists for even more.

The results of the recent drilling and, even more importantly, the exposures made by mining in the current open pit, give now an indication that mining probably can be extended to an area of about 20 acres, this quite subject to future enlargement by expansion into ground as yet unexplored.

The drill results indicated that 44% of the rock to be mined from the area will contain at least 15 lb. copper per ton on the average while

34% will contain an average of only 3 or 4 lb. per ton. 22% would go to waste.

Topography is such that the upper two hundred feet or so of the ore zone is mineable without much increase in the quantity of waste that also must be mined due to widening of the pit into wall rock as it deepens. A 20-acre area will produce 75 thousand tons per vertical foot; applying the 34% and 44% factors to 200 vertical feet indicates that this much depth should furnish almost 7 million tons of the higher grade of ore and 5 of the lower grade. Recoverable copper in this 12 million tons can reach a total of 100 million pounds. On this copper an operating profit of from 7 to 10 cents per pound is possible at current costs and market prices.

The figure of 12 million tons of both classes of ore is subject to increase on further and successful lateral exploration and development by a factor that could be, say, 1.5x, or even more. It is subject to increase for extension in depth by some larger factor. The latter may justifiably be assumed to be at least 2x, this requiring mining only to 400 feet of depth. It could be much more, but at some depth the problems of mining and even of extraction will have to be changed. Some of this change may be advantageous but some, especially in mining, are so likely to be adverse that assumptions at this time and stage of knowledge as to existence of ore at deeper horizons are dangerous. However, assuming continuity of ore grade to only the 400 level, the potential ore reserve justifiable may be thought of as 18 to 20 million tons of 15 lb. ore and 14 or 15 million tons of the lower grade. This infers about 300 million pounds of recoverable copper.

Vertical uniformity of ore grade is questionable for the reason that no useful criteria exists on tendency with depth of grade or distribution of mineralization to change, other than the general vertical persistence throughout more than a thousand feet of those scattered pipe-like bodies of ore stoped in the past. The latter justifies some hope that open pit mining extended to that large horizontal area possible would, indeed, produce ore of a reasonably uniform and persistent grade as the pit deepened.

It might be noted that shape of the characteristic mass stoped in the old mine was to some extent pipe-like and to some extent bean-like. The vertical dimensions were the greater, some pipes measuring 400 feet and one 600 feet. Widths generally were anywhere from 5 to 50 feet.

These masses were scattered erratically throughout the ore zone, this being a matrix consisting of either a tactite formed on the arcuate contact between a granitic mass and a limestone, or a silicified granite porphyry accompanying, or both.

It seems probable that the source of copper in the open pit mine on this zone will be mostly miniature editions of the pipe- or bean-like masses too small to have been either found or stoped by the underground miners of the past, occasionally a larger one previously missed, and random mineralized fissures and lenses not mineable by underground methods. If so, it signifies that a rather close control of shovel work in the pits must be anticipated and kept.

In recommending that any new money invested be used to support the existing production program rather than to further drill extensively, I have it still in mind that some continued program of drilling or other exploration work should continue as a normal adjunct of any production schedule. Once a steady operating profit is flowing, diversion of production income into an exploration account supporting such continuing drilling and exploration should be made at a rate hardly less than about 5¢ per ton ore mined.

Such low-key drilling should also continue as would be needed for control of day-to-day ore production, such control being necessary to prevent feeding undue amounts of waste or low grade to mill and to help maintain the advantage of a reasonable consistency in daily mill heads and hence in daily production. In many cases it is accomplished by the same drilling in the pit that provides the blast holes. In any event, it is an attribute of operation and not of exploration and properly is a mining charge.

Economics   Dependent on the effect of variations in copper market price on net payments to be made by smelter for mine product delivered, present outlook is for a return above operating costs amounting to 7 to 10 cents per pound copper produced and sold.

This assumes a steady mining gait of between 1,500 and 2,000 tons daily, these two figures representing generally the range to be expected after completion of the current program. The larger of these seems the probable one. It should result in a copper production of about 6 million pounds a year.

At only 7¢ per pound, annual working profit thus would be but about $400 thousand. It might be $500 thousand. In any event cash flow to profit would be about 2¢ per pound less, or from $100 to $150 thousand annually less, owing to interest and other prior charges.

Capitalized expense incurred in finally bringing the operation to the steady-gait, paying level hoped for may total $2.5 million*. The inferred return of $250-350 thousand before taxes is thus not enough. This prospect requires, and the ore potential seems to permit, if not also to require, expansion of the operation as fast as is feasible.

---

*Mr R. M. Pinder, corporation president, stated to me that as of May 1 he was "in" some $1.8 million. This may not all be capitalized. Purchases and cost of installation of equipment still necessary to bring the operation up to profitable steady gait should not exceed $0.5 million. Some capital must also be found to cover current and further loss operation and certain forward mining expenses (such as cost of mining and setting aside the low-grade heap-leach ore pending institution of actual heap leaching). Nevertheless a sound and profitable operation ought to be in being before consumption of capital has exceeded $2.5 million.

The ore potentials cited elsewhere above, if in time proven, can well permit an increase of the above gait by a factor of 5x or more. It is difficult to say by what factor capitalization would be increased by such expansion. It may be on the order of only 2x or less. The effect then would be to increase rate of return considerably.

Operating experience built up with time tends to increase economic efficiency of a plant, and increases of capacity through plant expansion are bound to. Thus if the market price of copper and average grade of ore do not change too greatly, future outlook is for a net operating return better than 7 to 10¢ per pound copper, and for an annual return increased accordingly.

While the necessary major expansion of plant is to be hoped for in the near future, the need for capital probably would indeed first require institution of a major drilling and exploration campaign. This might tentatively be scheduled now for the second or third year after operations had first settled down, since any such expansion could certainly benefit from and may even depend on experience and data gained from one to two years of operating at the more modest gait.

If the figure of 5¢ per pound indicated above as being available for return of capital (the 7¢ per pound operating net less 2¢ for prior charges), then to retire $2.5 million would require production of 50,000,000 pounds copper. As stated in early paragraphs above, this quantity seems virtually certain. Potentials, as noted, are much greater.

Plant deficiencies and improvements Amongst other problems not yet fully resolved, current operations suffer from presence of an unwieldy proportion of clayey material accompanying the ore at the horizon now being worked.

This presents particular problems in feeding ore from the coarse ore receiving bin onto the feed system ahead of the crusher. Instead of maintaining a capacious coarse ore bin, the operation must resort to stockpiling mine ore nearby and delivering same to feeder by bulldozer. This alternative to a large bin is the best possible under the circumstances and is effective, but also is expensive.

The same clayey material also makes trouble in fine crushing. At the time of my visit a screen ahead of the coarse crusher passed and so diverted from fine crushing all minus 2" material, which was being set aside. Much more ore thus was being mined than could be fine crushed and treated, with obvious economic disadvantage. The 2" size of the screen was too great for the purpose but was only temporary.

An apparently competent revolving wet scrubber was being constructed at the time for the laudable purpose of washing clay out of the fines passing the above screen. It was thought, probably correctly, that with the washer in operation the screen opening should be increased to 4" or even more.

Clay so washed out of the ore, subsequent crushing should then be free of stalls due to clay. More importantly, vat capacity should rise (by reason of a faster leaching cycle) and acid consumption should fall.

The washed rock would be returned from the scrubber to the crushing circuit. This circuit is generally effective and well installed except for the fact that the coarse crusher is fed by a rather long and steeply inclined conveyor, a source of much trouble. For obvious reasons standard practice is to avoid use of conveyors until after run-of-mine ore has passed the coarse crusher, and instead to deliver by pan feeder direct from coarse ore storage to coarse ore crusher. The particular crusher here is somewhat small for the rock sizes coming from mine. On account of these factors, coarse ore crushing here consumes too much operating and maintenance labor.

Together with the water the scrubber will discharge all material in the ore less than about 2 mesh in size. For the time being there is nothing to be done with it. This accordingly represents another loss, proportion unknown but possibly amounting to about 10% of total feed. Screen tests show that copper assay of screen fractions increases as the size of particle diminishes. These tests have not yet, however, fractionated the material below 120 mesh so that nothing is known of the tendency of the slime fraction (presumably largely or wholly the clayey material of the ore, the scrubber working efficiently) to carry copper.

Proportion volumetrically to total feed of the fines is unknown but by assay they threaten to carry off an undue part of the copper mined and will indeed do so if the volumetric proportion is large. This fact is known to the mine staff, and some thought has been devoted to it.

It seems most probable that the simplest and most advantageous treatment of the scrubber fines is to discharge them into a wet cyclone that will separate at a very small size, say -300 mesh. All above even this fine size, or at least -200, should make feed for vats. Presumably the material below -300 mesh will include most of, if not all, the clayey material of the orebody. Chance exists that this material may again be lower in copper content. In any event, more screen testing seems required, and also some effort to determine quantities involved as between non-slimed and slimed fines.

The alternative to use of a wet cyclone is to send scrubber fines to a leach plant that would employ agitators such as the Devereaux or Pachuca types developed in the past in cyanide mills, but installation of the necessary equipment would be many times more expensive than the wet cyclone, with correspondingly higher operating costs. The principal difficulty in the idea of the wet cyclone is that the quantity and the nature of the material to be dealt with may be quite critical to selection of the proper unit; and these two aspects must remain unknown until by actual operation of the scrubber for some time a competent and representative sample of the prospective feed of the wet cyclone has been produced and analyzed.

The clay problem will abate somewhat as the pit deepens as the material is largely a phenomenon of the upper hundred feet or so of the ore zone, but it will never disappear.

Resumption of effort to recover copper by heap leaching must wait on grading a site and putting down an asphalt sheet, and on installation of acid and water lines, of gathering sumps, and acid-proof pumps and return lines for conveying pregnant solutions to the precipitation plant already set up near the vats.

Winters at the elevation of the plant sometimes are and sometimes are not severe, and some of the plant is not yet winterized. Provision for heating solution and wash water may be necessary. Some units still must be housed.

Additions needed for mine plant include two end dump trucks to substitute for the wheeled scrapers presently in use. A large garage and shop must yet be built in which to house and repair trucks, tractors, and other equipment. Various other needs also exist.

The following briefs expense now contemplated for new equipment and installations:

New leach vats, 3000 T. cap'y, & auxiliaries	$100,000
Heap Leach inst. incl. pumps, lines, etc.	125,000
Wet Cyclone installation	50,000
Replace Coarse Crusher	40,000
Dump Trucks, Track Drill, Compressor	85,000
Garage Bldg. & Shop Equipt.	75,000
Other	25,000
Total	$500,000

Management Should you assume an interest in the operation above outlined your advances foreseeably can amount from 0.7 million to 1.2 million dollars.*

Since this, in proportion to the capital already committed, would be apparently less than a majority interest, much attention obviously must be given to means of policing and protecting it. Because the venture involves mining these means nevertheless must not be so designed for your needs as to deprive the field management of a reasonably free hand. This is particularly a necessity at the stage of development in which the operation finds itself at present, one in which a great deal of flexibility still is required because problems of operation still remain to be solved. In the beginning assumptions had to be made concerning application of techniques that were not perfected to an ore about which little was known. A process of trial and error has ensued. Today most of the error can be judged to be of the past, but some smaller amount still is possible. There also is the normal and ever present likelihood of variation (within limits) in the nature and quality of ore, from day to day. Irregular changes in weather, labor and materials supply, and such also will still affect the work.

All of this requires that the corporation and business management allow to the operations management a certain autonomy in respect to monthly expenses, to capital improvement, and even as to change of course and

*Equipment and new construction (see page 15) 0.5 million; existing obligations 0.3 million; loss operation, forward supplier and operation, and other 0.2-0.4 million.

target. The following are recommended as at least philosophical rules that may serve the requirements.

1. Obtain places on the Board of Directors of the U. S. Copper Corporation, in numbers proportional to the actual interest, or to the optional interest, that you will have as the result of your negotiations with the U. S. Company as to terms of agreement in advance funds.

2. Even if the extent of such actual or optional interest does not warrant your holding a majority of the directorships, you still must place a Director on an executive committee that has been given a high degree of control and numbers preferably no more than three. If still a minority, this place then should have at least veto power as to certain specified acts of the committee such as approval of items of major expense, or of major acquisitions or divestments, or substantial changes in operating schedules or course.

3. Require the present company management, in order to qualify it to receive your funds, draw up and present a competent and acceptable schedule covering for at least the next six months the contemplated equipment needs, work procedures and chronology, expenses, and production in terms of ore to be mined, copper to be produced, and dollar income anticipated.

4. This schedule, after acceptance and inception, to be subject to review each quarter year and to your approval of applications for revision desired on the part of operations management. After first six months, such a schedule then to be submitted in advance for each subsequent quarter year.

5. Require operations management to institute a competent system of cost and record keeping (which does not now exist) that will make possible weekly performance reports from field to you.

> These reports to be in terms of production both of ore from mine and metal from plant, of moneys actually spent and of value of obligations incurred and payable, of income received, and of income due on shipments made. Capital expense, exploration expense, and both immediate and deferred operating expense or charges should be identified. Expenses should be presented in terms of labor cost, supply cost, and contracted services, and should be distributed between mining costs, ore treatment and marketing (or realization) costs, and general expense.

This will require making a now much overdue addition of a competent bookkeeper to the field staff. It will require also institution of a good system of reports to field office on the part of operating foremen and others, and a warehouse stock control system. One man should suffice, though initially setting up the stock control system probably will require making an actual physical inventory of present supply stocks and other work that will require temporary help from the outside. It seems desirable that at some early date that existing capital equipment also be inventoried and appraised.

A second overdue addition to the staff is of a man both capable of and given time for executing (a) certain at least elementary engineering design of plant components, assembly and construction, and topographic surveys needed for engineered plant construction and also, beginning in the near future, for mining control; (b) the functions of an ore clerk; (c) supervision of if not actual performance of the constant chemical analyses required by the operation, and supervision and analysis of results of future exploration; (d) other normal functions of a plant engineer. It would be of advantage if the individual in this position was of such caliber as to serve in time as mine and plant superintendent as required occasionally to relieve the present staff. (It seems well to note here that the engineering functions above cited are now being undertaken by Mr. Lawrence and Mr. Casebolt, but are by no means being fully discharged because of gross overload.)

6. Retain title to major items of equipment as may be bought with your funds, such as trucks, track drill, large crusher, compressor, etc. Possibly assume title to or mortgage on certain principal items already on property.

The venture should have demonstrated its promise or lack of it within 6 months or a year. If lack, you would wish to retire. It will not be feasible then to recover that part of your funds that would have gone into covering periods of loss operation, costs of installation of equipment, little if any of costs of construction, and possibly not much of such of your funds as had covered certain present obligations of management.

To have title to heavy equipment items might return, through their sale, but 10 to 15% of your advance; it is not of great dollar importance but may be of value in the relationships between field and corporation management. It also offers opportunity for rental charges that would add slightly to leverage.

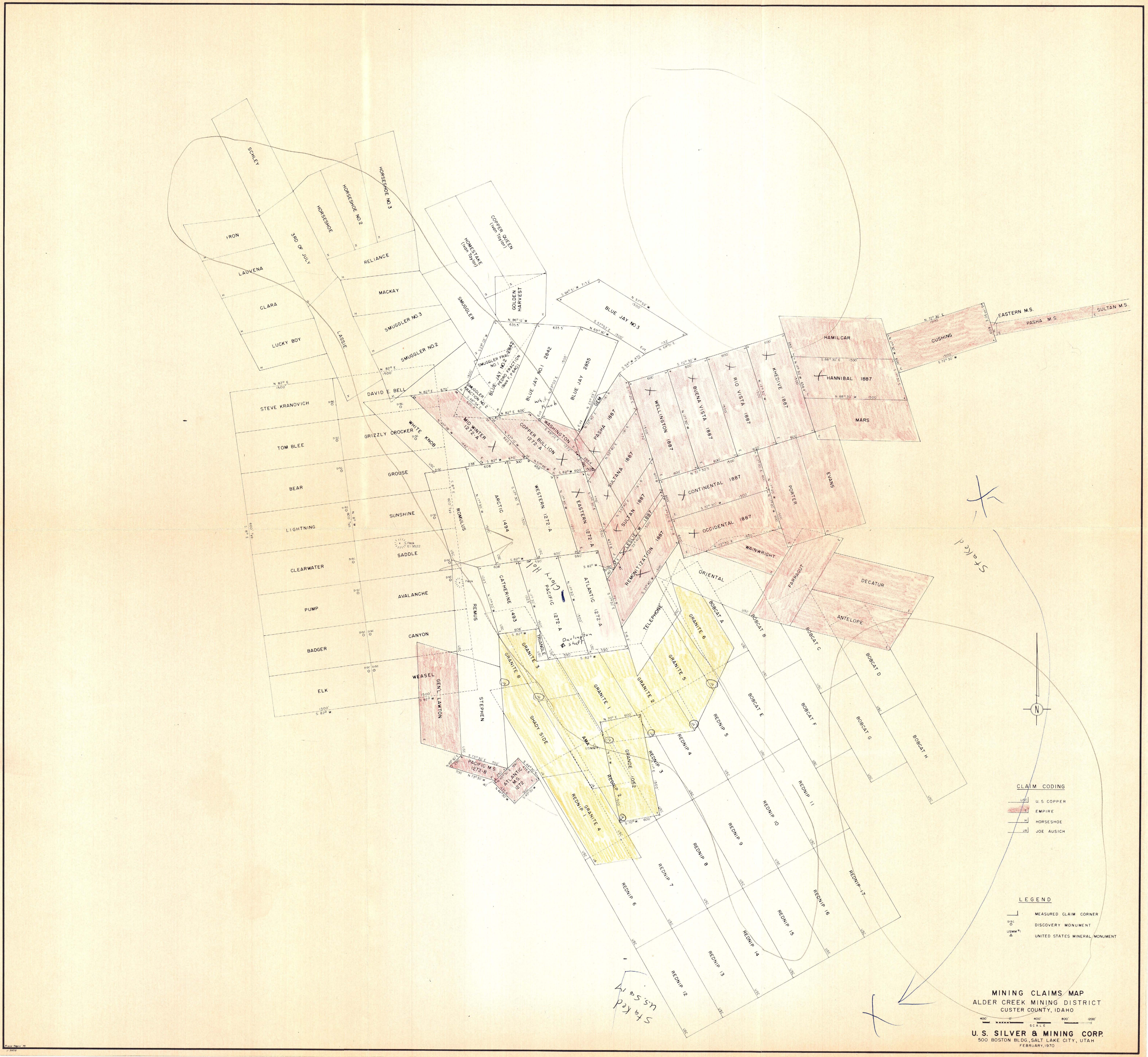
7. Attach some qualified person to your San Francisco staff to serve as the typical link needed under the circumstances between field and corporation management; to more or less continuously evaluate field results, progress, and competence; ot interpret to you the same, together

with validity of problems raised or found by the field; to assist in the solution of the latter, a clear need at the present time, and seemingly to be welcomed by the present operating staff.

8. Before any further commitments are made that might involve a cost to you of more than, say $10 thousand, a closer analysis should be made of existing data than was possible in the four day reconnaisance trip made by me earlier this month.

Philip R. Bradley

2801 Oak Knoll
Berkeley, California


MINING CLAIMS MAP
ALDER CREEK MINING DISTRICT
CUSTER COUNTY, IDAHO
SCALE
U. S. SILVER & MINING CORP.
500 BOSTON BLDG., SALT LAKE CITY, UTAH
FEBRUARY, 1970
CLAIM CODING
U.S. COPPER
EMPIRE
HORSESHOE
JOE AUSICH
LEGEND
MEASURED CLAIM CORNER
DISCOVERY MONUMENT
UNITED STATES MINERAL MONUMENT
SCHLEY
IRON
LAUVENA
CLARA
LUCKY BOY
3RD OF JULY
HORSESHOE
HORSESHOE NO.2
HORSESHOE NO.3
LASSIE
RELIANCE
MACKAY
SMUGGLER NO.3
SMUGGLER NO.2
SMUGGLER
HOMESTAKE
COPPER QUEEN
GOLDEN HARVEST
BLUE JAY NO.3
BLUE JAY NO.1 2842
BLUE JAY 2855
SMUGGLER FRACTION NO.1
BLUE JAY NO.2
DAVID E. BELL
STEVE KRANOVICH
TOM BLEE
BEAR
LIGHTNING
CLEARWATER
PUMP
BADGER
ELK
GRIZZLY CROCKER
WHITE KNOB
GROUSE
SUNSHINE
SADDLE
AVALANCHE
CANYON
WEASEL
GEN'L LAWTON
STEPHEN
ROMULUS
REMUS
MID WINTER 1272 A
COPPER BULLION 1272 A
WASHINGTON
ARCTIC 1494
WESTERN 1272 A
EASTERN 1272 A
CATHERINE 1493
PACIFIC 1272 A
ATLANTIC 1272 A
TRIANGLE
Darlington Shaft
Glory Hole
PACIFIC M.S. 1272 B
ATLANTIC M.S. 1272
PASHA 1887
GEM
SULTANA 1887
SULTAN 1887
LESLIE W. 1887
REMONITIZATION 1887
WELLINGTON 1887
BUENA VISTA 1887
RIO VISTA 1887
KHEDIVE 1887
CONTINENTAL 1887
OCCIDENTAL 1887
HAMILCAR
HANNIBAL 1887
MARS
CUSHING
EASTERN M.S.
PASHA M.S.
SULTAN M.S.
PORTER
EVANS
WAINWRIGHT
FARRAGUT
DECATUR
ANTELOPE
ORIENTAL
TELEPHONE
BOBCAT A
BOBCAT B
BOBCAT C
BOBCAT D
BOBCAT E
BOBCAT F
BOBCAT G
BOBCAT H
GRANITE 1
GRANITE 2
GRANITE 3
GRANITE 4
GRANITE 5
GRANITE 6
GRANITE B
SHADY SIDE
AMA
GRANDE 10&2
REDNIP 1
REDNIP 2
REDNIP 3
REDNIP 4
REDNIP 5
REDNIP 6
REDNIP 7
REDNIP 8
REDNIP 9
REDNIP 10
REDNIP 11
REDNIP 12
REDNIP 13
REDNIP 14
REDNIP 15
REDNIP 16
REDNIP 17
Staked
Staked U.S. 5a 17
N

Claim Map

Claim Map
USC.

Claim