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Richard Mieritz Mining Collection

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AMERICANA INVESTMENTS, INC.
PHOENIX, ARIZONA

DETACH AND RETAIN THIS STATEMENT
THE ATTACHED CHECK IS IN PAYMENT OF ITEMS DESCRIBED BELOW.
IF NOT CORRECT PLEASE NOTIFY US PROMPTLY. NO RECEIPT DESIRED.

DATE	INVOICE NUMBER	DESCRIPTION	AMOUNT	DEDUCTIONS		NET AMOUNT
				PARTICULARS	AMOUNT	
		(SALE) White Chief 2/12/64				100 00

METALS ENGINEERING CO.
P. O. BOX 597 — 201 E. 4th ST.
CASA GRANDE, ARIZONA

January 4, 1964

SUMMARY OF "OUT-OF- POCKET" EXPENSES TO ACCOUNT OF AMERICANA INVESTMENTS, INC.

October 17, 18 & 19, 1963:

3 men sampling White Chief (2 days time)

1 man @ $30.00	$60.00	
1 man @ $22.00	44.00	
1 man @ $12.00	24.00	
	128.00	
Plus Ins. & Taxes (10%)	12.80	
	$140.80	$140.80

Mileage (Pickup)

530 miles @ 10¢	53.00
Meals & expenses (3 men)	28.00

(No charge for Freemans time or car expenses)

October 21, 1963:

Ballams all day on preparing samples from Oatman and delivering to Tucson.

1 day	$30.00	
130 miles @ 10¢	13.00	
	$43.00	43.00

October 29, 1963:

Trip to Phoenix to meet with Hookers

110 miles @ 10¢ (No time charged)	11.00

November 4 & 5, 1963:

1 man 2 days @ $30.00	$60.00	
1 man 2 days @ $22.00	44.00	
	104.00	
Plus Ins. & Taxes (10%)	10.40	
	$114.40	114.40

Metals Engineering Co.
Summary of Out-of-Pocket
Expenses- Americana Investment
Co.

November 4 & 5, 1963 Con't.

530 miles car travel @ 10¢	$ 53.00
Room (4 men - Incl. Andy Zinkl)	47.12
Meals (4men) - 2 days	22.60

November 11, 1963:

½ trip to Phoenix to meet with Hookers and Zinkl.

110 miles @ 10¢ - $11.00 - ½ 5.50

November 21, 1963:

½ trip to Phoenix to meet with Hookers and Zinkl. 5.50

December 4, 1963:

Trip to Phoeinx to meet with the Hookers, Gen. Stone, etc. 11.00

Assaying Charges 37.50

Testing Costs 25.00

Total..... $597.42

G. A. Freeman
Metals Engineering Co.

GAF/vf

November 20, 1963

Americana Investment Corp.
4328 N. 56th Street
Phoenix, Arizona

Gentlemen: Re: WHITE CHIEF MINE

On Tuesday November 5, 1963, I accompanied John and Gary Hooker, along with George Freeman of Metal Engineering Company, to your White Chief Mine in the Oatman area.

My brief examination was for the purpose of evaluating the exploratory possibilities of your mine in the light of ore potential and production possibilities. Prior sampling by Metal Engineering Company and additional sampling on this date are the basis for my conclusions. Also influencing my conclusion is the over-all geological pattern of both the area and your vein.

I've had the advantage of studying Mr. Freeman's report and consider my comments here as supplemental to his statements.

I very forcefully concur with George Freeman's conclusions, most particularly his points number (1) and (3). The assays are extremely low and the vein is not in pattern with the good producing veins of the area.

Along the line of this latter factor it is noteworthy that not a single producing mine of the area had a strike direction similar to the White Chief; all producers were related to a general East-West strike direction and very close to the last intrusives which caused the major faulting and subsequent mineralization.

The area of the winze on the 190 level is a slight structural anomaly in your vein pattern, thereby being the most receptive to the better mineralization. Actually the samples immediately adjacent to and within the winze area are the best taken on the entire level, bearing out the greater susceptibility to better values.

However, these values are much too low to justify any optimism in depth should the anomaly continue downward. Actually it appears that the controlling factor was a slight change in strike direction at this point, and the vertical or

dip structure remained uniform -- so that the samples below this level would probably be similar to the level samples. This is borne out by the assays of the winze sample.

One very obvious fact is that no ore was stoped, mined, from this vein, even during the days of much lower operating costs. This would indicate that the entire development work on these levels did not open any mineable ore. My experience in old properties has shown that the old-time miner seldom, if ever, left any profit-making ore in place for someone to extract years later.

As to the possibility of ore being richer at depth, which is typical of the area, I do not feel this would be true on this structure; actually I would not be surprised to see a tightening of this vein at depth to just a few inches of low grade quartz ore. I say this because of its lack of direct relationship to the main mineralized zone and because of its adverse strike. Actually this vein is probably a pressure relief fracture which was slightly mineralized during one, or at the most two, of the five or six periods of mineralization of the area.

My very firm recommendation to Americana is to remove your equipment and leave the White Chief property to its owners.

Very truly yours,

AJ Zinkl

Andrew J. Zinkl
Registered Mining Engineer

AJZ/LL

c.c. George Freeman

REPORT OF THE WHITE CHIEF MINE
OATMEN, ARIZONA

NOVEMBER 13, 1963

Metals Engineering Company was primarily called on to examine the White Chief Mine for the purpose of determining whether the existing ore veins on the 190 foot level were high enough grade and of sufficient widths to warrant further exploration expenditures at depth. The geology of the district is well covered in previous reports including the Bureau of Mines.

There has been approximately 200 feet of drifting on the strongest vein or vein system that was found on the White Chief holdings. This vein strikes approximately North 10 degrees East with a dip of about 80 degrees. I would describe this as a zone of veins and fracture fills in andesites and represents overall widths of 3 feet to 5 feet. In sampling this vein system that was drifted on, we took bulk samples across the full width of the vein structure and in most cases across the full width of the drift since this would be the width that you would have to mine.

We actually sampled a total length of 155 feet, which previous sampling had indicated was the highest grade section. The average grade for this 155 feet was $5.45 with an average total width of about 4 feet. Previous sampling for this zone indicated an average grade of approximately $40.00 per ton. See attached sample map for sample assays and comparison.

Our sampling was certainly very discouraging but since previous samplings indicated an extremely high grade zone close to the winze, shown on attached map, it was decided to de-water this winze, which was sunk 35 feet below the 190 foot level, and sample it. We made a channel cut across the vein zone 8 feet above the bottom of this winze. This channel cut was made in 3 parts, 23" of the quartz veins in hanging wall, 37" of andesite in center containing a few quartz stringers and 12" of quartz vein on the

footwall. The assays of these 3 samples were as follows:

23" Hanging wall	o.o3 Au	0.3 Ag
37" Andesite	o.o4 Au	0.2 Ag
12" Footwall quartz	o.o1 Au	0.4 Ag

These assays were even more discouraging and also indicated that the chances were poor for the values to increase the depth. In addition to the values being lower in the winze, the ore bearing quartz was less predominant than in the drift immediately above.

SUMMARY:

In our opinion the White Chief Mine does not offer what we would consider even reasonable chances to develop an ore body of commercial interest. This decision was based on the following factors:

(1) The assay values that we came up with from our sampling of the vein on the 190 foot level were certainly much too low to be of any commercial interest.

(2) The sampling near the bottom of the 35 foot winze did not indicate that the chances were at all good for the values or vein strength to increase at depth as was the history of some of the veins in the Catman district.

(3) From a geologic standpoint, the only vein of interest as exposed at the White Chief Mine is striking contrary to all of the major producers in the district. That is, striking nearly at right angles to the main vein systems.

(4) Further, lets assume that the 200 foot of ore shoot exposed on the 190 foot level increased four times in grade and assayed $20.00 per ton at 100 feet greater depth and had an average of 4 feet in width, then in this 100 foot you would only have an 8000 ton block of ore. Your mining, hauling and milling cost would be at least $15.00 per ton. We would consider this a pretty small target to gamble the amount of money that would be required.

Mr. A. J. Zinkl , a Mining Engineer, was called on to study our sampling and results and also make an independent examination of the mine. He will submit a seperate report as to his evaluation of the mine.

Respectfully submitted,

METALS ENGINEERING COMPANY

By:

G. A. Freeman

Jack Ballam

METALS ENGINEERING CO.
P. O. BOX 597 — 201 E. 4th ST.
CASA GRANDE, ARIZONA

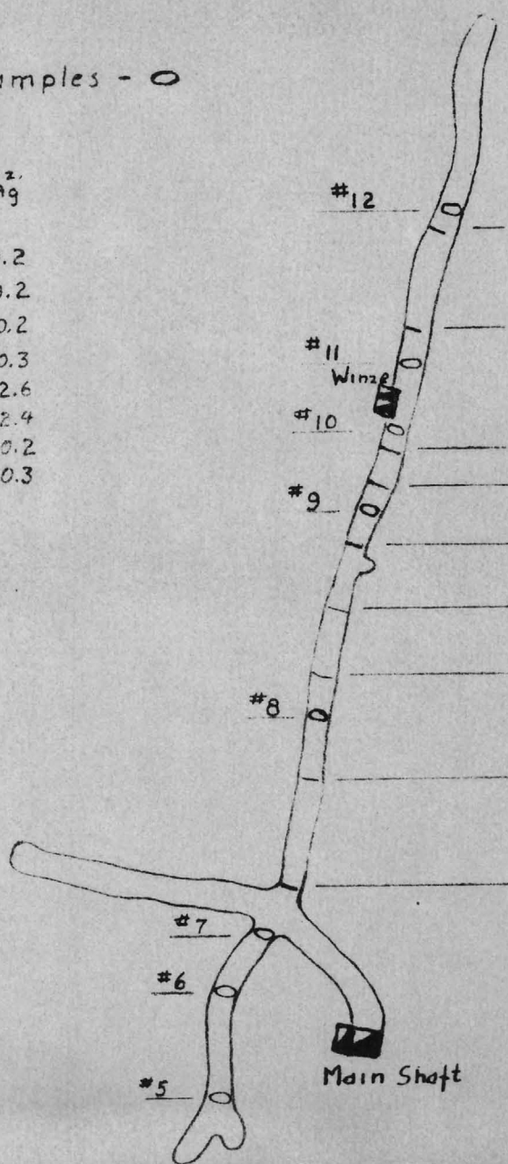
WHITE CHIEF MINE
OATMAN, ARIZONA

SAMPLE MAP - 190' LEVEL
NOV. 14, 1963

N

Wiscombe Samples - O

Sample No.	oz. Au	oz. Ag
#5	0.10	0.2
6	0.22	0.2
7	0.24	0.2
8	0.92	0.3
9	5.06	2.6
10	2.98	2.4
11	0.22	0.2
12	0.08	0.3



Metals Engineering Samples

Sample #	Cut		oz. Au	oz. Ag	Value/Ton
#12	1+55	1552 - 3'	0.21	0.4	#7.87
	1+32	1551 - 2'	0.38	0.5	13.95
#11 Winze					
#10	1+03	1545 - 6'	0.14	0.5	5.55
	0+95	1544 - 5'	0.08	0.3	3.19
#9	0+80	1546 - 5.5'	0.11	0.5	4.50
	0+65	1547 - 5'	0.04	0.2	1.66
	0+50	1548 - 3'	0.09	0.2	3.41
#8	0+25	1550 - 3'	0.02	0.3	1.09
	0+00	1549 - 3'	0.23	0.3	8.44

Winze Assays

HW - 4070 - 23" -	0.03	0.3
And. - 4071 - 37" -	0.04	0.2
FW - 4072 - 12" -	0.01	0.4

Scale 1" = 40'

J BALLAM - G FRAEMAN

File Copy

REPORT OF THE WHITE CHIEF MINE
OATMEN, ARIZONA

NOVEMBER 13, 1963

Metals Engineering Company was primarily called on to examine the White Chief Mine for the purpose of determining whether the existing ore veins on the 190 foot level were high enough grade and of sufficient widths to warrant further exploration expenditures at depth. The geology of the district is well covered in previous reports including the Bureau of Mines.

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(4) Further, lets assume that the 200 foot of ore shoot exposed on the 190 foot level increased four times in grade and assayed $20.00 per ton at 100 feet greater depth and had an average of 4 feet in width, then in this 100 foot you would only have an 8000 ton block of ore. Your mining, hauling and milling cost would be at least $15.00 per ton. We would consider this a pretty small target to gamble the amount of money that would be required.

Mr. A. J. Zinkl , a Mining Engineer, was called on to study our sampling and results and also make an independent examination of the mine. He will submit a seperate report as to his evaluation of the mine.

Respectfully submitted,

METALS ENGINEERING COMPANY

By:

G. A. Freeman

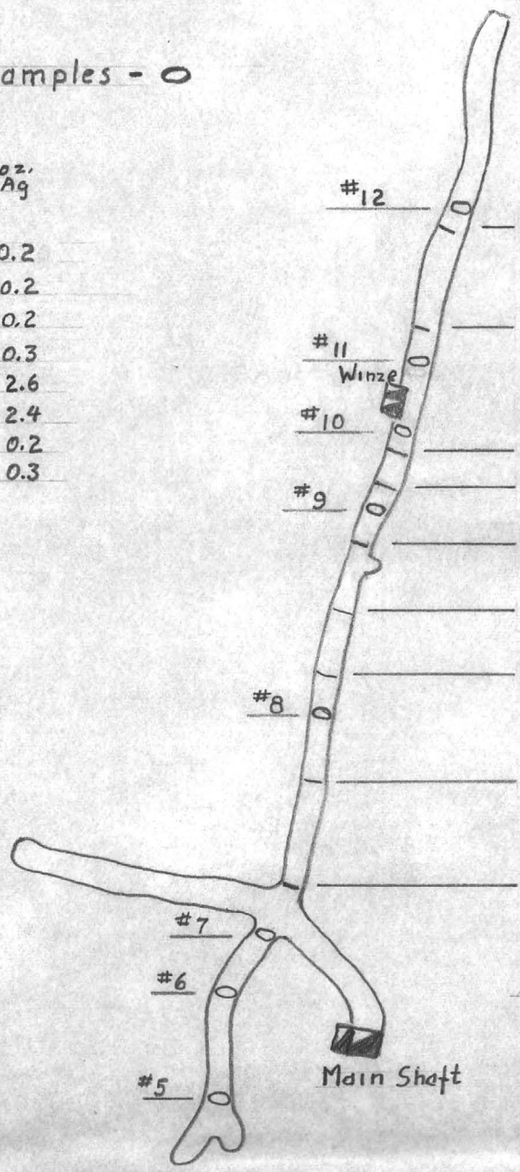
Jack Ballam

METALS ENGINEERING CO.
P. O. BOX 597 — 201 E. 4th ST.
CASA GRANDE, ARIZONA

WHITE CHIEF MINE
OATMAN, ARIZONA

SAMPLE MAP - 190' LEVEL
Nov. 14, 1963

N



Wiscombe Samples - O

Sample No.	oz. Au	oz. Ag
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Metals Engineering Samples

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0+95	1544	5'	0.08	0.3	3.19
0+80	1546	5.5'	0.11	0.5	4.50
0+65	1547	5'	0.04	0.2	1.66
0+50	1548	3'	0.09	0.2	3.41
0+25	1550	3'	0.02	0.3	1.09
0+00	1549	3'	0.23	0.3	8.44

Winze Assays

H.W.	4070	23"	0.03	0.3
And.	4071	37"	0.04	0.2
F.W.	4072	12"	0.01	0.4

SCALE - 1" = 40'

J. Ballam - G. Freeman

METALS ENGINEERING CO.
P. O. BOX 597 — 201 E. 4th ST.
CASA GRANDE, ARIZONA

WHITE CHIEF MINE
OATMAN, ARIZONA

SAMPLE MAP - 190' LEVEL

NOV. 14, 1963

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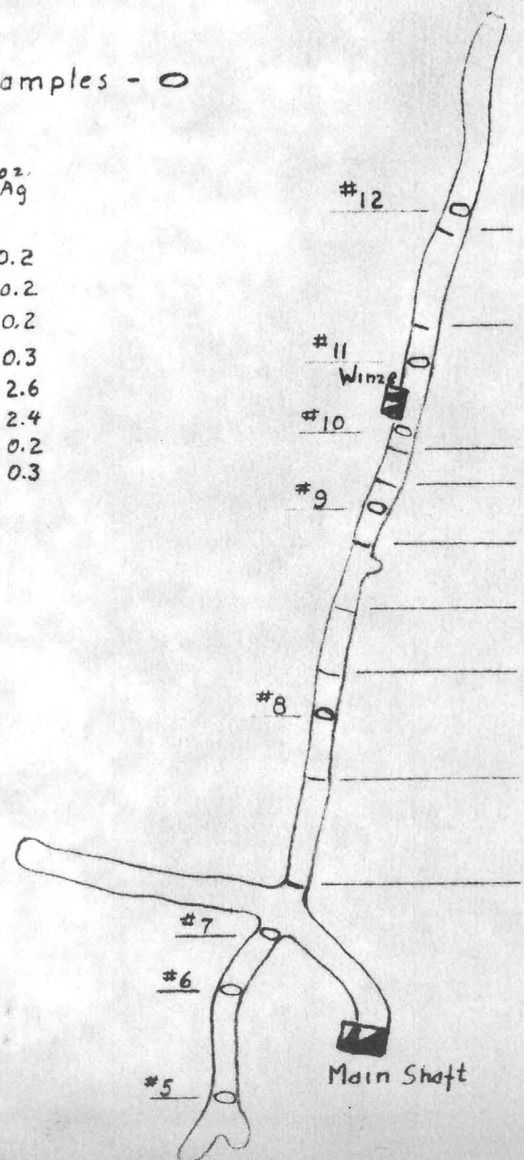

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0+65	1547 - 5'	0.04	0.2	1.66
0+50	1548 - 3'	0.09	0.2	3.41
0+25	1550 - 3'	0.02	0.3	1.09
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Winze Assays

HW	4070 - 23"	0.03	0.3
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METALS ENGINEERING CO.
P. O. BOX 597 — 201 E. 4th ST.
CASA GRANDE, ARIZONA

DENVER EQUIPMENT COMPANY
ORE TESTING DIVISION
Denver, Colorado

October 10, 1962

Americana Investments, Inc.
3015 East Thomas Road
Phoenix, Arizona

Attention: Mr. Leland M. Wiscombe

Reference: Order No. PW-70763

Gentlemen:

We are hereby submitting the following report covering the assays on the individual samples and batch concentration tests on a composite of the samples submitted from the Tom Reed Dump located at Oatman, Arizona.

Sample Identification

On September 17, 1962, we received via personal delivery by Mr. L. M. Wiscombe, Phoenix, Arizona, eight sacks of cyanide tailings having a gross weight of 459-pounds. For identification purposes this shipment was assigned our sample number 2935.

Object of Tests

The purpose of the test work was to determine the following:

1. The gold and silver content of the individual samples and the percentage of water soluble gold and silver.

2. The gold and silver content and water soluble values in a composite sample composed of 12-percent by weight of each of the red tailings samples and 40-percent by weight of the white tailings sample.

3. Laboratory beneficiation tests for the recovery of the gold and silver values from the composite sample using the Denver Mineral Jig, cyanidation and flotation.

Sample Preparation

Preparation of the tailings samples consisted of reducing the consolidated lumps contained in each sample to all minus 10-mesh, mixing thoroughly and quartering by repeated passes over a riffle

sampler. Three-fourths of the total sample was returned to storage, the remaining quarter was mixed and sampled to provide a head sample for analysis and material for testing.

Sample Description

Following is a brief description of the samples submitted, their weights and gold-silver assays:

Sample A - Red tailings, gross weight of 85-pounds and marked "Side of Dump."

Assay - oz/ton	Percent Water Soluble
Gold - 0.05	22.2
Silver - 0.10	29.1

Sample B - Red tailings, gross weight of 69-pounds and marked "5-feet from bottom south of 150-foot tunnel."

Assay - oz/ton	Percent Water Soluble
Gold - 0.07	14.6
Silver - 0.23	Nil

Sample C - Red tailings, gross weight of 44-pounds and marked "2-feet from bottom 100 North tunnel."

Assay - oz/ton	Percent Water Soluble
Gold - 0.03	27.0
Silver - 0.11	Nil

Sample D - Red tailings, gross weight of 61-pounds and marked "Tunnel 0-70' and 10 feet high."

Assay - oz/ton	Percent Water Soluble
Gold - 0.12	41.9
Silver 0.25	5.4

Sample E - Red tailing, gross weight 79-pounds and marked "Bank 25-foot from bottom."

Assay - oz/ton	Percent Water Soluble
Gold - 0.03	20.2
Silver - 0.07	Nil

Sample F - White tailings - gross weight of 96.5-pounds - No markings other than "White Tailings."

Assay - oz/ton	Percent Water Soluble
Gold - 0.04	18.3
Silver - 0.10	22.7

Sample G - Special sample, gross weight of 25-pounds and marked "Surface Composite Sample."

Assay - oz/ton	
Gold -0.03	EXCLUDED FROM
Silver - 0.03	COMPOSITE SAMPLE.

Sample H - Composite sample for testing and composed of 12-percent by weight of each of the red tailings and 40-percent by weight of the white tailings sample.

Assay - oz/ton	Percent Water Soluble
Gold - 0.04	35.9
Silver - 0.16	19.9

Additional assays and screen analyses of the individual samples are shown on data sheet number 5.

The percentage of water soluble gold and silver in the samples was based on the difference of assays of the samples as received and of the water washed head samples.

An examination of the heavy mineral concentrate obtained by panning

a portion of the composite sample showed no free gold or silver and only small amounts of iron oxides and base metal sulphides. Following is a screen analysis of the composite head sample as received with gold and silver assays of the screen sizes:

	% Weight	Assay oz/ton Au	Ag
Head Sample		0.04	0.16
Washed Heads	100.0	0.026	0.128
Heads + 150 Mesh	8.2	0.03	0.30
Heads + 200 Mesh	11.8	0.06	0.20
Heads - 200 Mesh	80.0	0.02	0.10
Water soluble by difference		0.014	0.032
% Water soluble, calculated		35.0	20.0

Results of Tests

Following is a brief description of the laboratory test procedure and of the results obtained in the beneficiation of the composite tailings sample. Tests Number 1 through 3 are described in detail on data sheets Number 1 to 4, inclusive.

Test Number 1
Composite Sample — Denver Mineral Jig

A 2500-gram charge of the composite sample was pulped at 50-percent solids to break any consolidated particles and then concentrated by the Denver Laboratory Mineral Jig to yield a concentrate and tailing. The jig concentrate which was produced at a ratio of concentration of 100 to 1 assayed 0.07 ounces of gold per ton, 0.27 ounces of silver per ton and contained 1.7-percent of the total gold and 1.7-percent of the total silver.

An analysis of the test products indicated that 36.5-percent of the gold and 11.7 percent of the silver was in a water soluble condition.

Test Number 1 is reported in detail on data sheet Number 1.

Test Number 2
Composite Sample — Carbon-Cyanidation and Flotation

A representative sample of the composite tailings was pulped to 33-percent solids and then agitated 4-hours at a solution strength of 0.25 pounds of sodium cyanide, 0.5 pounds of lime and 0.40 pounds of Norit wood charcoal EX per ton of solution. The pulp from cyanidation was then subjected to two (2) stages of flotation to produce a Number 1 concentrate using kerosene and a frother

and a Number 2 concentrate by the addition of xanthate.

The Number 1 flotation concentrate assayed 0.29 ounces of gold per ton, 0.10 ounce of silver per ton and contained 36.2-percent of the total gold and 4.2-percent of the total silver.

The Number 2 flotation concentrate resulting from xanthate flotation assayed 0.06 ounces of gold per ton, 0.08 ounces of silver per ton and contained 3.8-percent of the total gold and 1.4-percent of the total silver.

Test Number 2 is reported in detail on data sheets No. 2 and No. 3.

Test Number 3
Composite Sample — 24-Hour Cyanidation

Test Number 3 reported in detail on data sheet No. 4 investigated conventional 24-hour cyanidation of the composite tailings to determine if a longer agitation period would effect a higher extraction of the gold and silver.

The test procedure consisted of pulping a representative sample of the composite tailings to 33-percent solids and then agitating for 24-hours using a solution which contained 0.50-pounds of sodium cyanide and 1.0-pound of lime per ton of solution.

An assay of the washed tailings showed 0.025 ounces of gold and 0.06 ounces of silver per ton. Based on the head sample assay this would represent an extraction of 37.6-percent of the total gold and 62.4-percent of the total silver when using a 24-hour cyanidation period.

Remarks and Conclusions

An analysis of the individual samples showed gold assays which varied from 0.03 to 0.12 ounces per ton and silver assays which varied from 0.07 to 0.25 ounces per ton. A composite sample obtained by mixing 12-percent by weight of each of the red tailings and 40-percent by weight of the white tailings assayed 0.04 ounces per ton of gold, 0.16 ounces per ton of silver and contained 35.9-percent of the gold and 19.9-percent of the silver as a water soluble product.

Gravity concentration of the composite sample by the Denver Mineral Jig showed only a slight upgrading in the concentrate product

with very low gold and silver recoveries.

Four hour carbon-cyanidation and flotation recovered 40.0-percent of the total gold and 5.6-percent of the total silver in a product which assayed 0.21 ounces of gold per ton and 0.09 ounces of silver per ton.

Cyanidation of the composite tailings for 24-hours using conventional cyanide test procedures extracted 37.6-percent of the gold and 62.4-percent of the silver.

The remaining gold and silver values in the cyanide tailings do not appear to be easily recoverable and a high recovery of the gold and silver values is not indicated.

In view of the low metal content in the tailings and low gold-silver recoveries obtained, no recommendations are made for the treatment of the tailings represented by the submitted samples.

All remaining portions of the tailings samples will be discarded six months from the date of this report unless notified to the contrary.

Very truly yours

DENVER EQUIPMENT COMPANY

Henry C. Hurd Jr.
Henry C. Hurd, Jr.
Project Engineer

Richard W. Flagg
Richard W. Flagg
Chief Metallurgist

DENVER EQUIPMENT COMPANY
ORE TESTING DIVISION
Denver, Colorado

Report No. PW-70763

DENVER MINERAL JIG TEST DATA

Test No. 1

SAMPLE IDENTIFICATION: 2,500-grams of the composite tailings sample.

GRINDING: None

Preliminary grinding time, minutes

Final grinding time, minutes

Percent solids

JIG TEST PROCEDURE: A 2,500-gram charge of the composite tailings was concentrated by the Denver Laboratory Mineral Jig to produce a concentrate and tailing. The jig tailing was agitated for 30-minutes, thickened, filtered and washed several times on the filter for the removal of the water soluble gold and silver.

	PRODUCT	Percent Weight	ASSAYS oz/Ton			PERCENT RECOVERY		
			Au	Ag		Au	Ag	
	Head Sample	100.00	0.04	0.16		100.0	100.0	
1	Mineral Jig Conct.	1.00	0.07	0.27		1.7	1.7	
2	Mineral Jig Tailing	99.00	0.025	0.14		61.8	86.6	
	Water Solubles	-	0.015	0.02		36.5	11.7	
	Combine No. 1 and No. 2		0.025	0.14				

NOTES: *The water soluble gold and silver was taken as the difference between the head sample assay and calculated heads of the final products.

Test No. 1

A.

DENVER EQUIPMENT COMPANY 24 ORE TESTING DIVISION

Denver, Colorado

FLOTATION TEST DATA: CONDITIONS AND REAGENTS

SAMPLE IDENTIFICATION 2,500-grams of composite tailings sample REPORT NO. PW-70763 TEST NO. 2

TEST PROCEDURE: A 2,500-gram charge of the composite tailing sample was subjected to four hours of carbon-cyanidation at a solution strength of 0.25 pounds of sodium cyanide, 0.5 pounds of lime, and 0.4 pounds of finely ground wood charcoal per ton of solution. The pulp from cyanidation was subjected to two stages of flotation using kerosene and a frother in the first stage and xanthate in the second stage of flotation.

Grinding and Treatment				Reagents: Pounds per ton heads—(2)								
Operation	Time Min.	Percent Solids	pH	K.	A.	P.	Z.			NaCN	CaO	C
Grinding (1) None												
Cyanidation*		33		4 hours of agitation						0.50	1.0	0.80
Conditioner	4	33	10.6	0.08	0.02	0.05	-					
Flot. Conct. No.1	7	33	10.6	-	-	-	-					
Flot. Conct. No.2	5	-	-	-	-	-	0.08					

NOTES: GRINDING PERFORMED IN STANDARD DENVER 12"x5" DENVER BALL MILL. BALL CHARGE = 40 POUNDS, R.P.M. = 54.

CLASSIFICATION BY DECANTATION THROUGH LIMITING SCREEN, UNDECANTED SANDS REGROUND.

FLOTATION PERFORMED IN DENVER "SUB-A" LABORATORY FLOTATION MACHINE.

Grinding (1) None

Time, minutes

Classification, mesh

Sands reground, minutes

(2) Reagent Symbols:

K - Kerosene
A - Aerofroth 65
P - Pine Oil
Z - Dow Xanthate Z-5

NaCN - Sodium Cyanide
CaO - Calcium Oxide
C - Norit Wood Charcoal Ex, 90% minus 300 mesh.

DENVER EQUIPMENT COMPANY ORE TESTING DIVISION

Denver, Colorado

FLOTATION TEST DATA: METALLURGICAL RESULTS

IDENTIFICATION 2,500-grams of composite tailings sample REPORT NO. PW-70763 TEST NO. 2

	PRODUCT	Percent Weight	ASSAYS Oz/Ton Au	Ag
	Head Sample		0.04	0.16
	Calculated Heads	100.0	0.04	0.14
1	Flotation Conct. No. 1	5.0	0.29	0.10
2	Flotation Conct. No. 2	2.5	0.06	0.08
	Flotation-Cyanide Tail	92.5	0.025	0.125
	Total Conct. 182 Comb	5.2	0.21	0.09

An assay of the solution from the flotation cyanide tailing showed only trace amounts of gold plus silver.

	PRODUCT	Percent Weight	PERCENT RECOVERY Au	Ag
	Head Sample	100.0	100.0	100.0
1	Flotation Conct. No. 1	5.0	36.2	4.2
2	Flotation Conct. No. 2	2.5	3.8	1.4
	Flotation-Cyanide Tailing	92.5	60.0	94.4
	Total Conct. 1 & 2 combined	5.2	40.0	5.6

SCREEN ANALYSIS OF Tailing

Mesh	Percent Weight	ASSAYS Oz/Ton Au
+200	25.1	0.06
-200	74.9	0.015
Total	100.0	0.026

DENVER EQUIPMENT COMPANY
ORE TESTING DIVISION
Denver, Colorado

Report No. PW-70763
Test No. 3

CYANIDATION TEST DATA

SAMPLE IDENTIFICATION: 2,500-grams of composite mill tailings

GRINDING:

Test charge 2,500
Ground, minutes -
Classified, mesh -
Sands reground, minutes -
Percent solids -

AGITATION:

Time, hours 24
Percent solids 33

SOLUTION STRENGTH:

Pounds per ton of solution
NaCN 0.50
CaO 0.10

CHEMICAL CONSUMPTION:

Pounds per ton of heads
NaCN -
CaO 1.0

ASSAYS, ounces per ton

FEED:
Gold 0.04
Silver 0.16

SOLUTION:
Gold TR
Silver TR

TAILING:
Gold 0.025
Silver 0.06

RECOVERY, percent:*
Gold 37.6
Silver 62.4

SETTLING DATA:

F—2 to 1
R—0.375
D—1 to 1
A—4.4

F—Dilution ratio to start
R—Settling rate, ft./hr.
D—Discharge dilution ratio
A—Thickener area, sq. ft./ton/24 hours

Formula:

$$A = 1.33 \frac{F-D}{R} \times 1.25$$

NOTES: *Recovery of gold and silver determined by the difference between the composite head sample assay and cyanide tailing assay.

SCREEN ANALYSES OF TAILING

Mesh	%Wgt.	Au oz/Ton	Ag oz/Ton
+200	24.7	0.05	0.13
-200	75.3	0.02	TR
	100.0	0.027	0.07

DENVER EQUIPMENT COMPANY
ORE TESTING DIVISION
Denver, Colorado

SCREEN SIZES AND ASSAYS PW-70763

Sample A

	% Wgt	Assay Oz/T Au	Ag
Head Sample	100.0	0.05	0.10
Washed Heads	-----	0.039	0.07
Washed Heads Plus 150	9.7	0.08	0.24
Washed Heads Plus 200	20.7	0.05	0.23
Washed Heads Minus 200	69.6	0.03	TR
Water Soluble (Calculated)	---	0.011	0.03

Sample B

	% Wgt	Assay Oz/T Au	Ag
	100.0	0.07	0.23
	-----	0.06	0.23
	10.8	0.16	0.46
	13.9	0.10	0.30
	75.3	0.03	0.19
	----	0.01	nil

Sample C

	% Wgt	Assay Oz/T Au	Ag
Head Sample	100.0	0.03	0.11
Washed Heads	-----	0.021	0.13
Washed Heads Plus 150	6.4	0.07	0.20
Washed Heads Plus 200	13.6	0.04	0.12
Washed Heads minus 200	80.0	0.015	0.15
Water Soluble (calculated)	-----	0.008	nil

Sample D

	% Wgt	Assay Oz/T Au	Ag
	100.0	0.12	0.25
	-----	0.07	0.24
	11.1	0.16	0.54
	15.5	0.10	0.24
	73.4	0.05	0.13
	-----	0.05	0.01

Sample E

	% Wgt	Assay Oz/T Au	Ag
Head Sample	100.0	0.03	0.07
Washed Heads	-----	0.024	0.09
Washed Heads Plus 150	7.6	0.04	0.26
Washed Heads Plus 200	12.0	0.04	0.22
Washed Heads minus 200	80.4	0.02	0.06
Water Soluble (calculated)	-----	0.006	nil

Sample F

	% Wgt	Assay Oz/T Au	Ag
	100.0	0.04	0.10
	-----	0.033	0.035
	8.9	0.06	0.24
	13.1	0.03	0.07
	78.0	0.03	0.06
	-----	0.007	0.227

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DENVER EQUIPMENT COMPANY

1400 17th STREET, P. O. BOX 5268 ★ DENVER 17, COLORADO, U. S. A.

AIRMAIL
October 22, 1962

Mr. Leland M. Wiscombe
Americana Investments
Suite 7 - 3015 East Thomas Road
Phoenix 16, Arizona

Ref: Ore Test No. PW-70763

Dear Mr. Wiscombe:

This is in reply to your letter dated October 16 containing your comments concerning the recent test work we performed on your tailing samples.

Your letter has been reviewed by our laboratory staff with concern and they have pointed out that the test work was to consist of the following:

1. Assay the individual samples.
2. Determine distribution of values including the soluble content.
3. Investigate jigging and carbon cyanidation as the treatment method must be very economical.

During our conversation in regard to the beneficiation test work on the composite ore, it was stressed to us that counter-current decantation and/or filtration would be too costly for the economics involved. For this reason, carbon cyanidation test work was requested as well as jigging.

The screen analyses of the individual and composite samples revealed that regrinding of the plus 200-mesh fraction could possibly be beneficial to recovery, but the emphasis of a simple and inexpensive operation precluded the use of regrinding. Further to this, the composite sample was 80% minus 200-mesh and 75.9% of the total gold was contained in this fraction and in soluble form. The maximum extraction obtained was 40.0% of the total gold, regrinding of the plus 200-mesh fraction would not appreciably improve the overall recovery. If the extraction had been higher on the minus

200-mesh fraction, then regrinding would have been considered even though it would have represented additional plant operating costs. We do not recall specific instructions relative to investigating a regrind circuit but it certainly would have been included if the test results had definitely indicated it.

The recovery of soluble values, of course, is no particular problem. Test No. 1 showed the separation to be expected by jigging. Test No. 2 represented the recovery possible by carbon cyanidation and Test No. 3 showed what extended dissolution time (24-hours) using higher solution strengths under conventional cyanidation conditions could extract. Actually the cyanidation tests did not render any significant dissolution of the values as the percent extraction represented essentially only the values present in soluble form. The carbon cyanidation procedure also included flotation using xanthate to possibly recover a gold concentrate separate from the carbon product, but this step did not indicate any advantage of treatment of the tailing samples.

Our conclusion based on the test results of the samples submitted did not indicate a flowsheet in our opinion and we therefore stated this in our report. We, did not mean to imply that further sampling and test work together with other considerations could not be successful as we have no knowledge of the problem other than as represented by the samples tested.

We sincerely regret that we could not have submitted a more favorable report but we must emphasize that we honestly reported our findings which were obtained by the test work on the samples delivered to our laboratory.

Yours very truly

DENVER EQUIPMENT COMPANY

Frank A. Seeton

Frank A. Seeton, Manager
Metallurgical Division

FAS:tf

February 14, 1963

To: Mr. W. J. Nault

From: R. R. Mates

Subject: Re-Processing of Tom Reed Tailings for Recovery of Gold and Silver

Outline of Report:

1. Previous Test Work
2. Proposed Scope of Investigation
3. Test Experiments and Results
4. Conclusions to be Drawn
5. Further Recommendations.

1. PREVIOUS TEST WORK

A series of tests by W. J. Nault on Cyanide leaching of representative samples of tailings in the laboratory, gave results that indicated that a pilot run on a truck-size sample was adviseable. This was performed by a major research organisation in Denver, Colorado. The results apparently were sufficiently encouraging to warrant construction of a mill d esigned to treat 500 tons of dry tailings per day.

The flow sheet proposed included gathering and transportation of stockpiled tailings to a feed bin, followed by pulping; classification by cyclones into sand and slimes fraction; cyanide leaching of sand fraction and slime fraction separately; a regrind and recycle of the leached sands, so that ultimately everything reports through the slimes leach; contact either during leaching or afterward with fine carbon to adsorb the soluble gold and silver values; flotation of the loaded carbon for recovery by burning or caustic leaching. Additional recovery either for recycle back to this

circuit, or by separate methods, is to be performed by mineral jigs on the sand fractions and on the emerging tail stream.

The following Data Sheet is included to give an indication of what at least part of the feed is like.

NOTES ON:

Tom Reed Gold - Mill Operation

(Average Mill Grind in June, 1939)

Assay of Tailing Screen Size	Distribution	Assay	Distribution	% of Total
Plus 100 Mesh	4.63%	$1.40	0.07	12.50
Plus 150 Mesh	10.09%	$1.05	0.11	19.64
Plus 200 Mesh	11.68%	$ .70	0.08	14.29
Minus 200 Mesh - Sand	25.34%	$ .52	0.13	23.21
Minus 200 Mesh - Slimes	48.26%	$ .35	0.17	30.36

Tails Discharged as 53.70% Solids

Reagents /Aero Brand Cyanide - 0.23#/ tons of ore milled
/Sodium Cyanide - 0.35#/ tons of ore milled
/Lime - 4.14 pounds/ tons of ore milled

2. PROPOSED SCOPE OF INVESTIGATION

Using the flow sheet as proposed, determine a workable reagent combinations and the amounts necessary to bring about a satisfactory metallurgical recovery at reasonable cost. Point out any improvements in the flow sheet indicated by the test work that might be incorporated in the current building project without disrupting the overall plan. Specify the carbon to be purchased.

3. TEST WORK AND RESULTS

A. Preparation of Head Samples

Two drums of samples of the Tom Reed Tailings Pond, white tailings and red tailings, were available. These previously had

been taken July 25, 1960 for Sawyer Petroleum Co. They are believed to be representative of a large tonnage of the tailings pile. Each drum was put through an 8 mesh screen to break up the caked lumps and to remove any possible tramp inclusions. Each was split in half to obtain a working supply of feed material for test work. The retained halves were further split down to obtain the assay pulp samples.

Three more pond tailings samples submitted by W. J. Nault were prepared in the same manner for assay. The contents of these feed samples are as follows:

		Au	Ag
1.	Tom Reed White Tailings (Sawyer)	0.060	0.2
2.	" " Red Tailings (Sawyer)	0.20	0.3
3.	" " Top 47' (Nault)	0.03	0.3
4.	" " Bottom 53' (Nault)	0.025	0.3
5.	" " D1-T4-6'-6" (Nault)	0.03	0.3

B. Preliminary Testing of the Ore
Test Numbers (Ao2-12-27-1 and A02-12-28-1)

Two tests were run grinding the whole ore with Raven #15 carbon and perchlorethylene in the laboratory ball mill followed by flotation. The carbon products calculated out 0.15 oz Au/T 6.0 oz. Ag/T and 0.19 oz. Au/T 1.9 oz. Ag/T respectively. The tailings products were discarded on these preliminary tests.

C. First Indication of a Good Tail

After a consultation with Mr. W. J. Nault, it was proposed to wet screen the Tom Reed White Tailings to _ 150 mesh fractions. The leach was to be run on the -150 mesh fraction, since it is planned that everything will have to be at least this fine before it can leave the circuit. This was done. Three 1,000 Gm samples

were net screened using warm tap water. The screened out +150 M sands were dried and weighed 157 gm, 178 gm, and 168 gm, respectively. The average was 16.8% +150 M. This was assayed (AO 3-1-4-1) at 0.015 oz/T Au and trace of Ag. The assay is 1/4 the grade of the head sample in gold.

The slimes were combined and settled. Calculating by difference, there are 3,000 gm — 503 gm = 2497 gm of solids in the settled pulp, neglecting any solubles in the head samples. This amount just fits in the laboratory flotation cell without any dilution. The pH was 6.0 11b/T CaO was added and conditioned for 5 minutes; then .25 #/T cyanide was added. The pH was about 8.5 after 80 minutes of leach. 6gm or 4.8 lb/T Raven #15 was pugged with a small portion of the pulp and added to the cell for a contact of 30 minutes.

In order to float the carbon, the pulp was first conditioned with 2 drops of Solvenol (.05 gm/drop) and 2 drops of perchlorethylene. At this point, the pulp appeared to require dilution in order to obtain a good float. The pulp was split in half and diluted to about 25% solids. Each half was floated in the same manner. Two drops of Cresylic acid followed by one minute conditioning; 2 drops Aero Frother 77, conditioned one minute, then the rougher was floated. The two rougher concentrates were combined and cleaned twice. The cleaner tails were combined for assay. The carbon concentrate was filtered, but this is indeed a slow filter rate and would not be very practical in a plant. The results of this test were as follows:

Test No A03-1-4-1

Product	Grams	Au oz/T	Au WxA	Au % Distr	Ag oz/T	Ag WxA	Ag % Distr
+150 M Sands screened out of test	503	.015			Trace	---	
Ro Tails	1808	.005	9.10	68.5	Trace	--	
Cl Tails	625	.005	3.12	23.5	Trace	--	
Carbon Conc.	11.8	.09	1.06	8.0	.27	3.19	100.0
	2444.8	(.0054)	13.28	100.0	(.0013)	3.19	100.0

The tailings and cleaner tailings were certainly of an acceptable throw-away grade, .005 oz/T Au; but insufficient values reported to the carbon concentrate. It was evidently contained in the large quantity of screening solution.

D. SHORT 90-MINUTE CYANIDE LEACH AND PEERLESS 155 CARBON(pH 2.8) ADSORPTION AND FLOAT FROM COM REED WHITE TAILINGS HEAD SAMPLE NO. 1

Ore as received including the plus 150 mesh sands was leached at 50% solids in the Fagregren cell with 1/2 lb/T cyanide. The pH of the pulp was 8.0-8.5. No lime was added. After the first 20 minutes, the carbon was staged-in in 1lb. additions. The total leach and contact was for 90 minutes.

For the float, 1 drop perchlorethylene, 1 drop solvenol and .002 lb/T Z-6 reagent were added. After floating a clean carbon concentrate for 10 minutes, a drop of Shell Cresylic acid was added and a greyish mineral started to float. Floated this for ten minutes. The results of this test are as follows:

Test No. A03-1-12-1

	Weight Grams	Weight % Distr	Au oz/T	Au WxA	Au % Distr	Ag oz/T	Ag WxA	Ag % Distr
Ro Tail	1440	963	.025	36.0	96.3	0.1	144	92.8
Ro Conc	56	3.7	.025	1.4	3.7	0.2	11.2	7.2
	1496	100.0	(.025)	37.4	100.0	(0.104)	155.2	100.0

Obviously, there was no apparent up-grading, but the Metallurgical balance is not complete.

E. LONGER TEST OF 4 HOURS CYANIDE LEACH FOLLOWED BY 2 HOURS OF CONTACT WITH PEERLESS 155 Carbon (pH 2.8)

To 1,500 Gm Tom Reed White tailings in the laboratory Fag. float cell at 50% solids added 1/2 #/T Cyanide (soln. made from Aero brand cyanide flakes). The leach was carried out 4 hours. No lime was added. The pH was 8.5-9.0. Over the next 2 hours, 3 lb/T Peerless 155 Carbon was staged in. For the float, 1 drop perchlorethylene, 1 drop soluenol and .002 lb/T Z-6 reagent in solution were added and conditioned separately. After floating the above for 8 minutes, a drop of cresylic was added and floating was resumed for 12 minutes more. The results of this test were as follows: Test No. A03-1-14-1

	Weight		Au			Ag		
	Grams	% Distr	oz/T	WxA	%Distr	oz/T	WxA	%Distr
Ro Tail	1418	94.8	0.02	28.4	94.8	0.1	14.18	90.4
Ro Conc	75 / 1493	5.2 / 100.0	0.02 (0.02)	1.5 / 29.9	5.2 / 100.0	0.2 (0.105)	1.5 / 15.68	9.6 / 100.0
Ro Conc Filt. 540 ml			.003			0.8		

These results show that, though some values are taken into solution, apparently this carbon does not pick them all up. Since the ro-Tail is merely dried as is, solution and all these values would be incorporated in this product. Pond water would be returned to the circuit, but entrained values would remain in the pond unless filtering and washing were employed. Water dilution of tailings pulp will help but there are insufficient values to justify any expensive unit operation, such as filtering.

F. TEST ON WHOLE TAILINGS USING RAVEN #15 CARBON COLLECTION

We made an apparent tail .005 oz/T Au on the initial test on the -150 M fraction of Tom Reed White Tailings. We will try a somewhat similar test on unsized head sample. Some 1,500 grams of sample was pulped with distilled water to about 52% solids, and

conditioned with 1 lb/T CaO to pH 9.5-10.0 and 1/4 lb/T cyanide. After 10 minutes of conditioning, 2 lb/T Raven #15 Beads were added and the leach was continued for another 2 hours. The pulp was then split in half to allow dilutions. Each half was conditioned with 1 drop solvenol and 1 drop perchlorethylene. Aero 325 and Aero 343 were each added and conditioned to collect metallics. After 5 minutes of floating, cresylic was added to continue the froth. The concentrates from each half were combined, as were the Ro tails. Each was dried as-is. The results of this test are as follows:

Test No. A03-1-17-1

	Weight		Au	Ag
	Grams	% Distr	oz/T	oz/T
Ro Tail	1470	98.2	.05	0.3
Ro Conc	29.5	1.8	.06	Trace
	1499.5	100.0		

Again it is obvious that no headway was made on this test.

G. TESTING THE TOM REED RED TAILINGS SAMPLE USING RAVEN #15 BEADS

Pulped 1,500 gm solids with distilled water to 54.5% solids. Added 1 lb/T lime and 1/4 lb/T cyanide. Leached for 4 hours and contacted carbon for another 1.5 hours. Conditioned 4 drops perchlorethylene for 15 minutes, added 1 drop carbitol and floated for 25 minutes. The results of this test were as follows:

Test No A03-1-19-1

	Weight		Au			Ag
	Grams	% Distr	oz/T	oz/T	% Distr	oz/T
Ro Tail	1416	94.3	0.15	212.2	74.0	0.1
Ro Conc	84	5.7	0.90	75.6	26.0	Trace
	1500	100.0	(0.19)	287.8	100.0	(.094)

The heads on which this test was run had been reported 0.20 oz/T Au; 0.3 oz/T Ag. The gold checks closely, but the distribution is still unfavorable.

H. A SCREEN ANALYSIS TEST NO. A03-1-21-1

At ± 80 mesh on the + 150 mesh sands from Test No. A03-1-4-1, which had been reported .015 oz/T Au and trace Ag, was calculated to have .054 oz/T Au, very close to the head sample from which it came. Such discrepancies lead one to believe that the gold is not evely distributed in any of these samples.

I. TWENTY*FOUR HOUR CYANIDE LEACH ON TOM REED WHITE TAILINGS. NO CARBON USED IN THIS TEST.

A 1,500 gm charge of ore was placed in a jar and pulped with 1,000 ml distilled water, 1 lb/T lime and 1/4 lb/T cyanide was added. A stainless steel agitator and a stainless spatula as an aerator baffle were used. The pH was 8.5-9.0. The pulp was filtered and washed with weak cyanide solution,, followed by water. The residue was wet screened through 150 mesh. The samples dried were as follows:

	Weight		Au			Ag		
	Grams	% Distr	oz/T	WxA	%Distr	oz/T	Wxa	%Distr
+150 M Fraction	233	15.7	0.05	11.66	31.6	0.4	93.3	27.2
-150 M Fraction	1256	84.3	0.02	25.18	68.4	0.2	251.8	72.8
Total	1489	100.0	(0.025)	36.84	100.0	(0.23)	345.1	100.0

Based on the original heads, the 24-hour leach uner the conditions used, only got about 60% of the gold into solution. The fines leached better than the coarse. More work will have to be done on getting a better leach, or perhaps gravity methods of concentration will have to be employed.

J. REDUCTION OF TOM REED WHITE TAILING TO -150 MESH FOLLOWED BY LEACHING IN CONTACT WITH RAVEN #15 AND ACTIVATED CHARCOAL GRANUALS GROUND FOLLOWED BY FLOTATION WITH A DIFFERENT REAGENT COMBINATION.

By careful manipulation, 1,500 gm of ore was wet screened with a minimum of distilled water. The 0' size was dried and reduced by hand in porcelain mortar to -150 mesh. The solids and approximately 1,500 ml of the washing solution were combined in the laboratory float cell.

Some of the pulp and 1 lb/T Raven #15 and 1 lb/T activated charcoal granuals (DFC) were ground in the mortar. This was added along with 1 lb burned limerock to the cell. The pH was 9.5. Then 1/4 lb/T cyanide solution (9 parts KCN - 1 part Na CN) was added. The leach was run for 4 hours, adding an additional 1/8 lb/T cyanide combination at the final 1/2 hour. At the final 20 minutes, the leach was diluted with the remaining wash solution, about 500 ML, to allow adsorption of the values on the carbon prior to the flotation step.

A new frother combination, consisting of 1 part by volume c resylic acid, 1 part carbitol, and 1 part pine oil, was made up and used. One drop of this gave a strong, lasting froth. The rougher float took 20 minutes.

To the rougher tails, added .02#/T Cu SO_4; 1 drop Aero 325 --1% solution; 1 drop Aero 343--1% solution and 1 drop Aero 404 --1% solution. Floated for 5 minutes, then added another drop of frother and floated 5 minutes more.

The rougher concentrate was diluted and floated lightly for 5 minutes.

The tails were settled for 1 hour and dec anted of clear solution. This was added to the recycle carbon cleaner tail.

The results of this test are as follows:

Test No. A03-2-1-1

	Weight		Au			Ag		
	Grams	%Distr	oz/T	Wgt	%Distr	oz/T	Wgt	%Distr
Tails	1340	90.4	.02	26.8	80.9	0.8	1172	60.5
Cleaner Tails	108	7.3	.03	3.24	9.9	5.8	540	27.8
Carbon Conc	4.7	.3	.28	1.32	4.1	13.7	64.5	3.3
Scavenger Conc	28.5	2.0	.06	1.7	5.1	5.9	163	8.4
	1481.2	100.0	(.022)	33.16	100.0	(1.31)	1939.5	100.0

Again we see a loss of gold, but this time a gain in silver.

K. REDUCTION OF TOM REED RED TAILINGS TO -150 MESH BY DRY SCREENING AND CRUSHING IN PORCELAIN MORTAR BY HAND FOLLOWED BY CYANIDE LEACH WITH RAVEN #15 AND ACTIVATED CHARCOAL AND FLOTATION.

The red tailings were dry screened through 150 mesh. The screen analysis was as follows:

+150 M	299 gm
-150 M	1201 gm
Total	1500 gm

The 0' size was stage reduced by hand in the porcelain mortar to -150 mesh.

The leach was started at greater than 50% solids using distilled water for pulping and gradually diluted to slightly less than 50% solids by the end of the leach. 1 lb/T each of activated charcoal and Raven #15 were dry ground to -100 m in the mortar, then added to the pulp. 1 lb/T Burnt lime rock to pH 9.5-10.0 was added, then 1/4 lb/T mixed Cyanides (9 to 1). After 1½ hours, another 1/8 lb/T cyanide was added. The leach was continued for a total of 4 hours.

The float used 2 drops of mixed (CCPO) frother and 10 minutes to produce a rougher concentrate. To the pulp, to get a scavenger concentrate, were added .022 lb/T Cu SO_4; Aero 325 1% soln.(1 drop); Aero 343 1% soln.(1 drop); Aero 404 1%soln.(1 drop). An additional drop of CCPO frother was added and float lasted 10 minutes. The concentrate looked dirty.

The rougher concentrate was diluted and cleaned.

The tailings were settled and then decanted. Clear liquor was added to the cleaner tails recycle product.

The results of this test were as follows:

Test No. A03-2-4-1

	Weight		Au			Ag		
	Grams	%Distr.	oz/T	WxA	%Distr	oz/T	WxA	%Distr
Tails	1326	89.6	.20	265.0	80.8	1.0	1326	74.3
Carbon Conc	2.5	.2	1.20	3.0	.9	22.8	57	3.2
Scavenger Conc	29.5	2.0	0.60	17.7	5.4	3.0	88	4.9
Cleaner Tails	121.	8.2	0.35	42.3	12.9	2.6	314.2	17.6
Total	1479.0	100.0	(0.222)	328.0	100.0	(1.15)	1785.2	100.0

Red tailings head assay had been reported as 0.20 oz/T au and 0.3 oz/T Ag. The gold balance is fairly close, but gold recovery is still very poor.

The results to date have been very discouraging. Not only are poor extractions indicated, but getting any sort of a metallurgical balance seems only to be a matter of chance.

The reference book "Manual of Cyanidation" by Hamilton, suggests the addition of Copper sulfate, sodium sulfite, and sulfuric acid to pulps and solutions prior to assaying. In the future tests, the pulps will be thus treated and the assayer has been asked to add the prescribed amounts to the solutions prior to assaying. It is hoped that some improvement in balancing can be shown.

L. LEACHING TOM REED WHITE TAILINGS SLIMES WITH CYANIDE ACCOMPANIED BY POTASSIUM FERRI*CYANIDE OXIDANT (REFERENCE* CLENNELL) FOLLOWED BY CONTACT WITH GROUND -80 M ACTIVATED CHARCOAL AND FLOTATION.

A 2,000 gram sample of white tailings was deslimed with a mimimum of distilled water. The resulting sands were as follows:

+150 M Sands	145 grams
-150 M Sands	207 grams
Total	352

The slimes (2,000 -352 = 1648 gm) in settled slurry occupied approximately 1,900 ml. This pulp was leached in a

3 liter glass beaker, using a stainless steel impeller. To the slurry was added 1 lb/T burned lime rock 1/4 lb/T mixed cyanides (9-1). After the first hour, .05 lb/T potassium ferri-cyanide, made up fresh in solution, was added. At the second hour, 2 lb/T activated charcoal granuals freshly pulverized to -80 mesh was added. After another hour, 780 ml of decant solution from the original screening was added back in and the leach and contac t continued another hour for a total of 4 hours.

In the float, 1 drop of (CCPO) frother and 1 drop of solvenol was added. The froth looked much different than any before. It was more like a fatty acid or an amine float. The concentrate was greyish in color rather than the carbon black. A second drop of (CCPO) frother was added during the 10-minute float. The tail remaining in the cell was conditioned with 1 drop 1% Aero 325; 1 drop Aero 343; and 1 drop Aero 404; and another drop of (CCPO) frother. The scavenger float lasted for 10 minutes.

The rougher concentrate was diluted and cleaned for 10 minutes. Another drop of frother was used.

All of the products were decanted of 2.2 liters of clear liquors. This was sampled for assay.

The pulps were mixed with $CuSO_4$, Na_2SO_3, and H_2SO_4 solutions, as per Hamilton, then dried in the oven. After drying, it was noted that the dried salts easily flaked away from the edges of the enameled pan. This characteristic may be very important in having a chance to get a good accounting on dried pulps.

The results of this test are as follows:

Test No A03-2-7-1 & A03-2-6-1

	Weight		Au			Ag		
	Grams	%Distr	oz/T	WxA	%Distr	oz/T	WxA	%Distr
Tail	1444	85.9	.05	72.3	80.8	trace	--	--
Carbon Conc	21	1.2	.20	4.2	4.7	3.8	79.8	17.7
Scavenger Conc	48.5	2.8	.06	2.9	3.2	trace	--	--
Cleaner Tails	168.5	10.1	.06	10.1	11.3	2.2	371.0	82.3
Total	1682	100.0	(.053)	89.5	100.0	(.27)	450.8	100.0
+150 M Sand	145	41.2	.08	11.6	73.6	1.0	145	63.8
-150 M Sand	207	58.8	.02	4.14	26.4	0.4	82.8	36.2
	352	100.0	(.045)	15.74	100.0	(.65)	267.8	100.0
Calc Total Head	2034		(.052)	105.24		(.33)	678.6	
Decant Liquors	2.2	Liters	.002			trace		

It appears that the metallurgical balance is improved by addition of the chemicals to the pulps before assaying. The white tailings as calculated from the products is .052 oz/T Au and .33 oz/T Ag. The values were reported as .060 and .20 respectively.

The oxidant reagent combination appears effective in producing a silver tailing. But we are still not getting the gold. It is probably not being leached, or if it is, it is not being adsorbed and floated with the carbon.

M. TESTING VARIOUS GRADES OF CARBON FOR ADSORPTION FROM A GOLD BEARING SOLUTION.

A head sample of gold bearing solution was produced by compositing accumulated clear barren cyanide tail solutions fortified with some weak preg solution from a leach on a specimen gold ore sample. This solution head sample ran 0.020 oz/T Au and trace Ag. A number of 2 gram samples (1/4 lb/T) of the various carbons were agitated for a specified time with 500 ml of the head solution. Each slurry was then filtered immediately and the depleted solution was bottled for assay.

The following table shows the conditions used and the results:

Carbon	Minutes Agitation	Filtrate oz Au/T	oz Ag/T	Calculated % Au Removal on Carbon
Raven #15	30	0.02	Trace	0
Conductex SC(pH-5.0)	30	0.005	"	75
Activated Charcoal-60M	30	.003	"	85
" " Granuals	210	.002	"	90
" " "	15	.015	"	25
Super Spectra(pH-4.2)	15	.003	"	85
#999 (pH-3.9)	15	.002	"	90
Peerless 155 (pH-2.8)	15	.025	"	- 25

4. Conclusions to be Drawn

A. The initial test gave a good tail. Raven #15 Carbon was used on this test, therefore, it was assumed that it was a good carbon. The final series of tests shows it to be one of the poorest of the lot, along with Peerless 155, also used.

B. Even ground activated charcoal, one of the better carbons, did not produce a satisfactory tail, indicating that perhaps the values had not been put into solution. (See Test A03-2-7-1).

C. Coarse granular carbon will adsorb gold from clear solution, if given sufficient time.

D. #999 Black (pH 3.9) Carbon appears to be the most active of the six carbons tested in adsorbing Gold from a clear solution.

E. Assaying procedures should be modified to include the use of $CuSO_4$, Na_2SO_3, and H_2SO_4 in fixing the samples to minimize unaccountabilities.

5. Recommendations

A. Direct efforts to obtaining a good leach before going any further. Use oxidants or other chemical aids, if need be. The [illegible] of this step will have to be critically evaluated.

B. Employ graviety methods to get values into a concentrated fraction for separate treatment. Some graviety methods will be employed in the plant. The use of graviety test equipment is also adviseable.

C. Check out the possibility of using granulated charcoal in counter-current screening circuit for values recovery, as an alternative for flotation. (Reference: R.W. Krebs Patent #2,476,4 20 July 194 9. This has some strong arguments against flotation.)

D. Consider Dutch States Mines Stationary Screens for removal of Charcoal granuals from pulp. Krebs shows a trommel device.

E. Consider use of mid-whirl pumps for transfer of charcoal-bearing pulps and sand slurries. Presently it is available only in 3" and 4" sizes for alkaline or neutral service only.

F. Procure an ore blender for the laboratory preparation of samples.

RRM/tm
CC:L. Wiscombe

Robert R. Mates

Subject: Progress Report for Period Ending Feb. 15, 1963.

CHECKING THE EFFECTIVENESS OF ACTIVATED CHARCOAL GRANUALS AND CARBON #999 BLACK IN SLURRY.

Preliminary testing previously reported indicates that #999 Black and Granular Activated Charcoal are effective absorbers of gold from a clear gold-bearing liquor. This test and the next one will show that they are also effective in a slurry.

To simulate a flow sheet that would efficiently utilize granular charcoal, it is necessary to run the charcoal in stages counter-current to the flow of the gold-bearing pulp. To start, five stages were arbitrarily selected. Five samples of 50% slurry, each made from 100 grams of Tom Reed White Tailings, as received, and 100 ml gold-bearing clear head solution liquor (A03-2-8-1), were made up. Two grams (8 lb/T) charcoal was contacted with agitation for 40 minutes. The carbon and a few oversize rock pieces were retained on the screen. These were advanced to a 2nd stage. Similarly, through all five stages with the same carbon. The carbon product at the end was dried and weighed 3.3 grams. If the ore had been ground to 100% through 60 mesh, there would have been no oversize contamination contributing to the increase in weight. The assay of this product is 3.60 oz/T Au and 23/4 oz?T Ag. Since there was only a trace of Ag in the starting liquor, the charcoal is getting the Ag value from the Tom Reed Tailing. The following figures show the results:

Test No. A03-2-13-1

		Au			Ag		
		oz/T	W×A	%Distr	oz/T	W×A	%Distr
T.R.White Tailings	500 gm	.060	30.0		0.2	100	
Head Solution	500 ml	.020	10.0		Trace	--	
			40.0	100		100	
Loaded Char Product	3.3	3.6	11.9	29.8	23.4	77.2	77.2

This test is very encouraging. Apparently, in the five-stage operation with the 40-minute contact at each stage, 77% of the Ag was loaded from the tailings alone and the equivalent of more Au than was present in the solution portion of our feed slurry. In other words, without intentional leaching, the equivalent of all the gold that was in the solution plus 6.3% of the gold in the tailings was loaded. With more stages and a good leach, a good loading on char and the ash resulting from the burning of the loaded char can be expected.

If the char granuals are non-friable and properly sized, there will be no loss of fine-loaded carbon out with the tails, if there is any possibility of this happening, then a scavenger float cell could be put on the tail line to capture this loss. Aside from the operating advantages, the high grade product after burning makes the char process well worth investigating.

The activated char granuals were received some time ago, from D. F. C. and were used as received.

CONTACTING #999BLACK CARBON WITH GOLD SLURRY FOR 15 MINUTES FOLLOWED BY DILUTION AND FLOTATION OF CARBON.

As this test (A03-2-13-2) is for testing carbon loading rather than leaching, a synthetic 50% slurry of 500 gm Tom Reed

White Tailings and 500 ml gold head solution (A03-2-8-1) was contacted for 15 minutes with 4 gm #999 Black.

For the float, the pulp was diluted to about 20% solids. Pine oil frother alone did not appear to be effective when a drop was added. Three more drops of CCPO frother were staged in. A good float resulted and was continued for about 10 minutes. The sesults appear in the folbwing table.

		Au			Ag		
		oz/T	WxA	%Distr	Oz/T	WxA	% Distr
T. R. White Tailings	500 gm	.060	30.0		0.2	100	
Head Solution (A03-2-8-1)	500 ml	.020	10.0		Trace	1+-	
			40.0	Too.		100	100.0
Rougher Carbon Conc.	28.5	1.00	28.5	71.3	4.0	114	114.0

The Carbon concentrate contained the gold equivalent of all that was in solution plus a good portion of that in the T. R. Tailings. The recovered silver was more than was indicated in the head sample. "Fixing Reagents" were used on the conc.

The head samples should be run again using "Fixing Reagents." There may be more values to go after than we suspect.

Carbon #999 Black was not given any activation treatment, but used as received in air-tight jar.

This test confirms the effectiveness of #999 Black. The emphasis will now have to be directed to getting an effective leach.

CYANIDE LEACH OF TOM REED WHITE TAILINGS AT MINUS 200 MESH, FOLLOWED BY LOADING AND FLOATING #999 BLACK(ACTIVATED).

Previous difficulties of getting a gold recovery seemed to lie in poor extraction of the leach. The dissemination of the values may be such that within the given leach time, the values can only be reached by reducing the size of the sand particles.

The 1500 grams of T.R. White Tailings was pulped with distilled water and wet screened to 200 mesh. The wet sands were ground for 60 minutes in the laboratory ball mill with a portion of the 1 lb/T burnt lime rock; .3 lb/T mixed cyanide (1-1); and all of the 2 lb/T # 999 Black for the test. As the slurry in the mill was rather thick, it was diluted to about 60-65% solids with decant solution from the slimes treatment for the last 15 minutes of the grind. A screen test on the ground pulp showed all -200 M.

The slimes were scoured in the fag cell for 30 minutes with about half of the lime and 1/4 of the cyanide solution held out from the grind. The slimes were later combined with the ground sands for a 3.5 hour leach in the fag cell. It was impossible to keep the dilution down, so the cell was entirely full. The carbon was floated roughly for 10 minutes using 1 drop of CCPO frother. The tails were conditioned 1 minute with .10 lb/T Aero 325, then 1 minute with .10 lb/T Aero 343; followed by 2 drops of CCPO frother staged in during the 9 minutes of floating. The rougher concentrate was diluted and cleaned for 10 minutes with another drop of CCPO added after 5 minutes. The solution and a trace amount of floating carbon was decanted from the settled tailings and was added to the recycle cleaner tailings. "Fixing Reagents" were added to the products, after which they were dried overnight in the oven. The results of this test are tabulated below:

Test No. A03-2-14-1

R	Weight grams	%Distr	Au oz/T	Au WxA	Au %Distr	Ag oz/T	Ag Wxa	Ag Distr
Ro Tail(Decanted)	1153	77.2	.01	11.53	40.8	.6	693	64.6
Cleaner Tail (Recycle)	211	14.1	.03	6.33	22.4	1.0	211	19.7
Carbon Conc	11	.7	.40	4.4	15.6	4.4	48.4	4.5
Scavenger Conc	120	8.0	.05	6.0	21.2	1.0	120	11.2
Total	1495	100.0	(.019)	28.26	100.0	(.72)	1072.4	100.0

The .01 oz/T Au tail assay indicated a good gold extraction; however, the balance is dissappointingly low. Ag is high. These discrepancies cannot be explained.

Robert R. Mates

February 25, 1963

To: W. J. Nault

From: Robert R. Mates

Subject: Final Progress Report of Test Work Through Feb. 22, 1963

TEST OF USE OF ANION EXCHANGE RESIN IN RECOVERY OF PRECIOUS METAL FROM TOM REED WHITE TAILINGS PULP

Anion Exchange Resin (XE-123) treated with sodium cyanide was stage contacted (5 stages) with Tom Reed White Tailings which had been put through 48 M screen and pulped. Each contact was for 30 minutes. Overall ore to resin ration was 1500 gm to 500 ml(WSR). Separation was by decantation and washing on a screen. The entire pulp was screened at 48 Mesh for the final separation.

The residue was deslimed by decantation. The bulk of the pulp solution reported with the slimes. Results of the test are shown below:

TEST NO. A03-8-21-1

	Wt.	Au		Ag	
	Grams	oz/T	WXA	oz/T	WXA
1st Stage Resin	30.0	0.24	7.2	5.8	174
2nd " "	29.0	0.06	1.74	4.0	116
3rd " "	28.7	0.03	.86	2.0	58.5
4th " "	27.7	0.01	.28	2.0	55.5
5th " "	29.0	0.14	4.06	1.9	55.0
TOTAL RESIN	144.4	(.098)	14.14	(3.17)	459.0
Deslimed Residue	1182	0.015	17.7	Trace	0
Decanted Slimes Product	240	0.01	2.4	0.4	96
TOTAL RESIDUE	1422	(0.014)	20.1	(.067)	96
Calc. Head	1500	(0.028)	34.24	(.37)	555.0

This test shows a pretty good tail 0.014 oz Au/T. We might do better than this with a grind. However, it is felt that the gold

still in the coarse fraction of the residue is free and could be recovered by gravity methods, and/or amalgamation, or by drying, heating and electrostatic separation. The gold in the slimes could possibly be recovered by further treatment of this fraction by more resin contact.

No attempt was made to see how high the resin could be loaded with gold. Even the highest loading here is far from economic, since there is no method yet known to regenerate the resin. The resin is burned.

Caution must be taken to insure that pH control is maintained so that hydro-cyanic gas is not generated at any time. It is a deadly poison even in relatively small concentrations.

Robert R. Mates

Robert R. Mates

RRM:tm
cc:L. Wiscombe

REPORT ON THE PROPERTY OF TRIUMPH GOLD MINES, INC. OATMAN, ARIZONA.

BY

EDWARD F. CRUSKIE
MINING ENGINEER

June 1, 1948.

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INTRODUCTION:

The Triumph Gold Mines, Inc., of Oatman, Arizona was incorporated July 23, 1946, under the laws of the State of Arizona, to prospect and develop the original group of four Triumph claims in Section 21, Township 19 North, Range 20 West G. & S. R. meridian, Mohave County, Arizona. These claims are located in the Oatman portion of the San Francisco Mining District, about two and one-half miles southwest of the town of Oatman.

In September 1947 the Triumph Company acquired by purchase the White Chief property of five claims, located in Section 26 & 27, one and one-half miles south of Oatman, Arizona. The development work done here on the 200 foot level by the former operators had sufficed to disclose the presence of an ore body of promising tonnage and excellent grade. All criteria pointed to the possibility that further development would block out ore bodies that would be profitable to mine, and, in addition would be more immediately accessible with limited finances and at present day operating costs.

In addition to these two main groups the Triumph Company has acquired by purchase and locaation the intervening area of ground, and now has a total of 25 claims in a contiguous group, comprising approximately 450 acres of mining ground.

LOCATION AND ACCESSIBILITY:

The Oatman district, in which the holdings of the Triumph Gold Mines, Inc., are seated, is also called the San Francisco Mining District. It includes the Vivian, Gold Road, and Boundry Cone localities, covering an area of about 10 miles length by seven miles width on the western slopes of the southern portion of the Black Mountains, in western Mohave County, Arizona.

The lode claims of the company extend from the Vivian district on the west two miles southeasterly to what may be designated as the northern portion of the Boundary Cone district.

By road, Oatman, the principal town of the area, lies four miles northeast from the west end of the claims, and two miles northeast from the easterly or White Chief end of the claim group. Oatman is 29 miles, via U. S. Highway 66, from Kingman, the County Seat, population 4,200. Kingman is located on the Santa Fe Railway, and has good rail, air, and highway connections interstate and intrastate.

From Oatman and U. S. Highway 66, good secondary roads give acess to various mining properties, and to the operating plants of the Triumps Company

DISTRICT HISTORY AND PRODUCTION:

During the early sixties, soldiers from Camp Mohave, at the Colorado River, carried on prospecting in this region. In 1863 or 1864, John Moss is reported to have taken $240,000.00 worth of gold from a pocket in the Moss vein. The Hardy, Leland and Gold Dust veins were found soon afterwards, but the prominent outcrops of the Tom Reed and Gold Road veins remained untested for many years. The town of Silver city grew up at a watering place on Silver Creek about one mile south of the Moss lode, and a small mill was established at Hardyville, on the Colorado River. After the 1866 outbreak of the Hualpai Indians, the district was practically abandoned for several years.

A revival in activity took place in 1900 when rich ore was found in the Gold Road vein. In 1901, the Gold Road Company sank the Tom Reed and Ben Harrison shafts to a depth of 100 feet. The Leonora mill, at Hardyville, operated during part of 1901 and 1902 on ore from the Moss and Hardy veins. During 1903 and 1904, the Mohave Gold Mining Company did considerable work on the Leland property. The Blue Ridge Gold Mines Company produced ore from the Tom Reed vein during part of 1904 and 1905. In 1906, the Tom Reed Gold Mines Company purchased the mine, developed high-grade ore, and, in 1908 started production which continued through 1931. The Gold Road mine produced intermittently through 1931. The town of Oatman was started about 1912.

During 1915 and 1916, a $6,000,000.00 ore body was developed in the United Eastern Mine. This ore shoot did not outcrop, and its discovery prompted scores of mining ventures in the district. The advent of World War I with the usual war-born emphasis on industrial production, and inflated costs, hindered in most of these cases the additional financing necessary to explore a property after shaft-sinking is completed.

In 1916, the Big Jim Mining Company found an important ore body on their Big Jim claim, immediately northeast of the Gray Eagle and Black Eagle claims of the Tom Reed Company. Further work indicated that the Tom Reed or Gray Eagle vein is the upper, downfaulted portion of the Big Jim vein. The displacement, principally along the Mallery fault, amounts to about 400 feet. In 1917, the United Eastern Company purchased the Big Jim ground, but two years later the Tom Reed Company brought suit to establish its apex claim to the Big Jim vein. The courts, however, decided against the Tom Reed Company, saying the amount of horizontal displacement could not be proven.

In 1924, the United Eastern ore body became exhausted and after considerable diamond-drill prospecting the mine was closed.

DISTRICT HISTORY AND PRODUCTION: (Cont)

With the increase in the price of gold from $20.67 to $35.00 an ounce in 1933 a revival in gold-mining activities took place in the Oatman district, and production and exploration activiey continued up the start of World War II. The largest producers, the Tom Reed, Katherine, Vivian and Gold Road had a combined cyanide mill capacity of approximately 1,000 tons of ore per day. In addition to milling their own mine ores, the Tom Reed, Vivian, Katherine, and the Producers at Chloride, cyanided and treated custom ores from not only the district, but the entire county and portions of Nevada.

The total production of gold and silver recorded to the San Francisco Mining District by the Arizona Bureau of Mines, Bulletin 140 1(1936), is $34,675,000.00 in gold and $550,000.00 in silver.

With the increase in the gold price in 1933, considerable additional production was made. Data on this production, believed to be correct, and compiled by J. Carlton Bray, Mining Engineer, list the following for the period 1933-1942.

Tom Reed Mine - - - - - -	$ 3,860,000
Katherine - - - - - -	3,136,000
Gold Road - - - - - -	1,760,000
Producers (Pilgrim) - -	972,000
Vivian - - - - -	540,000
Telluride - - - - -	198,000

HISTORY OF TRIUMPH PROPERTIES:

Triumph Group: The original four claim holding of the Triumph Company, namely, the claims disignated Triumph 1-2-3-4, constitute the major part of the property held some 30 years ago by the Gild Edge Mining Company. This company, on what is now the Triumph 1 claim, sank a vertical shaft in the footwall to a depth which, after unwatering, is found to be 320 feet instead of a reported 200 feet. On the 100 foot level there is found to be a crosscut north, driven 240 feet. The presumed objective of this work was the cutting into and exploration of the vein now called the "North Vein". This crosscut fell short of its objective in distance required.

The 200 foot level has a 91 foot crosscut southwest from the shaft. This crosscut, it is calculated, requires an additional 130 feet to cut the southward dipping Vivian vein from the footwall side. There seems to be no doubt that the primary purpose in sinking this shaft in the first place was to attain this objective.

No other lateral work off this shaft is found. The shaft has been repaired to the bottom and is now in safe and operating condition.

HISTORY OF TRIUMPH PROPERTIES: (Cont)

On the Triumph 4 claim at the east end of the original group there is found to be about 400 feet of shallow tunnel work performed about 30 years ago. The area is difficult of access, on steep slopes. No records are available concerning the ore that apparently was hauled from here by burro teams, and present remaining vein exposures are marginal in grade.

White Chief Group: The White Chief group now consists of six claims designated White Chief 1-2-3, Ethel Morgan, Ethel Morgan 1 and Bullion Fraction.

The presently used White Chief shaft was first sunk to a depth of 100 feet in 1916, and a small amount of crosscutting done westward off this level without finding commercial ore. At the time of this work there were six adjoining active companies, all of which were either sinking or doing development work. Of these, the Boundary Cone and Orion properties were working in ore zones.

After a period of shut down, sinking was resumed on the White Chief shaft in 1926 to a depth of 190 feet, and a second level was established at the 183 foot depth. At this level a crosscut was driven northwest a distance of 40 feet, penetrating through the Golden Beauty vein, as it is now designated. No drifting was done on this vein, and the levels and shaft stood under water till about December 1946, the water standing within 70 feet of the collar.

In December 1946 a partnership called the White Chief Development Company obtained a lease on the White Chief property, unwatered the shaft and levels, and started development work. A crosscut was driven nearly due west a distance of 80 feet. This crosscut cut through two pronounced fissure veins, but, at the point of intersection with the crosscut, the grade of the vein filling is not commercial, averaging $2.00 to $3.00 per ton. Drifting was then started northward on the Golden Beauty vein, and it became immediately apparent that this is the main ore channel of the area. For the distance of 120 feet that the drifting was carried on, the vein averages from three to five feet in width, with hnaging wall margins indefinite, and the grade of the ore ranging from $10.00 per ton to $55.00 per ton over these widths.

The Triumph Company purchased the White Chief property from the White Chief Development Company in September 1947. The light prospecting headframe over the shaft was replaced by a heavy duty frame, some repair was made in the shaft, a change room built, and dumping arrangements, and surface layout, were improved. The drift north was resumed and driven an additional 150 feet, where a faulted zone was encountered, with values in the vein for the last forty feet ranging from $2.00 to $6.00 per ton.

HISTORY OF TRIUMPH PROPERTIES: (Cont)

White Chief group (cont): It seems apparent that the foot-wall in this area warrants crosscutting, but it was deemed more expedient to forego this recourse temporialy and to drift southward first. In the area plus 120 to plus 200 of this north drift unusual high grade ore was encountered, with the hanging wall margins of the drift assaying as high as $256.00 per ton over three foot width. Much of this ore shows coarse free gold in hand specimens.

The drift south on the Golden Beanty vein is advanced 65 feet. It is in commercial grade ore all the way, with the exception of a turn out portion of the drift, which was turned deliberately into the hanging wall zone, to leave at least a 15 foot pillar as the southwest wall of the shaft and manway at the station. The vein exposures of this south drift are averaging from two to four feet in width with face samples averaging from $8.00 to $56.00 per ton.

The present face at the 65 foot work shows an interesting vein structure with the Golden Beauty vein being crossed by an easterly dip slip with northerly dip. Vein filling is showing in the plane of the dip slip. Further work must be done here to determine whether this dip slip is the vein designated as the "C" vein, and to determine the relationship between the two fractures.

Blue Knob Group: The Blue Knob Group of the Triumph Company lies one and one-half miles south of the main Triumph shaft, and one and one-half miles southwest of the White Chief shaft. It comprises the claims known as Philfind, Philfind 1, Philfind 11, Tyran 11 and Tyray 1V. As a group they are situated in the southern half of section 28, T. 19 N., R 20 W. In order to have company assessment work apply for a group, these were linked to the Triumph group by location of the claims Good Luck 1, and Good Luck 11, and, on the east end of the Blue Knob, by location of the claims Alpha and Rita.

done

The work/on these claims is insufficient to prove notable structural continuity of any of the vein exposures, and, in addition, the greater portion of the area involved is mantled with erosional debris and gravels. Their merit lies chiefly in that they are situated so as to intercept the regional trent of the fracture systems striking from adjoining and nearby mines, some of which had had notable production.

GEOLOGY OF THE OATMAN DISTRICT:

The underlying rock forming the basement of the Oatman district is a highly-sheared granitic complex of pre-Cambrian age. Isolated patches of it are found in the western foothill portions at lower elevations, but in the northern portion of the district exposures of this complex are extensive.

GEOLOGY OF THE OATMAN DISTRICT: (Cont)

There the producting mines, as the Katherine, Frisco, Tyro, Sheeptrail, were dound in this basement formation.

Resting upon this granitic basement formation is an extensive series of Tertiary volcanic flows, agglomerates, and tuffs, which show considerable variety in composition and texture. These flows and associated tuffs are of such diverse types as olivine basalt at the basic end of the series and rhyolites at the acid end. Between the extremes are various intermediate types.

The best available evidence suggests that the Black Mountains are a fault block tilted to the east. As the major fractures of the Oatman District dip to the east at steep angles, and as these faults are of the normal type, the crest of the range would be depressed relative to the western portion of the district.

Intrusive into this thick series of flows are several prominent Tertiary intrusive of acid composition. In the vicinity of Mount Hardy these are exposed over an area of several square miles, and they have been designated as the Moss porphry, a quartz monzonite, and the Times Porphyry, a soda granite porphyry.

Several prominent plugs and numerous dykes of rhyolite porphyry cut through the granitic complex and the superimposed lavas, and commercial ore deposition appears to be intimately related to these minor intrusives.

ECONOMIC GEOLOGY OF THE OATMAN DISTRICT:

As outlined by exploratory effort to date, commercial ore deposition in the Oatman District, as well as in the northern portion, or Katherine District, is most prominent in the granitic complex, and in the lowermost flows of the Oatman lava section. These lava members have been disignated as follows, beginning from older to younger. Alcyone Trachyte, Esperanza Trachyte, Oatman Andesite, Gold Road Latite. The earliest developments in depth in the district were made on veins exposed in the latter two members of the lava section and production from these is therefore heaviest. As stated by Ransome and Lausen, however, the wall rocks here suffer little or no replacement, and there is no reason why ore shoots should not be found in any of the igneous rocks that occur in these districts.

Mineralization has been connected with faulting, the veins ossupying fissures or fractures in the enclosing rocks. Certain features of the occurrence of the vein filling in these fractures are important. Study has shown that there is a rhythmic alternation of quartz and calcite deposition in these fractures. In each vein, deposition begins with quartz and closes with calcite. Five stages have been recognized, each of which has its distinctive type of quartz, and they can usually, although not always, be recognized from each other. The calcite shows no such distinctive features.

ECONOMIC GEOLOGY OF THE OATMAN DISTRICT: (Cont)

The first stage of quartz filling which falls into commercial ore range is that called the third. It has a fine grained banded terture, variable color, and ranges in values per ton from $1.20 to $8.00. with the gold to silver ratio being two to three. This quartz is common in the Gold Road vein system only.

The next, or fourth stage quartz, is fine grained, pale green to yellow in color, and often shows casts of platy calcite. Range of values per ton is $4.00 to $20.00, with the ratio of gold to silver being 1 to 2. This quartz is abundant only in ore shoots.

The final, or fifth stage quartz, is fine to medium grained, usually banded, pale to deep honey yellow in color. Range of values per ton is $20.00 and upward. Ratio of gold to silver is four to one. This stage also is abundant only in ore shoots.

The feldspar, adularia, is a common constituent of the higher grade ore, and, in such ore, it is invariably associated with the yellow quartz. It usually occurs as white bands between layers of quartz, but in some specimens, plates of adularia and grains of quartz are intergrown. Gold is frequently found as an aggregate of grains in such intergrowths and thin plates of gold were found in adularia.

In the northwestern part of the Oatman District fluorite, gypsum, and kaolin, are commonly found in the veins. At the Moss mine free gold in the form of stout wires and plates was found enclosed in segregations of fluorite in the veins.

Pyrolusite, hematite and limonite, are quite common in the oxidized ores of the district. Porous quartz which contain an abundance of hematite frequently also carries small flakes and wires of free gold. Pyrite is rarely found in the veins, although it is quite common in the wall rock adjoining the veins. Traces of marcasite, lead, molybdenum, and copper are occasionally found in these veins.

VEIN STRUCTURES OF THE OATMAN DISTRICT:

Vein structures in the district vary from a simple tabular body of quartz and calcite with well defined walls, to lodes which consist of a large number of stringers or veins variable in width and separated from each other by barren rock. Small ramifying stringers may be found branching from the main vein into the wall rock. Frozen contacts are common, of one or both walls.

Usually ore shoots in the veins of the district are encased by pronounced alteration haloes of wall rock. Propylitic alteration and softening are most common. Concomitant with this is a general darkening of the rock due to oxidation and hydration. Where the wall rocks are of more acid type this alteration often becomes one of silicification and pyritization.

VEIN STRUCTURES OF THE OATMAN DISTRICT: (Cont)

These epithermal ore shoots of the Oatman district are lenticular in plan, as shown in development and stoping operations of the mines of the district. Unlike the lens-like ore bodies in many other districts, the ore bodies of this district are characterized by an impressive continuity in pitch length and breadth, as well as in thickness.

Strike faulting in and along the veins of the district is common. These faults, reopening a fissure zone filled with early stage, and usually low grade vein filling, serve as the ore channels for the later commercial ore surges. Where they at times leave a vein in drifting or stoping operations, there is a tendency on the part of operators to follow the vein, which all too frequently diminishes in values, and the strike fault linking with a minor dip slip into a parallel lens in the wall is overlooked. Frequent crosscutting and geologic control are required in the proper exploitation of these ore deposits.

ORE DEPOSITS OF THE TRIUMPH PROPERTY:

Triumph Group:

On the original Triumph group of four claims, the chief vein, toward the exploration of which the present effort is projected, is the Vivian vein. The Vivian mine is developed by three shafts, drifts, and tunnel to a depth of 270 feet. It was actively worked 40 years ago, and has a production estimated at $300,000.00 in gold and silver.

The Triumph shaft is located approximately 750 feet easterly across the Vivian wash from the portal of the Vivian tunnel. The line of cropping of the Vivian vein from the portal of the tunnel to the Triumph shaft collar is largely covered by wash gravel and sand. However, the chloritized, pyritic andesite on the Triumph dump is of the same nature as in the Vivian mine workings, and there appears to be no reason why the Vivian vein should not be found in the Triumph property. At the 200 foot level of the Triumph shaft there is found to be a 91 foot crosscut southwest. Driving this crosscut an additional 130 feet should cut the Vivian vein from the footwall side.

The Vivian vein, as worked on the Vivian property, years ago, is a strong vein, over half a mile long. It strikes north 75 degrees west, and dips 82 degrees south. The average width is three feet, and it consists of considerable fourth stage commercial quartz, with crystalline dark calcite and earlier stage quartz. The gold in the Vivian vein varies from fine to coarse colors, and in the early days of production values up to $9.500 per ton were reported.

Triumph Group (Cont):

The "North Vein" on the original Triumph group is approximately parallel to the Vivian vein. Shallow surface work on this vein shows it to be a promising fracture as to width and continuity, and, values obtained justify exploration of this vein at depth. The intervening area between the Triumph shaft and this "North Vein" is a well stringered zone and should be crosscut on several levels.

On the claims of the company that lie between the original Triumph group of four claims, and the White Chief group of six claims, little development work has veen done to date. Most of the area involved is covered with detrital material and gravels. The most promising showing is at the discovery of the Tyray 11 claim. Here a 25 foot shaft discloses a northerly striking, wide, fractured zone, in gray Alcyone trachyte. A width of three feet along the hanging wall margin assays up to $3.00 per ton in gold.

White Chief Group:

The most significant and main development conducted by the Triumph Company is on the White Chief, where drifting south on the Golden Beauty vein is in progress on the 200 foot level.

As shown on the White Chief map, four veins are outlined on the property near the shaft area; These veins are seated in the Alcyone Trachyte, and in the Oatman Andesite flows of the Oatman lava section. With the exception of the "C" vein, the prevailing strike is northeasterly. This trend is significant, showing parallelism with a pronounced rhyolite dike system to the west extending from the Boundary Cone Rhyolite plug to the Elephant Tooth plug. This rhyolite dike system strikes into the Tom Reed vein system on the north at a locus where heavily producing ore bodies were found.

The "Gloryhole" Vein was the first vein of this group in which ore was found, and it was this discovery which led to the sinking of the shaft in 1916. The site of the discovery is a pit 15 feet deep, with perhaps an additional 10 feet of depth obscured by caving ground. It is reported that the ore was a narrow talc seam of very high value. Sampling of present available exposures here fails to confirm the presence of high grade ore. In as much, however, as the fracture zone here has an impressive width, underground exploration of this vein is also warranted.

The "B" vein and the "C" vein, as exposed in surface cuts, are sheeted zones in the Alcyone Trachyte. Their content of vein filling at the surface horizon is meager, but, in as much as these zones will intersect the Golden Beauty vein, their possible influence on ore localization cannot be denied.

White Chief Group (Cont):

The "Golden Beauty" vein is a fault fissure vein, with the Alcyone Trachyte forming the footwall, and the Oatman Andesite the hanging wall. Pronounced strike faulting is evidenced in the vein zone and as shown by available slickensides the faulting is normal. The vein filling is chiefly quartz and rock breccia. Dark calcite is occasionally found. The quartz varies from coarse, glassy, copper stained, to fine drusy crystals lining small vug holes and areas of corrosion in the rock breccia. The broken ore is darkened by pulvurent manganese oxide. Adularia is commonly found associated with this drusy quartz. The gold found in this late stage quartz shows up as clusters of small grains. In the high grade portion of the north drift the Golden Beauty vein is a stringer lode from four to eight feet wide with no definite hanging wall exposed. The ore is an extremely hard quartz carrying small inclusions of rock breccia. It shows copper stain, and varies from banded chalcedonic quartz to finly crystalline drusy quartz. Some of the Quartz irself has a greenish tint but usually it is colorless. This quartz shows considerable well crystallized hematite. The gold occurs chiefly with this hematite as inclusions of small wires and foils of the metal.

The ore shoot as outlined to date by drifting on the Golden Beauty vein on the 200 foot level is 250 feet in length. It is undoubtedly a primary ore of commercial grade in itself. Repeated slight fault movement in and along this vein zone has enhanced the value of the ore by giving access to oxidizing solutions, which have apparently dissolved and leached away considerable of the calcite gangue in the vein, and made it porous and friable as a whole. Under similar favorable conditions in the productive veins of the district, this process has been found to have a dual result. Not only a negative enrichment by dissolving away of valueless constituents, as outlined above, but an accretion of value mechanically, by downward washing of grains and foils of gold freed from friable accociated hematite at upper horizons of the vein zone. This is in a measure analogous to cave deposits in linestone and required an open channel with vigorous circulation of meteoric waters.

BUILDINGS, EQUIPMENT, MATERIALS:

The assets of the corporation, in addition to the above described mining claims, consist of the following:

White Chief Shaft:

1-42'-8x8 Timber headframe, A-type	$1,000.00
1-25 h.p. Fairbanks Morse gas hoist	2,000.00
1-15x25 Hoist House	1,000.00
1-10x14 Change Room	200.00
1-Chicago Pneumatic, No. 8800, Diesel Air Compressor 315 Cu. Ft.	5,608.00
1-Sullivan 210 Cu. Ft. Air Compressor - 10x10	400.00
1-Air Receiver	150.00
1-Cameron No. 3 Pump	400.00
2-Piston Pumps	300.00

Shite Chief Shaft (Cont):

1-Water storage tank	250.00
7-Jackhammers	1,600.00
1-Drifter	545.00
1-Stoper	475.00
2-Mine cars	200.00
1-1942 Chevrolet Truck	1,025.00
12,000" 8x8 mine timber	960.00
4,000' 3"x12" mine timber	300.00
6,000' 2"x12" mine timber	750.00
600' 2" pipe	120.00
300' 12 pipe	42.00
2500' 3/4" pipe	250.00
200' 1/2" pipe	16.00
400' 8" Vent tube	314.00
3,000 lbs. 12# mine rail	200.00
800' shaft hanging rod stock 3/4"	95.00
Miscellaneous Tools and Fittings	1,000.00
Welding Outfit	150.00
White Chief Shaft - Total	$19,350.00

TRIUMPH SHAFT:

1-42 Ft. Headframe, A-type, constructed of 8x8 timber	1,000.00
1-25 H.P. Fairbanks Horse Gas Hoist	2,000.00
1-25x20 Hoist house	1,250.00
1-25x20 Blacksmith Shop	1,000.00
1-Shower and Change Room	300.00
1-Powder Magazine, 6x10	100.00
1-18,000 gallon , Steel water storage tank	1,000.00
2-1,000 gallon supply tanks	300.00
Milling equipment, gravity concentrators	8,000.00
Miscellaneous Tools, Fittings and Equipment	1,496.00
Triumph Shaft - Total	$16,446.00
White Chief Shaft Total	19,350.00
Total Assets, Triumph and White Chief Shafts	$35,796.00

ORE RESERVES:

On the basis of the development work, or drifting, that has opened up a length of 250 feet of Golden Beauty vein on the 200 level of the White Chief operation of the Triumph Company, and bearing in mind the geologic criteria and history of similar productive veins of the district, when at a similar stage of development, I am making an estimate of ore reserves, as follows:

Ore in sight - 12,500 tons @ $22.00 per ton - gross value	275,000.00
Possible Ore - 41,200 tons @ 15.00 per ton - gross value	618,000.00

RECOMMENDATIONS AND CONCLUSIONS:

The White Chief operation of the Triumph Company has opened up the most significant discovery of a virgin ore body found in the Oatman district in nearly 30 years. The operation has reached a point where the development program should be expanded, and moderate milling facilities provided for the property. The company has an opportunity to purchase for $30,000 a used 100 ton per day capacity mill, and should do so. If, while further development work is going on, the mine operation should fail to supply the capacity of this mill at any time, custom milling of ores from the district should balance the temporary deficiency.

The following mine development program is recommended;

(1) The placing of the Triumph shaft operation on a standby basis, and concentration of all effort on development of the White Chief mine.

(2) The raising out of a well-timbered, two compartment vertical shaft, from a point in the north drift 260 feet from the present operating shaft. This raising operation can be done concurrently with drifting southward on the Golden Beauty vein, and other lateral exploration and development of the 200 foot level.

(3) Sinking of the newly raised shaft 125 feet, and cutting out stations for lateral work north and south on the Golden Beauty ore shoot already opened on the 200 level.

(4) Bringing in electric power to the property by erection of approximately a quarter mile power line to link with the public utility main line.

(5) Installation of 75 h.p. electric hoist at the collar of the new shaft, and erection of permanent surface plant here.

(6) Retimbering of shaft in present use, leaving present surface plant as is, so as to have an auxiliary operating shaft and escapeway.
(7) Purchase and erection of used cyanide mail of 100 ton per day capacity near the collar site of the new shaft.

The development program outlined above is required to place this mine on a production basis. I am of the opinion that the completion of this development program will insure the productive future of the mine, and pay dividends to all participants in this unusually meritorious venture.

Respectfully submitted,

Edward F. Cruskie
Mining Engineer
Oatman, Arizona
June 1, 1948.

CERTIFICATE OF ASSAY FROM LABORATORY OF
R. V. McALLISTER
ASSAYERS, CHEMISTS AND METALLURGISTS
Front Street
Kingman, Arizona

June 1, 1948.

TRIUMPH GOLD MINES, INC.

OFFICE NO.	OWNER'S MARK	GOLD, OZ.	PER TON VALUE	SILVER, OZ.	PER TON VALUE	TOTAL VLUE GOLD & SILVER
16105	# 1	0.54	18.90	0.60	0.~~72~~ 54	19.62
16106	2	0.03	1.05	0.20	0.18	1.25
16107	3	0.10	3.50	0.20	0.18	3.68
16126	1 Grab 15 cars	0.20	7.00	0.40	0.36	7.36
16127	2	1.04	36.40	0.40	0.~~54~~ 36	36.94
16128	3	0.14	4.90	0.30	0.27	5.17
16129	Triumph	2.43	85.05	2.20	1.98	87.03
16140	1 C	0.04	1.40	0.20	0.18	1.58
16141	2 C	0.46	16.10	0.60	0.54	16.64
16142	3 C	0.03	1.05	0.20	0.18	1.23
16143	4 C	0.06	2.10	0.60	0.54	2.64
16134	1 C	1.38	48.30	1.30	1.17	49.47
16135	2 C	0.07	2.45	0.20	0.18	2.63
16136	4 Grab	0.26	9.10	0.60	0.54	9.64
16137	5 Grab	0.20	7.00	0.40	0.36	7.36
16230		2.24	75.40	15.80	14.22	92.62
16260	1 C Feb. 26	2.89	101.15	2.60	2.34	103.49
16407	1 C Mar. 17	0.17	5.95	1.40	1.26	7.21
16318	1 C Mar. 4	0.72	25.20	1.40	1.26	26.46
16474	1 C Mar. 22	1.71	59.65	1.90	1.71	61.36
16644	1 C Apr. 4	0.10	3.50	0.40	0.36	3.86
16740	1 May 1	1.63	57.05	1.30	1.17	58.22
16751	1 C May 4	0.03	1.05	0.20	0.18	1.32
16752	2 C May 3	1.62	56.70	1.80	1.62	58.32
16753	Screenings on dump May 3	1.28	44.80	1.40	1.26	46.06
16763	1 C May 6	0.22	7.70	2.40	2.16	9.86
16750	C May 7	2.11	73.85	1.20	1.08	74.93
16795	F May 9	0.17	5.95	0.20	0.18	6.13
		4.14	172.90	2.40	2.15 ~~16~~	175.05
16796	10	0.18	6.50	0.20	0.18	6.68

HAMLEY & HAMLEY.

Average $37.79 per ton

211019	1 C	0.25		0.35	8.25
211021	2 C	0.28		0.03	7.80
211022	3 C	6.61		4.09	233.80

200 level drift. Average $37.3 per ton

ARIZONA TESTING LABORATORIES

A DIVISION OF CLAUDE E. McLEAN & SON LABORATORIES, INC.
PHONE ALpine 3-6272 817 WEST MADISON ST. P. O. BOX 1888 PHOENIX 1

Chemists . . . Engineers

For Americana Investments
3015 East Thomas Road
Phoenix, Arizona

Date September 26, 1962

Sample of Ore

Received: 9-24-62

Submitted by: Mr. Tom Parker

Material marked: White Chief

ASSAY CERTIFICATE

Gold figured at $ 35.00 per ounce.

Silver figured at $ 1.00 per ounce.

Lab. No.	Identification	Gold		Silver		Percentages	
		Oz. per Ton	Value	Oz. per Ton	Value		
155302	#1 Winze, 30' S.end 6' wide	0.06	$2.10	0.60	$0.60		
155303	#2 Winze, 20' S. end 6' wide	0.63	23.80	0.50	0.50		
155304	#3 Winze, 10' - 8'wide	0.26	9.10	0.50	0.50		
155305	#4 Top of Winze back of drift 5'	0.52	18.20	0.50	0.50		
155306	#5, 0-90', 5' on Footwall	0.10	3.50	0.20	0.20		
155307	#6, 0-25', 3' on Footwall	0.22	7.70	0.20	0.20		
155308	#7, 4' top of drift	0.24	8.40	0.20	0.20		
155309	#8, 0-60', 5' bottom	0.92	32.20	0.30	0.30		
155390	#9, 1-20', 5'	5.06	177.10	2.60	2.60		
155391	#10, 1-30', 3' H.W.	2.98	104.30	2.40	2.40		
155392	#11, 1-60, 5'	0.22	7.70	0.20	0.20		
155393	#12, 2-00, 5' bottom	0.08	2.80	0.30	0.30		

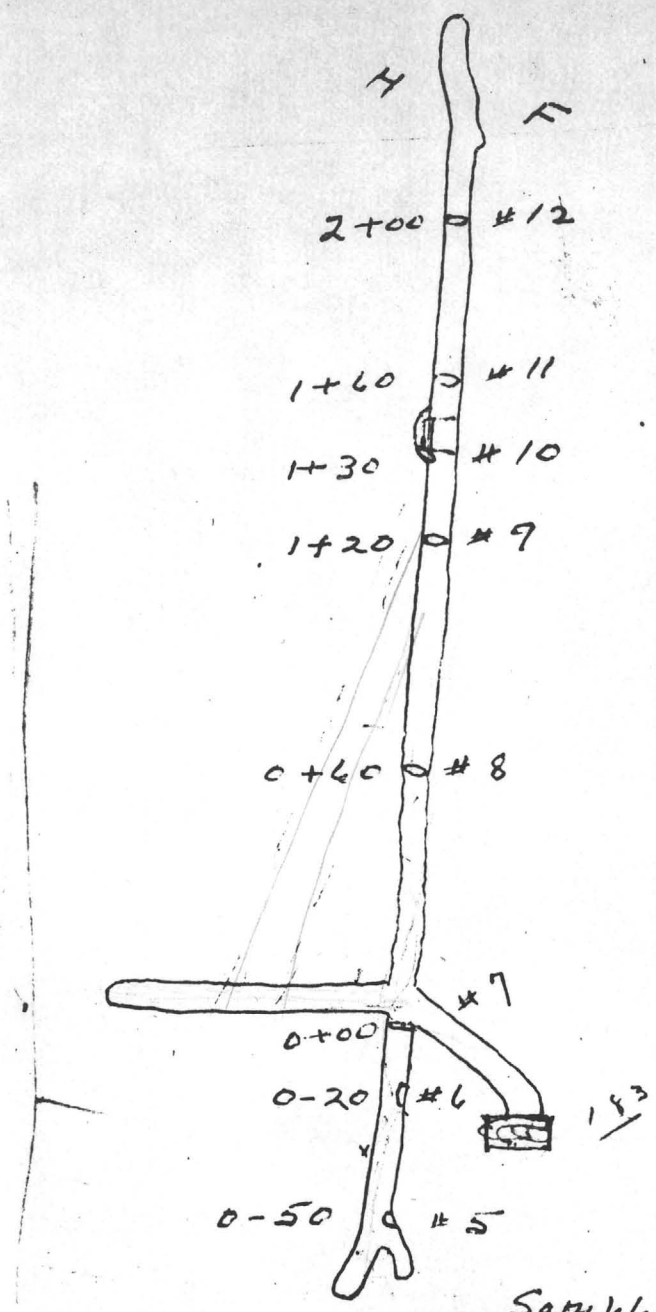
REGISTERED ASSAYER
CERTIFICATE EXPIRES
CLAUDE E. McLEAN
DEC. 31, 1962
ARIZONA, U.S.A.

Respectfully submitted,
ARIZONA TESTING LABORATORIES

Claude E. McLean

10M AMPCO

White Cliff
Sample Map
Sept 22, 1962



Sample No.	Au	Ag.	# Tot
# 5	.10	.20	3.70
# 6	.22	.20	7.90
# 7	.24	.20	8.60
# 8	.92	.30	32.50
# 9	5.06	2.60	179.70
# 10	2.98	2.40	106.70
# 11	.22	.20	7.90
# 12	.08	.30	3.10

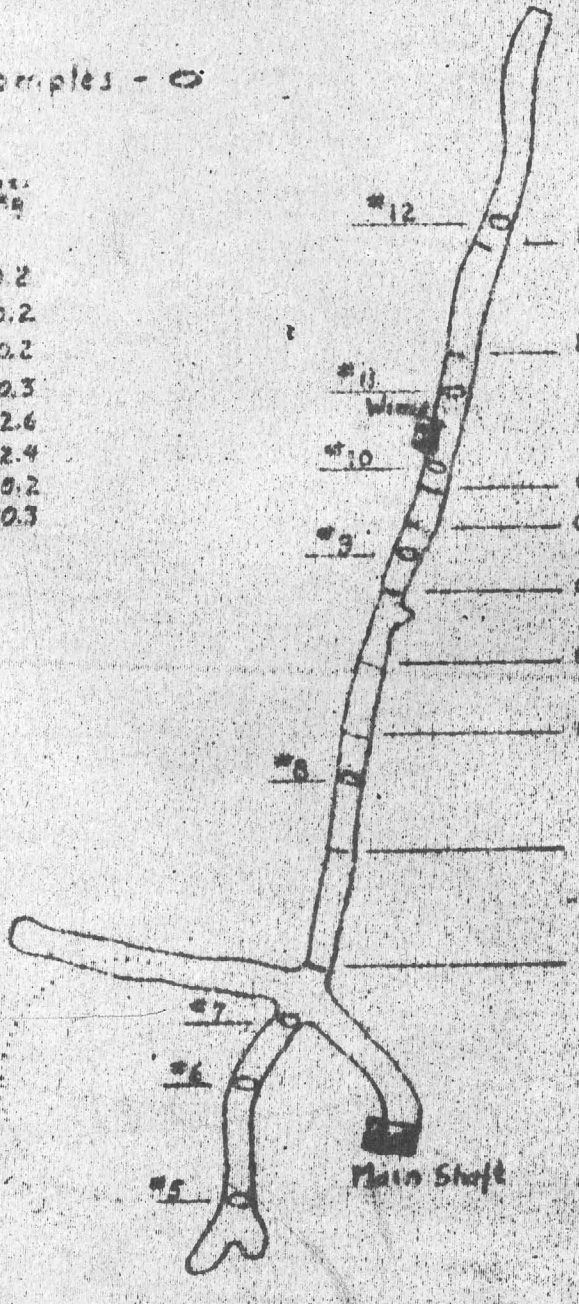
WHITE CHIEF MINE
OATMAN, ARIZONA

SAMPLE MAP - 190' LEVEL
Nov. 14, 1963

N

Wiscombe Samples - ○

Sample No.	oz. Au	oz. Ag
#5	0.10	0.2
6	0.22	0.2
7	0.24	0.2
8	0.92	0.3
9	5.06	2.6
10	2.98	2.4
11	0.22	0.2
12	0.08	0.3



Metals Engineering Samples ↓

Sample #	Cut	oz. Au	oz. Ag	Value/Ton
1+55	1552 - 3'	0.21	0.4	#7.87
1+32	1551 - 2'	0.38	0.5	13.95
1+03	1545 - 6'	0.14	0.5	5.55
0+95	1544 - 5'	0.08	0.3	3.19
0+80	1546 - 3.5'	0.11	0.5	4.50
0+65	1547 - 5'	0.04	0.2	1.64
0+50	1548 - 3'	0.09	0.2	3.41
0+25	1550 - 3'	0.02	0.3	1.09
0+00	1549 - 3'	0.23	0.3	8.44

Shop No. 1544-52
File No. .....

Date 25 OCT 1963

CHAS. A. DIEHL
(Registered No. 682)

Arizona Assay Office

Phone ALpine 3-4001

815 North First Street
Phoenix, Arizona
P. O. Box 1148

VALUES
Latest Quotation

1 oz. Gold.....
1 oz. Silver.....
1 lb. Copper.....
1 lb. Lead.....
1 lb. Zinc.....

THIS CERTIFIES
Samples submitted for assay contain as follows:

HOOKER MINING CO

Short Ton ..... 2000 Lbs.
Short Ton Unit ..... 20 Lbs.
Long Ton ..... 2240 Lbs.
Long Ton Unit ..... 22.4 Lbs.

MARKS	SILVER PER TON Ozs.	SILVER PER TON Tenths	VALUE PER TON	GOLD PER TON Ozs.	GOLD PER TON 100ths	VALUE PER TON	TOTAL VALUE PER TON of Gold & Silver	PERCENTAGE	REMARKS
1544 / 0 - 95					.10	$3.50			
1545 / 1 - 03					.20	$7.00			
1546 / 0 - 80					.02	$ .70			
1547 / 0 - 65					.08	$2.80			
1548 / 0 - 60					.02	$ .70			
1549 / 0 - 00					.41	$14.35			
1550 / 0 - 25					.02	$ .70			
1551 / 1 - 32					.44	$15.40			
1552 / 1 - 55					.30	$10.50			

REGISTERED ASSAYER
CERTIFICATE EXPIRES
J CK E. A.
STONE
ARIZONA, U.S.

Charges $ 22.50

Assayer .....

ANDY CHUKA, PRINT

MEMORANDUM

TO

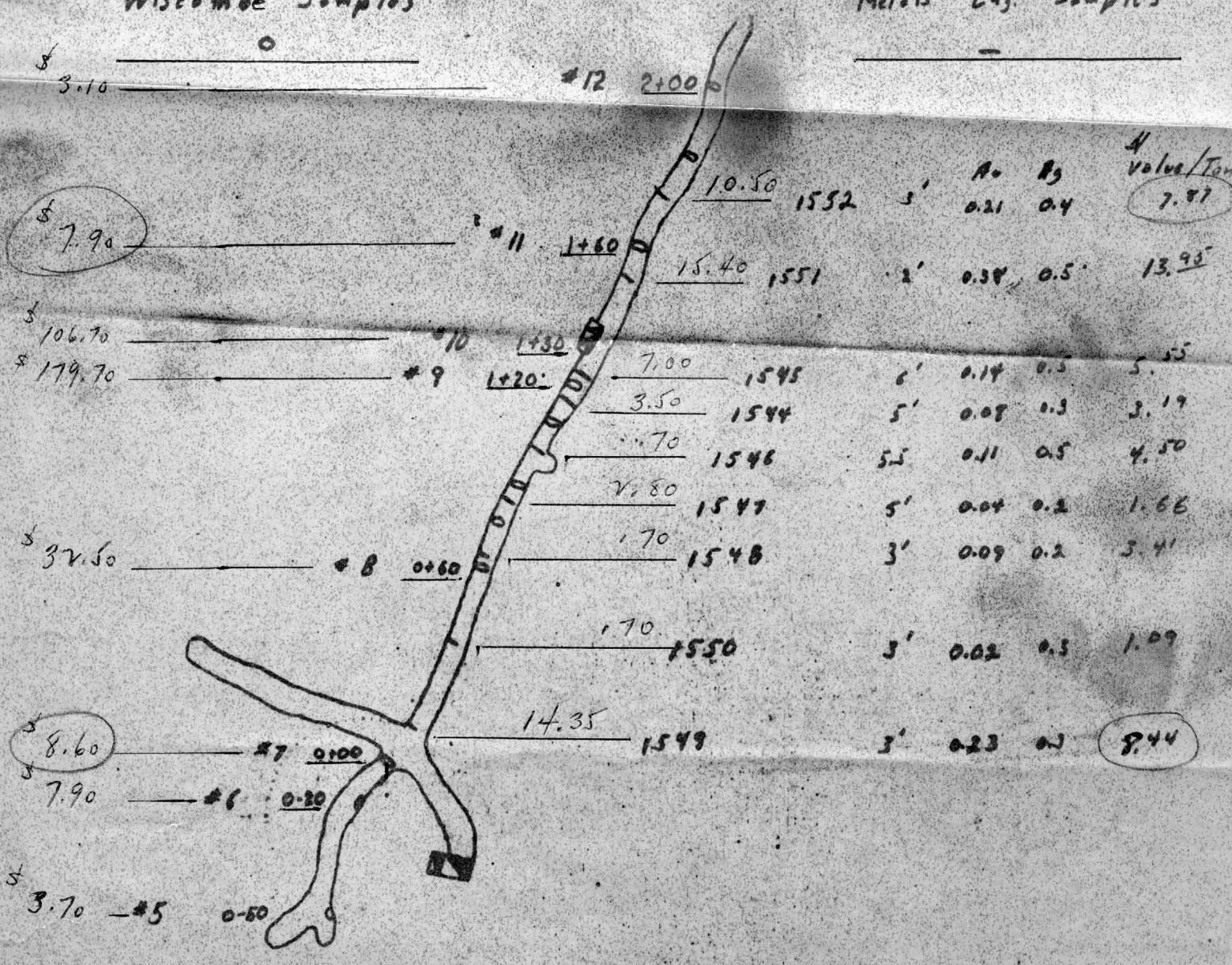
DATE

FROM

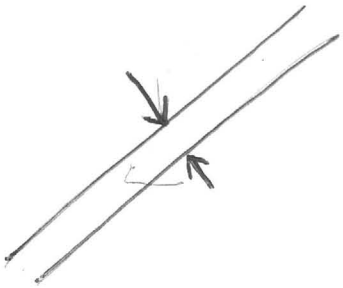
SUBJECT

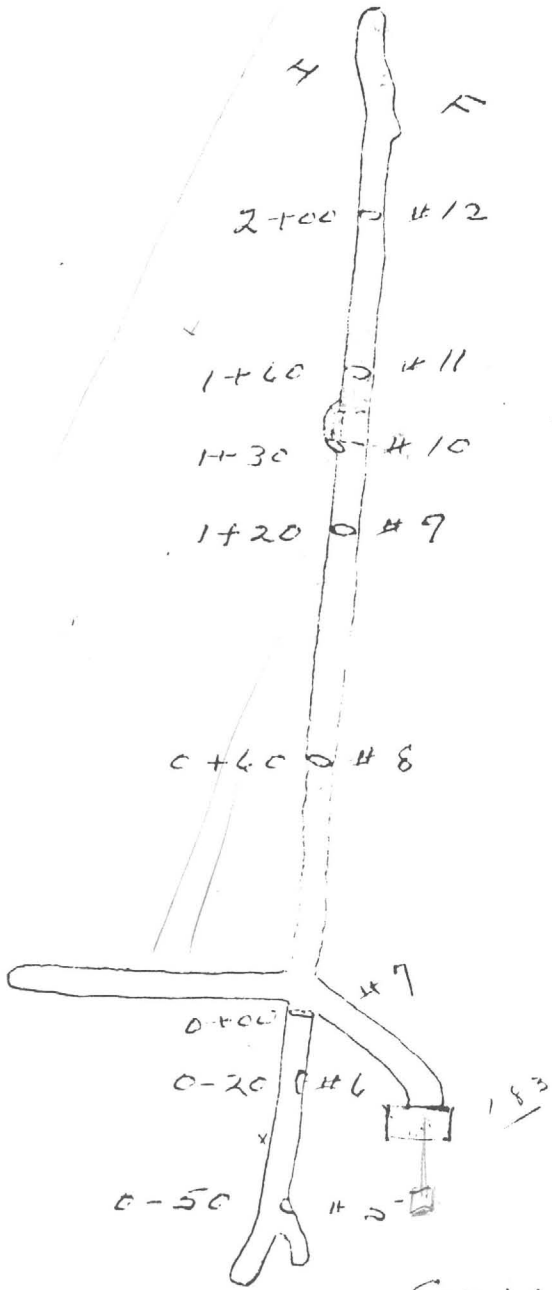
Wiscombe Samples — 0

Metals Eng. Samples — —



White Chief
Sample Map
Sept 22, 1912





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Winze

REGISTERED ASSAYER
CERTIFICATE EXPIRES
CLAUDE E. McLEAN
DEC. 31, 196[illegible]
ARIZONA, U.S.A.

Respectfully submitted,
ARIZONA TESTING LABORATORIES

Claude E. McLean

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MEMORANDUM

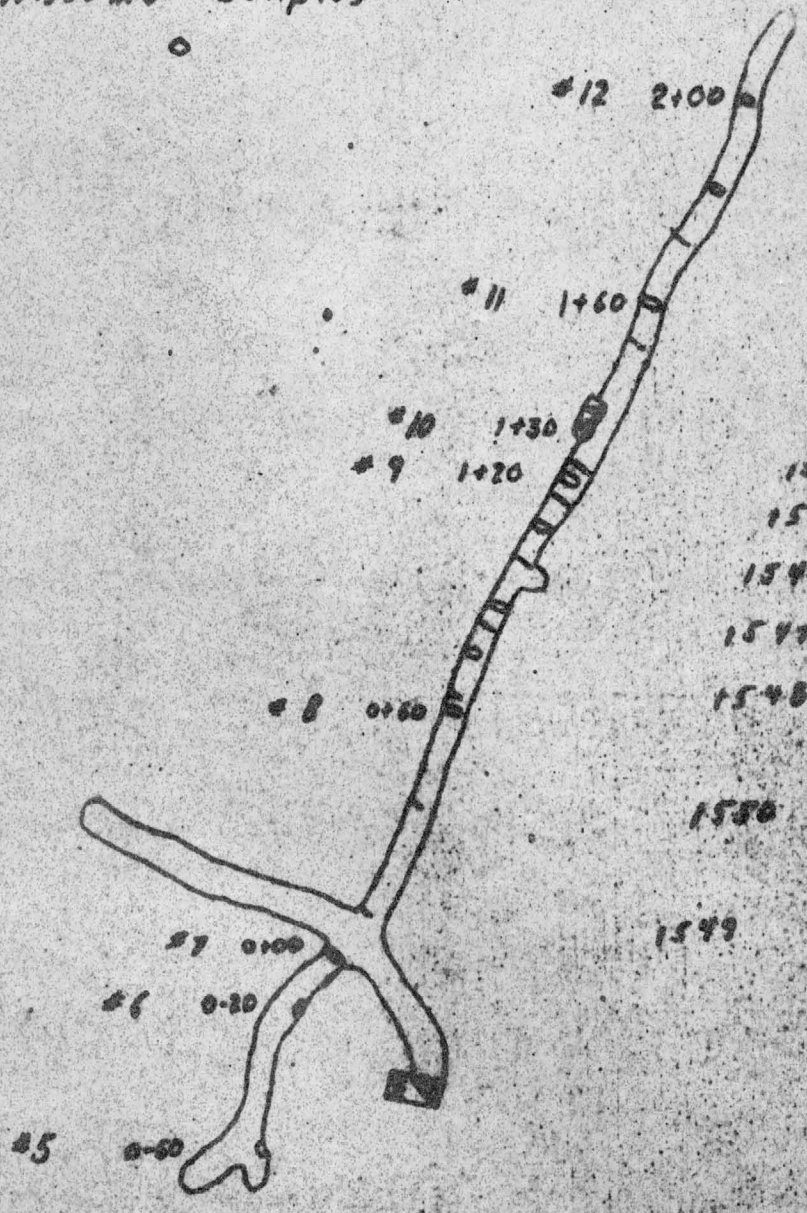
TO

DATE

FROM

SUBJECT

Wiscomb Samples



Metals Eng. Samples

		Au	Ag	Value/Ton
1552	3'	0.21	0.4	7.87
1551	3'	0.38	0.5	13.95
1545	5'	0.14	0.2	5.55
1544	5'	0.08	0.3	3.19
1546	5'	0.11	0.5	4.50
1547	5'	0.04	0.2	1.66
1548	3'	0.09	0.2	3.41
1550	3'	0.02	0.3	1.09
1549	3'	0.23	0.5	8.44

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