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SHARONSTEEL • Mining Division

AN **NVE** COMPANY

SHARON STEEL CORPORATION

19TH FLOOR UNIVERSITY CLUB BLDG.
SALT LAKE CITY, UTAH 84111

TELEPHONE (801) 355-5301

FOR INTER-OFFICE COMMUNICATION

May 18, 1982

To: Mr. E. Peter Matthies, Vice President
and General Manager

From: William T. Worthington, Chief Geologist

Exploration Proposal

Gold Road Project

SHARONSTEEL • Mining Division

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May 18, 1982

Gold Road Project Proposed Exploration Program, 1982

Abstract

The Gold Road Project is located 20 miles west of Kingman, Arizona and consists of 18 patented lode claims, 21 unpatented lode claims, 4 patented millsites, 7 unpatented millsites and 4 leased unpatented lode claims.

The annual direct holding cost is \$3,000.

Past production records indicate a total production, through various operating periods, of 1,691,000 tons at 0.34 ounces of gold per ton along with minor silver values.

The Gold Road Mine was shut down by government order in 1942 and was subsequently stripped of all equipment and allowed to flood. The mine is in a deteriorated condition with considerable caving of the workings having occurred.

Estimated probable ore reserves are 556,000 tons at 0.24 ounces gold and 0.28 ounces silver per ton. Inferred reserves are estimated at 503,000 tons approaching the same grade.

An exploration drilling program of a total of 6 holes is recommended for this project at a total cost of approximately \$375,000. This drilling program will evaluate the probability of possible extension to known reserves along the southern strike and down the dip of the vein.

This project is considered to have good prospect potential for the discovery of additional reserves.

A leased property, four unpatented claims totaling 61 acres owned by Gold Road Red Top Mining Company, is recommended for purchase. This property is estimated to contain a considerable tonnage of inferred ore and has substantial potential for additional discoveries but is made unattractive by a high royalty commitment.

The purchase price of the Red Top property is estimated at \$80,000.

The total cost of this proposed program is projected at \$455,000.

Pertinent maps relative to the project are included herewith.

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FOR INTER-OFFICE COMMUNICATION

May 18, 1982

Gold Road Mine Project Exploration Proposal, 1982

Location - Logistics

This project is located in Mohave County, Arizona, in Township 19 North, Ranges 19 and 20 West; San Francisco Mining District, near the old mining town of Oatman. The project is 18 miles east of the Colorado River and 20 miles west of the small city of Kingman, the principal population, transportation and supply center for this northwest area of Arizona.

The project is readily accessible by paved road, and power, natural gas and telephone lines cross the property. Water for mining and milling purposes, although barely adequate in the past, is available from a number of scattered, small, natural springs in the area.

Property

The property owned by Sharon is composed of 18 patented claims (253 acres); 21 unpatented claims (431 acres); 4 patented millsites (18 acres) and 7 unpatented millsites (14 acres). Sharon leases four unpatented claims (61 acres), known as the Red Top Group, under an agreement extending until the year 2002.

These claims form a contiguous, elongate group which is four miles long in a northerly-southerly direction and one-half mile wide at its maximum.

The annual, minimum direct, holding costs total \$3,000.

Historical Background

The Gold Road Mine was discovered in 1903 and was operated by a French Company until 1911 when it was acquired by the U. S. Smelting Company. U. S. Smelting operated the mine during the periods 1911 through 1917; 1924 through 1928 and 1935 through 1942 when it was closed by government order L-208 which closed most domestic gold mines.

All of the surface mine and the milling plant were subsequently removed with much of the equipment being transferred to Company operations in New Mexico. The property has remained inactive since 1942.

Property Condition

Maintenance of mine workings was suspended after closure in 1942 and all surface and underground facilities and equipment were stripped and removed by 1951. As a consequence of this abandonment, the surface and underground workings are in a deteriorated, overall poor condition with considerable caving of the workings having occurred. The mine is inaccessible and the workings have flooded with approximately 62 million gallons of water which stands at about the 300 foot level.

Past Production

Total production through 1942 is estimated at 1,691,000 tons of ore, most of which was milled in the facilities on site. The production was as follows: French Company - 1903 to 1911, 327,000 tons at an estimated grade of 0.60 ounces gold per ton; U. S. Company - 1911 through 1928, 564,000 tons at 0.33 ounces gold and 0.35 ounces silver per ton; U. S. Company 1935 through 1942, 800,000 tons at 0.24 ounces gold and 0.19 ounces silver per ton.

These ore tonnages and grades represent averaged, mine-run production and reflect a historically averaged mine dilution factor of 25 to 30%, a factor inherent to the mining of this type of ore occurrence.

The principal mining method employed was shrink-fill stoping which was particularly suitable to the near vertical attitude of the average 4.5 foot wide, fissure-vein ore occurrence and the host wall rock conditions.

During the later production period 75,000 to 100,000 tons of earlier produced mill tailings were reprocessed at intervals in order to maintain a full mill-circuit load due to a deficit of mine ore production. These old tailings contained approximately .08 ounces gold per ton, which could be partially recovered with the later, improved, milling process.

Mined ore production ranged from an early low of 150 tons per day to a later high of about 400 tons per day. The milling facilities through these periods were rated at approximately the mine production tonnage and were of the cyanide, agitated-vat-leach process. Gold recovery ranged from a low of 75% to a high of 92%; the silver ranged from 40% to 70%. The higher ranges of recovery were achieved during the later operating periods.

Ore production from the other principal mines of this district, the Tom Reed and United Eastern, up to 1942, amounted to 1,900,000 tons at an average mine diluted grade of 0.59 ounces gold per ton. This ore was of the same type as that produced from the Gold Road Mine and was milled by the cyanide leach process at their facilities.

Present Ore Reserves

At the time of the mine operations closure in 1942 the ore reserves, as estimated by the U. S. Company, were listed at 556,000 tons of proven ore at an average grade of 0.24 ounces gold and 0.28 ounces silver per ton. In addition, there has been estimated to be 503,000 tons of possible ore approaching

the same grade. The estimates include a mine-run dilution factor of 25%.

The, so called, proven reserves are not a certainty in that the majority of the ore blocks are exposed on one or two sides and the minority on three sides. The ore would, therefore, be more properly designated as being in the probable class.

The possible ore reserves listed above are based upon blocks exposed only partially on one side, on blocks projected from mine workings and on two widely spaced drill holes which indicate a significant diminution of the tonnage and grade. The possible ore should, therefore, more accurately be designated as being in the, somewhat lower, inferred reserve class.

The probable ore reserve blocks are scattered throughout the entire 6000 foot length by 900 foot depth of the mined and developed area of the vein. The minability of many of these ore blocks is uncertain as many are small and of difficult access, while others appear to have been left to the last as mine workings support.

The inferred ore reserves area, as projected in 1942 by the then available evidence, is along the strike of the vein in a southerly direction for a total distance of up to 1500 feet from the nearest mine workings and up to 1000 feet of vein dip interval. This projection places about half of the inferred ore potential into the adjacent Red Top property which is under lease to Sharon. Of lesser inferred ore potential is the down-dip projected extension of the ore bodies beneath the previously mined deepest levels of the mine.

There are approximately 1,000,000 tons of mill tailings on the Gold Road property. In the 1930's these tailings were thoroughly sampled by drilling, with the findings that about 120,000 tons, the French Company tailings, averaged about .08 ounces gold per ton. The remaining tailings contain .03 ounces of gold or less per ton. As previously stated, up to 100,000 tons of the French tailings were reworked during the last period of operation.

Geology - Topography

The Gold Road Mine is located in the southern portion of the Black Mountains which consist of ruggedly dissected, gently eastward dipping, layered blocks of Tertiary, volcanic flow rock units which overlie a basement complex of Pre-Cambrian gneiss and granite. The topography is rugged to moderate with elevations ranging from 2,000 to 4,200 feet. The climate is arid with moderate rainfall; hot summer and cool winter temperatures.

The principal rock formations in ascending order are the gneiss-granite basement, overlain by a thick series of trachytes, andesite, latite, tuffs, rhyolite and basalt. Intrusive into parts of these are monzonitic, granitic and rhyolitic porphyrys. The most important ore bearing host formation of these rock units is andesite. These formations are cut by numerous faults of prevailing northwesterly strike and northeastward dip.

The Gold Road vein is a complex normal fault fissure system which strikes northwesterly and dips at about 80 degrees to the northeast in the upper levels of the mine and decreases to a 60 degree dip in the lowest levels. The vein is a compound stringer lode throughout much of its extent, which consists of two or more veins separated by wall rock with stringer veinlets in between. Much of the vein is lenticular in all dimensions which may pinch and swell from 1 foot to 25 feet, and in places the vein zone may reach 80 foot widths but only a small portion would contain ore mineralization. The average width as mined is 4.7 feet.

The ore mineralization is of the epithermal type; that is, formed at a depth within 3000 feet of the surface by ascending, circulating mineralized solutions in the temperature range of 50 to 200 degrees Centigrade. The vein filling is mineralogically of simple character, consisting mainly of quartz, calcite and adularia with associated free gold in the ore bodies. Either quartz or calcite may predominate and vary widely throughout the vein. Ore grade mineralization usually occurs where both minerals are present. Areas of the vein that consist entirely of either quartz or calcite are generally low grade or barren.

The metallic minerals consist of free gold and rare pyrite and chalcopyrite. The gold is characteristically fine grained and generally can be seen only in rich ore. The vein appears banded in cross section, showing that the minerals were deposited in successive layers from the walls to the middle of the fissure during five, somewhat, distinct stages. The gold content increased with each stage from an average of 0.06 ounces per ton during the first stage to 1.0 ounces or above during the final stage.

The ore bodies of the Gold Road and other mines in the district are characteristically lenticular in plan and irregular in longitudinal section. The bodies are generally confined to a fairly definite zone of vertical range, the parts of the vein above and, especially, below this zone being barren. Very little ore has been found below a depth of 1000 feet.

This vertical range ore zoning observation has been tested to a high degree during the past several years by deep exploration drilling on other principal mines in the district and has proven to be substantially correct, although exceptions may exist. This zoning remains to be tested at the Gold Road Mine, but indications are that it will conform to the district type ore occurrence of a series of ore bodies "bunched" along the vein strike at close spaced intervals and all occurring within a general vertical range.

Exploration To Date

The only exploration work of significance since the mine ceased operations has been the core drilling of two holes from the surface to intercept the area of inferred ore reserves along the south strike of the vein within the Red Top property. These two holes were spaced 600 feet apart along the vein strike and at the same intercept level, approximately the 700 foot level of the mine workings. The GR-1 hole was drilled in 1973

and penetrated 4 feet of vein which assayed 0.112 ounces of gold per ton; GR-2 was drilled in 1974 and penetrated 2.1 feet of vein which averaged 0.53 ounces of gold per ton. At a minimum vein mining width of 5 feet, with barren wall rock, the average grade of the vein intercepts would be 0.10 ounces of gold for GR-1 and 0.28 ounces of gold for hole GR-2. As with the other ore reserves, an average 25%, as mined, dilution factor would apply to the reduction of these grades.

As previously stated, these two drill holes have tended to diminish the tonnage and grade estimate in the area of inferred ore along the south strike of the vein.

Current geologic data is sufficient and reliable enough to justify proceeding with an initial exploration drilling phase. Concurrent with this other exploration work, such as geologic mapping, interpretive geologic-mineralogic studies, geochemical and geophysical surveys, should be continued and extended.

Potential Ore Reserves

Based upon studies of the Gold Road Mine data, recent geologic studies and exploration drilling of others in the same district, and on numerous published geologic study reports, a reasonably attractive exploration potential is apparent for the discovery of additional ore reserves.

The principal area of recognized potential lies along the projected extension of the vein beyond the southernmost limits of existing mine workings. This extension has been proven to exist for 1500 feet by the previously described GR-1 and GR-2 drill holes. Beyond this proven extension, the vein is projected for a possible strike distance of about 9,000 feet. Although a great deal of potential vein area remains to be tested to the south, the strength of the mineralization does appear to be diminishing in this direction and only a small fraction of this distance may have good ore potential. This southerly vein mineralization projection has a downward plunge (rake) of 10-25 degrees, this being due to a volcanic rock unit identified as the "Red Sill," which appears to have dammed the upward extension of the vein mineralization. Because of the plunge, potential areas of mineralization are progressively deeper with increasing distance to the south and thus may be subject to "bottoming out," due to the epithermal vertical range zoning.

Proposed Exploration

Although a significant mineral reserve is indicated to be present in the Gold Road Mine, it is insufficient to justify the high cost of re-opening the mine and does not represent a profit potential at existing prices and production costs. It is, therefore, necessary to explore for additional ore reserves in the mine area that would improve the potential. Targets of good merit are present and are herewith recommended to be tested by a surface diamond drilling program.

It is recommended that a diamond drilling program consisting of about six drill holes be conducted on the property. The holes would be

drilled at various angles to explore for additional ore along the southerly projection of the vein and to test for extensions at depth below previously indicated ore zones. The total planned footage to be drilled in this program approximates 12,500 feet and is expected to cost a total of \$375,000 under a drilling contract agreement. The operations would be conducted with one drill rig operating three shifts per day and would require six to seven months for completion.

This drilling approach, if successful in its objective, would then justify and require that the mine be reopened for further exploration and evaluation. A successful program should consist of five or six ore holes indicating a minimum recoverable tonnage potential of 1.5 to 2 million tons of ore at a significantly improved grade and/or average ore width.

Preliminary estimates have been made of the work required to reopen the southern end of the mine to the extent of providing access for underground exploration should the drilling warrant it. Such a program would cost 2.0 to 2.5 million dollars and would require approximately 18 months to complete. This work would include rehabilitation of the No. 3 shaft, dewatering of the mine to the 700 level and rehabilitation of the 700 level and Line Road Tunnel from No. 3 shaft to their southern limits. It would not include any exploration work, diamond drilling, opening of other mine levels or the northern end of the mine, new development work, permanent surface facilities, a mill, tailings pond or other such projects. It would provide access for underground exploration only.

In the event it should prove feasible to place the mine into production the full cost of a 500 ton per day operation, including exploration, reopening, development, surface plant, mill and tailings pond is estimated at from 15 to 20 million dollars. Time required to complete such a project from commencement of drilling to initial production is estimated at from 3 to 4 years.

It is, of course, much more likely that the proposed drilling will provide less conclusive results than indicated above. In this event, depending upon the kind of results forthcoming from the initial drilling, a follow-up exploration program, probably consisting of additional surface diamond drilling, will be recommended.

Property

The southerly extension of the Gold Road vein, where diamond drill holes GR-1 and GR-2 have exposed low grade gold ore and where much of the exploration will be conducted for possible ore extensions, is located on the leased Gold Road Red Top Mining Company property. This property consists of four unpatented lode mining claims totaling approximately 61 acres. Sharon has fully complied with a work commitment agreement and is now only obligated to perform annual assessment work on the property. The lease, with extensions, can be continued in effect into the year 2002.

A significant portion of the Gold Road inferred ore reserve and the major portion of the potential for vein extension and new discovery

are located within this property. Several of the proposed exploration holes will be drilled to assess this potential.

The lease contains a sliding royalty scale that provides for payment of 3% of the gross value of ore on ore valued at \$15.01 to \$20.00 per ton and increasing 1% in royalty with each \$5.00 increase in value per ton to a maximum 20% royalty on gross value of ore with a gross value of \$100.00 per ton and over. The gross value is determined by first computing the gross weight of each metal by multiplying the dry weight in short tons of each lot of such ore by the weight of each metal contained in an average ton of such lot and then multiplying the weight of such metal so computed by the average price prevailing during the calendar month in which the ore is sampled at the receiving mill, smelter or other plant.

This royalty is extraordinarily high, especially when it is remembered that it is paid on gross value of contained metal and not on recovered metal. The price of gold would have made a major upturn before Red Top ore would even approach being profitable under this schedule.

During negotiations with officials of the Gold Road Red Top Mining Company it has been indicated that the property could be purchased for \$80,000. This purchase is recommended and must be made if the Gold Road Mine is to have any chance of being economically viable.

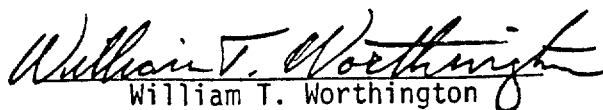
Cost

The total cost of this program, as here presented, is \$455,000.

Maps

Accompanying this Report-Proposal are the following described maps:

- Plate 1 - Gold Road Project; geographic location.
- Plate 2 - Gold Road Project; geographic location.
- Plate 3 - Gold Road Project; property, topography.
- Plate 4 - Gold Road Project; old photographs.
- Plate 5 - Gold Road Project; long section, previous mining.
- Plate 6 - Gold Road Project; long section, ore reserves, proposed drill holes.
- Plate 7 - Gold Road Project; section through No. 3 shaft.
- Plate 8 - Gold Road Project; district geology, topography.
- Plate 9 - Gold Road Project; geologic section, vein.
- Plate 10- Gold Road Project; property, topography, vein trace.


William T. Worthington

WTW:ms

APPROVED:

Gaylon W. Hansen
Manager Exploration

September 6, 1983

J. D. Sell
J. R. Stringham
S. A. Anzalone

*call JRS as he
will contact
Worthington*
*Tues
call = Thursday, PM*

Sharon Steel

Data review of Gold Road, AZ and Tintic, UT has been scheduled for September 19, 20 at Sharon's office, Vanadium, NM (SAA knows location).

J. D. Sell should call Worthington at Sharon's Miami office (305-866-7771) Thursday, September 15th to confirm meeting and inform Worthington of your arrival time at his office.

You should review any other of their properties that you want.


W. L. Kurtz

WLK/cg

December 2, 1983

To: R. L. Brown
From: J. R. Stringham

Sharon Steel's Gold Road Mine

Attached are the results of my analysis of the profit potential of the Gold Road Mine of Sharon Steel. You should have on hand Sal Anzalone's report on the ore reserves upon which this analysis is based. The returns on investment are calculated for three purchase conditions and a graph prepared to demonstrate the relative returns. You will note that the ROI is 19.5% even if we pay \$4 million for the property.

I'm attaching summaries of how I arrived at the production costs and capital investments. You may wish to discuss those at greater length to satisfy yourself that they are reasonable.



J. R. Stringham

JRS:mek

cc: W.L. Kurtz
J.D. Sell
S.A. Anzalone

GOLD ROAD #10

177,560 100% year: 58 years @ 0.2450% A/T/M,

Purchase Price 4 M; Cost 5440M, Tax 40%

15% depletion
88% Recovery
\$400/100 M

12-1-83

LPS

	CAPITAL INVEST.	NSR	ROYALTY		NSR AFTER ROYALTY	OPER. COST	TOTAL OPER. COST	CASH FLOW BEFORE TAX	DEPREC.	DEPLET.	TAXABLE INCOME	INCOME TAX	CASH FLOW AFTER TAX
			NSR	NET PROFIT									
1	\$6.05					2.65	2.65	(8.7)			(2.65)	(1.22)	(8.7)
2	\$3.75					1.65	1.65	(5.4)			(1.65)	(0.76)	(5.4)
3		15.09			or Cap. +1.51	4.59	11.1	3.99	0.31	1.98	1.98	0.91	3.94
4		15.69				9.59	9.59	5.50	1.01	2.25	2.25	1.04	5.50
5		15.09				9.59	9.59	5.50	1.25	2.13	2.13	0.95	4.55
6		15.09				9.59	9.59	5.50	1.22	2.14	2.14	0.98	4.52
7		15.09				9.59	9.59	5.50	1.22	2.14	2.14	0.98	4.52
8		12.03			or Cap. -1.51	8.15	6.64	6.19	0.77	1.92	1.92	1.03	5.31
9													
10													
11	GOLD ROAD #10 with 4 M Payment - 19.5% ROI												
12													
13	GOLD ROAD #11 with \$241 PAYMENT = 25.1% ROI												
14													
15	GOLD ROAD #12 with 0 PAYMENT = 32.7% ROI												
16													

SUM OF CASH FLOW STREAM = 14340.00

GR#10

SUM OF CASH FLOW STREAM = 18340.00

GOLD ROAD #11

SUM OF CASH FLOW STREAM = 16340.00

GOLD ROAD #12

	YEAR	CASH FLOW		D.C.F.
19				
20	1	-8700.000	-8700.000	
	2	-5400.000	-4519.606	
	3	3990.000	2795.031	
	4	5550.000	3253.969	
	5	4550.000	2232.743	
	6	4520.000	1856.404	
	7	4520.000	1553.744	
	8	5310.000	1527.715	

	YEAR	CASH FLOW		D.C.F.
	1	-4700.000	-4700.000	
	2	-5400.000	-4069.610	
	3	3990.000	2266.162	
	4	5550.000	2375.582	
	5	4550.000	1467.734	
	6	4520.000	1098.838	
	7	4520.000	828.119	
	8	5310.000	733.175	

	YEAR	CASH FLOW		D.C.F.
	1	-6700.000	-6700.000	
	2	-5400.000	-4317.956	
	3	3990.000	2551.184	
	4	5550.000	2837.568	
	5	4550.000	1860.155	
	6	4520.000	1477.613	
	7	4520.000	1181.531	
	8	5310.000	1109.905	

TOTAL 14340 0.000

TOTAL 18340 0.000

TOTAL 16340 0.000

RETURN ON INVESTMENT = 19.479 %

RETURN ON INVESTMENT = 32.691 %

RETURN ON INVESTMENT = 25.059 %

GOLD ROAD #10

177,500 tons/yr: 585 years @ 0.245% A/TM,

Purchase Price \$4 M; Cost \$4/TM, TAT 40%

15% dilution
88% Recovery
\$400/oz HL

12-1-83

JRS

#M	CAPITAL INVEST.	NSR	ROYALTY	NET PROFIT	NSR AFTER ROYALTY	OPER. COST	TOTAL OPER. COST	CASH FLOW BEFORE TAX	DEPREC.	DEPLET.	TAXABLE INCOME	INCOME TAX	CASH FLOW AFTER TAX
1	\$6.05					2.65	2.65	(8.7)			(2.65)	(1.22)	(8.7)
2	\$3.75					1.65	1.65	(5.4)			(1.65)	(0.76)	(5.4)
3		15.09			or Cap. + 1.51	9.59	11.1	3.99	0.31	1.98	1.98	-0.91	3.99
4		15.09				9.59	9.59	5.50	1.01	2.25	2.25	1.04	5.50
5		15.09				9.59	9.59	5.50	1.25	2.13	2.13	0.98	4.55
6		15.09				9.59	9.59	5.50	1.22	2.14	2.14	0.98	4.52
7		15.09				9.59	9.59	5.50	1.22	2.14	2.14	0.98	4.52
8		12.83			or Cap. - 1.51	8.15	6.64	6.19	0.79	1.92	1.92	0.98	5.31
9													
10													
11	GOLD ROAD #10 with \$4 M Payment = 19.5% ROI												
12													
13	GOLD ROAD #11 with \$2 M PAYMENT = 25.1% ROI												
14													
15	GOLD ROAD #12 with 0 PAYMENT = 32.7% ROI												

17	SUM OF CASH FLOW STREAM = 14340.00			SUM OF CASH FLOW STREAM = 18340.00			SUM OF CASH FLOW STREAM = 16340.00		
18	GR#10			GOLD ROAD #11			GOLD ROAD #12		
19	YEAR	CASH FLOW	D.C.F.	YEAR	CASH FLOW	D.C.F.	YEAR	CASH FLOW	D.C.F.
20	1	-8700.000	-8700.000	1	-4700.000	-4700.000	1	-6700.000	-6700.000
	2	-5400.000	-4519.606	2	-5400.000	-4069.610	2	-5400.000	-4317.956
	3	3990.000	2795.031	3	3990.000	2266.162	3	3990.000	2551.184
	4	5550.000	3253.969	4	5550.000	2375.582	4	5550.000	2837.568
	5	4550.000	2232.743	5	4550.000	1467.734	5	4550.000	1860.155
	6	4520.000	1856.404	6	4520.000	1098.838	6	4520.000	1477.613
	7	4520.000	1553.744	7	4520.000	828.119	7	4520.000	1181.531
	8	5310.000	1527.715	8	5310.000	733.175	8	5310.000	1109.905
	TOTAL	14340	0.000	TOTAL	18340	0.000	TOTAL	16340	0.000

RETURN ON INVESTMENT = 19.479 %

RETURN ON INVESTMENT = 32.691 %

RETURN ON INVESTMENT = 25.059 %

GOLD ROAD

MINEABLE RESERVE

Assured & Probable

Calculated to a mined width of at least 5 feet: 343,700 tons @ 0.281 oz. Au/ton
Diluted by 15%: 395,000 tons @ 0.245 oz. Au/ton

Possible

200,000 tons
360,000 tons

560,000 tons
Diluted by 15% 644,000 tons @ 0.245 oz. Au/ton

Total

1,039,000 tons

Time Period

1,039,000/177,500 = 5.85 years

Values/ton 0.245 X .88 X 400 = 86.24
Shipping & Refining .245 X .88 X 3 = 0.65

85.59

say 85/ton

Mine, Mill & O.H. Costs

Net Mine

54

\$31

Per Year: Values 85 X 177,500 = 15,090,000
(1st 5 years) Costs 54 X 177,500 = 9,585,000
Net 31 5,505,000

6th Year 15,090,000 X .85 = 12,827,000
9,585,000 X .85 = 8,147,000
5,505,000 X .85 = 4,680,000

DEPLETION

Cost Depletion

4,000,000/1,039,000 X 177,500 = \$683,000/year

15% Depletion

50%

Year 1	\$2.26M	\$1.98M
2	2.26	2.25
3	2.26	2.13
4	2.26	2.14
5	2.26	2.14
6	1.92	2.70

CAPITAL INVESTMENT

I. Mill

(Study by G. Hansen, Sharon, 1981)

Mill Construction	\$5.0M
Environmental Study	0.2
Tailings Pond, Offices	<u>0.6</u>
Total	\$5.8M*

* Don Crowell and Al Raihl think, with the little information that we provided, that this figure is not unreasonable. They both suggest further research of Sharon's Silver City files to properly assess this estimate.

II. Mine

(Red path estimate of rehabilitation, JRS estimate of mine development to provide 500 tpd)

Rehabilitation	\$2.65M
Development	.93
U.G. Equipment	<u>.70</u>
	\$4.28M

III. Resume'

Mill	\$5.8M
Mine	<u>4.3</u>
Total	\$10.1M

MINING COSTS

- I. Sharon's figures escalated to present using Galena's rate of escalation from 1973 to 1983.

Mine	\$15.52 @ 10.79%/year	\$43.25/ton
Mill	2.50 @ 8.13	5.46
Overhead	<u>2.00 @ 8.90</u>	<u>4.69</u>
Total	\$20.02	\$53.40/ton

- II. Coeur's figures - less timber and pumping plus \$2.00 extra reagent cost.

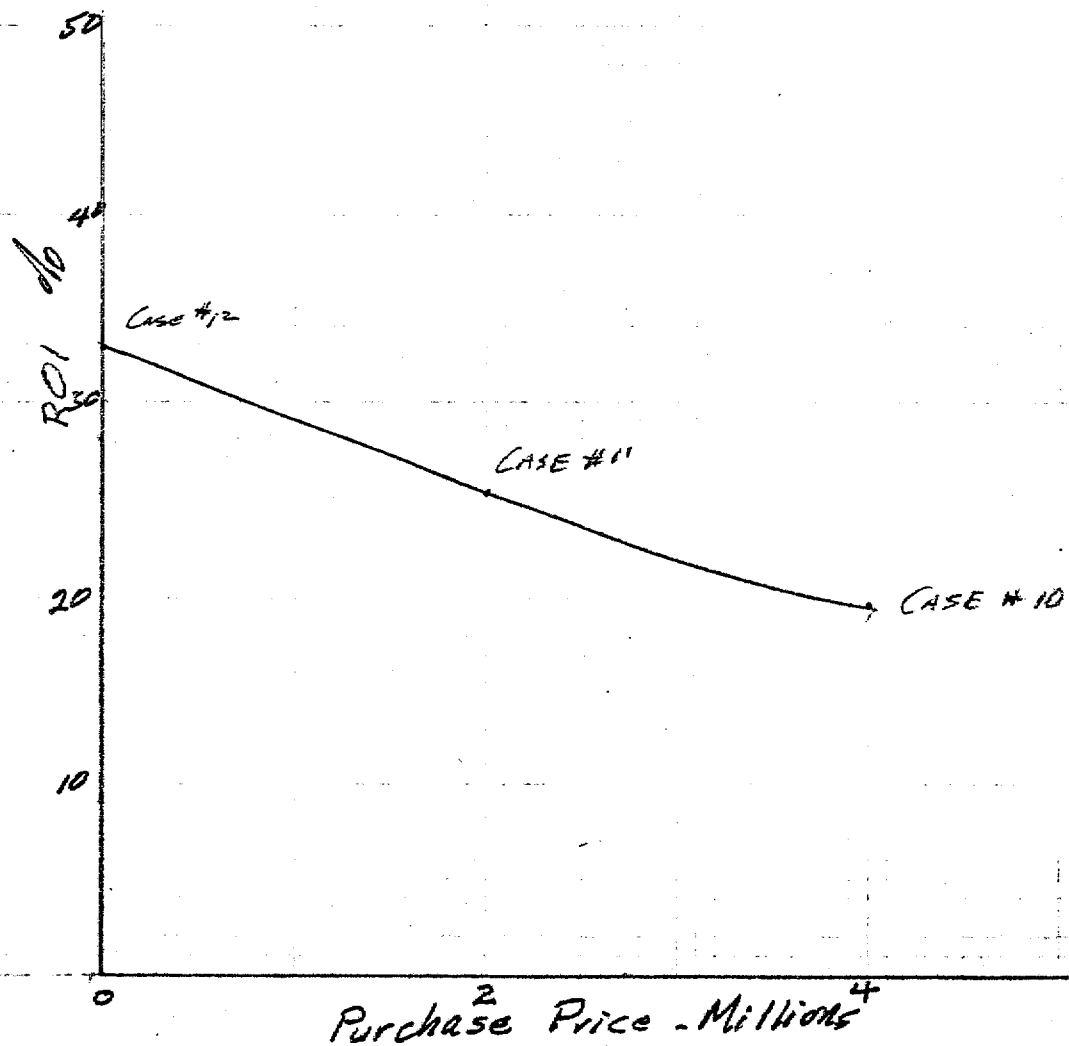
Mine	\$37.29 - \$12	=	\$25.29
Mill	3.80 + 2	=	5.80
Overhead			<u>19.91</u>
			\$51.00

Use:

\$54.00 for total cost

GOLD ROAD

12-1-R3



Near
mineral
concentrations

530,000

1/2 mill
detritals to 5 ft min.
.276 cost/ton

Gold Road, AZ

plus

560,000

.208 detritals to 5' min.

Total 1,090,000

.241

What's Mineable (5' dia).

345,000

.281

+ 15%
mining
deduction

355,000 less .245

in Escalante

December 15, 1983

R. L. Brown
New York Office

Gold Road Mine
Sharon Steel
Arizona

In Denver I gave you Anzalone's report and Stringham's preliminary financial analysis of Sharon Steel's Gold Road property at Oatman, Mohave County, Arizona. At separate times I briefly discussed the property with T. C. Osborne and R. J. Kupsch.

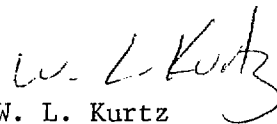
Several features of the Gold Road Mine should be emphasized:

- When the mine shut down in 1942 mining was actually taking place within more than a dozen of the blocks in the SAA/JRS 400,000 ton mineable reserve. I see no reason to resample any of the blocks--SAA/JRS have seen the actual pencil originals of the assays taken by USSMR.
- The better grade reserve blocks on the east coincide with a bend in the vein and a shallowing of dip. This suggests that at some point to the east the grade may diminish somewhat but not necessarily limit mineable grade.
- The 600,000 ton inferred ore is defined by ore blocks on the west and two diamond drill holes. Ore in this block could be confirmed and developed as mining proceeds in the proven area.
- It is important to note that the diamond drill hole at the east margin of the inferred block cut 5.0' of 0.10 Au. Thus mineralization has not been limited to the east and exploration chances for developing an additional 500,000 to 1,000,000 tons east of the drill hole are rated excellent. The vertical extent of ore may be less due to the presence of an upper unfavorable vein forming unit.
- Tests conducted for USSMR show that 88% recovery can be obtained by flotation. Cyanide tests indicate 95% recovery should be obtained with a minus 200 mesh grind. Note that JRS used 88% as a recovery figure in a cyanide plant. 95% would add \$7/ton net mine.
- As SAA points out renegotiating the Red Top lease is imperative, since \$80-85 ore requires a 16% royalty and +\$100 ore 20%. At one time Sharon thought they could buy Red Top out for \$70-80,000 cash.

R. L. Brown
Page 2
December 15, 1983

I am attaching a longitudinal section showing the SAA reserve blocks and grades and a list of SAA's data for these blocks. You will note that most of the higher tonnage blocks are also higher grade and cluster at the east end. Obviously one can play with these blocks to arrive at a smaller tonnage with a higher grade.

As I have previously stated I believe we can operate this property at a good profit and recommend an outright purchase, including the Sharon ground at Tintic, Utah. Perhaps offer \$750,000 and be prepared to go as high as \$2 million.


W. L. Kurtz

WLK/cg

Attachments

cc: JDSell (w/o Attach)
JRStringham "
SAAanzalone "

December 22, 1983

R. L. Brown
New York Office

Sharon Steel

E. Peter Matthies, Vice President, has resigned from Sharon Steel and is new President and CEO of Storm King Mines (Colorado coal). Sharon has not yet appointed a new Vice President of Natural Resources and now just might be an excellent time to consummate an agreement on the Gold Road - Tintic and/or all of Sharon's mining properties.

A Jim Newcombe (303-866-7771) is handling the job until Posner appoints a new man.


W. L. Kurtz

WLK/cg

cc: JDS
JRS
SAA

January 10, 1984

R. L. Brown
New York OfficeGold Road Mine
Arizona

Concerning Mr. T. C. Osborne's January 6, 1984 memorandum on the Gold Road Mine I have the following comments:

1. Sharon has a current, valid lease on the Red Top claims.
2. The Fischer-Watt holdings, plus most of the other land between the Tom Reed and Gold Road, have been acquired by Royal American Petroleum, a private well-financed company. They have approached us to participate in the development of their holdings which include 1.5 million tons of tailings @0.27, a reported 200,000 tons @0.2 gold in the United Western Mine (Fischer-Watt-Hecla project), a number of 15,000-20,000 ton shoots in the Tom Reed Mine, plus additional potential outlined by Fischer-Watt. Cimetta Engineering has indicated that regrinding the tails is feasible (Don Crowell is presently critically reviewing this work) and has developed a plan for processing the tails at 1,000 tpd in conjunction with mining underground ores at 500 tpd.
3. I personally do not believe the mine needs to be opened up to be re-examined. I accept the ore block data of the United States Smelting Refinery and Mining Company which was the data at hand when the mine closed because of WW II.
4. The figures for equipping the mine and mill are reasonably good figures developed by Stringham and Crowell based on the St. Cloud Mine's mill and Redpath's rehabilitation estimate.
5. Stringham studied the ore blocks and included in Anzalone's reserve are only those thought to be easily accessible.
6. This "decent little vein" is probably less than 60% mined out. Ore does exist above the contact ("erosion surface") as shown on the long section. We have traced the vein at the surface for a 1,000 feet east of drill hole No. 1. Some of the veins in the district have a convex shape with the ends containing deeper ore. Thus at least a million tons of mineable ore seems a sure bet to me.
7. Predicting metal prices is difficult but with operating cost of \pm \$200 per ounce the downside risks seem quite limited.

8. As with so many small mines, good profitability rests with good mine management.

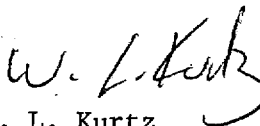
Perhaps an attractive way to develop the district would be for Asarco-Royal American Oil to joint venture the Gold Road-Tom Reed et al. I believe (but have not yet inquired) that Royal American would be interested in sharing the costs of developing a 1,500 tpd plant that would process 1,000 tpd of tails and 500 tpd of underground ore based on reserves of:

3 million tons tailings @ 0.025	Tom Reed, United Eastern, Gold Road
400,000 tons proven 0.24	Gold Road
800,000 tons indicated 0.24	Gold Road, United Western
600,000 tons possible 0.24	Gold Road, Tom Reed, Others

Without the Royal American ground I would still rate the Gold Road property the best of any Asarco exploration properties in the western U.S. because of these features:

1. Known reserve and additional potential;
2. Can be put into production easily, quickly, and with low capital investment;
3. Very low downside risk;
4. Cash flow of 3-5 million after tax.

Incidentally Tony Cimetta says that one of his senior men who resigned in December and is now representing some European interests has tied up the Gold Road properties. My phone call to Sharon suggests this is not true but indicates the pressures are mounting. I think we should make Sharon an initial proposal by mid-January.


W. L. Kurtz

WLK/cg

cc: JDSell
JRStringham
SAAnzalone

January 12, 1984

Mr. R. L. Brown
New York Office

Gold Road Mine
Arizona

Enclosed are:

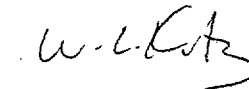
Report: Gold Road Mine by T. P. Billings, September 5, 1945.

This is essentially a report reviewing the history and ore reserve with a detailed description of the condition of existing stopes and ore blocks and suggests a plan for reopening the mine at 500 tpd.

Red Top Lease

Composite level plan and long section @ 1" = 200'

Other maps (level, stope, etc.) are in Bayard and we can drive over and procure these if you need them at this time.


W. L. Kurtz

WLK/cg

Attachments

cc: JDSell (w/o attach.)
JRStringham "
SAAanzalone "

GOLDROAD MINE

Goldroad, Arizona.

.HISTORY OF PRODUCTION.

.ORE RESERVES.

.PRESENT CONDITION OF PROPERTY.

.COSTS.

.EXPECTED ECONOMIC RESULTS.

.REHABILITATION.

-o-

by T. P. Billings

Sept 5/1945

TPB:LS

cc-orig & 2 to Mr. Mulock
-Mr. L. H. Duriez
-Mr. R. N. Hunt

*no circulation
made of report;
copies held up.
During next returned
31*

HISTORY OF PRODUCTION

The following tabulation shows production from the Goldroad Mine from 1903 to 1928:

<u>Operators</u>	<u>Years</u>	<u>Tonnage</u>	<u>Estimated Heads per Ton</u>	<u>Gross Production</u>	<u>Gross Earnings</u>
✓ Prior Owners:...	1903-1911	327,165	\$12.50	\$3,416,299	?
USSRAMCO:...	1911-1916	500,104	7.70	3,196,341	\$442,752.54
Lessees:....	1917-1922	352	?	3,203	?
USSRAMCO:...	1923-1924	60,965	12.50	683,977	46,335.85
Lessees:....	1925-1928	<u>2,847</u>	<u>?</u>	<u>37,169</u>	?
		<u>891,433</u>	<u>\$9.80</u>	<u>\$7,336,989</u>	

(next page for 1935-1942)

from 1935 to the shutdown in August 1942: -dry ton basis-

	<u>1935</u>		<u>1936</u>	
	<u>USERRAMCO</u>	<u>Lessee</u>	<u>USERRAMCO</u>	<u>Lessee</u>
Mine				
Prod'n:.....	7,567	1,581	7,756	3,255
Processed				
at Mill:.....				
Total				
Millheads:.....	\$89,301.25	\$15,852.38	\$87,423.40	\$52,772.99
Precipitates				
Produced:.....				
Net Amount				
Received:.....	\$54,853.81	4,997.13 (royalty)	\$52,145.13	\$11,569.28 (royalty)
Operating				
Margin:.....				
" per dry ton:..				
Other Revenue:..				
" per ton:....				
% Economic				
Return:.....*				
Mine Ore				
Prod'n Cost:....				
" per ton:..				
Milling Cost:....				
" per ton:..				

Note: - no production 1929 - 1934.

LOSSES:
1934 & 1935:.. \$ 78,638.42
1936:..... \$104,043.04

USSR:MCO 1937 - 1941	USCRA:MCO 8 months 1942 Jan. - Aug. inc.
657,583 (X) + 0.23 A-1/K ₁	98,153
690,943 (2)	98,980
\$4,941,354.77	\$786,578.95
\$4,511,512.55	\$728,167.81
\$4,400,470.25	\$712,676.03
\$1,032,799.80	\$202,531.00
\$1.495	\$2.446
\$30,874.53	\$4,802.99
\$0.044	\$0.049
89.05%	90.60%
\$2,474,932.49	\$383,146.91
\$3.764	\$3.904
\$897,912.95	\$124,641.32
\$1.299	\$1.259

• .246 Ag. .192 Au. average weighted assay per ton, includes tailing
skimmings and reclaimed water.

~~##~~ \$6.369 per ton.

Note: Total Operating Margin & Other Revenue
for 1937-1941, inc. = \$1,063,674.33
or \$1.539 per ton.

(X)
$$\begin{array}{r} 657,583 \\ 33,360 \end{array} \times \begin{array}{l} .24 \\ .03 \end{array} = \begin{array}{r} 157819.92 \\ 1000.00 \\ \hline 158,820.72 \div 690,943 = \\ \text{W.L.K} \\ 1/12/07 \\ \text{.23} \end{array}$$

contind:
History of Production:

The accompanying longitudinal section of the Goldroad Mine (Exhibit A) shows the portions of the vein mined during these several periods. The areas tinted yellow were stoped prior to the acquisition of the property by USSRMECO while those tinted blue were mined by the latter company during its first period of operations which terminated in 1916. The areas tinted brown were stoped during the period of operations 1923-1924; areas tinted red were stoped subsequent to 1928. The present ore reserve is shown with their block numbers.

Reference is made to report by R. T. Walker dated March 30, 1935 which covers the geology and proposed resumption of operations as of that time.

--

ORE RESERVE

As of September 1, 1942 an ore estimate was made of the Goldroad Mine as

follows:

<i>total all</i>						
<u>Balance</u> <u>Sept. 1, 1942</u>	<u>Balance Sept. 1, 1942</u> <u>Broken Ore</u> <u>in Stones</u>	<u>Ore</u> <u>In Place</u>	<u>Developed</u> <u>One Side</u>	<u>Developed</u> <u>Two Sides</u>	<u>Developed</u> <u>Three Sides</u>	<u>Probable</u> <u>Ore</u>
451,496	9,499	441,997	115,055	137,964	161,075	27,903
<u>Goldroad Red Top Mine</u>						
<u>4,642</u> <u>456,138</u>	<u>4,642</u> <u>446,639</u>	<u>4,642</u> <u>165,717</u>				

Weighted average assay:.... 0.325 oz. Au per ton.

In discussing the possibilities of ore extensions beyond the present ore reserves with Mr. Duriez and Mr. Elgin there is estimated a possible tonnage of 413,000 tons. Both Mr. Duriez and Mr. Elgin believe that the dilution from mining by the shrinkage stoppe method used at the Goldroad Mine and now used at our Bullfrog property at Bayard, will be less than that obtained in the operations from 1937 to August 31, 1942 - and have advanced fairly good reasons for a reduction of

-4-

dilution in the mining of the present ore reserves and future ore. However in checking the records from 1937-1942, inclusive, for dilution which are based on weighted average assays of car sampling and resultant millheads, the dilution showed a reduction of 24.619% in the grade of the ore.]

- -

PRESENT CONDITIONS AT THE PROPERTY

As of September 30, 1944 Mr. Walter Elgin, former ^{assistant} superintendent of the Goldroad Mine, wrote a letter covering the conditions of the property with a general plan for re-opening the mine. During the shutdown period of this property, since September 1942, there has been maintained a small crew of men (under the direction of Mr. Gardner former Mine Foreman at this property) who have kept in repair the main mine workings available above the water level. Their work, in large part in the early periods, was on the lower levels and the work has retreated upwards as the water level was allowed to rise. The only equipment remaining on the property used in the mining operation is an old, converted steam hoist at the No. 1 Shaft which hoist I understand required frequent repairs and was too small for a full skip load and generally regarded as unsatisfactory; also a small hoist at the No. 3 Shaft which should be replaced. The power substation was left intact consisting of the following transformers:

three 50-KVA 2300/400
three 333-KVA 44,000/2300

Also, there is a small 250 cubic foot compressor. The No. 1 Shaft, which is the main hoisting shaft of two different sized compartments, is in good condition and has its collar at surface at the head of the mill where the ore is dumped by skip direct into a bin. The present collar of the No. 3 Shaft is on the Lineroad Tunnel Level. It is proposed to retimber the No. 3 Shaft to surface and there construct a new hoist house and erect the compressors. Also, it is proposed to erect the

changehouse, timekeeper's office and lamphouse at the portal of the Lineroad Tunnel and hoist and lower the men in the mine through the No. 3 Shaft by equipping it with an adequate hoist and double deck cages to be operated in balance, and also use this shaft for lowering supplies and as a general service shaft. This shaft is in fairly good condition, however, the station at the Lineroad Tunnel should be enlarged and other repairs made in this area. Already in the No. 3 Shaft there is a 5" air column from the present collar at the Lineroad Tunnel down to the 700 Level. The mine tracks on the different levels in the mine were left in place while the piping was taken out. However, on the 700 Level which will continue to be the main haulage level, it is proposed to replace the 20# rail with 30# rail. During the last operation of this mine the wage rate was low as compared with the present or immediate future, and labor was quite efficient, so that the results obtained were exceptionally good although working with low air pressure, poor ventilation and very little mechanization. But with the present wage scale and inefficiency of labor it now appears advisable to provide more mechanization and improve working conditions in the mine, by better ventilation, sufficient ore passes and sufficient compressor capacity, which we believe in large part will offset the present high cost of labor and its inefficiency. The additional mechanization in general consists of high power drills on account of the hard vein rock; more mechanized tramming; the use of air operated mucking machines, and other minor changes toward reducing labor. When it is decided to re-open the property the first things to do are make provision for a larger hoist at the No. 1 Shaft; erect a compressor or two, depending on the size but preferably two; and install pipe lines on the levels above the water level so that stope preparation and development can proceed while the mill, shops, etc. are being erected. Mr. Elgin has outlined certain development work that could be got underway (during the construction period) on the Lineroad Tunnel, 100 Level, 350 Level and 500 Level, and have about 50,000 tons of broken ore reserve at the time the mill is started. Also, simultaneous with the development work on these

New reported
at 50 level

levels, the unwatering of the mine could be carried on.

As of September 1, 1945 the water had raised in the mine to a point about 15' below the 600 Level and there was estimated to have accumulated as of that date 47,304,000 gallons of water. There is already installed in the No. 1 Shaft a 5" water column and at our Lark property in Utah there are in storage, pumps with auxiliary equipment which should provide for pumping out the mine at the rate of 500 gpm. [The present inflow is estimated at 30 gpm,] so pumping at the rate of 500 gpm there would be a net gain of 470 gpm - and the above estimated accumulation would require 70 days to pump out to a point 30' below the 600 Level. The pumping out of the accumulation for each month after September 1, 1945 will require an additional two days. Following is an estimated cost of the pumping expense for 70 days:

Labor, etc:.....	\$6415.	
Equipment and Supplies:...	992.	
Freight:...	131.	
Power:...	3162.	
Contingencies:1. 10% ..	1070.	<u>\$11,770.</u>

Pumps with auxiliaries to be borrowed from Lark where they are now in storage. For each month accumulation after September 1, 1945, add \$301.00 to the above total cost.

COSTSdry ton basis.

Following is a tabulation of the different items entering into the total
estimated cost per ton of ore:

Production:.....	150,000 tons per year.	180,000 tons per year.
	<u>Per Dry Ton</u>	<u>Per Dry Ton</u>
Mine Development:....	\$0.50	\$0.50
Broken Ore Extracted:...	1.75	1.75
Pulling Stopes:...	.15	.15
Tramming:...	.30	.26
Hoisting:...	.25	.22
Handling Ore & Waste At Surface:...	.005	.005
Camp Water System:...	.017	.016
Mine Drainage:...	.032	.03
Maintenance Old Workings:...	.12	.10
Miscellaneous Underground Expense:...	.285	.285
Shop Overhead Expense:...	.033	.03
Assaying:...	.058	.058
Engineering:...	.07	.07
Miscellaneous Surface Expense:...	.10	.085
Ore Sorting:...	<u>.03</u>	<u>.03</u>
Total Cost at Mine:...	\$3.70	\$3.589
General Office and Overhead:...	<u>.53</u>	<u>.50</u>
Total Cost - Goldroad Mine Ore:.	\$4.23	\$4.089
Total Milling Cost:...	<u>1.40</u>	<u>1.273</u>
Total Mining and Milling Cost:..	<u>\$5.63</u>	<u>\$5.362</u>

- -

Mine Development:

The year 1941 was a full year of operation at the Goldroad Mine and I consider it fairly representative of their experiences and can be used as a basis for estimating costs, along with other factors. In 1941 Development Expense shows the following:

71.91% labor
 22.91 supplies
 5.18 other expense

The present machiner's rate at Bayard which probably will be applied in the event of resuming operations in the near future at Goldroad, is \$7.87 per shift including

the shift differential - while for 1941 the average machiner's rate was \$5.38 (an increase of \$2.49 or 46.1% greater). 71.91% of the Development cost being labor would amount to an increase in cost of 33%. In 1941 the Development write-off charge was 40¢ per ton, however, the actual cost amounted to 30¢ per ton. In view of the proposed additional development work, I estimate 50¢ for Mine Development.

Broken Ore Extracted:

In 1941 the cost of Stope Preparation and Stoping was as follows:

72.50% labor
20.3% supplies
7.19% other

Then 72.5% multiplied by the increase in the wage rate of 46.1% would amount to an apparent increase in cost of 33.5% on account of labor. The cost for this item in 1941 was \$1.534. I have estimated a cost of \$1.75 per ton under the present wage rate. I have discussed this estimate with both Mr. Elgin and Mr. Duriez and, in view of the proposed improvements in facilities and their experience in the Bullfrog Mine using a similar mining method (shrinkage stoping), they feel as I do that this estimated cost is ample.

Pulling Stopes:

The 1941 cost for this item was 12.9¢ per ton. I have estimated a cost of 15¢ per ton which is nearly 1/3d greater and, in view of the experience at the Bullfrog Mine and proposed new facilities, should be sufficient.

Tramming:

In 1941 the tramming cost was 32¢ per ton. I arbitrarily estimate this tramming cost at 30¢ per ton which is based on better facilities in the way of ore passes and mechanized haulage.

Hoisting:

In 1941 the cost of this item was 23.7¢ per ton. My estimated cost for hoisting is 25¢ per ton which, with a larger proposed hoisting facility and servicing through the No. 3 Shaft, should be ample.

The next 4 items: Handling Ore and Waste at Surface; Camp Water System; Mine Drainage; Maintenance Old Workings; I have increased about a 1/3d over that of 1941, based on the increased labor cost.

Miscellaneous
Underground Expenses:

In 1941 this item was 18.4¢ per ton. My estimated cost is 28.5¢ per ton which is about a 50% increase. The additional cost provided for in my estimate is based on the additional cost that will be required for ventilation besides the additional labor cost.

Shop Overhead Expense and Assaying I have increased each approximately 1/3d over that of 1941.

Engineering:

The cost per ton in 1941 was 2.6¢. I have arbitrarily increased this item to 7¢ per ton which is better than 100% and I base it on more engineering that will be required at this property.

Miscellaneous
Surface Expense:

In 1941 the cost was 7.3¢ per ton. In my estimate I have increased this to 10¢ per ton.

Ore Sorting

In 1941 the cost was 7.3¢ per ton. My estimate shows this 3¢ per ton. In discussing this Ore Sorting with Mr. Brough and Mr. Duriez they do not feel that it is worthwhile to use 4 men on this job and indicated that one man would be all that is required.

My estimate for Total Cost at Mine is \$3.70 per ton as compared with a total cost in 1941 of \$3.183 per ton, on a mine production of 510 tons per day working an average of 25 days per month, while an increase of 18% to 600 tons per day the cost is estimated at \$3.59 per ton. On Page 7 are shown these itemized costs.

The General Office and Overhead Cost in 1941 was 42.7¢ per ton. In my estimate this is shown as 53¢ per ton. The items that make up this General Office and Overhead are: office expense, taxes, compensation insurance and other insurance. To arrive at this 53¢ per ton I used 25¢ per ton for office expense - practically doubling that used in 1942 which was 12.2¢ per ton - I base this increase on the fact that a great deal more will be required of the office force at greater pay than existed in 1941. Under taxes the cost in 1941 was 28.4¢ per ton and in making up the 53¢ overhead estimated cost, I used 30¢ per ton for this item which I feel should be adequate. Under compensation insurance and other insurance in 1941 the cost was 17.2¢ per ton while I have used an estimated cost of 15¢ per ton - a reduction. This reduction is based on an expected reduction in our compensation insurance on account of being self-insurers. The above cost is based on a mine production of 510 tons per day, while on 600 tons per day this cost is estimated at 50¢ per ton.

My estimated Total Cost - Goldroad Mine Ore is \$4.23 per ton as compared with \$3.61 per ton in 1941, based on 510 tons per day and \$4.09 per ton based on 600 tons per day.

The Milling Cost in 1941 was \$1.164 per ton. Mr. Brough has kindly submitted an estimate of \$1.40 per ton in the event operations are resumed at the Goldroad property - an increase of 23.6¢ per ton. Mr. Brough bases this increase due to the increase in labor and material costs. The following is quoted from Mr. Brough's letter of May 30, 1945 addressed to Mr. Duriez:

*Goldroad Mill Costs On the Basis of Bayard
Mill Labor and Material Costs

Direct charges:....	\$1.093
Indirect charges:..	.274
Plus 2.4% for inefficient labor:..	.033
Total Costs:...	<u>\$1.400</u>

The first two figures are arrived at by adding 30% and 20% to the 1941 labor and material costs, respectively. Nothing was added to power because Mr. Hart informed me, last December, that the old power contract was still in force. To these is added a charge for inefficiency of labor, which should apply for the first year, at least.

contnd:

Mr. Brough's letter:

Costs for 1941 are as follows:

Direct charges:...	\$0.931
Indirect charges:...	<u>.233</u>
Total Costs:...	<u>\$1.164</u>

The above estimate is made on the supposition that the mill would be operated at or near capacity, after suggested alterations to bins and crushing plant have been made."

The above cost of \$1.40 per ton (Pg.10) is based on milling an average of about 425 tons per day, 30 days a month. Mr. Brough submitted an estimated cost of \$1.27 per ton on milling an average of 500 tons per day.

- -

EXPECTED ECONOMIC RESULTS

.dry ton basis.

The present Ore Reserve of 456,138 tons, allowing for a 24.62% reduction in grade from dilution of barren material totals 605,113 tons and Possible Ore from extensions is estimated at 413,000 tons and allowing for a 24.62% reduction in grade from dilution of barren material totals 547,836 tons. The weighted average assay for the 456,138 tons is .325 oz. gold.

During the 5-year period 1937-1941, inclusive, the ratio of silver to gold equals 1.181, therefore, $1.181 \times .325 = .3838$ oz. silver, which I have assumed.

Allowing for a reduction of 24.62% in grade the average assay for the 605,113 tons reserve and 547,836 tons possible is calculated as follows:

.245 oz. gold.
.289 oz. silver.

With the present domestic price for silver, the Goldroad operation received 71.111¢ per ounce of silver and \$34.9125 per ounce of gold. During the 5-year period 1937-1941, inclusive, the metal value of the Reclaimed Water, Tailings and Skimmings, amounted to 36.6¢ per ton of Mine Production, while for the 2-year period 1940-1941, inclusive, (the metal value of the Reclaimed Water, Tailings and Skimmings, amounted to 18¢ per ton of Mine Production,) and for the 8 months in 1942 the metal value of these items was 19.3¢ per ton. Therefore, I have credited the Gross Value per Ton with 18¢ for these items.

\$3.5536 Gross Gold Value per Ton of Ore.
 0.2055 Gross Silver Value per Ton of Ore.
 .1800 Average Gross Value Reclaimed Water,
 Tailings and Skimmings.
\$8.9391 Gross Value per Ton of Ore.

The Estimated Economic Return for the 5-year period 1937-1941, inclusive, of the Gross Value per Ton averaged 89.05% which multiplied by \$8.9391 (Gross Value per Ton) gives \$7.96 Net Value of Precipitates per ton of ore. After deducting the total Production Cost (Mine and Mill) of \$5.63 per ton leaves \$2.33 as Operating Margin per ton.

Also during the 5-year period 1937-1941, inclusive, there was Other Income amounting to an average of 4¢ per ton which added to the Operating Margin of \$2.33 totals \$2.37 expected Total Income per ton of ore.

The Diluted Ore Reserve considered assured is 605,113 which multiplied by \$2.37 amounts to \$1,434,117.81 - and, at the rate of 150,000 tons mined per year would be mined out in 4.03 years.

The Diluted Possible Ore Reserves of 547,886 tons multiplied by \$2.37 equals \$1,298,489.82 - which, at the rate of 150,000 tons mined per year means an additional life of 3.65 years.

Estimated TOTAL INCOME expected is \$2,732,607.63. 1,152,999 extracted at the rate of 150,000 tons per year would give a period of 7.68 years.

\$2,401,793³⁵
 1,013,415
 6.76 years.

if diluted material is assumed to be in place see notes for calculations using .025% for overhead.

REHABILITATION OF MINE AND MILL

1: Unwatering Goldroad Mine:

September 1, 1945 - an estimated 47,304,000 gallons of water accumulated.

Pumping at the rate of 500 gallons per minute with inflow of 30 gallons per minute will require 70 days. Accumulation of water for each month after September 1, 1945 will require an additional two days to pump out. Estimated cost of pumping water as of September 1, 1945:

Labor, etc:.....	\$6415.
Equipment and supplies:...	992.
Freight:....	131.
Power:.....	3162.
Contingencies - 10% -:....	<u>1070</u> <u>\$11,770.</u>

Pumps with auxiliaries to be borrowed from Lark now in storage. For each month accumulation after September 1st add \$301 to the above total cost.

2: Ventilation:

The Goldroad Mine is a hot one which in most part is due to high rock temperature, although a shallow mine and is not properly ventilated. I am advised that the No. 1 Shaft is always downcast and around the No. 5 Shaft and stopes to surface in this vicinity it is always upcast; while in the Nos: 2 and 3 Shafts the air current is downcast in summer and upcast in winter. Also, with stopes connected through from different levels and with surface results in short circuiting and a large portion of the incoming air not being used to an advantage in the places where work is in progress. We all believe it is advisable to provide for improvement in the ventilation. This can be accomplished as follows: Coordinate the development easterly on the different levels so as to provide connections of the levels before stoping operations are started; an air raise connection of the east end of the Lineroad Tunnel with surface; a fan of from 40,000 to 60,000 cubic feet per minute at 3 to 4 1/2 inch water gage as an

exhaust fan; doors and bulkheads properly placed for the distribution of the air; and small auxiliary blowers in the headings in progress of driving. A survey should be made of the surface in the vicinity of the east end of the mine in the Red Top area to determine the length of a raise required. This proposed air raise could be driven on the vein and may be justified as an exploration project.

To accomplish the above would require an expenditure of about \$20,000 or at the rate of 2 cents per ton charged to a possible 1,000,000 tons. The power expense of operating the fan is estimated around per day or cents per ton. For comparison purposes the cost of ventilation at the Lark and U.S. mines are 2 and 6 cents, respectively; while at the Goldroad operation from January 1937 to September 1942 this cost averaged about 1 cent per ton, however, in the estimated cost of production of the ore reserve 3 cents per ton is used for this item.

3: Underground Haulage:

The only mechanical haulage in the mine during the former operation was on the 700 Level (which was the main haulage level) while on the other levels it was all hand tramping.

700 Level: On this level there was installed an electric locomotive with generator set. This equipment has been transferred to Bayard and it is planned to use it on the 670 Level of the Bullfrog No. 2 Shaft to serve the Bullfrog and tributary mines. This equipment was very satisfactory at the Goldroad Mine and the conditions existing on the 700 Level will permit this type of haulage.

The present track on the 700 Level is 20# rail, 24 inch gage, and this has been under water and will no doubt need considerable overhauling and inasmuch as this is the main haulage level where from 500 to 600 tons of ore and waste per day will be transported, it appears advisable to replace this 20# rail with heavier rail. The 20# rail can be salvaged for the upper levels. The estimated cost of replacing the 20# rail with 30# rail is \$1368.00 per 1000 feet and for the 6500 feet required is estimated at \$8892.00 after crediting \$1083.00 for salvaging 43 1/3 tons of 20#

First 9 months of 1942 a Tramway locomotive was operated on the 100 level. Handling about 125 tons per day 25 surface line

rail at \$25 per ton leaves a net expense of \$7809.00. On the 700 Level an 8-ton trolley locomotive, 275 volts, 4/0 wire, using 2-ton cars on 24 inch gage, 30# rail, is recommended. A study of the haulage requirements on the 700 Level for a 500-ton per day operation shows advantages in the operation of a trolley locomotive over a large battery locomotive; also, the first cost of the trolley setup is appreciable less and the present level is large enough to permit its use safely. The ore will be hauled from a distance of about 6600 feet from the No. 1 Shaft or an average

distance of 4000 feet to the shaft. In the event this rail is changed we should consider going to a 40# rail at an estimated net expense of \$9122 or an increase of \$1332 over the 30# rail.

Tons of ore hauled per day:...	500
Tons of ore hauled per shift:...	250
Tons of ore per car:...	2
Cars per shift:...	125
Weight of locomotive:...	8-ton
Cars pulled by 8-ton locomotive:...	21
Number of trains required per shift:...	6
Average length of haul (round-trip):...	2,000'
Time allowed per round trip:...	70 minutes.

Loading Time:...	25 minutes.
Unloading Time:...	25 "
Total:...	50 "
Traveling time allowed:..	20 "
Total:...	70 "

$$\text{Speed required} = \frac{8000}{20} = 400 \text{ f.p.m.} = 4.5 \text{ miles per hour.}$$

The drawbar pulls are as follows:

Empty train up 3/4% grade:....	1305#
Loaded train down 3/4% grade:..	1070#
Loaded train up 3/4% grade:....	3200#

$$\text{Locomotive weight required} = 5 \times 3200\# = 16,000\#.$$

Comparing trolley and battery locomotive haulage the following points are of

notes:

- (1) One battery will not last out the shift. It requires a battery of 41 KWH per shift to operate the six trips. The 48-cell, 19 plat MVM or KMD batteries have a capacity of 29 KWH.
- (2) The maximum speed of a loaded battery locomotive is 3 m.p.h. No require 4.5%.

*notes are much clearer
in places are as much as
2 to 2.5%*

- (3) The cost of the trolley locomotive, trolley wire in place and the motor-generator set is about \$1000 less than the cost of the battery locomotive, two batteries and a motor-generator set.

From the above there is no alternative but to use a trolley locomotive for this haulage.

It is planned to use a Mancha Little Trammer on each of the 100, 350 and 500 Levels, using the present 20# rail at 18 inch gage and 1-ton cars.

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A: Hoisting: On the No. 1 Shaft, the main hoisting shaft at the head of the mill and through which it is planned to hoist all the production, there is erected an old converted air hoist. According to the men connected with the former operation at this property, this hoist was not adequate for the requirements and constantly developed trouble and breakdowns which was a constant worry to the operators. It is advisable to consider the purchase of a new hoist for this No. 1 Shaft and erect it back of the present hoist. The present hoist can be used for pumping out the mine and certain preliminary preparation underground until such time the new hoist can be erected. For a safe and economic operation a new hoist for this shaft should have about the following specifications:

The hoist should have double drums not less than 30" in diameter and about 24" face, both drums to have separate brakes and clutches. The clutch and brakes should have asbestos brake blocks. Brakes should preferably be of the post type. Flanges should be not less than 6". The hoist should be capable of hoisting an out of balance load of 6600# at a speed of 500 feet per minute requiring not less than a 100 HP motor. The motor should be for 440 volts complete with heavy duty grids and control.

ropes and cables could be removed from shaft and shaft made safe for heavy traffic

The headframe with dumping mechanism now erected on surface at the collar of the No. 1 Shaft is a remodeled contraption of light timber, a real fire hazard and according to Mr. Cannon will require about \$1500.00 in repairs. So, if a larger hoist is erected here a new headframe should be considered.

It is planned to use the No. 3 Shaft of the Goldroad Mine primarily as a service shaft, that is, for lowering and hoisting men, materials and supplies. At present there is erected a hoist on the Lineroad Tunnel for operation through this shaft. This No. 3 Shaft is quite central to the ore reserve and probably future operations, so considerable saving of time can be obtained by the men in getting to their places of work and in the delivery of supplies and equipment through this shaft. The shaft is a 2-compartment affair and in fairly good condition, and the present operation here provides for one hoisting compartment and a counterweight. The station on the Lineroad Tunnel at the shaft is small and in bad condition, so it is planned to repair this shaft to surface and there erect a hoist and compressor-house, so that the hoistman can also watch the compressors. The following I believe covers the specifications at this shaft for hoisting:

The hoist at the No. 3 Shaft at present is a Vulcan Denver #23 and it is assumed that it will be used as a material and men handling hoist. It has two drums 28" in diameter with 21" faces. Each drum is separately clutched and braked. The rope speed is 350 feet per minute with a 30 HP motor. One compartment of the shaft will be used for hoisting and the second compartment for a counterweight. The cage will be a double compartment cage with a weight of not more than 3000# and will carry a net load of 2000#. This will be equivalent to 12 men, eight 10"x10"x8' timbers, etc. The counterweight should be altered to handle 40 100# lead pigs or a gross weight of about 4200#. This would cause the hoist motor to lift an unbalanced load of 1200# when the cage is being lowered and 1300# when it is being raised. This will reduce the peaks and heating and be well within the capacity of the 30 HP motor.

The present hoist proposed to be installed on surface at the collar of the No. 3 Shaft will require the construction of a small wooden headframe at an estimated cost of \$2500.00.

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5: Compressed Air and Piping

At present there is a small 250 cubic foot compressor on the property and for an efficient operation of 500 tons per day with necessary development work there should be provided at least 2500 cubic feet additional, and preferably in two identical units of the latest type each operated by synchronous motors at 2300 volts, and designed for an elevation of about 3000 feet above sea level. When considering the purchase of second hand compressors, particular care should be taken in determining the design especially for extreme elevations in order that we are sure the horsepower provided will be adequate for an elevation of about 3000 feet above sea level.

The pipe has all been removed from underground except the main 5" air column down the No. 3 Shaft. We have found it to be good practice not to use, on the main levels, airlines under 2 inch and water lines under 1 inch. Also, from our experience with Victaulic couplings, we recommend that these be used on all pipe 2 inches or greater. Following is a comparison of the cost of Victaulic couplings with the thread type but we feel this additional first cost will be compensated by saving of labor and provide for better efficiency:

Comparative Prices of Various Pipe Couplings

<u>Size</u>	<u>Standard Sleeve</u>	<u>Ground Joint Union</u>	<u>Flanged Union</u>	<u>Victaulic</u>	<u>Average for 9 Sleeve Couplings plus 1 Standard Union</u>
4"	\$1.47	\$5.80	\$11.53	\$1.85	\$1.88
3"	.94	2.90	7.38	1.30	1.14
2"	.29	1.21	3.07	.90	.38
1"	.12	.42	-	.75	.15

Savings in labor, salvage values, reduced maintenance, lesser loss through leakage, and convenience of having every joint a union, clearly justify use of Victaulic couplings on 4" and 3" sizes, probably on 2", and perhaps even 1".

6: Electric Power: The electric power is furnished at the mine by the Citizens Utilities Company at 44,000 volts. The power substation now on the property consists of the mill transformer station with the following transformers:

one 10KVA, 2300/240/120 volts,
three 50KVA, 2300/440 volts,
three 33KVA, 44,000/2300 volts,

and the mine transformer station with the following transformers:

one 10KVA, 2300/240/120 volts,
three 100KVA, 2300/460/240.

The above are new as originally erected, in good condition, owned by USSRAMCO and located near the No. 1 Shaft and mill. Also, from one of the above transformer stations there is constructed a powerline to the No. 3 Shaft site. During the former operation, the above facilities were adequate and the power cost was 1.1 cents per KWH with the demand based upon the three highest average fifteen minute peaks during the month. However, with the planned increase in mechanization and additional hoisting and compressor capacity the transformer setup and powerline to the No. 3 Shaft should be checked.

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7: Re-Opening and Preliminary Stone Preparation: It is estimated that about three months will be required to erect equipment in the mill ready for a milling operation after delivery of the equipment. And during this period the mine could be pumped out and made ready for production of about 500 tons per day. The following is an estimate of rehabilitating the mine, carrying on certain development, providing improved facilities such as ore passes and installing larger rail on the 700 Level and breaking about 20,000 tons of additional ore in the stopes ready for transporting and hoisting to the mill:

Pumping expense:.... \$12,000.

Larger rail on 700 Level:... 9,000.

*Stope Preparation, Broken Ore Reserves,
etc. Crew 50 men - 3750 manshifts -
3 months -!... 60,000.

*Includes supplies, other, repairs, etc.
General Office Overhead, but no hoisting of ore or
tramping on the 700 main haulage level.

The Broken Ore Reserves in the Goldroad Mine increased from 2000 tons in the early part of 1937 when the mill was started, to a maximum of 64,744 tons in March 1941. This Broken Ore Reserve was gradually reduced to 45,122 tons as of June 1, 1942 when breaking ore was discontinued preparatory to a shutdown, and as of August 31, 1942 there is estimated, as per cost statement, 9,499 tons. Our first plan was to obtain about 50,000 tons in reserve and maintain it at this amount in order to provide a steady flow to the mill and result in better control of grade. However, we believe that the above \$60,000 as an ore production expense will provide for stope preparation, and the breaking of an average 250 tons of ore per day so that in a period of three months (the time estimated to get the mill in operation) there will be approximately 30,000 tons of Broken Ore available in the mine, and should also provide about 450 feet of drifting and raising.

The above general program is of course subject to change, depending upon the progress made in rehabilitation of the mill and in the availability of working places in the mine along with a supply of good miners.

During the shutdown of the Goldroad Mine there has been maintained on the property a repair crew and watchman at an expense of around \$2500 per month so we would expect the mine in general good condition.

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As of September 1944 Mr. W. F. Elgin in a memorandum to me outlined a plan of work to re-open the Goldroad Mine as follows:

100 Level, Block 179, 1700E and W Stopes: The east part of the stope has been emptied and the west part is just ready for mining. The west stope will break into the 1340 stope as it goes up, it is in the footwall, and there is some doubt as to the amount of ore that we will be able to get. The east stope can be re-entered in two ways, first, by sinking a short winze to connect with the east raise of the 1340 Stope and then underhand mining the block. Second, and probably the best method would be to re-enter the 1800 Stope and Sublevel just below the Lineroad and drift to the 1340 East Raise and then underhand.

100 Level, Blocks 10, 11 and 183, 2870 Stope: We have never gotten to the back of the 2870 Stope except in the extreme east 45 feet of it. Old samples show a width of 4.5 feet and 0.20 oz. gold. To reach the back we can slab off both the hanging and footwall, which show lowgrade ore. Then by mining from the east to the west, the

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Line Road	5203-5405	201'	3.62'	0.465 an
350 Level	4975-5109	135'	3.81'	0.364
500 Level	4979-5270	292'	8.79'	0.36

700 Level	4899-4975	76'	4.06'	0.212
	4975-5035	62'	4.4	0.174
	5035-5070	35'	4.3	0.116

15' to 11' for overall width of 30'

±4900'E to 4930E 50'-4.8 0.332

545 N

Easting on 100 Level 5344 E

back of the remaining part of the stope can be reached. The east end is now about half filled and stoping can be started at once. It would be some time before any large tonnage could be drawn. Any ore drawn could be dropped through 3590 ore pass.

100 Level, Block 29, 3600 HW Stope: Has been pulled dry but can easily be re-entered as it is very low and not very wide. The east end is still in the preparation stage. This stope should average about 0.20 ozs. and is well ventilated. The ore would be dropped to the 700 Level through 3590 ore pass.

100 Level, Block 34, 4560 Stope: Is a small stope of narrow but good grade ore and is ready for chute construction. The tonnage from this stope is uncertain as we do not know too much about the old stope in the center of the block.

100 Level, Blocks 157 and 172, 4820 and 5040 Stopes: The vein will average 4.49 feet wide and 0.53 oz. The west 4820 Stope has been pulled and the walls were caving. The east manway should be in place and will keep the caved waste out of the newly mined ore. We can start mining 100 feet that is prepared and begin knocking down the back in the 5040 Stope at once. This ore could be dropped down the 4975 Raise or taken back and dumped in the 3590 ore pass. This is the hardest rock that has ever been found in the mine and it will require a good heavy stoper and lots of air to drill it.

350 Level, Block 160, 1975 Raise: This raise should be finished as soon as possible, so that the footwall workings can be ventilated and so that sublevels can be driven from the raise to explore the extent of the ore just above the 350 Level. The ore only comes through to the level at the raise but is 255 feet long on the 100 Level. There is an ore pass at 2040 that goes through to the 700 Level. This ore pass is on the H.W. Vein.

350 Level, 2890 Stope: Blocks 41, 42 and probably Block 147 can be mined from here. The 2890 Stope is full of broken ore, 9009 tons; the walls are bad and the vein had very spotty values. The back of the stope appeared to be improving. The manways are still open but two of them are pretty badly crushed. Block 147 is in the hanging wall of the old 2700 Stope and we had started 2890 Raise off the end of the 2890 Stope with the intention of connecting into the upper part of the 2700 Stope. We then intended to underhand ore in Block 147. It is also possible to enter the 2700 Stope by making a very short winze from the 100 Level. The stope is empty. Ore from 2890 Stope would be dropped to the 700 Level through the 2700 ore pass.

350 Level, Blocks 44 and 174, 3320 and 3420 Stopes: The 3320 Stope has been emptied and the best way to enter it would be from the 3420 Stope which is in the preparation stage. The 3420 Stope is on the footwall vein and the 3320 Stope is on the junction of three strands. 3320 Stope, or Block 44, is doubtful ore as a large fault is visible in the west end and may cut the vein. Block 174 is in the footwall of the old 3500 Stope and contains spotty values, being largely calcite. The old 3500 Stope is evidently on the center strand and old stope reports show some very high grade ore in the 3590 Raise, this is not estimated. (see note on photostatic copy of 50 scale map). There is also another vein in the hanging wall and it is exposed in the 3590 hanging wall drift, this vein will be discussed under Development. A very large tonnage is possible from this vicinity and it will be cheap mining. The ore will be better than the ore exposed in the 3420 Stope as the stope preparation work was done well in the footwall.

350 Level, Blocks 46 and 175: Block 46 is on the footwall and can be started at once. Block 175 is hanging wall ore and some drifting will have to be done before all of it can be prepared for stoping. However, the hanging vein appears to be on a much flatter dip than the footwall and most of it might be mined from the footwall

drift. There should be a large tonnage of good grade ore in these two blocks and there will be no big tramming problem.

Block 47 and 176 are also ready for mining.

500 Level, Block 56: We had planned on making this the next new stope on this level. Drift samples do not indicate much, samples average 5.58 feet at 0.162 oz. However, we know that there are two veins in the stopes on either side of the block. The footwall vein was a very good one on the 350 Level. We believe that we can mine blocks 56 and 148 and other ore from here.

500 Level, Block 57: could be started at the same time and should give a fair tonnage of good ore. This block assays better than it looks as there is a high grade stringer in the vein.

500 Level, Block 164, 4600 Stope: is partially prepared and ready for mining. The vein is 4.89 feet wide and assays 0.384 oz. There will probably be a lean zone in this stope, similar to the lean zone in the 4725 Stope. We went through the low grade in the 4725 Stope and the ore came back in again. The track from the 4600 Stope to the 4975 Raise is very crooked and steep, we were driving an inclined raise from the west end of the 700 Level 4900 Stope and were intending to drop the ore from the 4600 and 4725 Stopes through this raise.

500 Level, Block 171, 4725 Stope: This stope is full of broken ore and is ready for immediate mining. This is good ore and the vein is 7.28 feet wide and assays 0.244 oz. gold, in the stope back. The block averages 5.6' wide and 0.329 oz. gold. All of these blocks are averaged on the conservative side and I am sure that the ore will average practically what the average of the block shows, even when the stope is diluted by overbreak.

500 Level, Block 178, 4975 Stope: Mining has just been started and 9 chute holes are in. It would probably be better to take down the entire back as this is very good ore, averaging 8.79 feet in width and 0.36 oz. This will be one of the best stopes that we have had.

700 Level, Block 130, 4100 Stope: The stope is partially prepared and should be continued as fast as possible. The west manway would be driven as a raise and the west compartment would be used as an ore pass until the east side of the stope is finished. The ore on the level is not too good but was improving as we raised above the level.

700 Level, Block 150, 4900 Stope: We have gone far enough with the 4900 Stope to see that the stope should be extended to the west. The 4900 hanging wall drift ends just west of the last chute and should be extended as soon as possible. While on the subject of this drift, would recommend that we start at the 4530 Crosscut and follow the vein or fault to the east until a connection is made with the 4900 hanging wall drift, about 350 feet of drifting will be required. We show 2050 tons of broken ore in the 4900 Stope. The east two chutes were pulled dry and the angle of repose leaves a triangle of broken ore that extends to the west manway. The ore is very wide at the west manway and it will probably be necessary to abandon this and start another more to the west. The 4850 Raise lacks about 70 feet of connecting to the 500 Level, this should be finished for ventilation and for use as an ore pass for the stopes above.

Also in his memorandum, Mr. Elgin listed certain development work as follows:

L.R.T. Red Top: The Red Top drift is now in good condition and can be entered at once. The drift is far advanced of any other drift. There are now two raises, the 6040 and 6120. They are connected at a point 150 feet above the Lineroad Tunnel. They have shown a definite rake in the ore. The drift that connects them is still in ore and should be extended to the east until it has passed out of the ore, then another raise driven. If the ore continues to rake as it did in the first 150 feet, then sublevel off again and repeat the raising as long as possible. The vein on the surface shows no value and when the values in the raises become too low, raise vertically to the surface. The mountain drops off fast to the north and two to three hundred feet of raising may be saved by leaving the vein. This would solve the ventilation problem on the upper levels for years.

Lineroad Tunnel, 5545 E: This is near the end of the ore on company ground and 200 feet ahead of the face of the 100 level. The last ventilation from the 100 Level comes up through the old 4800 Winze. A winze could be started at approximately 5545E and when the 100 Level is advanced, a raise could be started to connect with this winze. It will require approximately 250 feet of raising and sinking to make this connection.

100 Level: At 2550E we have started a raise with an incline branch east and west. So far no commercial ore has been found but both branches are in quartz and average from 0.08 to 0.10 oz. The raises are above the fault that cuts the ore in the stopes below the 100 Level. Our experience in the last few years has shown that barren zones in quartz are not large and we have high hopes for this raise. There is approximately 1000 feet between the 1340 Stope and the 2550 Raise that is unexplored except for the old 100 Level drift.

100 Level, 4140E: A raise should be driven to prospect the ground between our 3800 Stope and the old 4200 Raise. The ore undoubtedly makes in the footwall above the 100 Level. This is proven by the records of the old 4000 Raise and the 4200 Raise, in which high grade ore was found 9 to 12 feet in the footwall at an elevation of 120 feet above the level.

100 Level, 4510 to 4555E: There is a small vein of high grade ore showing here. It also shows in the sublevel off of the 4560 Raise 120 feet above the level. The old 4560 Raise is on the hanging wall vein.

4560 Sublevel, off 4560 Raise: This sublevel should be driven to the east and connected with the old sublevel from the 4800 Stope. The 4800 Drift shows a wide, hard glassy quartz and is probably in the hanging wall from the 4560 Sublevel Drift, both veins should be explored as any ore found in this vicinity is likely to be good grade. The sublevel should also be driven to the west to explore Block 33.

350 Level, 1975 Raise: This raise has been driven to a height of 110 feet in a vein that averaged 3.35 feet in width and with an assay value of 0.205 oz. It will require 140 feet of raising to connect to the 100 Level. This raise will open up block No. 160 which contains 6850 tons of ore that averages 0.589 oz. gold. An intermediate drift will probably have to be made as the ore does not extend to the 350 Level except at the base of the raise.

350 Level, 1975 Footwall Drift: Should be extended 200 feet and a 40 foot raise driven at 1600E. This would connect with the old 300 Level and give access to that level east of the very bad ground in the No. 2 Shaft district. At 1500E on the 300

Level a raise could be driven on the footwall vein (it shows some value) until solid, unbroken ground is encountered. Old sections show that the Sharp Stope was cut by a fault at about 50 feet above the 300 Level. When in solid ground and above the Sharp Stope, crosscut to the hanging and look for the continuation of the Sharp Orebody. Considerable prospecting may be required in this area. We have ore down the No. 2 Shaft for about 80 feet below the 100 Level and it is doubtful if we will ever be able to get to the shaft on the 300 Level again. Therefore, the ore around the shaft and under the 1200 and 1340 Stopes can be mined by coming up under it.

350 Level, 3590E: The vein has apparently split into three strands. The 3420 Stope will mine the footwall strand, the center west of the 3590 Raise is already mined. We have exposed the hanging strand with the 3590 hanging wall crosscut and drift. This drift should be extended west to 3400E and extended east to 3860E, where it will join the footwall vein. At 3600E there is a raise from the stope below and this would be used as an ore pass, passing the ore directly to the 3500 ore storage stope and grizzly chamber on the 700 Level.

350 Level, E. Drift: The face is now at 5108E and there crosscuts to both walls. Samples of the vein at the crosscut show 18.5 feet of stranded ore that assays 0.216 oz. gold. When the drift is far enough out to be under the proposed 100 Level 5545 Raise, start another so that if desired a hoist can be installed on the Lineroad and hoist supplies in one shaft or raise from the Lineroad to the 100, 350, 500 and 700 Levels.

500 Level, Footwall Drift at No. 3 Shaft: Diamond drill hole No. 5-A cut 9 feet of vein that assayed 0.175 oz. gold. The hole cut the vein 35 to 40 feet ahead of the face. The drift should be extended and the vein followed until it rejoins the main Sharp Stope Orebody, between 1300 and 1400E. The vein will probably split and the footwall strand may lead to the excellent ore that is 100 feet in the footwall of the Sharp Stope on and just below the 300 Level at 1100 to 1200E. A serious effort should be made to get under that ore as it is about 5 feet wide and assays 1.00 oz. gold. We mined in the 1100 underhand stope until it became too dangerous. Should the 500 Level be too low to catch this vein, then the 350 Level 1975 Drift should be extended until it is into the cave. The old operations mined the Sharp body and when they pulled the stope on the 700 Level they noticed some of this footwall ore coming out of the chutes. The tonnage that can be recovered is an unknown amount. When we re-opened the 700 Level we pulled several thousand tons out of the Sharp Stope and recovered some of the footwall ore. Even with the enormous amount of waste that we had to handle, this paid and the only reason that we quit was because we were afraid that we would cave the ore above the 300 Level and that the cave might extend through to the surface.

500 Level, East Drift: The face is now at 5275E and is still in ore. The ore has split and just a few feet back from the face there was ore in both walls. There is a small cross fault in the face but ore shows underneath it.

700 Level, 4530E: The vein turns into the hanging wall here and should be followed until it connects with the 4900 W. Drift. This was mentioned under Stopping.

700 Level F. Drift: Is far behind the other drifts and the face is at 5066E. The vein was faulted just a few feet back from the face and is just beginning to come in again. The assays are low but regardless of the assays it should be advanced as soon as possible. The hanging wall should be watched closely and prospected frequently as it may be the main vein to follow.

900 Level, West from No. 3 Shaft: The drift is plugged at 1560E and we have never seen the 1540 Winze or the 1540 Sheave Raise. The 1560 Raise was almost entirely plugged at the 700 Level. There is a raise at 1740 that is open and we were on the sublevel, or 800 Level, two or three times. Everything was so dirty, muddy and slick that we could not tell much about it. Could not get past the 1690 Raise as there was about 4 feet of mud behind a small cave. We did get to the east face and think that it should be driven. There is some good ore above this 800 Level.

We were down the winze from the 700 Level, at 1400E, but could not get all the way through to the 800 Level. There is good ore here on what appears to be the junction of the footwall and hanging wall veins. This junction shows in the east face of the 800 Level. From the old reports, it seems that the two veins are separated in the west end of the 800 Level for there are crosscuts to the hanging and foot. Since there is ore all along beneath the 800 Level and since the junction of the two veins is evidently lower going west, it would seem that the 900 Level west drift should be extended.

There is some doubt about whether we should drive the 900 east or drive the 800. There is nothing of value on the 900 east and it might be well to explore the 800 first.

The foregoing outlined development does not list the footages required to attain the objectives so I have scaled off the longitudinal section (50' scale) which shows the ore blocks and their respective assays and tonnages - and estimated that the work indicated by the foregoing will require 6,000 feet of drifting and 2700 feet of raising. In 1938 there was performed 2028 feet of drifts and crosscuts, 987 feet of raising, and 832 feet of deep hole drilling as compared to 3311 feet, 369 feet and 1926 feet, respectively, in 1941. In view of the number of faces that will be available and that more raising appears necessary than the average in past years, the above work could be accomplished in the first two years including 2000 feet of diamond drilling, at estimated costs of: \$17 per foot of drifting, \$25 per foot of raising, and \$3 per foot of diamond drilling - or a total development cost of \$90,750.00 which is approximately 50 cents per ton in case the mill capacity is increased to an average of 500 tons per day.

Water:

The following are Mr. Elgin's comments on the water situation:

Had not the mine closed when it did, we would have had difficulty holding up the milling rate as we did not have enough water. As it was, we had to shutdown the mill two or three times for a few hours. Water might be found in small amounts by diamond drilling through the dike that parallels the vein to the south. More water can be developed at Hance Well and in Antelope Spring but this is a long distance from the mine.

The Tom Reed Mine owns two springs in Cottonwood Canyon. These were reported to have decreased considerably during the summer of 1944 but never did go dry. They have a very poor line to the spring and there are many leaks and flat spots in the pipe where it has been bent. There is water going to waste down the Canyon after it leaves the spring. Mr. Brough and I have often thought of filling a Water Right far enough up the canyon that there would be a gravity flow to the mill tanks.

There is another likely spot that might be looked into. It is about 1/4 mile below the Reed Spring - the main canyon narrows here and a branch canyon takes off to the northeast. In the "Y" there is a shallow shaft that has several feet of water in it. This shaft might be deepened and drifts run under both canyons with an excellent chance of developing water. This would have to be pumped. Again, there is the Babb Spring in the next canyon to the west. This spring was the water supply for the United Eastern and is supposed to have made between 25 and 30 gallons per minute. This would have to be pumped.

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Therefore to assure an adequate water supply would require an expenditure estimated at \$10,000 - a charge of 1 cent per ton to a possible 1,000,000 tons of ore.

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March 15, 1984

To: W. L. Kurtz

From: T. Dalla Vista

Gold Road Mine
Sharon Steel Files

A review of the Gold Road Mine files at Sharon Steel's Bullfrog Exploration office was conducted during the first ten days of March.

The primary purpose of this work was to verify whether any discrepancies existed between the grade carried on the reserve and production/mill heads grade. Also, to identify any potential problems which could adversely affect mining operations.

The following paragraphs outline, in detail, the above and other pertinent data. Part of the information contained herein was obtained from verbal communication with Gail Cutney, Mine Foreman at Gold Road during 1935-42.

1. PRODUCTION

- a) Total (1903-1942) 1,600,000 tons @ .25 oz/T Au
By U.S.S. & R. Co.
1911-1928 564,268 tons @ .332 oz/T Au & .164 oz/T Ag
1928-1935 Shut Down
1935-1942 800,000 tons @ .24 oz/T Au & .23 oz/T Ag
- b) During the last two years of operations, production was as follows:
1941 154,436 tons @ .218 oz/T Au & .224 oz/T Ag
1942 (8 mos.) 98,980 tons @ .223 oz/T Au & .238 oz/T Ag
- c) 1941 Management projection of future production from 1942 to 1949.
154,000 tons/yr. @ .241 oz/T Au and .285 oz/T Ag
- d) Dilution:
Average mining dilution experienced during the last seven years of operations was 25%.
Gail Cutney reported that "pickers" were used to discard waste material from the muck on the way to the mill, thus somewhat increasing the heads value.

e) Mining Methods

- 1) Shrinkage stopes - 74%
- 2) Open stopes (some selective mining done) - 26%

f) Costs Ratio (1935-1942)

Labor	-	55.62%
Supplies	-	21.21%
Contingencies	-	23.17%

2. MILLING

Tests run on 122 tons averaging .194 oz/T Au and .26 oz/T Ag produced the following results:

	<u>Recoveries</u>	
	<u>Au</u>	<u>Ag</u>
a) Based on sample mill heads	91.58	38.50
Based on calculated mill heads	89.45	38.80
Indicated recovery on tailings	76.85	NA
Undissolved loss in tailings	.0173 oz/T	.169 oz/T
Dissolved loss in tailings	.0010	Tr

b) Average Grinding Size

	<u>Percentage</u>
Minus 65 Plus 100 Mesh	.66
" 100 " 150	5.65
" 150 " 200	15.77
" 200	77.91

Tests also proved that for acceptable liberation of gold particles, it would require a grind finer than minus 100 mesh. Flotation is fast and positive and generally the ore is very clean with no earthy or argillaceous slimes. Also, ground material settles and filters fast.

3. WATER

a) Mine Water

Flow	30 GPM
Level as of 10/15/43	715' (#3 shaft)
Level as of 2/01/49	559'
Rise in 5 Years	156'

Estimated volume to be pumped out (total dewatering) 47,304,000 gals.

Present level estimate $\pm 230'$ below Line Road Tunnel.

b) Water Supplies

- 1) Antelope Springs
- 2) Meadow Well

4. VENTILATION

Air Flow:

No. 1 shaft	Downcast
No. 5 shaft	Upcast
Stopes	Upcast
Nos. 2 & 3 shafts	Upcast (winter); Downcast (summer)

5. PROBLEMS EXPERIENCED DURING OPERATIONS

- High drilling costs related to underhand stopes (possibly caused by abrasiveness of vein material coupled with difficulty in washing out drill cuttings).
- Vuggyness of vein caused holes to be ravelly, resulting in excessive stuck drill steel.
- Hangingwall andesite, east of coordinate 3000, badly decomposed. (Re: 3885 stope below and 3900 stope above 450 ft. level.)
- Hi-humidity affecting stopes (east section) due to poor ventilation.
- At 5640 east ground caving and timbers decayed due to high T & H (100 Lv 4975 raise).

6. ORE RESERVE AND PRODUCTION GRADE COMPARISON

*because of
poor gas
when re-opening
area!*

The following comparative data were selected from stopes which were productive during the last two years of operation (1941/1942). This period of operation was singled out because it should better reflect future mining operations.

a) <u>500 Level</u>	Block No.	Tonnage	Au (oz/T)	Vein and/or Stope Width
Reserve	178	17,450	.356	6.49 Ft.
Production	4975 Stope	451	.352	8.07 Ft.
b) <u>500 Level</u>				
Reserve	171	7,002	.273	5.24 Ft.
Production	4725 Stope	1,899	.256	8.50 Ft.
c) <u>700 Level</u>				
Reserve	130	24,360	.22	5.2 Ft.
Production	4100 Stope	1,308	.149	6.6 Ft.
d) <u>700 Level</u>				
Reserve	150 FW	11,233	.329	5.6 Ft.
Production	4850 Rs/ 4900 Stope	4,218	.165	8.46 Ft.

	<u>Block No.</u>	<u>Tonnage</u>	<u>Au (oz/T)</u>	<u>Vein and/or Stope Width</u>
e) <u>100 Level</u>				
Reserve	34	2,108	.422	2.42 Ft.
Reserve	172 & 173	48,193	.530	4.49 Ft.
Production	4820 E. Stope	10,185	.67	5.41 Ft.
f) <u>500 Level</u>				
Reserve	164	2,716	.384	4.89 Ft.
Production	4600 Stope	359	.271	6.54 Ft.
g) <u>350 Level</u>				
Reserve	173	12,096	.469	3.96 Ft.
Production	4975 Rs. 350 E.Dr. 100 E.Dr. }	2,153	.346	5.91 Ft.

Consistently, production figures compare favorably with existing reserves in the majority of blocks where some degree of production took place.

7. COMPARATIVE LIST OF MONTHLY PRODUCTION SHOWING MINE GRADE VS. MILL GRADE

<u>Year</u>	<u>Mine</u>	<u>Oz/T Au</u>	<u>Mill</u>	<u>Oz/T Au</u>
1916	1553 T @	.36	1560 T @	.306
	1757	.27	1680	.502
	1773	.51	1781	.178
	1575	.467	1560	.306
	1347	.312	1430	.329
	<u>1536</u>	<u>.46</u>	<u>1400</u>	<u>.335</u>
Totals	9541 T @	.398	9411 T @	.396*

*Possible upgrading by "pickers" but not thought to substantially affect the overall grade.

1914	2229 T @	.259	2158 T @	.232
	2164	.286	2139	.256
	2010	.30	2315	.315
	2113	.46	2056	.452
	2101	.326	2100	.297
	<u>1915</u>	<u>.403</u>	<u>2100</u>	<u>.342</u>
Totals	12532 T @	.292	12868 T @	.32

8. RESERVE AS OF 1942

Proven 456,138 tons .332 Au .164 Ag

a) Broken muck in stopes

Stope 2890	9009 Tons	.15 Au	.20 Ag
Stope 4725	490 Tons	.26 Au	.20 Ag

- b) Indicated by diamond drill holes

206,872 tons @ .24 oz/T Au and .23 oz/T Ag in following blocks:

<u>Block Coordinates</u>	<u>Tons</u>
5750 to 6410 East	11,728
5560 to 5750 East	6,752
5320 to 5560 East	5,483
5100 to 5310 East	10,380
5040 to 5310 East	8,077
5310 to 6410 East	<u>104,447</u>
Total	206,872 Tons

9. POSSIBLE TARGETS (IMMEDIATE)

- a) Deeper horizons below the 700 level primarily in the easterly section.
- ✓ b) Areas within eastward dipping latite overlying "red sill" which would be separated by the "red sill" from present most easterly stopes (300A, 450A Lv).
- c) Ground east of present 500 and 700 levels.
- d) Possible targets within the 350, 500 and 700 levels on FW veins which exist between 1500E and 2800E SW of main vein.
- e) Drill H.W. from face of west Dr off 2002 x-cut on FW vein (350 level) to test H.W. strands on FW vein, which leaves drift just west of 2002 x-cut.
- f) x-cut along H.W. to investigate DDH on 100 level at 1425 East location, which produced low grade (.09 oz/T Au).
- ✓ g) Extend x-cut (Lr level) from #3 sta., if waste can be handled.

10. MINE CONDITIONS (INSPECTION 1949)

- a) 100 Level

From #2 shaft to 4975 raise in good condition, from 4975 raise to face in bad shape.

- b) 350 Level

From 1840 E to 4975 raise plus 120 feet in good condition.

- c) 500 Level

From #3 shaft to 4975 raise in good condition, except some stress between 3450 E and 3500 E. Well ventilated to this point. Last 280 feet, to face, drift is hot and humid due to lack of ventilation.

d) #3 Shaft

In fair condition, except top 45' (repaired 1984). Timbers missing, possibly collapsed, from 170' below station on the Line Road adit (1984).

e) #5 Shaft

Timbers in bad shape.

f) Line Road Adit

- 1) Spilled between 3220 E and 3300 E.
 - 2) Stopped out area below floor between coordinates 3370 E and 3440 E could show some settling.
 - 3) Stopped area from 4680 E to 4745 E closely timbered.
 - 4) Ground caving at 5640 E with timbers beyond this point decayed.
- g) Levels 700, 800 and 900 not inspected because under water (1949) but should be in fair shape according to G. Cutney.

Summarizing, the grade mined at Gold Road (accounting for dilution) reflects remarkably well the grade carried on the reserve. There is no reason to believe that even better grade could be achieved by careful mining, thus keeping dilution to a minimum.



Tony Dalla Vista

TDV/cg

cc: JDS
JRS
TES

ASARCO

Exploration Department

July 3, 1984

To: R. L. Brown

From: W. L. Kurtz

Sharon Steel
Gold Road, AZ
Sunnyside Mine, CO

In a short phone conversation Allan Bell, head of Sharon's Natural Resource Department, indicated he was still evaluating Sharon's properties, and I got the distinct impression that he is going to try to resurrect Sharon's Mining Department rather than spin off or joint venture their properties.

He has been spending 90% of his time with the Standard Metals bankruptcy and hints that Sharon is trying to get title to the property and then operate it themselves. He apparently has not studied Gold Road and it may be some time before he does so.

Though it will make him mad, I will call again after Labor Day.



W. L. Kurtz

WLK:mek

cc: D. M. Smith
J. D. Sell/J. R. Stringham

Gold Road
Mohave Co, AZ

FROM: W. L. KURTZ

Gold Road
Arizona

TO: IRS/SAA

Sept 23, 1943

New York Exploration Dept
believes you should physically
look at gold road surface +
underground if you are
going to make positive recommendations
to acquire property for Sharon

1 CC: JDSch

January 17, 1984

To: J. D. Sell

From: F. R. Koutz

Robert Smith M.S. Thesis
Fluid Inclusions
Oatman District
Mohave County, AZ

Bob Smith, an M.S. student at U. of Arizona (621-2988), called to talk^(Jan 12) about fluid inclusion heating stages. He also talked about some of his findings (now being written up) on fluid inclusion research at Oatman initially sponsored by Fischer-Watt.

All of Mr. Smith's samples in contrast to many collected by Clifton (AIME Preprint 80-143) were from outcrop and drill core. His samples from the Kokomo vein averaged 230-245°C and the Leland Butte-Ben Harrison area: 220-250°C--both with salinities around 1.6 wt.% NaCl equivalent. Mr. Smith spent 300 hours on one specimen from the Gold Road vein outcrop studying 35 different bands in the quartz, obtaining only 20 reliable fluid inclusion homogenization temperatures mostly in the 180-190°C range but with a total range of 75-320°C-which could be indicative of boiling. (I would have gotten more samples rather than spend so much time on one sample.)

Mr. Smith finds that the nice fluid inclusion story told by Mr. Clifton and others breaks down with detailed work and temperature/depth horizons cannot be easily correlated across offset vein segments. Although Mr. Smith finds multiple periods of boiling, lack of boiling does not always indicate the bottom of a vein/ore shoot. He also says that the presence of high salinity (NaCl daughter) fluid inclusions has not been confirmed in the Boundary Cone area and the story reported by Clifton and others is false and based on solid inclusions that appeared to one of the authors to be homogenizing fluid inclusions. Fischer-Watt (as explained in the preprint) was looking for a reason why the area should not be drilled and the supposed presence of high salinity/high temperature inclusions was taken to indicate that the productive levels had been eroded away. In short the preprint is a classic example of both the use and misuse of fluid inclusion data--although their conclusions may be right--but for the wrong reason.

Mr. Smith did say that he found no evidence of liquid CO₂ in the fluid inclusions that Fischer-Watt has suggested was present although Bob Bodnar (USGS/ Penn State-ex U of A-Red Mtn) did find CO₂ in Gold Road inclusions under the crushing stage. Mr. Bodnar is among those fluid inclusion consultants who claim a "secret method" of determining productive vs. unproductive veins. In short there is not a simple fluid inclusion story in the Oatman District. If more detailed information on the district becomes necessary I can probably obtain this from Mr. Smith.


F. R. Koutz

FRK/cg

cc: WLK

February 27, 1984

To: W. L. Kurtz

From: T. Dalla Vista

Gold Road Mine Sampling
Arizona *Moheave Co.*

Six days were spent on the property by the undersigned, T. Benavidez, and H. Stone.

Three ladders, totaling 82 feet, were replaced and two landings were installed in the upper section of the #3 shaft.

The lower part of the shaft to about 280 feet below the collar was found to be in fair condition. However, no timbering exists below this point.

The Line Road adit is caved at about 1200 feet from the station, with several hundred tons of muck filling the drift.

The 107 sub-level (110 feet below Line Road level) is also caved about 30 feet from the shaft.

Attached is a list of samples collected and a map showing locations.



T. Dalla Vista

TDV/cg

Attachments

cc: JDSell

Sample Descriptions

Gold Road Mine - Line Road Adit

<u>Sample No.</u>	<u>Width (Ft.)</u>	<u>Au (opt)</u>	<u>Remarks</u>
GR-1	3	.135	Qtz. vein. Fine grained, jasperoid wt. adularia and chalcedony wavy banding. Wk. FeOx staining.
GR- 1A	1	.025	H.W. vein contact, strongly oxidized bx zone.
GR- 2	4	1.075	Qtz. vein. Bx'ed wt. angular andesite frags. Adularia bands. St. bx at F.W. contact.
GR- 3	3	.305	Qtz. vein. Intensive brecciation near H.W. Several qtz. vlts. in H.W. andesite
GR- 4	2	.840	Same as GR-3.
GR- 5	6	.265	Bx'ed zone with intensive qtz. veining (1" to 18" wd). Andesite frags. in qtz. matrix.
GR- 6	2.4	.020	Massive qtz. vein. Adularia banding. Wall rock frags. near H.W. and F.W.
GR- 6A	5	.010	Calcite structure. Grayish color about 15' SW of main qtz. vein. Coarse grained.
GR- 7	7	.130	Qtz. bx zone. Junction point of qtz. and calcite veins.
GR- 7A	1.2	.010	F.W. contact of vein. Calcite zone merge wt. qtz. vein and narrows to 1.2'.
GR- 8	4.5	.045	Qtz. bx zone wt. wall rock frags. F.W. about 1.5' calcite.
GR- 9	6.5	.035	Qtz. vein. Pinkish color wt. adularia/chalcedony banding. CuOx staining. F.W. 1.5' calc.
GR-10	3.6	.290	Same as GR-9.
GR-11	3.5	.050	St. bx zone near F.W. contact. Otherwise massive qtz. vein.
GR-12	3.8	.155	Qtz. vein, intensive adularia banding. 4" bx (andesite frags.) in center and parallel to vein.
GR-13	2.6	.120	Qtz. vein. Massive and banded wt. angular wall rock frags. near contact.
GR-13A	1	.020	Bx. qtz frags in FeOx groundamss at F.W. of vein.

Sample Descriptions - Gold Road Mine - Page 2

<u>Sample No.</u>	<u>Width (Ft.)</u>	<u>Au (opt)</u>	<u>Remarks</u>
GR-14	2	.010	Strongly bx'ed vein. Several qtz. vlts. in F.W. and.
GR-15	3	.290	Vein splits. Stringers and vlts. and with st. brecciation near F.W. contact.
GR-16	3	.080	Massive qtz. vein. 8" FeOx bx at F.W. contact. (Taken in raise.)
GR-17	1.5	.005	Taken 36' N.E. of #3 shaft station. Calcite vein in H.W. about 4" wide.



September 13, 1985

To: J. D. Sell

From: F. R. Koutz

Mineralogy/Fluid Inclusion
Thesis Review
Oatman District
Mohave County, Arizona

I have obtained from the author, Robert C. Smith, a copy of his 1984 M.S. Thesis from the University of Arizona: "Mineralogic and Fluid-Inclusion Studies of Epithermal Gold-Quartz Veins in the Oatman District, Northwestern Arizona," 233 p., 39 figs., 27 tables, 1 plate. Bob Smith worked on this thesis from 1980-1984 under the direction of J.M. Guilbert. Mr. Smith is now a consulting geologist (AZ Reg. Geol. No. 16291) specializing in fluid inclusion studies--a rather slow business. He also helps to manufacture fluid inclusion heating/cooling stages for SGE, Inc. of the Geosciences Dept. and has been known to put a few geochem. samples in sacks or stake claims for various mining companies. The thesis was initially sponsored by Fischer-Watt(P. Durning), apparently later in part by Exploration Research Labs (G. Clifton).

After a thorough review of the district lithology, structure, vein mineralogy alteration and ore occurrence (3.8 m.t. production @ 0.58 opt Au and 0.17 opt Ag), the thesis attempts to characterize various veins and vein stages with mineralogic studies by a plethora of instrumentation and methods and with fluid inclusion studies of homogenization and freezing temperatures and C&S gas contents by gas chromatography--mass spectroscopy (GC-MS) at Exploration Research Labs. Samples were taken from outcrop of 6 veins over the district realizing that the upper and lower productive elevations in the district were domed and with deeper erosion in the central part of the district spatial comparisons would be somewhat tenuous.

Work concentrated on a complete Gold Road vein cut, wallrock to wallrock, in which 60 bands were correlated to 35 stages by mineralogy, color, thickness and texture. Smith found that Lausen's (1931), AZ Bur. Mines Bull. 131, 5-stage classification was reliable only 57% of the time. Even on Lausen's original samples at UofA stages could not be reliably classified between veins--a conclusion made by most who have studied the vein fillings over the district.

Smith found that catholuminescence, UV-fluorescence, visible and near-IR spectrometry and electron microprobe studies of mineralogy were not useful in differentiating vein stages. Microprobe analysis did suggest, but not confirm that the "greasy-green" color of late stage Au-productive vein stages was probably caused by finely divided (<1%) FeMg silicates--possibly chlorite or corrensite (1:1 interstatified trioctahedral chlorite and vermiculite or smectite).

Only one grain of electrum was found and microprobed--which contained 29-30% Ag, .13-.16% Te, .03-.05% Se, .015% S, .03% As, .08% Fe and .015% Cu. This electrum with Te from the Gold Road vein is consistent to what we found (1984) in surface and underground bulk cuts from the Gold Road vein. Surprisingly no Au, Ag, Te or other elements of interest by microprobe scanning of polished vein slabs from the Gold Road vein were found. This is probably due to polishing problems. However, AA-Au and Ag analyses of the bands (by Skyline) showed <.02-23 (4.4 mean) ppm Au and 2.2-55 (15 mean) ppm Ag. Reasonable lower detection limits for the probe are about 2-5 ppm Au or Ag, so some should have been detected unless the electrum were plucked by polishing. Surprisingly Au, Ag and Au/Ag decreased with time--opposite that which Lausen found for the district in general; although the Gold Road Bands may be all part of the same fourth/fifth Lausen vein stage, ^{pure} Au content was confirmed to be proportional to adularia. Over the district Smith's Ag/Au analyses averaged 5/1--quite different than the 1/3 production ratio. We do know that Ag recovery was not as efficient as Au and that mining concentrated on late Au-rich/Ag-poor stages.

Fluid inclusion homogenization temperatures were 205-255°C for primary and 175-335°C for secondary inclusions. The wide range of secondary inclusion temperatures and inclusion textures are suggestive of boiling, but no vapor-rich inclusions were noted. Inclusions contained 1.5 weight % NaCl equivalent, no daughter minerals and about 65 bars of vapor (but no liquid) CO₂. Higher temperatures were found in the central, more deeply eroded, portion of the district. Overall usable fluid inclusions were not common--a common occurrence itself in epithermal systems and quite frustrating to detailed study.

Fluid inclusion gases were removed from carefully cleaned samples by decrepitation and carbon and sulfur species identified and quantified by GC-MS. Comparisons of individual species do not appear to be particularly useful and no correlations with color of individual bands could be found. The ratio of total C/S vs total oxidized (C+S)/reduced (C+S) decreases with time and is low compared to other epithermal systems and is more similar to porphyry copper or other mesothermal deposits. Smith suggests this is because there was more sulfur in gas versus minerals in the low sulfur Oatman hydrothermal system. Smith suggests that boiling of hydrothermal fluids (more common in the central part of the district) which would increase the ratio of oxidized/reduced gas is responsible for many of the noted changes in the samples. The pH rise by vapor escape from boiling probably contributes to Au precipitation.

This study provides little new information applicable to exploration at Oatman. The sweeping conclusions based on preliminary and cursory fluid inclusion studies reported in Clifton, Buchanan and Durning (1980 AIME preprint 80-143) were found on further work not to be supportable and are not included in this study nor in the revision by Durning and Buchanan

(1984-AGS Digest XV). The present fluid inclusion data suggests that Au-Ag mineralization occurred at depths from 1000 to 2850 feet. The presence of adularia is a better guide to Au mineralization than greasy-green to yellow quartz color. Adularia is especially common in vein stages that contain wallrock fragments (tectonism). Fluid inclusion temperatures are probably as good as analyses of gas contents for determining mineralization centers in the Oatman District. GC-MS gas analysis has been a fairly standard analytical tool for the last 15-20 years and is particularly useful in the petroleum industry, and apparently in the study of alteration associated with oil pools and sediment hosted mineral deposits. Although for Oatman it appears to show more noise than meaningful data, developments in this analytical field should be kept up with.

FRK:mek


F. R. Koutz

++Mossback (Crown Res./Sutton) 259 + 5 pat. claims (Ex-1200 HIMCO claims)
Brad Peek (Geol)

1-5 m.t. @ 0.07 opt Au (Ex-potential-disproved) Meridian (1982)
8 1982 RDH 3735' - Calc. \pm Qtz HW stockwork of Mossback vein
a few 5-10' intercepts of .X-1 ppm Au hit.
20 1984 RDH 5000', Sec. 16 hot spring (jar + alunite), NY +N vein
a few 0.0X ppm Au zones hit in v. strong dissem. py zones \pm "shale"
Have data maps-data-worth low priority recon. Moss Porphyry margin.

**Secret Pass/Tincup Mine (Santa Fe/Pacific Mining) Fred Jenkins, Mgr. ABC
Martin Steinpress, Geol. (+WSA)

90 DH, 6 areas, about \$1 million spent
74,000T @ 0.11 .02 c/o 5.6/1 w/o Tincup (op)
108,000T @ 0.053 .02 c/o 3/1 w/o FM area (op)
400,000T @ 0.07 .01 c/o NW Tincup UG "Geologic Resource"
including 92,000T @ 0.20 .10 c/o NW Tincup UG "Geologic Resource"
 \pm 45,000 oz. total, \pm fair potential for additional UG Geol. Res. \pm op
mineral in vicinity, complex land situation (State + RR checkerboard,
Pat + Unpat. claims). Stockwork (not vein) mineraliz. in pE Gr + T
and.- not detachment. Have data packet for detailed field review; SF
wants \pm \$200,000 buy out.

*Moss Mine (Moss and Ruth veins) Production: 64T @ 280! opt Au (1864)
Total: 25-30,000 oz. Au to 1930's

200,000T @ 0.11 opt Au, 0.6 opt Ag to 300' level:	1940 Tonapah
	Belmont
	+
19,000T @ 0.29 Au + 1.4 Ag UG (300' level)	F.M. Stevens
400,000T @ 0.1 opt Potential	(Asarco)
90,000T @ 0.13 opt Au (op) Bill Faulta drill data (F-W/WLK notes)	
500,000T @ 0.06 opt includes above. Peter Drobeck 3/87. Harrison	
Resources pouring shaft collar for retimbering main shaft 3/11/87.	

**Gold Road:

395,000T @ .245 opt Au diluted to 5'+15% = Mineable SAA/JRS
or 556,000T @ .24 opt Au +0.28 Ag 1942 USSR&M
plus 560,000T @ .21 opt Au inferred (Gold Road-Red Top) Sharon Steel
Tails: 20,000T @ .08 opt Au Drilled 1930's (80,000T remilled 1930's-'42)
 \pm 1,000,000T @ .02-.03 opt Au - production records.
1,691,000T @ 0.34 opt Au - production
Sharon Steel (1987) feasibility study; maybe looking for partner--
turned down Echo Bay (1986).

*Datman: United Western \pm 200,000T @ 0.2 opt Au (vn) Fischer-Watt (Sun
River Au)

Tom Reed Tails 1.5 m.t. @ 0.025 opt Au
Vivian Area: \pm 100,000T @ 0.2-0.3 opt Au Sun River Gold Potential (UG)
or \pm 125,000T @ 0.25 opt Au Sun River Gold calc. from GCNL 2/12/87
Oro Fino Vn pot: 20,000T @ 0.3 opt Au Pete Drobeck 1987 Drilling (BX)

largest automobile manufacturer in the United States at both boards have approved the acquisition.

March 20, 1987

To: J.D. Sell/W.L. Kurtz

From: F.R. Koutz

Gold Road Mine
Sharon Steel
Mohave County, AZ

I talked with Bill Worthington (505-537-3884, Vanadium, N.M.) concerning the status of Sharon Steel's Gold Road properties. Apparently Echo Bay completely reviewed the mine data last year and made Sharon a lease or JV offer for the property in November--which they turned down. Echo Bay had some kind of scheme of building a central mill between Gold Road and their properties at Congress! which seemed farfetched. Sharon then decided to study the possibility of a "spin-off" gold company with Canadian help and new shares issued. They had American Mine Services do a detailed feasibility study, completed Feb. 1987, which showed that a 300 TPD operation would produce gold at \$248/oz. Sharon's people (they still have 12 experienced supervisors on the payroll) thought this figure was a little low. Sharon is studying whether they should try to operate themselves. They are leaning towards bringing in or leasing out to an experienced operator, if they can find one interested in operating rather than selling stock and if they can't raise the money themselves.

Sharon has been approached several times by the Canadian successors to Fischer-Watt's claims in the district (this is probably Sun River Gold Corp; Pete Drobeck, Geol.), but they have no track record and Sharon does not want to be part of another stock scheme.

Two years ago Sharon started a new DDH some 2500' SE of their previous drilling. The hole is stopped at 469' and will be deepened for assessment work in future years. The vein should be cut at 1000'. The long step out was decided on as negative results closer in to previous Red Top intercepts might kill the project while positive results this far SE are extremely significant. Other assessment work has been dozing old and new roads and drill sites.

Allan Bell in Miami Beach is still Sharon's Natural Resources Division VP. I told Mr. Worthington to keep Asarco in mind.

FRK:mek


F. R. Koutz

ASARCO

Exploration Department
Western USA

March 24, 1987

R.L. Brown
New York Office

Gold Road Mine
Oatman District
Mohave Co., AZ

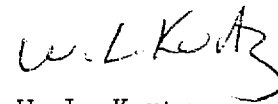
You undoubtedly recall that Mr. Stringham and I spent considerable time trying to convince Asarco (translate as Tom Osborne) that the Gold Road Mine owned by Sharon Steel and the Red Top extension leased by Sharon Steel represented an excellent opportunity for Asarco. With the "new" 150,000 ounce minimum requirement Gold Road should be acquired and put into production by Asarco as Asarco's first gold mine in the good old USA

Proven reserves and exploration potential remains as previously reported (395,000 tons proven at a diluted grade of .245 oz/t and +500,000 tons additional ore at same grade indicated by two drill holes). To be profitable the mine will require an excellent small underground mine manager (e.g. Jim Stringham).

Please remember this is the best known reserve in the Oatman District an old epithermal vein mining camp that produced over 2.2 million ounces of gold.

Attached is a memorandum by Koutz discussing recent business activity at Gold Road.

WLK:mek
Att.


W. L. Kurtz

cc: J.D. Sell
F.R. Koutz

ASARCO

Southwestern Exploration Division

March 20, 1987

To: J.P. Sell/W.L. Kurtz

not open-pit, but could get us on the road to ore?!
JS

From: F.R. Koutz

Gold Road Mine
Sharon Steel
Mohave County, AZ

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FRK:mek


F. R. Koutz

ASARCO

Exploration Department
Western USA

W. L. Kurtz
Manager

January 27, 1984

Mr. Robert Weagel
Sharon Steel Corporation
Natural Resources Division
6917 Collins Avenue
Miami Beach, FL 33141

Dear Mr. Weagel:

This confirms our phone conversation of today that Asarco will enter the #3 shaft at Gold Road to make an examination of the Line Road Tunnel level and collect a few samples of the vein.

This work will be supervised by Tony Dalla Vista and Asarco assumes all responsibility for the liability of their men. We agree to close the shaft opening when we finish.

We will complete this work by the end of February 1984.

Very truly yours,

W. L. Kurtz
W. L. Kurtz /g

WLK/cg

cc: JDSell
TDalla Vista

GEOLOGICAL REPORT
ON THE
UNITED WESTERN PROJECT
SAN FRANCISCO MINING DISTRICT
MOHAVE COUNTY, ARIZONA

FOR
SUN RIVER GOLD CORP.

April 7th, 1987
Vancouver, B.C.

W.G. Timmins, P.Eng.
W.G.T. CONSULTANTS LTD.

W.G.T. CONSULTANTS LTD.
CONSULTING GEOLOGISTS

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SUMMARY

The Oatman camp in the San Francisco Mining District has produced 3,800,000 tons of ore averaging 0.58 oz. gold/ton and 0.17 oz. silver/ton from 1897 to 1942, mainly on the Tom Reed and Gold Road veins.

The United Western Mine is located on the Tom Reed vein system from which approximately 2,000,000 tons were mined.

The United Western Mine produced 40,000 tons averaging 0.30 oz. gold/ton prior to 1940.

The property is underlain by the favourable Oatman Formation consisting of latite flows, latite tuffs and flow breccias.

Vein structures of quartz-calcite-adularia contain fine-grained free gold.

Reserves have been calculated in the area of underground workings of the United Western Mine between the 400 and 950 levels as follows:

<u>Class</u>	<u>Tons</u>	<u>Grade (oz. Au/ton)</u>
Reasonably assumed	118,054	0.259
Inferred	107,608	0.268

In addition there is excellent potential for the development of substantial additional reserves along the strike of the vein system both to the northwest and southeast.

A three-phased exploration and development program consisting of extensive surface drilling to expand reserves and rehabilitation and dewatering of the shaft to enable further evaluation

including mapping, sampling, metallurgical studies and feasibility in preparation for mining, is recommended.

The total estimated cost of the three phases of the exploration and development program is \$1,104,000.00 Canadian.

INTRODUCTION

At the request of the Board of Directors of Sun River Gold Corp., the writer has prepared the following geological report which is based on a personal examination of the property during January 18 - 23, 1987 and from a thorough study of past exploration carried out during 1979 through 1982 and previous historical data.

The purpose of the examination was to calculate reserves and to recommend an exploration and development program that would assist in fully evaluating the economic potential of the United Western Mine deposit.

PROPERTY

The property is comprised of four patented claims and 25 lode mining claims held by location. They are as follows:

<u>Patented Claim</u>	<u>Mineral Survey No.</u>
Arizona Lode	8301
United Western	3285
United Western No. 1	3288
United Western No. 2	3288

<u>Lode Claims</u>	<u>Book</u>	<u>Page</u>	<u>AMC No.</u>
Mace	271	56	6629
Mace Extension	271	57	6630
FWW 1-23 Inclusive			263542-263563 inclusive

OWNERSHIP

The claims are all held under option to purchase or lease by Sun River Gold Inc., a wholly owned subsidiary of Sun River Gold Corp., of Vancouver, British Columbia. The claims are in good standing.

LOCATION

The property is located immediately north of the town of Oatman on Highway 66, some 28 miles (45 km) west of Kingman in Mohave County, northwestern Arizona, some 100 miles (160 km) southeast of Las Vegas, Nevada.

The claims are situated in T19N; R20W; Sections 10, and 15.

ACCESS

The property is accessible by paved Highway 66 and the gravelled Silver Creek Road which pass through the claims, about 1/2 mile (0.3 km) north of Oatman. Oatman may be reached by means of paved Highway from Kingman, a distance of 28 miles (45 km), or by paved and gravel road from Bullhead City on the Colorado River, a distance of approximately 15 miles (24 km).

TOPOGRAPHY

The area exhibits gentle to moderate relief on the property with elevations in the Oatman District ranging from 2400 feet (731 meters) to 4400 feet (1341 meters).

CLIMATE

The climate within the district is arid with average winter temperatures about 45 degrees F. (7 degrees C.) and during the summer months the average reaches into the 90's F (32 degrees C.). The annual rainfall is less than 10 inches (25 cm).

VEGETATION

Vegetation is sparse and consists of a desert variety of grasses, cactii, sagebush, thorny brush and greasewood.

WATER

Water exists in the mine workings and would be available from wells and possibly from the local supply of the town of Oatman.

POWER

Hydro-electric power would be available.

SUPPLIES AND ACCOMMODATIONS

Most supplies would be readily available from the larger communities in the area.

Local communities have ample accommodation and employees would be able to commute to the site thus alleviating the costs of any camp construction and maintenance.

HISTORY

Gold was first discovered in the Oatman (San Francisco) District in 1863 with the main discoveries of the Gold Road and Tom Reed Veins in 1900 and 1901 respectively.

Rich ore was discovered on the Tom Reed Vein in 1916 and in the same year the United Eastern ore body was discovered.

In 1924 the United Eastern Mine closed down and the Tom Reed Mine shut down in the mid 1930's.

The Gold Road Mine operated from 1900 to 1916, briefly during 1922 and again from 1934 to 1942.

Between 1897 and 1942, the Oatman District produced 2.2 million ounces of gold and 800,000 ounces of silver from over 3.8 million tons of ore at an average grade of 0.58 ounces of gold per ton and 0.17 ounces of silver per ton.

Production ceased in 1942 with WWII emergency legislation. The low gold price after the war was not conducive to exploration and development in the district.

In recent years the increase in the price of gold has resulted in a renewed interest in the area to re-evaluate the gold potential in the Oatman District.

Underground development of the United Western Mine commenced around 1915. A total of 6,700 feet (2,040 m) of drifting and cross cutting was done along with 2,750 feet (838 m) of shafts and raises. Production to 1940, when the mine closed down due to the closure of custom milling facilities, was 40,000 tons at a grade of 0.300 ounces of gold per ton.

REGIONAL GEOLOGY

The Oatman district lies on the western flank of the Black Mountains, a fault-bounded Tertiary volcanic pile composed of trachyte, latite, rhyolite, and basalt situated at the southern end of the Basin and Range province.

The Black Mountains are deeply incised on its western flank, but are little eroded on its eastern. Exposures of Precambrian basement rocks are found on the western margin and north end of the district. Except for the capping basalt, the volcanic center appears to have been in the Oatman area.

The volcanic pile has a 10 degree easterly dip. The Tertiary volcanic rocks rest on a Precambrian basement of schist, gneiss, and granite (Figure 2). The Precambrian rocks are overlain by the Eocene Alcyone Formation, a sequence of trachytic tuffs, latite flows, tuff breccias, lahars and minor sedimentary rocks. The Alcyone Formation was intruded by the Times Porphyry, a rock very similar in composition to the Alcyone, and therefore interpreted as being comagmatic with the Alcyone. The Alcyone Formation is unconformably overlain by the Esperanza Formation, a trachytic flow.

The Esperanza Formation is conformably overlain by the Miocene Oatman and Gold Road Formations. The Oatman Andesite, as it is usually called, is a sequence of massive to vesicular biotite-poor latite flows, latite tuffs, flow breccias and includes minor sedimentary beds. The Oatman Andesite is approximately 1000 feet (305 m) thick at Oatman, but thins rapidly from the central district. Conformably overlying the Oatman andesite is the Gold Road Formation, a sequence of biotite-rich latite flows and minor ash flows and breccias. Maximum thickness is about 800 feet (244 m).

Generalized Stratigraphy

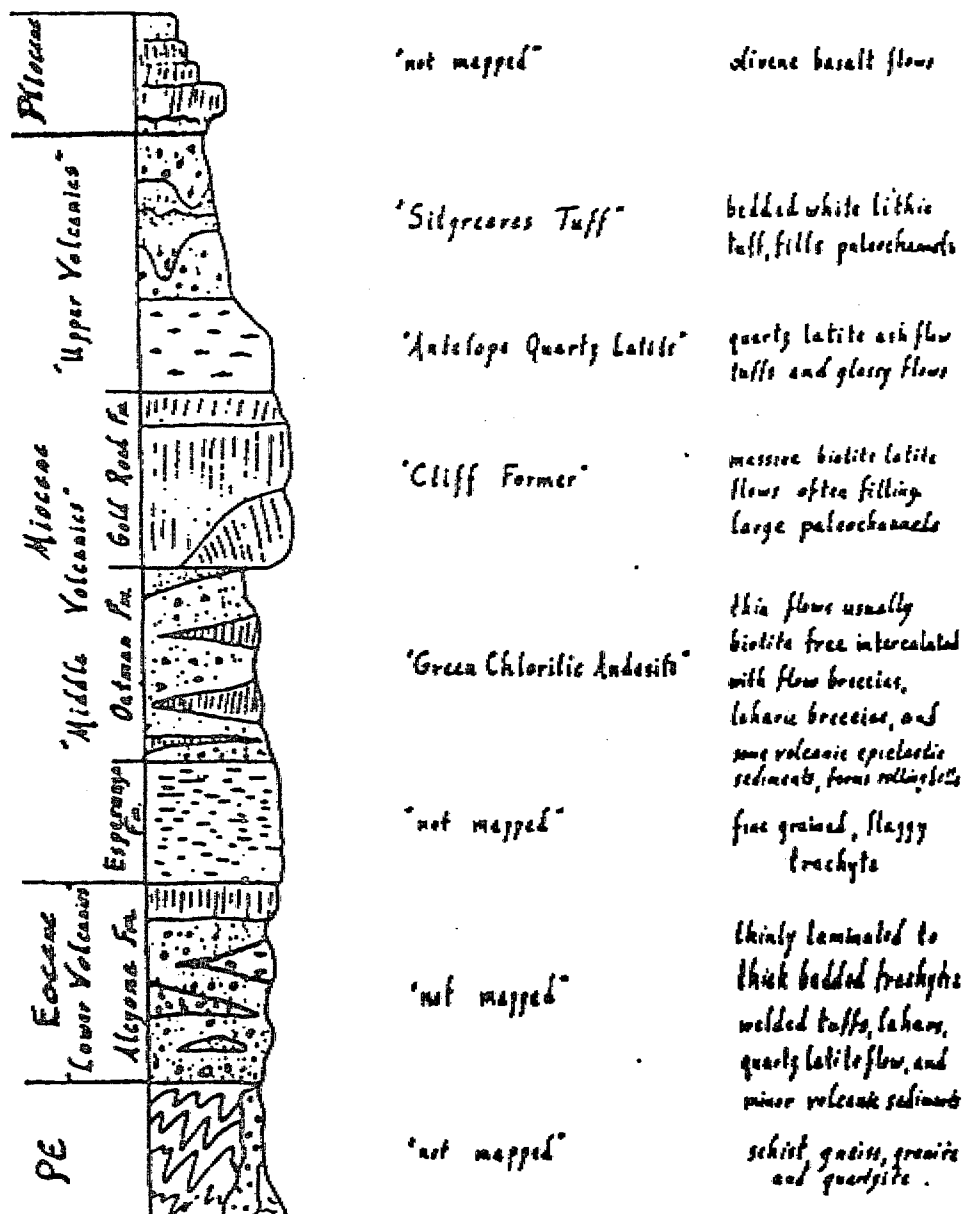


Figure 2

after Knight and Winston

The Esperanza, Oatman, and Gold Road Formations are collectively termed the Middle Volcanics. Based on the similarity of distribution of the Oatman and Gold Road Formations and on their petrologic character, it is probable that these rocks were comagmatic and originated from vents nearby.

The Middle Volcanics are unconformably overlain by the Upper Volcanics, a series of trachytes, quartz latites and rhyolite tuffs. Included are the Antelope Quartz Latite and Sitgreaves Tuff.

The Middle Volcanics and probably the Upper Volcanics, as well, were intruded by a quartz monzonite pluton, the Moss Porphyry. It is believed that this intrusive rock was comagmatic with the Middle and Upper Volcanics and is interpreted to be a late-stage epizonal pluton which intruded its own volcanic cover.

Also intruding the Middle Volcanics and probably the Upper Volcanics are conspicuously white rhyolite dikes and domes. This rhyolite is known as the Elephants Tooth Rhyolite. The name originated from the prominent white spire outcropping just east of the town of Oatman.

LOCAL GEOLOGY

The property is underlain by the Oatman Formation, a series of massive to vesicular pyroxene latite flows, latite tuffs and flow breccias. The formation is approximately 1,000 (305 m) thick, is of uniform character, and is more porphyritic than other volcanics in the area.

The main structure passing through the property is the Tom Reed vein which is fault-related. At the United Western shaft the vein strikes at 330 degrees with a steep dip to the southwest.

The vein system at the shaft collar has a total width of about 40 feet (12 m) with 10 to 15 feet (3 to 4.6 m) of waste separating two veins.

The vein filling shows evidence of movement along the fissure with brecciation of the quartz and calcite.

The alteration along the vein surface is propylitic.

A rhyolite dyke cuts across the vein and strikes about east-west with a southerly dip.

MINERALIZATION

The mineral deposits at Oatman are typical of epithermal precious metals veins in Tertiary volcanic rocks. They consist of quartz-calcite-adularia open space fillings along fissure veins within which definite ore shoots containing fine-grained free gold occur. Sulfide content is generally low, commonly less than one quarter of one percent.

The ore shoots have a limited vertical range of a few hundred to 1200 feet and exhibit a strong vertical zonation of alteration-mineralization. Many of the ore shoots are blind at the surface, showing only weak alteration with little or barren vein matter.

In addition to primary zonation, many of the ore bodies have been cut and displaced by post-mineral faults which commonly closely follow the attitude of the veins, further complicating the mineralization geometry.

The characteristics of ore shoots which have been mined are outlined on Table I. Those ore shoots having greater widths seem to be localized in areas of dilatency along the Tom Reed vein where significant bends in strike or dip have occurred. Bends in

strike producing a concave north trace or steepening of dip have produced the greatest ore shoot widths, such as the United Eastern with widths up to 45 feet.

Mineralization has occurred in most of the ore shoots in several stages of quartz and calcite. Lausen (1935) recognized five stages. The better grade ore shoots typically contain bands of the late stage green and yellow quartz which occur as streaks through the lower grade early quartz and calcite. This makes for rather erratic distribution of values in a given ore shoot.

In general, the ore shoots are best developed in those portions of the vein which consist of nearly massive quartz-calcite although a significant portion of the vein zones, particularly the Tom Reed vein zone consists of a stockwork of quartz and calcite veinlets.

Two principal vein systems, the Gold Road system and the Tom Reed system, have been responsible for most of the production in the district.

TABLE I
SUMMARY OF OREBODY CHARACTERISTICS MINED FROM THE
TOM REED-UNITED EASTERN AND GOLD ROAD VEINS, OATMAN, ARIZONA

<u>Orebody</u>	<u>Tonnage</u>	<u>Grade</u> <u>O/T Au</u>	<u>Maximum Dimensions</u>		
			<u>Length</u>	<u>Width</u>	<u>Height</u>
United Eastern	550,000	1.10	450	45	700
Tip Top	250,000	0.70	500	20	1,300
Ben Harrison	250,000	0.70	650	20	750
Big Jim					
Aztec	500,000	0.75	1,950	35	800
Black Eagle	200,000	0.50	350	10	1,000
United American	140,000	0.50	300	10	1,000
United Western	40,000	0.30	990	6	300
Gold Road	1,500,000	0.32	6,200	22	1,300
Telluride	20,000	1.0	200	2-3	200

The United Western mine on the northwest extension of the Tom Reed vein zone, produced an average of 0.3 oz. t. gold from principally 3 levels: the 500 level, the 700 level, and the 780 level.

The Tom Reed vein at United Western dips from 53 degrees to 75 degrees to the southwest and occurs at a major bend to the northeast in strike. Average mining width was about 6 feet although the vein commonly ranged from 3 to 12 feet. Most of this ore consisted of friable calcite with white quartz.

Two diamond drill holes were completed on the northwest extension of the United Western zone, one by Fischer-Watt, hole no. 79-5, and one by Hecla, hole no. 82-5. Drillhole 79-5 intersected the United Western vein below the 700 level and cut 7 feet of massive calcite which assayed .115 oz. gold/ton. In addition, this hole also intersected at a depth of 716 feet another split of the vein which assayed 0.424 oz. gold/ton gold over 3 feet.

Although widths are commonly 3 to 12 feet, drilling to the southeast of the United Western workings intersected two veins having true widths of 8.0 feet grading 0.216 oz. gold per ton and 30.0 feet grading 0.201 oz. gold per ton indicating a dilatant zone.

Sampling on the 100 level at the United Western Shaft in 1915, indicates the possibility of a mineralized zone at upper levels as shown below.

Sampling on 100' level

No. 1	Select	0.406 oz. Au/ton
2	4.0'	0.203 oz. Au/ton
3	5.0'	0.097 oz. Au/ton
4	6.5'	0.135 oz. Au/ton
5	4.0'	0.063 oz. Au/ton
6	15.0'	0.068 oz. Au/ton
7	4.5'	0.121 oz. Au/ton
8	--	0.145 oz. Au/ton
10	--	0.068 oz. Au/ton
11	Select near No. 4	2.622 oz. Au/ton

Gold mineralization at the Katherine Mine, about 20 miles (32 km) north of Oatman, occurs in Precambrian rocks, which occur below the Oatman Andesite series, and has many similarities to the mineralization at Oatman.

ORE RESERVES

Reserve calculations are based on 1940 underground plans and sections, showing numerous stope assays and assay widths at the time of the mine shutdown, thus the data remains valid for present calculations. The United Western Mine was forced into closure due to the cessation of the milling facility which was processing their ore at the time.

Tonnages classified as reasonably assured were calculated by extensions immediately adjoining old stopes and workings on the 500, 700, 780, and 850 levels.

Inferred tonnages were calculated using only 100 foot extensions above the 500 level and below the 850 level and 100 feet laterally beyond the reasonably assured tonnages.

A tonnage factor of 12 cubic feet per ton was applied throughout.

All blocks are shown on the longitudinal section accompanying this report.

A summary of the calculated tonnage blocks is shown in Table II and ore reserves are shown in Table III as follows:

TABLE II

SUMMARY OF UNDERGROUND TONNAGE BLOCKS
UNITED WESTERN MINE

Reasonably Assured

<u>Block</u>	<u>Tons</u>	<u>Grade</u>	<u>Width(ft.)</u>	<u>Tons x Grade</u>
A	24,235	0.306	3.34	7,415.9
B	15,587	0.259	3.68	4,037.0
C	12,577	0.203	4.56	2,553.0
D	11,552	0.225	3.78	2,599.0
E	2,133	0.308	4.25	656.9
F	5,021	0.158	5.38	793.3
G	12,257	0.420	3.17	5,147.9
H	<u>34,692</u>	0.213	4.60	<u>7,389.4</u>
TOTALS	118,054			30,592.89

Average 118,054 tons grading 0.259 oz. Au/ton across 4.10 ft.

Inferred

I	41,333	0.267	4.0	11,035.9
J	27,708	0.287	3.5	7,952.2
K	6,167	0.214	4.0	1,319.7
L	12,000	0.384	3.0	4,608.0
M	<u>20,400</u>	0.190	4.0	<u>3,876.0</u>
TOTALS	107,608			28,791.8

Average 107,608 tons grading 0.268 oz. Au/ton across 3.70 ft.

TABLE III

UNDERGROUND ORE RESERVES
UNITED WESTERN MINE

<u>Class</u>	<u>Tons</u>	<u>Av. Width (ft.)</u>	<u>Av. Grade (oz. Au/ton)</u>
Reasonably Assured	118,054	4.1	0.259
Inferred	107,608	3.7	0.268

POTENTIAL RESERVES

Fischer-Watt Mining Co. Ltd., was actively engaged in gold exploration in the Oatman District from 1979 to 1982 and entered into a joint venture with Hecla Mining Company and Canadian Natural Resources in 1981.

Due to Hecla's budgetary restraints in late 1982, the joint venture was terminated, however the joint venture did carry out extensive geological mapping, compilation of subsurface data, and diamond drilling in the Tom Reed vein system area.

Drilling of the northwest extension of the United Western vein intersected a seven-foot interval assaying 0.115 oz. gold/ton. Additional sampling in the same hole by Hecla in 1982 revealed a parallel vein of which three feet assayed 0.424 oz. gold/ton.

To the northwest on the 500 level of the United Western workings, two vein structures are mapped, providing an additional minimum 500 feet of potential.

In addition Hecla also indicated a parallel target zone some 1,000 feet in length in this area.

To the southeast of the workings diamond drilling intersected two intervals exhibiting true widths of 8.0 feet averaging 0.216 oz. gold/ton and 30 feet averaging 0.201 oz. gold/ton. A second hole intersected 7.0 feet grading 0.16 oz. gold/ton.

Geological and limited drilling data indicate that the northwest and southeast vein system extensions of the United Western Mine provide excellent potential for additional ore reserves.

CONCLUSIONS

The Oatman camp in the San Francisco Mining District has produced 3,300,000 tons of ore averaging 0.58 oz. gold/ton and 0.17 oz. silver/ton from 1897 to 1942.

The United Western Mine property consisting of 4 patented claims and 25 lode claims is held by Sun River Gold Corp., and is located on the main Tom Reed gold vein system from which approximately 2,000,000 tons were mined. Grades ranged from 0.30 oz. gold/ton to 1.10 oz. gold/ton.

The United Western Mine produced 40,000 tons with an average grade of 0.30 oz. gold/ton prior to 1940.

The property is underlain by the favourable Oatman Formation consisting of latite flows, latite tuffs and flow breccias.

The vein structures consist of quartz-calcite-adularia open space fillings along fissures containing fine-grained free gold.

Reserves have been calculated in the area of underground workings of the United Western Mine between the 400 and 950 levels consisting of the following:

<u>Class</u>	<u>Tons</u>	<u>Grade (oz. Au/ton)</u>
Reasonably Assured	118,054	0.259
Inferred	107,608	0.268

There is also excellent potential for developing additional reserves along the strike of the vein system both to the northwest and southeast.

A three-phased exploration and development program is recommended.

W. G. T. CONSULTANTS LTD.
CONSULTING GEOLOGISTS

RECOMMENDED EXPLORATION AND DEVELOPMENT PROGRAM

Extensive diamond drilling should be undertaken to test the known extensions of the United Western Vein system above and below the 500 and 850 levels.

Drilling to the northwest will be required to test the parallel structure encountered in previous drilling and to test an additional target area outlined by Hecla Mining Company which is located on the Western, Western No. 1 and Western No. 2 patented claims.

Drilling to the southeast is required to confirm and establish continuity and further extension in that direction.

The drill program will assist in delineating the mineralized sections of the vein structures along strike and to depth providing a substantial increase to the existing reserves.

A third phase consisting of the rehabilitation and dewatering of the United Western Mine shaft should be carried out to provide access to the mine workings to enable further evaluation including mapping, sampling and metallurgical studies to feasibility in preparation for mining.

ESTIMATED COST OF EXPLORATION AND DEVELOPMENT

Phase I

Canadian Dollars

1)	Surface Diamond Drilling (NQ Wireline) 4,000 feet @ \$30/ft., all inclusive	\$ 120,000.00
2)	Engineering, supervision, assays, etc.	10,000.00
3)	Contingencies @ 20%	<u>26,000.00</u>
TOTAL COST PHASE I		\$ 156,000.00

Phase II

1)	Surface diamond drilling (NQ Wireline) 6,000 feet @ \$30/ft., all inclusive	\$ 180,000.00
2)	Engineering, supervision, assays, etc.	20,000.00
3)	Contingencies @ 20%	<u>40,000.00</u>
TOTAL COST PHASE II		\$ 240,000.00

Phase III

1)	Equipment purchase and rentals	\$ 100,000.00
2)	Head frame	100,000.00
3)	Rehabilitation of shaft to 700 level	100,000.00
4)	Ventilation	30,000.00
5)	Underground drilling (parallel structures)	100,000.00
6)	Mapping, sampling, analysis, metallurgical studies, etc.	50,000.00
7)	Feasibility Studies	35,000.00
8)	Engineering, supervision, etc.	75,000.00
9)	Contingencies @ 20%	<u>118,000.00</u>
TOTAL COST PHASE III		\$ 708,000.00

TOTAL COST BOTH PHASES \$1,104,000.00

Respectfully submitted,



W.G. Timmins, P.Eng.
W.G.T. CONSULTANTS LTD.

April 7th, 1987

W.G.T. CONSULTANTS LTD.
CONSULTING GEOLOGISTS

CERTIFICATE

I, WILLIAM G. TIMMINS of the City of Vancouver, in the Province of British Columbia hereby certify that:

1. I am a geologist with offices at #550 - 1100 Melville Street, Vancouver, B.C., practising my profession for 22 years.
2. I am a graduate of the Haileybury School of Mines, Haileybury, Ontario, and attended Michigan Technological University.
3. I am a member of the Association of Professional Engineers of British Columbia.
4. I have no direct or indirect interest in either the property or securities of Sun River Gold Corp., or its affiliates, nor do I expect to receive any such interest.
5. This report is based on a study of published reports and maps, government reports, historical data, and an examination of the property carried out personally from January 18th - 23rd, 1987.
6. I hereby consent to the use of this report by the Company in connection with a prospectus or a statement of material facts relating to the raising of funds for this project.

DATED at Vancouver, in the Province of British Columbia, this 7th day of April, 1987.



WILLIAM G. TIMMINS, P.Eng.

REFERENCES

- Lausen, Carl. 1931. Geology and ore deposits, Oatman and Katherine Districts, Arizona. Ariz. Bureau Mines Bull. 131.
- Ransome, R.L. 1923. Geology of the Oatman Gold District, Arizona, USGS Bull. 743.
- Snell, J.C. 1983. Geological report on the Times-La Paz and Gold Range properties, Oatman District, Mohave County, Arizona. Private report prepared for United Southern Mines, Inc., Denver Colorado.
- Snell, J.C. 1982. Geological study and mine development program on the North Start and Gold Reed Mines, Oatman gold camp, San Francisco District, Mohave County, Arizona. Private report prepared for United Southern Mines, Inc., Kingman, Arizona.
- Parrish, I.S. 1980. Exploration Potential, Oatman District, Mohave County, Arizona.
- Muchow, D.R. 1981. Summary of Exploration Drilling by Fischer-Watt Mining Company in the Oatman District, Mohave County, Arizona.
- Knight, L.H., and Winston, M.R., 1982. Progress Report, Oatman Project, Mohave County, Arizona.
- Various news articles, letters, correspondence, maps, sections, Oatman Area, Arizona.

APPENDIX

ECONOMIC CONSIDERATIONS

The calculations involving the economics of the United Western Mine deposit are based on the following facts and assumptions:

The calculations are based on production rates of 250, 300, and 400 tons per day and the price of gold per ounce is expressed in increments of \$50.00 per ounce starting with \$400.00 through to \$550.00 per ounce.

Ore Reserves

225,662 tons in place at a weighted average grade of 0.263 oz. Au/ton.

Mining Dilution assumed at 15%

Reserves:

259,511 tons grading 0.224 oz. Au/ton

Milling Loss assumed at 5%

Reserves:

259,511 tons grading 0.212 oz. Au/ton (recoverable)

Potential Reserves

Estimated at an additional 300,000 plus tons of approximately the same grade (0.212 oz. Au/ton recoverable).

Mining and Milling Costs

Estimated at \$40.00/ton

Mining and Milling Rates are estimated at 330 days per year.

Cost estimates include an additional \$500,000/year to cover intangible costs.

Shares issued and outstanding estimated at 3,000,000

All dollar values are expressed in U.S. funds.

No consideration has been given for contained silver values.

It is estimated that it will require approximately \$3,000,000 to place the property into production.

\$400/oz. Au

<u>Production Rate</u>	<u>Recoverable oz. Au/ton</u>	<u>Tons Mined & Milled/year</u>	<u>oz. Au Produced/yr.</u>	<u>Gross Revenue/yr.</u>	<u>Costs/yr.</u>	<u>Net Before Taxes/yr.</u>	<u>Earning /Share</u>
250TPD	0.212	82,500	17,490	6,996,000	3,800,000	3,196,000	1.06
300TPD	0.212	99,000	20,988	8,395,200	4,460,000	3,935,200	1.31
400TPD	0.212	132,000	27,984	11,193,600	5,780,000	5,413,600	1.80

\$450/oz. Au

250TPD	0.212	82,500	17,490	7,870,500	3,800,000	4,070,500	1.36
300TPD	0.212	99,000	20,988	9,444,600	4,460,000	4,984,600	1.66
400TPD	0.212	132,000	27,984	12,592,800	5,780,000	6,812,800	2.27

\$500/oz. Au

250TPD	0.212	82,500	17,490	8,745,000	3,800,000	4,945,000	1.65
300TPD	0.212	99,000	20,988	10,494,000	4,460,000	6,034,000	2.01
400TPD	0.212	132,000	27,984	13,992,000	5,780,000	8,212,000	2.74

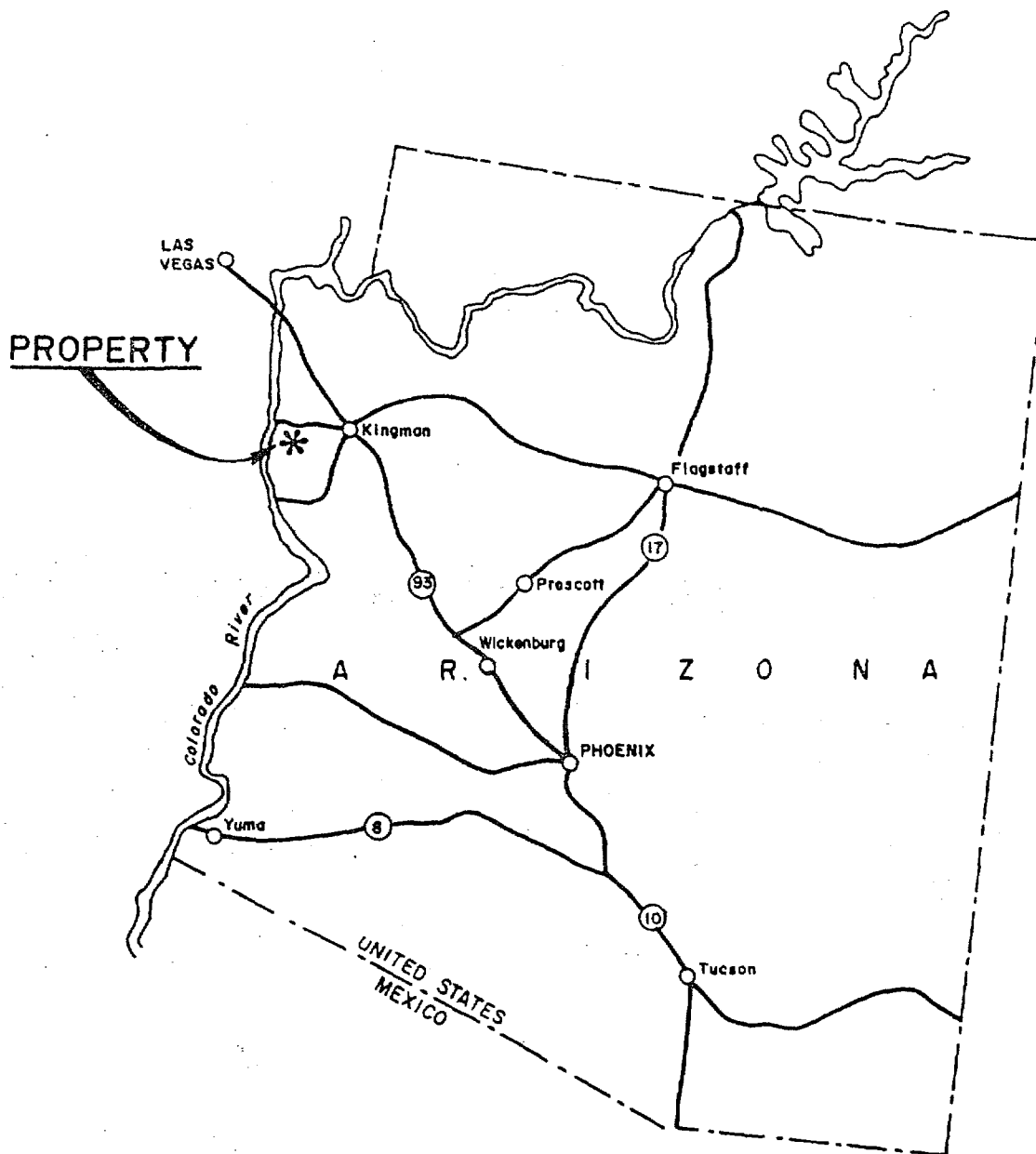
\$550/oz. Au

250TPD	0.212	82,500	17,490	9,619,500	3,800,000	4,819,500	1.94
300TPD	0.212	99,000	20,988	11,543,400	4,460,000	7,083,400	2.36
400TPD	0.212	132,000	27,984	15,391,200	5,780,000	9,611,200	3.20

Earnings per Share per Year

No consideration has been given to depletion allowances, depreciation or taxes.

The number of shares currently outstanding is approximately 3,000,000.



SUN RIVER GOLD CORP.

OATMAN PROJECT, ARIZONA

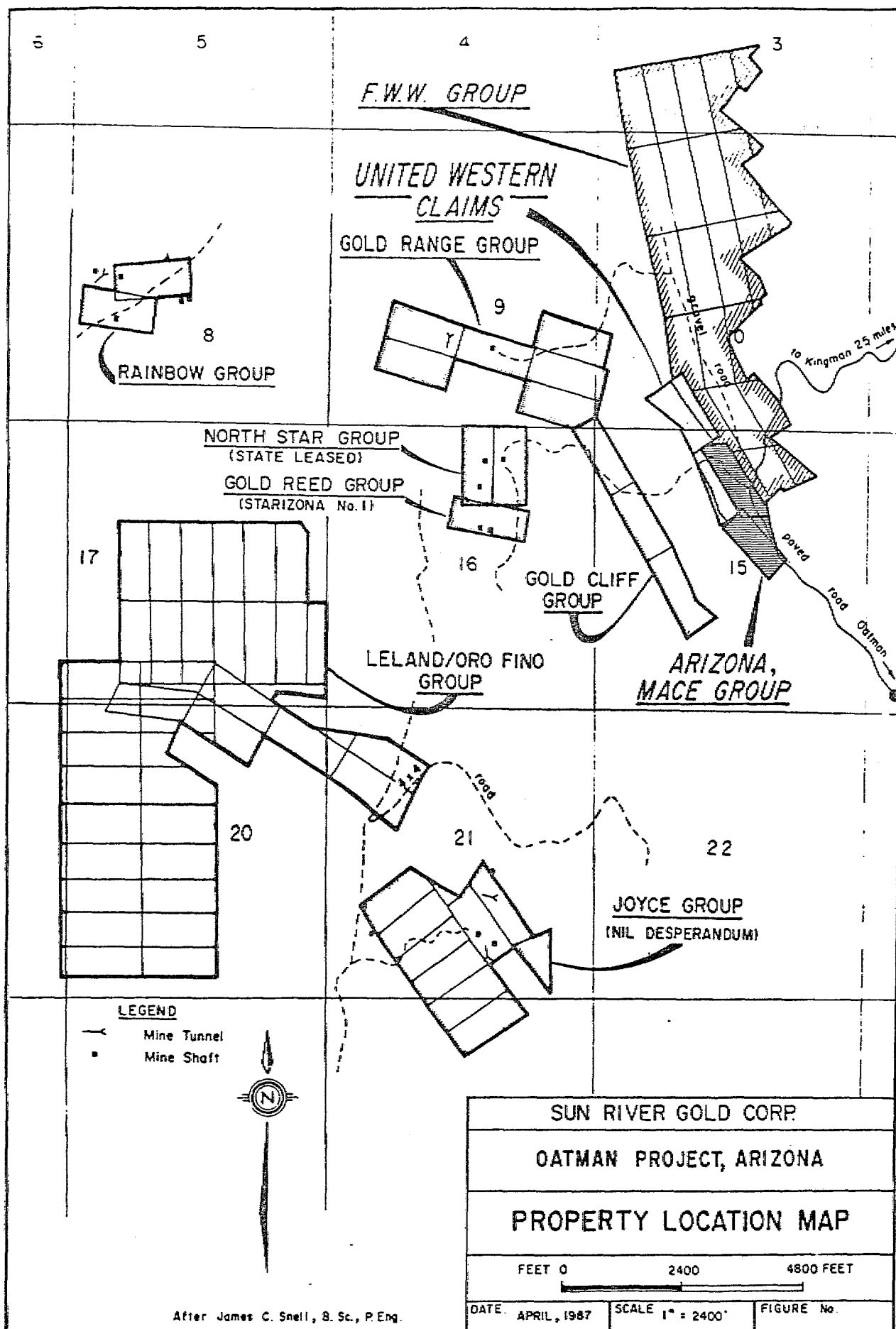
LOCATION MAP

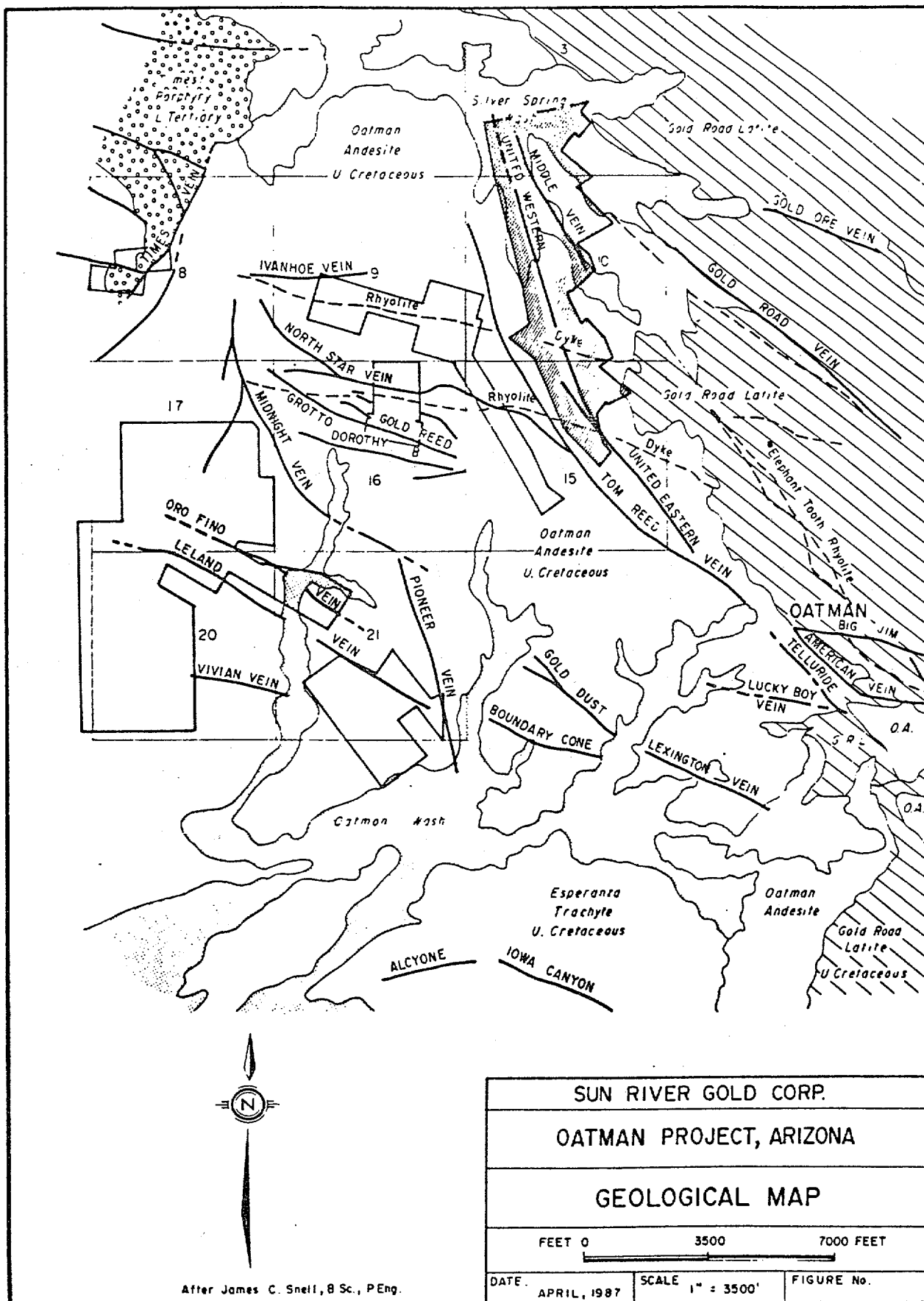
MILES 0 25 50 100 150 MILES

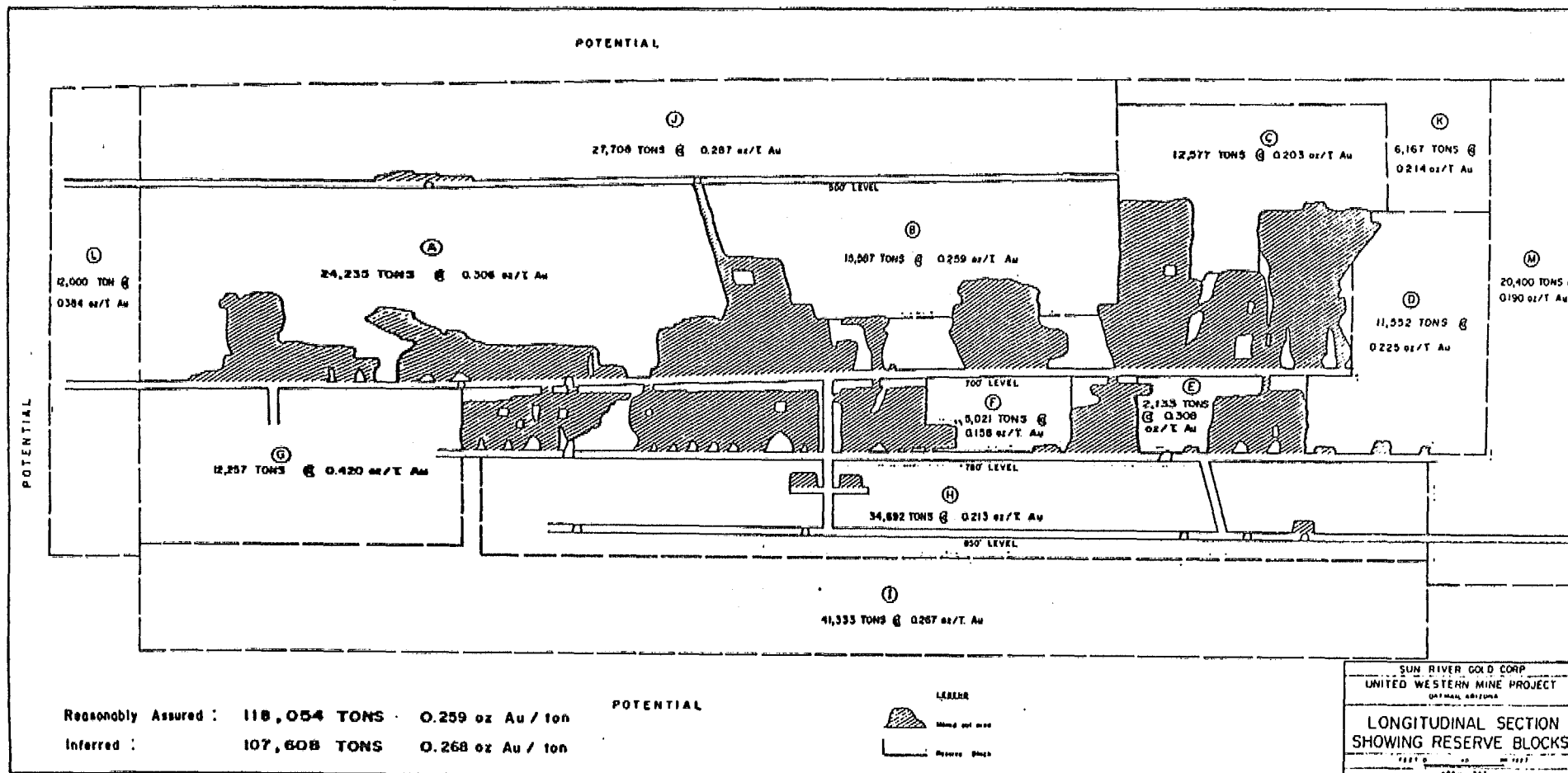
DATE: APRIL, 1967

SCALE: As Shown

FIGURE No.









CERTIFICATE OF THE DIRECTORS AND PROMOTERS OF THE ISSUER


The foregoing constitutes full, true and plain disclosure of all material facts relating to the securities offered by this Statement of Material Facts.


DATED: December 14, 1987.


THOMAS ROBERT TOUGH
Chief Executive Officer/
President/Director/Promoter


CORWIN COE
Director


TATJANA KATHARINA PESSL
Chief Financial Officer/
Secretary-Treasurer/
Director


GERARD O'DOWD
Director

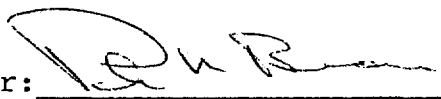

MURRAY MOXLEY
Director

CERTIFICATE OF THE AGENTS

To the best of our knowledge, information and belief, the foregoing constitutes full, true and plain disclosure of all material facts relating to the securities offered by this Statement of Material Facts.

DATED: December 14, 1987.


CANARIM INVESTMENT CORPORATION
LTD.

Per: 

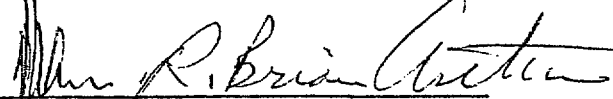
MCDERMID, ST. LAWRENCE LIMITED

Per: 

JEFFERSON SECURITIES INC.

Per: 

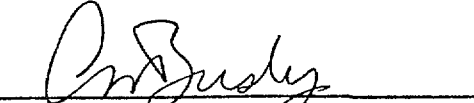
GEORGIA PACIFIC SECURITIES
CORPORATION

Per: 

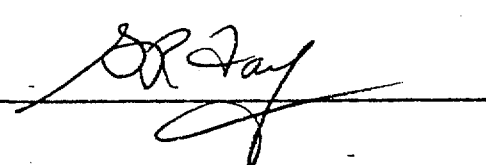
BRINK, HUDSON & LEFEVER LTD.

Per: 

DAVIDSON PARTNERS LIMITED

Per: 

CONTINENTAL CARLISLE DOUGLAS

Per: 



Joseph G. Colwell



Alan R. Bell



C. George Lynn



Donald H. Freas



Thomas J. O'Neil



Millet L. Wei

Joseph G. Colwell, Jr. has been promoted to production manager/chief engineer of the Wrens, GA, and Edisto, SC, locations of J. M. Huber, Clay Division. Colwell will be responsible for engineering, maintenance, mining, and processing operations at both locations.

Alan R. Bell has joined Nevada Goldfields, Inc. as vice president, general manager, and member of the board of directors. Bell was previously executive vice president of Sharon Steel Corp. and has extensive international mining experience.

John Mathes & Associates, Inc. has announced the addition of **C. George Lynn** to the firm's Columbia, IL, office as senior hydrologist and manager of the Environmental Sciences Group.

Donald H. Freas has been named group president of the newly formed metals/mineral group of Applied Industrial Materials Corp. to be headquartered in Pittsburgh, PA.

Cyprus Gold Division has announced the appointment of **Thomas J. O'Neil**, vice president, South Pacific operations. O'Neil will be headquartered in Sydney, Australia, and will be responsible for all Cyprus Gold's operations in that region.

Two Bethlehem Steel Corp. engineers, **Millet L. Wei** and **Kripa N. Singh**, have won first place in the Association of Iron and Steel Engineers' Kelly Award program for their technical paper, "Design of Refractory Lining and Steel Shell for Caster Ladles."

Brian W. Hester has joined the partners of Derry, Michener, Booth, and Wahn, consulting mining engineers and geologists, as senior associate. Hester joins DMBW from a gold exploration project in Tanzania where he was exploration manager.

The newly formed International Process Research Corp. (formerly Colorado School of Mines Research Institute) has announced the appointment of **Gordon C. (Bud) Presley** as manager of industrial minerals.

Patrick E. Phillips has joined Echo Bay Mines in their Denver office as manager of development analysis.

Anthony B. Szwilski is leaving the University of Kentucky to begin a consulting assignment on engineering education and energy-related projects in Indonesia sponsored by the Asian Development Bank.

A University of Missouri-Rolla alumni achievement award was presented to **Henry P. Whaley** by the MSM-UMR alumni association. The award recognizes outstanding service in the areas of business, academics, or civic achievement. Whaley is senior vice president of Cliffs Mining Co., Cleveland, OH.

The firm of Don Galloway, PE, has announced its association with **R. E. (Ralph) Romero**. Romero has 31 years professional experience in the design and construction of fueling and lubrication systems for commercial, industrial, municipal, and airport facilities. Romero was formerly vice president-engineering with J-8 Equipment Co.

Ronald Wardle has joined Basic Chemicals as director of technology and is now responsible for magnesia and related product development, process development, and customer technical support. Wardle was previously employed by Combustion Engineering in quality assurance, technical application, and process development.

Raul Eyzaguirre has been named manager of Corporacion Minera Nor Peru, SA, succeeding **Carlos Jahnke** who has retired. Corporacion Minera Nor Peru is a subsidiary of Asarco and operates the Quiruvilca silver-copper-lead-zinc mine in Peru. Eyzaguirre had been serving as chairman of Centromin, the Peruvian government's mining, smelting, and refining complex, since 1985.

The board of directors of Placer Dome, Inc. has confirmed the appointment of **W. Anthony Triggs** as senior vice president, project development. Triggs was formerly vice president, project development, and will now be responsible for evaluation and development of new mining properties.

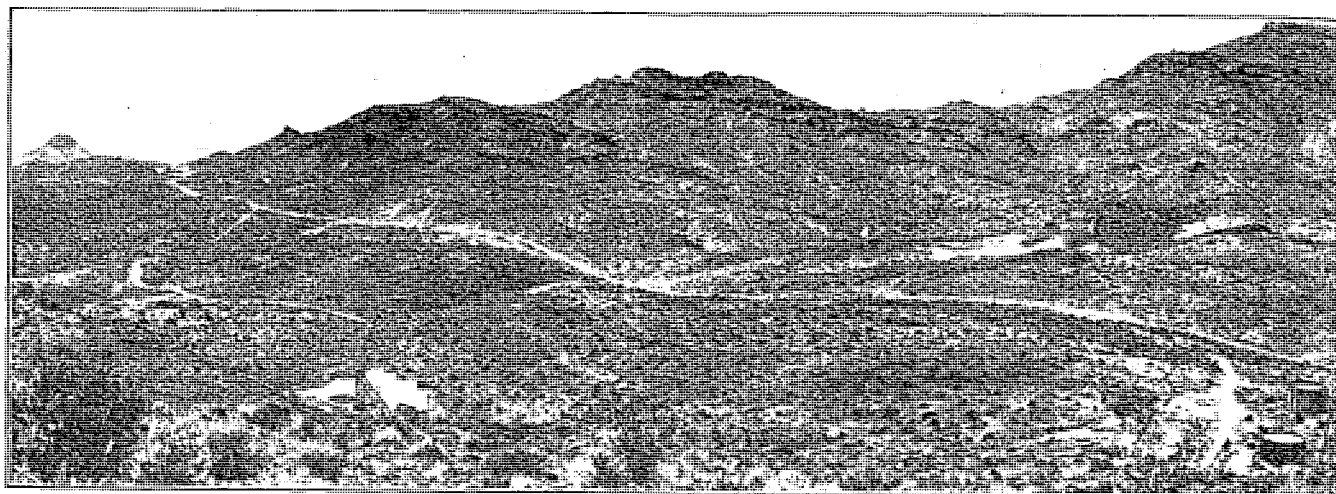
James R. Latick has retired from USS Minnesota Ore Operations after 30 years service. Latick had been division manager-mining since 1986 and has had recent experience in open pit natural iron mines and the Minntac operation on the Iron Range. His career with USS began in 1957 as a management trainee in the Hibbing-Chisholm district of the Oliver Iron Mining Co.

Anthony P. Tripolino, formerly vice president and general manager of Electro-Coal Transfer, will become president of Electro-Coal Transfer and vice president-administration and terminal operations of TECO Transport and Trade.

The appointment of **Michael C. Korb** as division general manager of the Panther Valley Division of Bethlehem Mines has been announced by **Robert M. Funk**. Korb has also been appointed division general manager of Drummond Dolomite, Inc., a Bethlehem stone quarrying subsidiary in Michigan.

Mike Read has joined MINEsoft, Ltd. as director of programming. Read's duties will include directing upgrades to the company's software systems, documentation, and customer support.

Homestake Mining Co. has announced that **R. Ray Beebe**, vice president project development, has assumed additional responsibility for corporate engineering, metallurgy, and mine engineering for the activities of Homestake International Minerals, Ltd.



United Western.

Main Road to Gold Road.

PANORAMIC VIEW OF A

RECEIVED**FEB 29 1988****EXPLORATION DEPARTMENT**

The Oatman District, Arizona

By HOWARD D. SMITH*

A recent mining discovery has revived interest in the Oatman district. Therefore some notes gathered during a recent short inspection may be of interest.

This district is situated in the Black Mountains, in Arizona, but only 20 miles north-east of Needles, in California. Passengers from the west, now that the Colorado river is low, can cross in a launch at Needles and proceed thence by regular motor-service. Heavy freight is hauled from Kingman or Topock, both in Arizona, the first named being the best railroad station for Eastern passengers. A railroad to serve the Leland property, a few miles south-west of Oatman, was constructed in 1903, but what remained after severe floods was hauled away. Even the ties are now being used for timbers in the United Eastern mine. There is little likelihood of this railroad being re-built, as all the ore produced in the district is treated locally and motor-trucks will probably prove economical for freight.

About 1863 John Moss discovered what has continued to be known since as the Moss mine at a point four miles north-west of the Gold Road mine. Some \$200,000 was produced shortly after the discovery from a small hole on the surface. This showing attracted prospectors to the district. After the rush of the first-comers, little was done until 1902, when the Gold Road mine was opened up. Shortly afterward that picturesque character, Col. Thomas Ewing, promoted a number of properties. While some rich material was produced, his enterprises on the whole were not profitable. Fresh work is to be done, however, on his Victor and Virgin property, now owned by the Orion Mining & Milling Company.

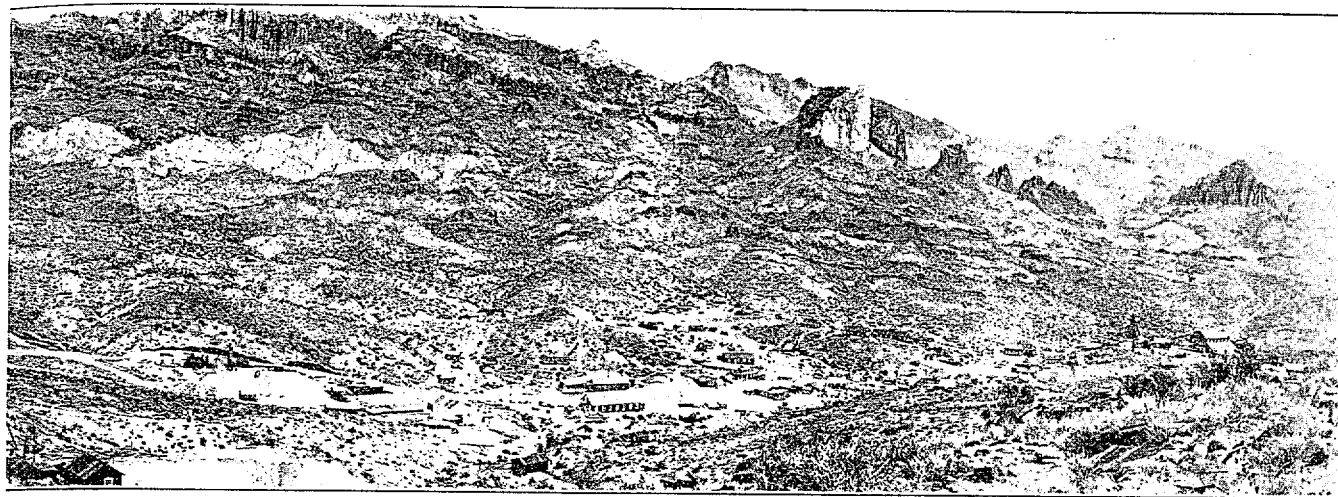
The prevailing country-rock is chloritic andesite, occurring both as intrusions and as flows, according to F. C. Schrader in U. S. Geological Survey Bulletin No. 397. This intrudes or overlies an older andesite, which in turn overlies a gneissoid granite. In the north-eastern part of the district, extending toward the Gold Road mine, the chloritic andesite is covered by undifferentiated flows of andesite, rhyolite, and other volcanic rocks. Overlying, and intrusive into the rocks heretofore mentioned, are prominent dikes of rhyolite, and finally more andesite and basalt, both as sills and dikes.

For those without the technical knowledge possessed by the geologists of the Survey, the official report, although highly interesting, is difficult of comprehension. As far as I can ascertain, the miners in the district have got little out of it except the idea that the flows of chloritic andesite would have a maximum thickness of 800 ft. and that, as the veins are not likely to be profitable in the underlying older andesite, the maximum depth for successful exploration would be 800 ft. The reported existence of \$20 ore in quantity on the 1075-ft. level of the Tom Reed mine, where the collar and bottom are both in the chloritic andesite, is held to discredit Mr. Schrader. If, however, the chloritic andesite, is considered as an intrusion at this point, which is not unlikely, then this find of ore at 1075 ft. is not out of harmony with Mr. Schrader's expressed views. I hope to gain some additional knowledge at first-hand in the future, but at present I doubt whether the formation of ore-shoots can be shown to be controlled by the particular andesite serving as the wall-rock. As a constructive criticism it might be pointed out that if the learned geologists of the Survey would keep in mind the fact

*Mining Engineer, Kohl Bdg., San Francisco.

*WARR - Came across
this while seeking
something else.*

J.M.S.



United Eastern.

Town of Oatman.

Tom Reed.

PORTION OF THE OATMAN DISTRICT.

that they are writing such bulletins primarily for those without technical advantages, more benefit would be derived from their writings.

In general, the veins are strong, persistent, and readily followed. Some, it appears, have been traced for miles. Widths of over 20 ft. are common. Where a more complete substitution of quartz for calcite is observed, the outcrops have resisted weathering and can be seen for miles.

The prevailing strike is SE-NW, with dips at high angles NE, but occasionally SW. Veins are commonly frozen to the country-rock, and the hanging wall is usually the best defined.

The gangue consists in large measure of altered rock cemented by calcite and quartz. Schrader mentions adularia, a variety of orthoclase free from sodium, as another gangue-mineral. The ore is well oxidized. The value of the ore is due almost exclusively to its gold content. Silver is present in small proportion. According to my information the belief seems justified that more gold is found where the quartz replacement of calcite is most nearly complete. Experience in both the Tom Reed and Gold Road mines indicates that the richest orebodies are found from about the 300-ft. level down to around the 500-ft. level. Below that a marked diminution in richness is noted. While rich quartz is often found near the surface, the masses are usually small and irregular. The presence of vugs and manganese oxide in the upper portions of the veins suggests that leaching may have brought about a secondary enrichment at about the 500-ft. level in the mines.

At the present time, the most interesting territory is that shown in the accompanying panoramic photograph. For some years past the Tom Reed company has operated successfully. Three ore-shoots on a vein striking $N50^{\circ}W$ and dipping $75^{\circ}NE$ have been exposed. The workings extend north-west from the main shaft, which is now 800 ft. deep. The longest shoot in the upper levels had a pronounced rake to the north-west, but when

followed by a winze about 1200 ft. north-west from the main shaft, the major axis and the north-west limit of the stopes became nearly vertical. This limit is reported to be about 240 ft. south-east of the United Eastern end-line. It may be mentioned that the Tom Reed company heretofore has pursued a policy of secrecy, at least in regard to the underground workings, no visitors being allowed access thereto. The absence of authentic information gave birth to many rumors. Messrs. Long and McIver, then working in the Tom Reed mine, became erroneously convinced, it appears, that the Tom Reed westerly ore-shoot was passing into the ground now owned by the United Eastern. They accordingly obtained an option on this ground, and started a shaft approximately 365 ft. north-west of the Tom Reed end-line. After working slowly for some time, owing to limited means the co-operation of Frank S. Keith and associates was obtained, and the shaft sunk to the 565-ft. level. According to the management, at the 307-ft. level a cross-cut shows 4 ft. of ore assaying \$14 per ton; at 460 ft. a corresponding cross-cut 22 ft. long exposed ore assaying \$22 per ton, and on the 565-ft. level, the vein shows 22 ft. of ore assaying \$35 per ton. On the foot-wall side of the vein on the 460 and 565-ft. levels, there is, in addition, 15 to 16 ft. of material assaying about \$3 per ton. On the 460-ft. level about 100 ft. of development north-west on the hanging-wall side has exposed low-grade ore, better ore being found, presumably, on the foot-wall side of the drift, as indicated in the cross-cut. Drifts 28 ft. north-west and 120 ft. south-east on the hanging wall at the 565-ft. level show an average of approximately \$50 per ton for the width of the drift. Thus it appears that the concealing of information by the Tom Reed company was a powerful factor in bringing about the development of this promising property. Much difference of opinion exists as to whether the United Eastern vein, with a hanging-wall strike in the south-east drift of the 565-ft. level of approximately $N14^{\circ}W$ and a dip $75^{\circ}NE$, is a continuation of the vein exploited by

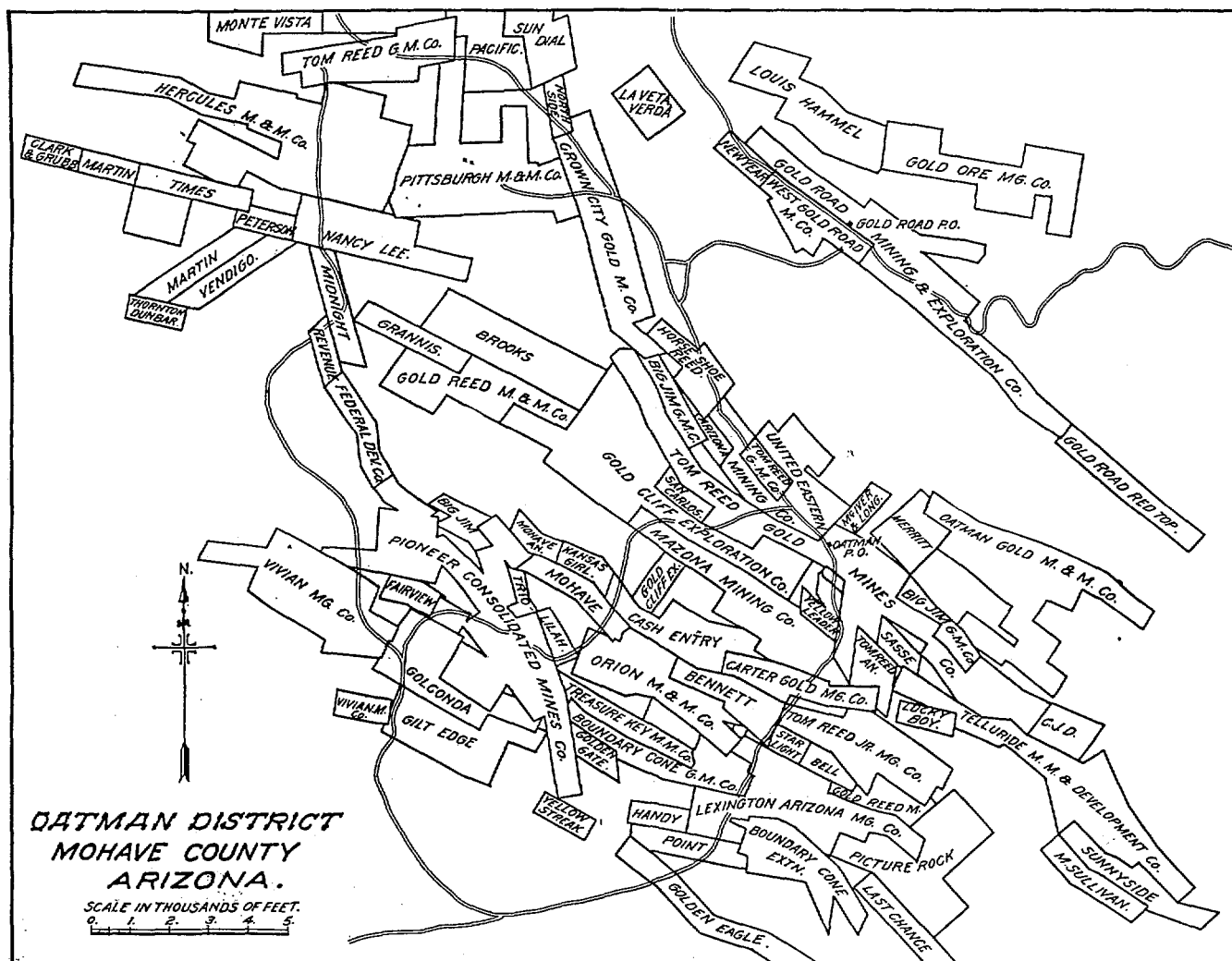
the Tom Reed company, which has a strike of approximately N 50° W and a similar dip. Going N 50° W from the main shaft of the Tom Reed leads in the direction of the Oatman shaft of the Tom Reed. This, together with the information that a level follows the vein continuously between these shafts, would indicate the contrary. The course followed in the United Eastern workings thus far, provided it persists, will cause an intersection shortly with the Tom Reed workings. The use of air-drills at the United Eastern will hasten the connection suggested.

The property of the United Western is situated about a mile north-west of Oatman. W. H. Weed states that, in his opinion, this is a continuation of the deposit of the United Eastern. A shaft is being sunk a few hundred feet from the eastern boundary of the property. Extensive lateral development is proposed when the 400-ft. level is reached. Mr. Weed expresses the opinion that the geologic evidence is just as favorable for the occurrence of ore as that observed in the United Eastern. It is to be hoped that the venture here will also result favorably.

The Gold Road mine, now operated by the United States Smelting, Refining & Mining Co., is about a mile and a half north of the Tom Reed property. It is in-

cluded in the Oatman district. The vein, though occurring in the undifferentiated volcanic rock overlying the chlorite andesite, is similar to the Tom Reed as regards width, extent, and character of the gangue. The gold content is lower, however. One of the orebodies cropped strongly, so that large open stopes can be seen for some distance near the No. 1 shaft. Although the metallurgical treatment is similar, namely, crushing in cyanide solutions, without amalgamation, followed by treatment in Pachuca vats and Dorr thickeners, the color of the Tom Reed tailing is much darker. This indicates a difference not apparent in the untreated ore. It is interesting to note that Dorr thickeners appear to be particularly suited to conditions in this district, where little silver is present, weak solutions are used, and slime settles quickly.

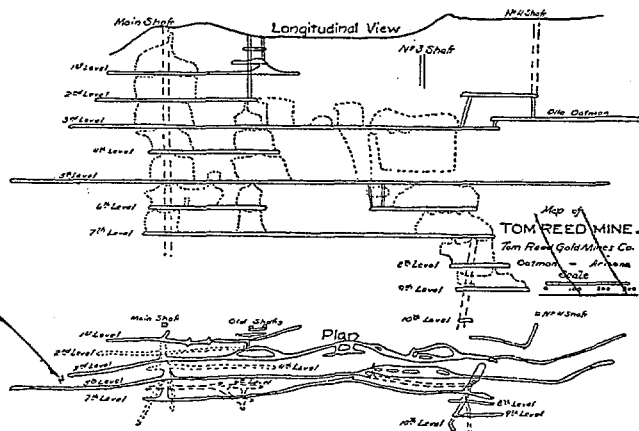
The old Moss mine, where the original discovery was made, is now to be developed under option by the owners of the Gold Road mine. A mining boom now being well under way, development is proposed and is actually being started on many other properties. Oatman itself has outgrown, almost overnight, its site in the vicinity of the mining plant of the Tom Reed property. A new town, Old Trails, located a little over a mile down the arroyo, is projected. A wide avenue in the dry bottom is already



MAP OF CLAIMS IN THE OATMAN DISTRICT. PREPARED BY HIAFF & COLWELL, LOS ANGELES.

cleared of rocks. The district, to one familiar with boom camps, has a subdued air, particularly after dark. This quiet may be credited in part to prohibition. Let us hope that the energies formerly so prodigally wasted in dissipation will here be expended on actual work underground. A touch of human interest was given to the daily work of Oatman by the coming of Joe Parrizo with two motor-loads of relatives. He had recently received a payment for his interest in what is now the United Eastern property, and insists on having his relatives, whom he had not seen for 20 years (so the story goes) enjoy his success with him. Thus even prohibition does not wholly kill the romance of mining.

Many other properties are either doing development work, or propose to do so. No other bodies of material sufficiently valuable to justify the erection of a treatment plant are known to exist besides those already mentioned. The local slogan is "Gain depth," the inference being that this will ensure finding a mine. The United



PLAN AND SECTION OF TOM REED WORKINGS.

Eastern is instanced as conclusive evidence on this point. As against this, it may be mentioned that development on the United Eastern is at present limited, and the experience of the Tom Reed may be repeated, which followed with a stope to within fifty feet of the surface, a shoot discovered at depth. Further, there is the fact that a considerable number of shafts have been sunk, in some cases over 500 ft., without success. Inadequate lateral development therefrom is reported, however.

The problem in this district is not so much the finding of veins, as the discovery of ore-shoots in them. Nothing sufficiently tangible was brought to my attention to use as the basis for a theory to guide operators in the search for ore. That considerable areas in the veins without ore must be expected is certainly to be inferred from experiences in the Gold Road and Tom Reed mines. Consequently, sufficient capital should be provided for adequate lateral exploration before shaft-sinking is started. It remains to be proved whether the large amount of work now under way in the district will demonstrate that there are signs sufficient to make the successful search for ore something more than a fortuitous gamble.

Copy of a Contract for Tin Ores Between European Smelters and Bolivian Miners

DG
Copy.

TERMS OF SALE OF A. & C. TIN ORE FOR ACCOUNT OF
MESSRS. ABELLI & CO.

The whole of the shipments of Bolivian Tin Ore to your consignment of the mark:

A. & C. to the extent of about 80 (eighty) tons monthly during the months of March-August, 1914, both inclusive, without guarantee of quantity.

A.

Sellers to have the option of supplying any shortage in any month from surplus shipments in subsequent months, provided such shipments are made not later than the last months mentioned in the preceding clause. Bill of Lading date to be decisive as to the period of shipment.

The maximum quantity tenderable under this Contract is 576 (five hundred & seventy-six) tons, say 96 (ninety-six) tons per month per six months.

WEIGHING, SAMPLING, AND DELIVERY in bulk in ore-yard, Liverpool, at Sellers' expense.

TIN ASSAYS TO BE EXCHANGED by same post, to two points of decimals, and agreed between buyers and Mr. A. H. Knight, Liverpool, acting for Sellers. Should these results differ to the extent of more than half of one per cent, then either Buyers or Sellers to have the option of claiming non-agreement, and a reference sample to be sent to either Messrs. W. H. Christoe & Sons, Truro, or Messrs. F. Claudet, Ltd., London, and their result taken as final if between the results of the original assays, but if otherwise then the original assay of either Buyers or Sellers nearer thereto to be taken as the agreed assay, losers paying cost of reference.

MOISTURE LOSS to be also agreed between Buyers and Mr. A. H. Knight, on sealed bottle samples drawn at time of weighing.

TIN PRICE REGULATING THIS CONTRACT to be the Mean of the Cash and Three Months official prices of Standard Tin, as quoted by the London Metal Exchange on the market days comprised in the period of 14 (fourteen) days following the day on which each steamer bringing Ore under this Contract is reported in Liverpool Customs Bill of Entry.

Broken prices to be regulated and agreed as follows:

2s.6d. and under	7s.6d., 5s.
7s.6d. " "	12s.6d., 10s.
12s.6d. " "	17s.6d., 15s.
17s.6d. " "	2s.6d., 20s.