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Item 5

P I N E G R O V E M I N E S

Y E R I N G T O N N E V A D A

Reports by

Hubert C. Taber

David J. Blackner

Metallurgical Reports

Southwestern Engineering Corporation

Dump Sampling

H. B. Menardi

Foreword

Edward Thornton

FOREWORD

In submitting the enclosed reports wish to state that I have talked to both Mr. Taber and Mr. Menardi and both are conservative and reliable men yet there is quite a divergence in their statements.

Mr. Taber operated in 1928 for some Los Angeles brokers who came a cropper through the Julian affair before he really started. He did quite a lot of work on the mine and took a great number of samples in the mine and states he also took over 100 samples on the large dump, and had a surveyor measure it up with a transit. The following is what I have gathered from a couple of talks with him before and after talking with Mr. Menardi.

The mine above the tunnel level has been pretty well worked out yet probably 30 to 40,000 tons can be rawhided that will run say $10.00 on the old price.

The ore is irregular, that is to say you may have a face of $40.00 ore toda; and very low grade tomorrow, but by careful minning a good grade can be maintained. The ground breaks easily and stands fairly well.

Below the tunnel level the mine is under water, but it only makes 25 to 50 gals. per minute. Mr. Taber never unwatered but states that he shipped some high grade ore that was proported to have come from the inclined shaft. It is stated that the 1800 ft. drift from the 200 ft. vertical shaft contains some high grade ore. This drift was driven to prove up a law suit on apex rights.

Mr. Taber is of the opinion that development below the tunnel level will prove up a very valuable property as the ore is apparently better in depth than at the surface.

While as shown on plate I the veins finger at the surface and these are numbers of stringers. Mr. Taber is of the opinion that it would not make a large low grade body.

Mr. Taber states that the results of his sampling the large dump was $3.50 per ton old price and that it contains 200,000 tons.

Mr. Menardi went up to sample that Pine Grave dumps in 1930 in connection with a proposed mill installations. He says in his report that he "squared" up the dump and figures 107,000 tons. He made two trips taking about a dozen samples each time. He gave an average of $2.20 per ton.

Mr. Menardi believes that there should be 50,000 tons of scattered dumps exclusive of the tailings that will average better than the large dump.

He made a very cursory examination of the mine but states that the lower mill tunnel dump gives on an average of $1.60 per ton which in checked by Mr. Taber and that the tunnel is a cross-section of the entire hill and that it all seems mineralized. He feels that nearer the surface where there is a greater prefonderance of fingering that the whole hill may pay to work.

Regarding the report on the dumps by David J. Blackner, I know nothing about him. He gives $4.20 for one half of the large dump and $5.95 for the other half or an average of $5.07. He calculates the tonnage at 200,000 tons. He also believes that there are possibilities for a large low grade.

It would seem there was no reason not to expect profitable ore below the tunnel level and very probably a profitable mine can be developed.

From what has been described to me as the condition of the fingering and taking into consideration the valueo of the dumps I am inclined to agree with Mr. Menardi and Mr. Blackner, that there are possibilities of a large low grade deposit.

Water

Water is obtained from springs that can furnish about 75 gal. per minute, however on the other side of the hill a tunnel of some 45,00 feet long was driven to cut the veins but was stopped before reaching them. This tunnel is making 150 gals, per minute so feel there will be no difficulty in the matter of water supply.

EQUIPMENT BUILDINGS

The equipment has been sold or stolen, but the buildings should be in fair condition, at least cost very little to make them usable.

FINANCIAL OUTCOME

In setting up the financial out-come I have been in doubt as to what figures to use in view of the divergence of opinions and results.

The eleven shallow pit sample taken over an area of 125 ft. by 500 ft. hardly seem enough to determine the value with any degree of accuracey. Mr. Taber took over 100 samples and figures $3.50 as conservative and is very positive regarding the tonnage.

Probably $3.00 would be a conservative approximation on value and 175000 on tonnage, in the large dump and 50,000 tons in the scattered dump disregarding the tailings dump which Mr. Menardi reported at $2.20 per ton.

While Mr. Menardi uses 90% extraction and it will be probably attained in practice, particularly if the concentrates are cyanided, as a lower grade concentrate can be made with higher extraction, it seems safer however to use 85% on the dump ores and 90% on the mine ores.

$3.00 old price	85% Recovery	$5.25 new price
2.55 old price		4.46 new price

The costs are based on a 250 ton mill as there would be several years run on the dump alone.

Milling Cost	$.90
Handling dumps and overhead.	$.40
Total	$1.30

Value	$2.55 old price	$4.46 new price
Costs	$1.30	$1.30
Gross Profit	$1.25 " "	$3.16 " "
Corporation and State Taxes.	.19	.42
Net Profit	$1.06 " "	$2.74 " "

While there is no actual measurable ore in sight Mr. Taber states there are 31 places ready where ore can be stoped now and several others can be prepared. By rawhiding the mine I would judge there is a very probable chance that at least 30,000 tons of $10.00 ore can be extracted.

As the mine is opened up and the development done in the upper levels a mining cost of $2.00 and $1.50 development for the lower levels should be ample.

After two or three months run on the dump ore the tonnage could be cut 50 tons and that amount supplied by the mine.

Value	$10.00 old price	$17.50 new price
	90% Recovery	
Value	$9.00 old price	$15.75 new price

Milling Cost	.90		.90	
Mining Cost	$3.50		$3.50	
Total Cost		$4.40		$ 4.40
Gross Profits		$4.60		$11.35
Corporation and State Taxes.		.60		1.41
Net Profit		4.00		9.94

This would give a total indicated profit of :

Dumps	$616,500.00
Mine	297,200.00
Total	$913,700.00

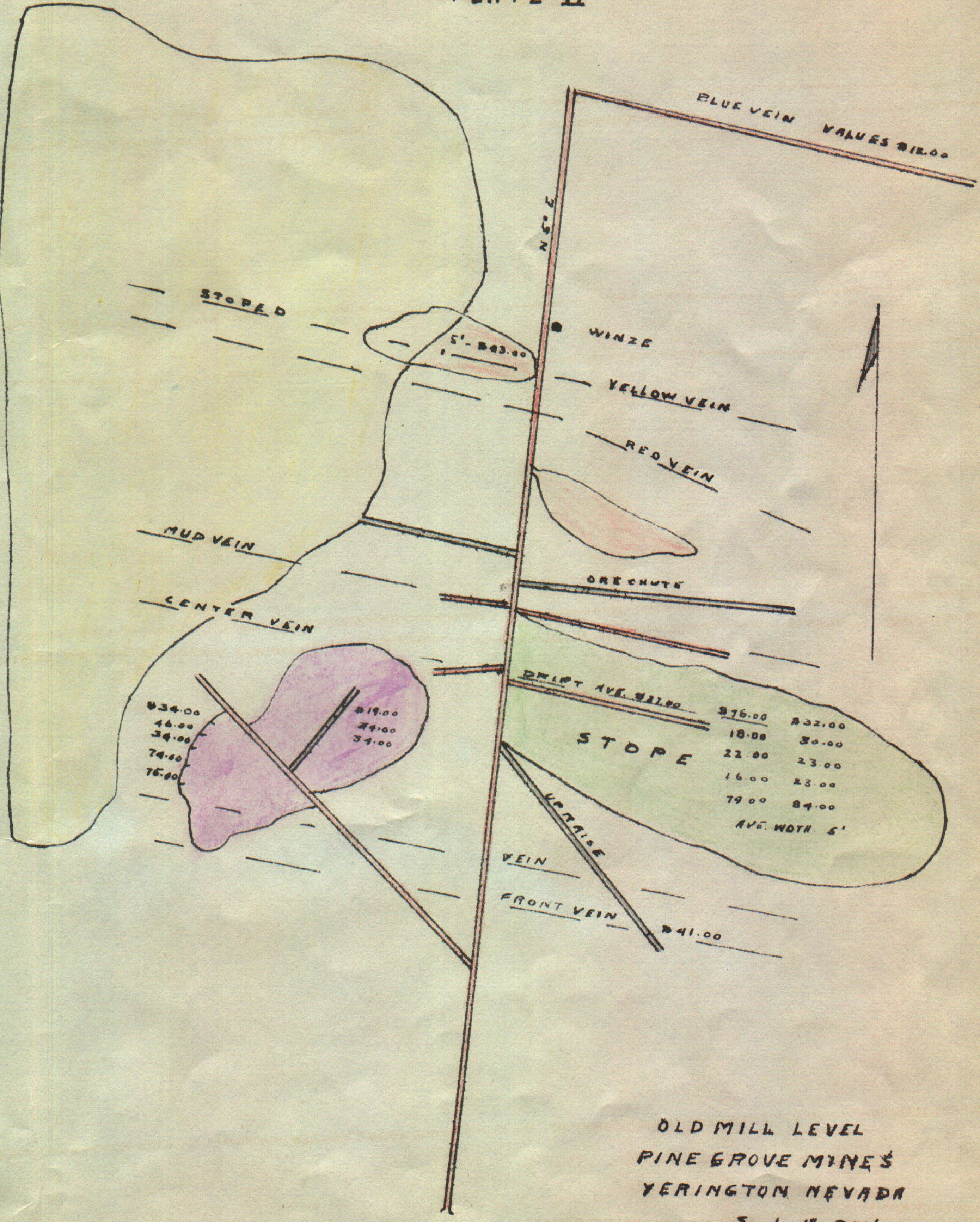
I have checked up with several engineers and machinery salesmen and feel reasonably sure that we could equip the property with good second hand machinery for not to exceed $60,000 and possibly considerably less in case we were fortunate in locating some good buys. We would require another $15,000 per working capital so believe that $75,000 should put the property on an earning basis of say $25,000 monthly.

Feb. 27, 1934

Respectfully submitted

Edward Thornton

PLATE II



OLD MILL LEVEL
PINE GROVE MINES
YERINGTON NEVADA
Scale 1" = 200'

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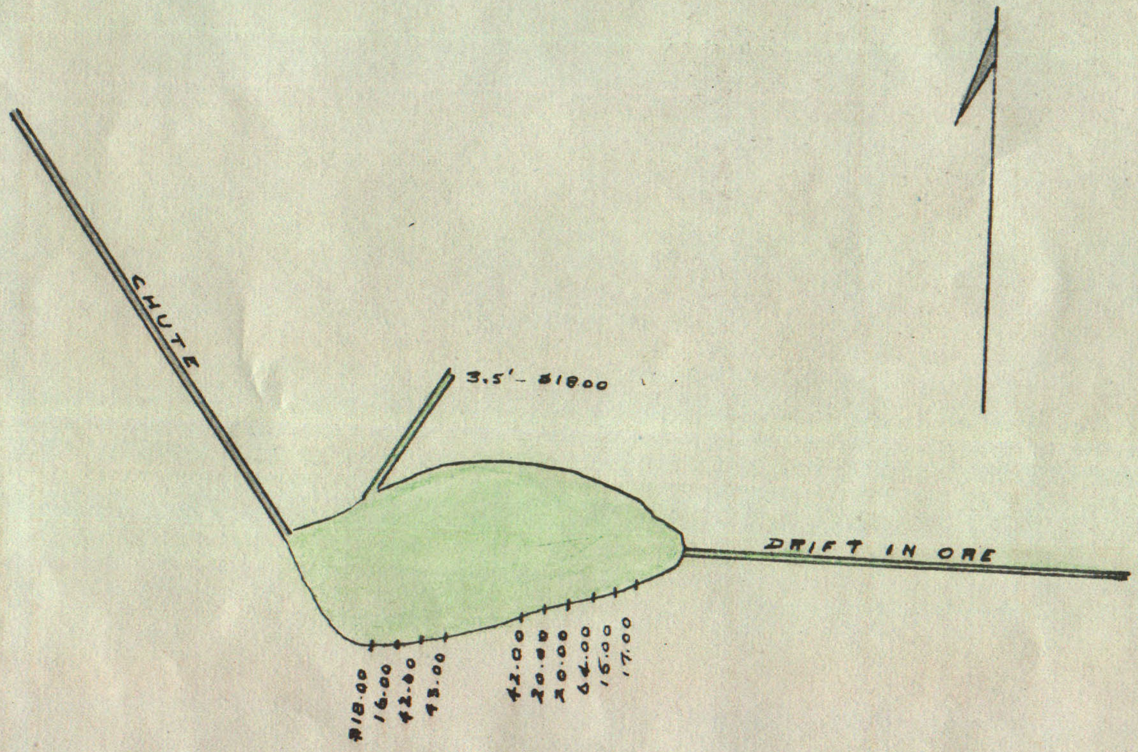
PLATE III



SUB-LEVEL
PINE GROVE MINES
YERINGTON NEVADA
Scale 1" = 200'

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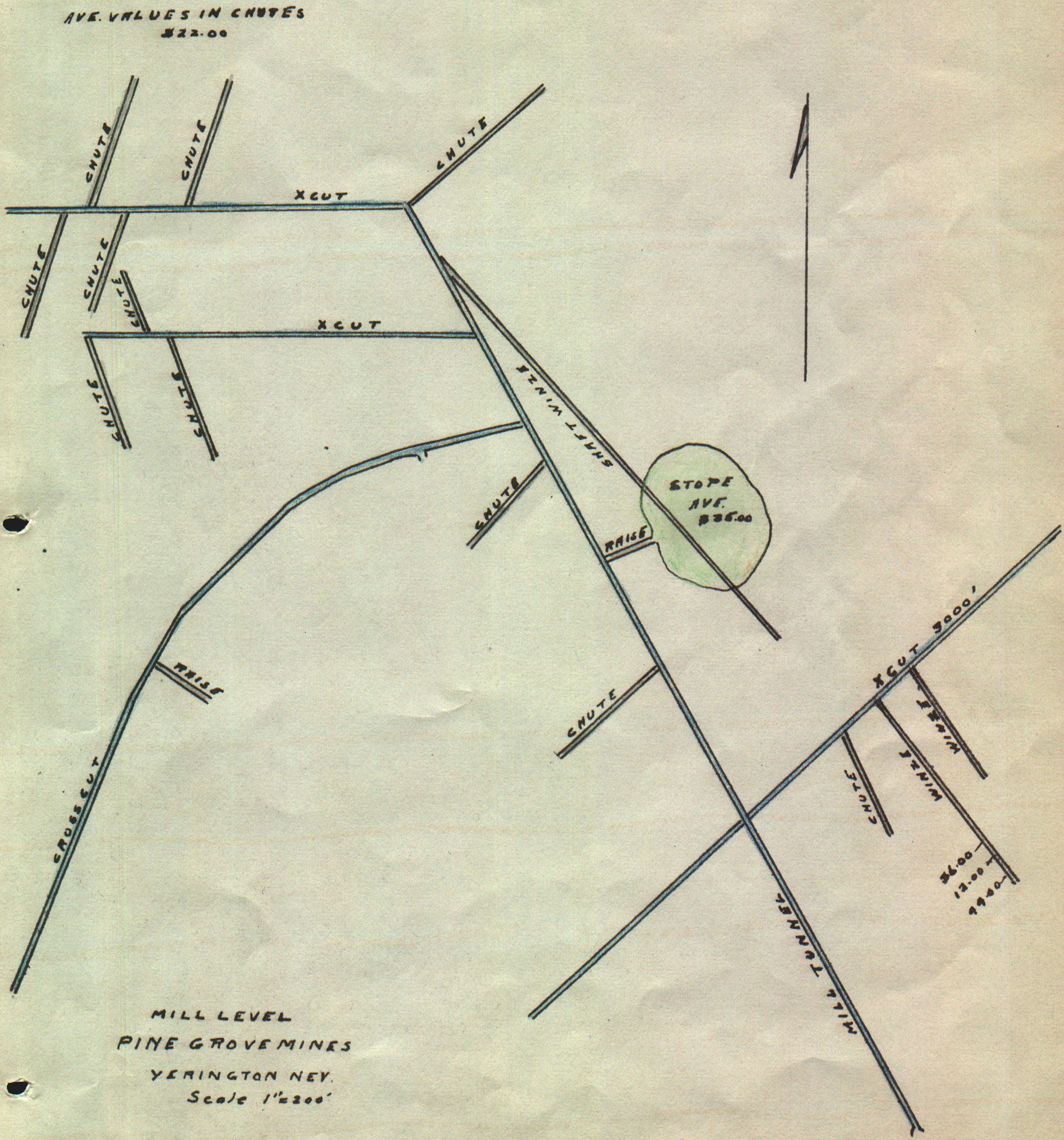
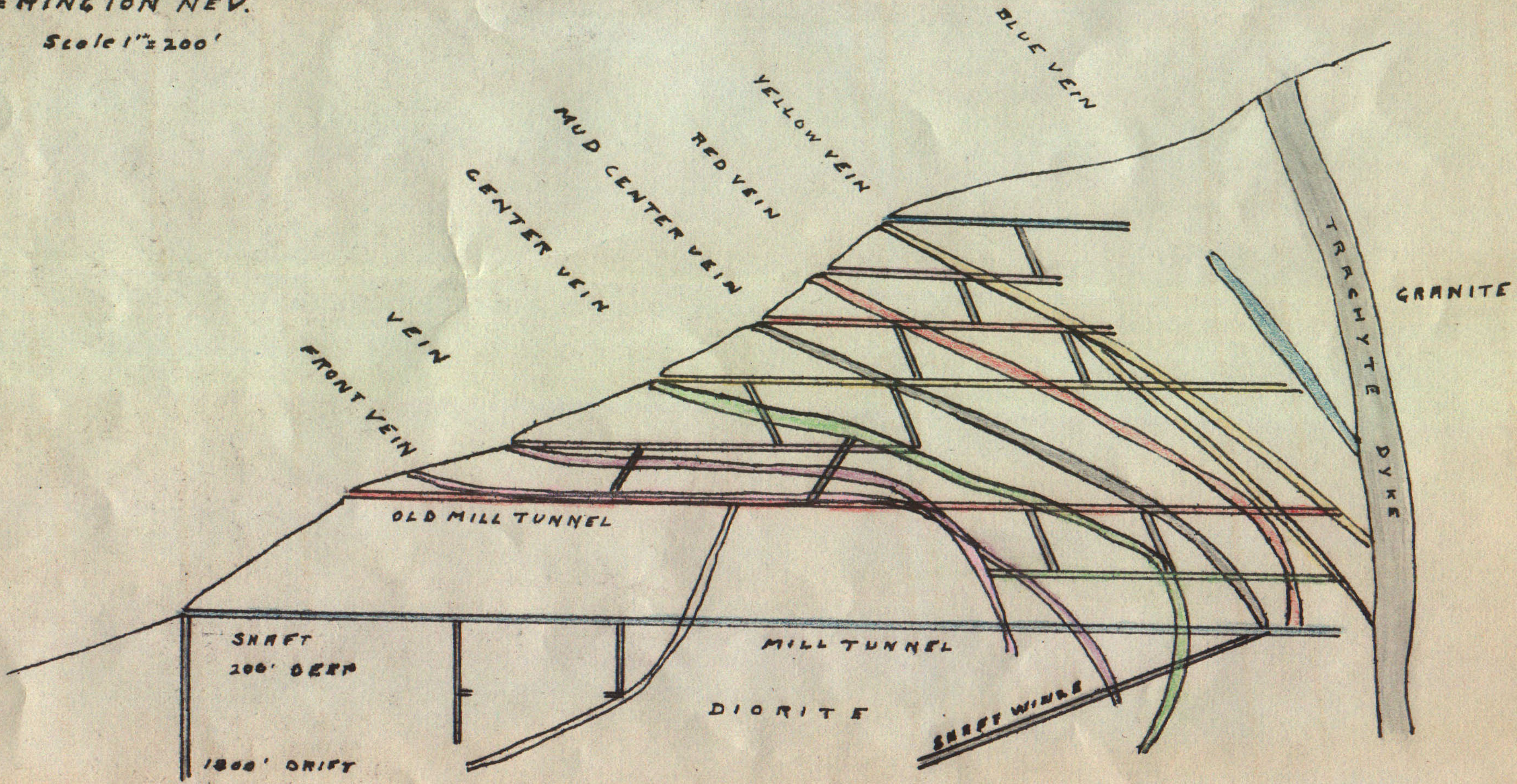
PLATE IV



MILL LEVEL
PINE GROVE MINES
YERINGTON NEV.
Scale 1"=200'

PLATE I

SECTION
PINE GROVE MINES
YEMINGTON NEV.
Scale 1" = 200'



Item (182)

PINE GROVE MINING PROPERTY
By
HUBERT C. TABER, E.M.

LOCATION:

Pine Grove Mining District, Mineral County, Nevada, 25 miles south of Yerrington, the local railroad station, trading and shipping point. Good roads and private telephone line to the mine.

PROPERTIES:

Eight patented claims, containing approximately 160 acres; three mining claims held by location containing about 60 acres; one tunnel site located for drainage and transportation; permit from United States Government on four sections (2560 acres) in timber reserve, for water timber and fuel for mining purposes; Government permit for right of way 20 feet side and two miles long for water line connecting springs with mines.

GEOLOGY:

The vein system on this property is a part of the general vein system as classified by Peel, which includes all of the well known mining camps of Western Nevada, such as Virginia City, Carson City, Mound House, Wabusca or Mason Valley Group which includes the Pine Grave Distract at the southern end; thence on to Mina, Tonopah, Divide and Gold Field, also the smaller camps in the southern part of Nevada; thence to Oatman, Arizona and the gold camps of Southern Arizona; thence on through Old Mexico.

While the mine is located in a strictly gold belt, gold, silver, copper and lead have been deposited along the vein system, depending upon the period of deposition.

The uplifting and faulting of this particular zone is metamorphic Diorite, with intrusionsof Dolomite and dykes of Trachyte Porphyry.

Original vein filling is composed of highly silicious Diorite. Later vein replacement consists of Hemetite Quartz, colcoloring from dark red to pale yellow; the light yellow veins showing a large amount of porphyry Trachyte, Pyrite, Azurite and Mlachite. A highly crystalized quartz along the foot-wall is characteristic of the high grade ores. Sugar quartz, Hornblend and Pyritic Quartz make the vein filling, the Talc and slikenside on either wall.

There are four distinct veins traversing this property, all dipping into the hill at angles of from 12 to 30 degrees, the vertical distance between thm varying from 10 to 60 feet. They continue from 300 to 700 feet where, encountering a large

Trachyte dyke, they nearly unite and dip downward following the contour of the dyke. There are also two blanket veins on the opposite side of the gulch that dip towards, and should unite with these other veins at depth. Three large intrusive Trachyte dykes cross the vein system at right angles, causing three distinct ore shoots on all veins on this property.

ORE AND TREATMENT:

Between the lower mill tunnel and the old mill tunnel there is one ore body, I estimated at 10,000 tons, giving assay values running from $12.00 to $40.00 per ton. A mill run of 100 tons drawn from this shoot gave mill heads of $20.00 per ton. (See assay sheet shown on Plate 3). Other large ore bodies estimated at 20,000 tons are partially developed and can be made available by a very small outlay for upraises and drifting.

The gold content is from 75% to 85% free milling. The values in the production zone vary from $5.00 to $100.00 per ton. The free gold yields readily to amalgamation, the iron pyrites and sulphides to concentration and vanning. Silver content about 1/3 ounces per ton and copper content 1/2 to 1%.

The milling areas, throughout the mine, run from $10.00 to $20.00 per ton, width of veins from 3 to 12 feet with high gradd streaks from 6 inches to 18 inches wide, showing assay values from $60.00 to $100.00 per ton.

MINE DEVELOPMENT:

Estimated underground workings is 14 miles. This includes tunnels, drifts, crosscuts, upraises and winzes. From the best data obtainable, it is estimated that this property has produced over $7,000,000.00 from above the lower tunnel level and the most of it came from above the old mill tunnel. One series of stopes, shown on Plate 2, produced almost $4,000,000.00 above the mill tunnel level, a cross-cut about 150 feet in length from the lower tunnel will reach this ore shoot on that level and should disclose a large body of pay ore. In the old mill tunnel level, 800 feet in length, there are ten drifts and cross-cuts. In these drifts and cross-cuts are more than 15 upraises. Ten of these chutes are well timbered, clear from waste and ready for production. Three winzes used as ore chutes to the lower tunneling are newly timbered and are placed at convenient points for storing ore and tramming from the lower tunnel. Transportation track, air and water lines to all points. There is also a series of levels on the different veins above this tunnel leading to these chutes, for the economical handling of ore. Adjacent to the portal of the old mill tunnel is a shaft rep rted 200 feet deep with a drift leading off for 1,800 feet, following the footwall of the vein from the opposite hill. It is reported that this shaft contains high grade ore. This shaft takes all the drainage from the mine above the lower tunnel level and is now full of water, which is being used a a winter supply for the mill.

Lower mill tunnel 1,000 feet long with 5 main cross-cuts and 11 upraises, newly timbered, track rails just reballasted, water line and main air line installed. Through this tunnel is handled the ores from the old mill tunnel and sub-levels between; the richest body of ore in sight is above this level and below the old mill tunnel. (See plate2).

EQUIPMENT:

Buildings and equipment consist of the following:-

Office and boarding house-10 rooms.
Manager's house and bath, 5 rooms.
Three houses, 4 rooms each.
Two miner's cabins.
One two-story bunk house.
Two barns.
Two garages.
Two Blacksmith shops.
One compressor house.
One mill building, 5 floors, large enough for 200 ton mill, now equipped with 100 ton mill.
Ore bins, 200 tons capacity.
Jaw ore-crusher, 9 x 12.
Conveyor and automatic Feed.
One Agnew Centrepact "Model L" 100 ton mill.
Two Centrepact amalgamators.
Two concentrating tables.
Two 8 inch vanners.
One mill, 10 stamps of 1250 pounds each, not in use.
One mill, 2 stamps of 850 pounds each, not set up.
Two 5½ ft. Huntington mills.
Tow amalgamating tables, 3 plates, each.
One 14 x 20 compound air compressor.
One 40 h. p. semi-Diesel engine.
One 4 x 6 air compressor.
One 6 h. p. Fairbanks-Morse engine.
One 2 h. p. Fairbanks-Morse engine.
Also miscellaneous lot of mining equipment.

I would recommend: an immediate change in the mill to comply with flow sheet shown on Plate 5 of this report; the driving of a cross-cut from lower tunnel, about 150 feet, to get under the big stope shown on Plate 2; making a 30 foot upraise from lower tunnel to reach orein sub-level, assay sheet shown on Plate 3; opening of barn tunnel driven to center vein, which, on account of being caved at portal was never stoped although high-grade ore was exposed; the purchase of a 50 h. p. semi-Diesel engine and the readjustment of power plants. Some changes will be required in the water line. The operation company could then confine themselves to production and milling for some time on practically blocked-out ore that is exposed on two and three sides. If these recommendations are followed, I predict a bright future for this property. Title is perfect and immediate possession can be taken.

Respectfully submitted,
Hubert C. Taber, E. M.

June 20, 1928.

Mr. Richard Day:

The following is a brief report of my examination of the waste dump at Pine Grove:

In the month of July, 1928, I went to the Pine Grove Mine, at Pine Grove, Nevada, for the purpose of determing the value of the dumps.

There is not less than 500,000 tons of ore in the several dumps at this mine.

On one dump, (the one on which the assay laboratory is built), I gave a most thorough examination, both for a true value and the cubical contents.

This dump contains 200,000 tons of ore. One-half of this dump averaged $4.20 per ton and the other half averaged $5.95 per ton of 2000 lbs. One-half ton of ore was taken and crushed to one-quarter inch size and then quartered down to about 20 lbs., each sample being taken in this manner, and the samples were assayed by the Universal Engineering & Equipment Co., Hollywood, Calif. The above samples were not only taken from the surface of this dump but were also taken from pits dug into the dump. These dumps are all on the hillside so that no ore would have to be elevated to the mill, and tailings would flow from the mill by gravity.

I took a great many samples of other dumps and all could be worked by modern methods and pay a fair return on the investment. The lowest value I found was $1.65 per ton in a little dump on the hillside one-half mile from the main workings. I am confident that the whole hill will pay to work as there are rich spots in the mine, (of which I was through about 4000 ft., wide and 250 ft., high.

I do not know how much water may be developed, but in looking down a shaft of which the collar is on a level with the main tunnel of the mine the water stood at 20 ft., below surface.

Power is not a serious matter in the West and Southwest as crude-oil is cheap, and in a great many places power may be generated at the mill for less than the power company's charge for a continuous service; and the first cost of installation of a full diesil-electric plant will not be greater than the cost of building a power line. In over twenty years experience handling low grade ores in localities from the Mississippi River west; this is the greatest opportunity for a sound investment in mining of which I have come in contact, and certainly believe that with proper management and a mill of sufficient capacity that the operators of this mine would meet with success.

Respectfully submitted,

David J. Blackner,

Vallicita, Calif.

Metallurgical Reports

Southwestern Engineering Corporation

Sample No. 2356

Description of Sample:

Two sacks of dump ore were received, already crushed to approximately one-half inch. The contents of the two sacks were mixed together as one sample. The ore composing the sample was partially oxidized character and contained gold as the only important value. A large portion of the gold was observed to be present in the free state. Minerals identified were iron pyrite and iron oxides.

After mixing, the sample was further crushed, again thoroughly misod, sampled and assayed with the gollowing results;

Gold	.22	Ounce per ton.
Silver	.10	" " "

Object of Testing:

The object in testing was to determine the amenability of the ore to the following combination of processes:

(1) Flotation followed by gravity concentration of the flotation tailing.

(2) Amalgamation with flotation treatment of the amalgamation tailing.

(3) Gravity concentration at relatively coarse mesh followed by regrinding and flotation treatment of the gravity tailing.

Testing was limited to the combinations described as it was considered undesirable to resort to cyanidation unless the other metmods failed to give satisfactory results.

Conclusions:

The ore was found readily amenable to a combination of amalgamation and flotation. Fairly good results were also indicated by a combination of gravity concentration and flotation as described under object of Testing. Flotation followed by table concentration of the glotation tailing did not result in the satisfactory recoveries obtained by other combination methods.

In the amalgamation-flotation treatment (Test No. 5) the extraction by amalgamation was unusually high. One of the ef ects of this high extraction was to deplete the flotation feed to such an extent as to result in comparatively low grade flotation concentrates. Actual practice would perhaps result in somewhat higher grade flotation concentrates, particularly if part of the pyrite could be successfully depressed without lowering the recovery materially. Results of the amalgamation-flotation test are given as follows:

Extraction by Amalgamation	78.01	Per cent
Recovery by Flotation	10.00	" "
Total Recovery	88.01	" "

The flotation concentrate assayed 1.20 ounces gold per ton and was 1.57 per cent by weight, or 1.57 tons from 100 tons of ore.

Increase of approximately one percent in the glotation recovery would probably result from retreatment of the middling product.

In event that the flotation concentrate reported above would not return a profit by shipment to a smelter and could not be further incresed in grade as before mentioned, it would be necessary to subject it to treatment locally. In such case cyanidation would be indicated.

Gravity concentration on tables at relatively coarse mesh followed by regrinding the tailing and floating indicated that there would be recovered from 100 tons of Ore:

.86 tons of table concentrate

Assaying	Containing
Gold 9.87 oz. per ton	54.79 per cent of the gold

and .99 ton of flotation concentrate

Assaying	Containing
Gold 3.10 oz. per ton	19.80 per cent of the gold

or, 1.85 tons of combined table-glotation conct.

Assaying	Containing
Gold 6.25 oz/ per ton	74.59 per cent of the gold

The flotation middling resulting from this test was much higher in both assay and per cent of the total gold than in the amalgamation-flotation treatment. It is estimated that the final recovery would be increased approximately 4 per cent by retreatment of the middling, bringing the total recovery up to 78.59 per cent.

Judging from the tailing assays alone, which were .02 oz. gold for the amalgamation-flotation test and .03 oz gold for the gravity concentration-flotation test, the difference in recoveries between the two methods does not appear as great as when the recoveries based upon the calculated heads are considered. It should be noted that in testing an ore in which a large proportion of the gold is present in the free state, it is difficult to prevent loss of some gold particles during laboratory manipulation. Any loss has the effect of lowering the calculated head (which is the gold only actually accounted for) resulting in lower indicated recovery than would be the case than if the gold not accounted for should report in the amalgam or in the concentrates. The calculated head in Test No. 5 was .188 oz. and in Test No. 6. .155 oz as compared with the original head assay of .22 oz.

From the foregoing comments it may be concluded that actual practice would likely result in much closer agreement between recoveries obtained from the two methods. However, even after making due allowance for the possible corrections above mentioned, it is still indicated that the amalgamation-flotation will result in higher recovery than gravity concentration-flotation.

The most practical degree of granding was not definitely established in the preliminary tests conducted. In the amalgamation-flotation combination the ore was reduced to 4.7 per cent plus 100 mesh with 74.3 per cent passing 200 mesh as shown by a sizing test on the flotation feed. The table tailing in this test by tabling and flotation was finally reground to 5.95 per cent plus 100 mesh with 72.55 per cent minus 200 mesh as shown by a sizing test on the flotation tailing. It was indicated from the results of two of the flotation-tabling tests that the coarser grinding than that above quoted could be practiced without deleterious influence upon recovery. Reference is made to test No. 1 and No. 2 in which the grind was varied from 10.5 per cent plus 100 mesh with 65.5 per cent minus 200 mesh in Test No. 1 to 4.7 per cent plus 100 mesh with 74.3 per cent minus 200 in Test No. 2, resulting in a tailing assay of .01 oz. lower with the coarser grinding in Test No. 1.

Recommendations:

In view of the foregoing comments and observations, it is recommended that the ore should receive more exhaustive investigation. The primary object of further investigation would be to establish more definitely certain details not possible to determine in the limited time available before rendering the present report. Also, it is recommended that the possibility of discarding a sand table tailing should be investigated before definitely deciding on the process to be employed. Points which should receive further attention would be the approximate coarseness to which grinding could be carried without lowering the recovery, and whether or not a higher grade flotation concentrate could be produced in connection with amalgamation.

Summary of Preliminary Tests:

A total of six tests were conducted on the sample received. Four tests were by flotation followed by tabling, one test was by amalgamation-flotation and one test was by tabling followed by flotation. The readers' attention is directed to the attached data sheets for the details of the various tests.

The reagent adjustment for the two tests discussed in the present report were as given below.

Amalgamation-flotation Test (No. 5):

	Pounds per ton	Where Added
Potassium Ethyl Xanthate	.15	Rougher
P. E. Oil	.5	"
Sodium Sulphide	1.42	"
Yarmor Pine Oil	.15	"

Tabling-Flotation Test (NO. 6):

P. E. Oil	.5	Ball Mill
Sodium Sulphide	1.42	" "
Patassium ethyl xanthate	.15	Rougher
YarmorPine Oil	.15	"

In each case the rougher concentrate was cleaned once without the addition of more flotation reagents.

For actual operations, particularly when amalgamation is followed by flotation, a five minute conditioning period would be desirable ahead of flotation.

Sizing Tests on the two tests under discussion follow.

Test No. 3 Amalgamation-flotation, Flotation feed:

Mesh	
Plus 48	.0 per cent
Minus 48 plus 65	.07 " "
" 65 " 100	4.00 " "
" 100 " 150	7.00 " "
" 150 " 200	14.00 " "
" 200	74.30 " "

Test No. 5 Tabling-flotation:

Mesh	Table Feed	Sands Reground for Retabling	Flotation Tailing
	All Minus 20 Mesh		
Plus 48		6.0 Per cent	.0 Per cent
Minus 48 Plus 65		11.0 " "	.95 " "
" 65 " 100		13.5 " "	5.00 " "
" 100 " 150		9.0 " "	7.50 " "
2 150 " 200		11.0 " "	14.00 " "
" 200		56.5 " "	72.55 " "

In the amalgamation-flotation test the ore was reduced to the size reported before being submitted to amalgamation in a revolving bottle.

SOUTHWESTERN ENGINEERING CORPORATION

By

Robert Lord:A

7-7-30

SOUTHWESTERN ENGINEERING CORP.

METALLURGICAL TESTING DEPARTMENT

JULY 25, 1930

SAMPLE NO. 2356.

The present report is intended to supplement a report made on the same sample under date of July 5, 1930, entitled, "Preliminary report of tests conducted on a sample of ore received from Mr. William A. Ingoldsby."

A description of the sample was given in the report to which reference has just been made.

OBJECT OF ADDITIONAL TESTS:

The main object of the additional tests was to carry out the suggestions made under "Recommendations" in the preliminary report.

In the work covered by the present report it was desired to determine the following points:

(1) Whether or not coarser grinding could be practiced for amalgamation-flotation.

(2) Whether or not higher grade flotation concentrates could be produced following amalgamation.

(3) Whether or not a sand table tailing could be rejected at -20 mesh of sufficiently low gold content to not warrant its further treatment.

(4) Amenability to amalgamation and tabling at 40 mesh.

CONCLUSIONS:

In the preliminary report to which reference has been made, a combination of amalgamation and flotation was recommended. It is concluded from the results of the tests forming the subject of the present report that a combination of amalgamation and flotation as recommended in the preliminary report is best suited for the economical treatment of ore represented by the sample. Results of additional testing confirmed the conclusions drawn in the preliminary report.

Tests also indicate that there is a tendency for extraction by amalgamation to decrease with a coarser grand as will be noted by the following tabulation which shows amount of gold extracted by amalgamation where the ore was reduced to various degrees of fineness:

Apparent effect of Grind
on amalgamation extraction.

Test Number	Percent plus 100	Percent Minus 200	Percent Extraction
12	20.0	58.0	55.37
7	10.5	65.5	72.21
10	10.5	65.5	67.87
11	4.7	74.3	71.42
13	4.7	74.3	74.42
* 5	4.7	74.3	79.01

Note (*) Taken from preliminary report.

A comparison of the foregoing results indicates that granding to 4.7 percent plus 100 mesh and 74.3 percent minus 200 mesh seems about right for best amalgamtion extraction, although it is the writer's opinion that considerably coarser grinding than the figures just mentioned will be possible in actual operation due mainly to the effect of classification.

The ore appears rather spotted as regards free gold content, and it si believed this somewhat spotted condition causes part of the variation noted in amalgamtion extraction. For example, in Test #12 two separate charges of ore, each consisting of 700 grams, were amalgamated. The gold in the amalgams from each charge was determined as follows:

	Milligrams of gold	Ounces gold per ton ore
Charge #1	3.00	.125
" #2	1.88	.078

Results of Test #12 indicated that higher grade flotation concentrates can be produced following amalgamtion than were reported in the preliminary report. This was accomplished by reducing the flotation reagents and time of flotation contact. Raising the grade of the flotation concentrate was accompanied by a decrease in the actual gold recovery in the concentrate. Retreatment of the middling product should raise the actual recovery in the concentrate from 7.76% to between 10 and 12%, making the total amalgamation-flotation recovery above 85%.

Highest recovery was obtained in Test #11 which was a duplication of Test #5 covered by the preliminary report. Without considering the flotation middling, the actual recovery by amalgamation and in the flotation concentrate in this test was 35.59%. The flotation concentrate assayed 1.60 ozs. gold per ton. In most of the additional tests the calculated heads were low, which probably is due, as pointed out in the preliminary report, to loss of free gold in laboratory manipulation. As previously noted, reporting the unaccounted for gold in the amalgam or concentrates would add materially to the recoveries.

Rejecting a sand table tailing at minus 20 mesh isnot considered desirable as the rejected sand assayed .05 oz. or more in gold.

Amalgamation at approximately 40 mesh followed by tabling is not considered as suitable a combination as amalgamation at finer mesh following by flotation. The main reasons for this conclusion are that the amalgamation recovery was comparatively low and the final tailing assay was high, making a low total recovery. It is believed that fine gold escaping amalgamation will be recovered better by flotation than by table concentration.

SUMMARY OF TESTING:

A total of seven tests were involved in the additional work covered by this report. Reference numbers are as follows:

Test No.		Combination process employed
7, 10, 11, 13	-	Amalgamation-flotation
12	-	Amalgamation-tabling
8, 9	-	Tabling-flotation

Results obtained from these various combinations are compared below:

AMALGAMATION-FLOTATION:

Test No.	Calculated Head	Tailing Assay ox.gold	Wt.% of conct.	Conct. assay ox.gold	Amal. Extraction	Total Recovery	Grinding 100 mesh	Grinding 200 mesh
7	.33	.05	1.36	2.51	72.21	82.57	10.50	65.50
10	.162	.02	1.72	1.56	67.87	84.43	10.50	65.50
11	.177	.02	1.57	1.60	71.42	85.59	4.70	74.30
13	.175	.02	.43	3.16	74.42	82.18	4.70	74.30

AMALGAMATION-TABLING

Test No	Calculated Head	Tailing Assay oz.gold	Wt.% of conct.	Conct. assay oz.gold	Amal. Extraction	Total Recovery	Grinding 100 mesh	200 mesh
12	.184	.06	1.50	1.52	55.37	67.79	20.0	58.0

TABLING-FLOTATION

8	.205	.038	1.80	9.40	-----	82.50	5.00	81.00
9	.198	.043	1.40	10.91	-----	77.25	9.00	76.00

The above sizing tests were made on the feed to flotation. In test #8 and #9, minus 20 mesh table sand tailings were rejected. Results of a rough sizing test made on the rejected sands are as follows:

Mesh	Test No. 8	Test No. 9
Minus 20 plus 100	73.5	79.0
" 200	5.5	6.0

Flotation Reagents:

Flotation reagents used in Test #13 in which the higher grade flotation concentrate was produced following amalgamation are listed below:

	Pounds per ton	Where added
Potassium ethyl xanthate	.10	Rougher
Reagent #208	.03	"
Yarmor pine oil	.10	"

Amalgamation Tests:

All amalgamation tests were made by grinding the ore to the fineness shown and then rolling in bottles with mercury for one hour. The mercury and amalgam were then panned out. Amalgomation extraction is based on the actual gold contained in the amalgam.

Particularly if heavy oils or sodium sulphide is to be employed, a five minute condition period ahead of flotation seems desirable.

Southwestern Engineering Corporation,

By Robert Lord.

Report by

H. B. Menardi

August 18, 1930

Mr. William A. Ingoldsby,
130 S. Broadway,
Los Angeles, Calif.

Dear Sir:

Complying with your request just received over the phone and also confirming our recent conversation, we are pleased to submit herewith a list of assaying showing results of samples from the large dump at Pine Grove together with our opinion regarding average values which you might expect to find in the upper dumps from which we did not take any samples.

List of Dump Samples:

1.	.175 oz. old per ton
2.	.03
3.	.06
4.	.04
5.	.27
6.	.17
7.	.12
8.	.08
9.	.04
10.	.07
11.	.04
12.	.07

Average - .097 = $1.94

A sample of old mill tailings which are underneath a portion of this ore dump gave returns of $7.80 per ton. If there are 5,000 tons of this tailing the calculated average of the entire dump would be approximately $2.25 per ton.

It is our opinion that the upper dump will average higher than this main dump, included in this sampling. These upper dumps came from workings which followed the higher grade portions of the ore some near the durface while the lower ore dump includes a considerable portion of ore from cross-cuts which cut entirely through the zone.

The fact that shipments have been made from screenings from the upper dumps also indicates higher grade ore. These upper dumps would be somewhat more expensive to load and transport to the mill but we believe this increased expense would be more than off set by the higher value. We would expect that sampling and measuring would indicate that there is at least an additional 50,000 tons of ore just as good or better than the 100,000 tons or ore in the main dump which is averaged at $2.25. Metallurgical tests indicate a recovery of 90% consequently the operating revenue can be assumed at $2.00 per ton. We would certainly recommend however, that additional samples be taken from the upper dumps in order that this estimate be based upon observed data rather than upon an estimate. We would also suggest that further metallurgical tests be made in order to determine the treatment of the concentrate and also to make sure that the old mill tailings respond to the same treatment as has been indicated for the dump.

In making a preliminary estimate of operating costs we are assuming first class equipment and substantial construction together with efficient and economical operating mangement. We are assuming that power will cost 1¢ per kilowat hour. On this basis we believe that direct milling costs of 75¢ per ton will be obtained and that the cost of transporting the dumps to the mill and indirect operating expense will not exceed 50¢ per ton or a total of $1.25 per ton. This leaves an operating profit of 75¢ per ton or $112,500.00 for the 150,000 tons of dump ore.

If additional sampling and detailed estimates confirm these opinions, this property offies a very attractive venture in that returns from the dump ore will more than pay for the cost of the mill installation. We also believe that is is quite possible you will find a considerable tonnage in the mine which would be technically classed as proven or blocked out ore. This ore would undoubtedly be considerably higher grade than the dump ores and constitute a considerable source of revenue.

We wish to suggest that you sample the upper dumps and ore exposed in the mine very thoroughly and completely before making any definite plans or commitments in connection with mill construction.

Yours very truly

SOUGHWESTERN ENGINEERING CORP.

Signed:

H. B. Menardi

Mr. Geo. A. Taylor, 8-1-30

Assays by Southestern Engineering Corporation laboratory are as follows:

Sample Number	Ounces Gold Per ton	Value per Ton
1	.06	$1.20
2	.19	3.80
3	.03	.60
4	.09	1.80
5	.03	.60
6	.06	1.20
7	.03	.60
8	.02	.40
9	.38	7.60
10	.11	2.20
11	.08	1.60
Avg.	.0982	$1.964

Samples Nos. 1 to 4 inclusive were taken from the top surface of the main dump. The attached sketch shows the location of the small pits included in each sample. About 3 to 6" of the surface was removed and then one small shovel ful taken from the bottom of each pit. The samples weighed approximately 25 pounds each.

Samples Nos. 5 to 8 inclusive were taken from the slopes of the dump as indicated. In these pits the surface of the dump was shoveled away to a depth of from 6 to 12" and the sample taken from the bottom of the pit.

Sample Nos. 9 is a tailing sample taken from one pit under the edge of the large mine dump. This material is tailing from one of the early mill operations and is probably fairly high grade, as the material panned very well. There is an indeterminate amount of this material under the mine dump which would be recovered along with the mine dump. None of this tailing material was included in the mine dump samples.

Sample No. 10 represents tailing across the draw from the large mine dump. This sample was taken from shallow pits in the same manner as the mine dump samples.

Sample No. 11 is one general sample taken from pits along the slope of the large mine dump from the lower mill tunnel and is indicative of the general average of this dump.

In order to estimate the tonnage in the large mine dump "squared up" measurements of the width, length and depth of the ore were made. The length of the dump is 500 feet. The average depth at the lower end, not taking into consideration a gullpy of indeterminate depth, is 60 feet. The width at the upper end of the dump was taken as 8 feet, therefore the average depth, assuming that the ground slope underneath the dump is uniform, is 34 feet. The dump is 118 feet wide at the upper end and 125 feet plus at the lower end. The average width was taken as 125 feet. The weight of the ore is assumed at 100 pounds per cubic foot, therefore the calculated tonnage of the dump is 107,000 tons.

To determine the percentage of fines in the dump a rough screen analysis was made on samples 1, 3, 5, and 7. On the basis of these screen analyses it is estim ted that 57.7 per cent of the dump will pass a .371" opening (Tyler standard screen). One sample, No. 11, was taken from the dump of the lower mill tunnel. This dump is darker in color than the large dump and is not so thoroughly weathered, but visual inspection showed sulphides throughout.

A number of smaller dumps from upper workings are available and it is probable that these dumps will be higher in value than the large dump. The loading and transportation of these dumps to the mill offers no particular problem. These dumps should all be carefully sampled and measured in order to arrive at a definite estimate of tonnage and values.

Hurried trips were made through the lower mill level and the old mill level, which is approximately 75 feet above the lower mill tunnel. In each of these levels a main-cut runs through the ore zone at right angles to the strike. A series

Mr. Geo. A. Taylor,

of parallel veins existed which has been drifted and raised on. The wall rock between these veins was all mineralized and showed sulphides. It is quite possible that the entire ore zone might carry sufficient values for milling on a large scale, but of course this could only be definitely determined by thorough sampling. The ground is quite soft and most of the openings require timbering.

It is recommended that in addition to the dump sampling recommended above, the underground workings be thoroughly sampled with the view of determining whether the wall rock between the veins is of sufficient value to consider mining the ore zone as a whole on a large scale.

Very truly yours,

Signed:
H. B. Menardi

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Ed Lawrence ——————

(W & P)

veinlets in shear zone, in "tactite zone"
volcanics contain molybdenite,
quartz, secondary copper

U.S.B.M. data — 1957

W & P Mining Corp
Last Chance mine
35 mi S. of Yerington
500 tons mined of 1.50 %
MoS_2 — no ore milled