275
Item 15

-2340 0008-

TOULON MILL - EXHIBIT NO. 2
J. H. Wren & Company
George Crerar Mill Reports
Mill Inventory
Toulon Tailings Assays

TREATMENT OF COMPLEX AND REFRACTORY ME[illegible]

2340 0008

275

Item 15

GEORGE CRERAR

MINERAL DRESSING ENGINEER

BRICELAND, CALIFORNIA

July 21, 1955

Nev-Tah Oil and Mining Company
Gazette Building
Reno, Nevada

Attention: Mr. A.L. Damon, General Manager

Dear Sir:

The following are the screen test results on Stormy Day ore treated at the Toulon Mill of the Wolfram Company. Samples taken on the last two days of operation in January 1955, but assumed to be representative of the 1257.55 wet tons of ore treated in the above mill, as follows:

Stormy Day Mill Heads

		% WO_3
-10 plus 14 mesh	15.2%	0.25
-14 " 20 "	18.9%	0.28
-20 " 28 "	13.6%	0.41
-28 " 35 "	9.2%	0.56
-35 " 48 "	8.5%	0.63
-48 " 65 "	8.5%	0.63
-65 " 100 "	5.9%	0.62
-100 " 150 "	5.9%	0.62
-150	14.5%	0.62

Equated average on the basis of the above results - 0.4675% WO_3.

A head sample of above ore assayed 0.47% WO_3.

A special sample of Stormy Day ore marked "Stope A" yielded the following screen test results. The sample was crushed thru 10 mesh by laboratory equipment at Nevada Mass. It was screened oftener than would be done in standard mill practice.

		% WO_3
-10 plus 14 mesh	13.1%	0.22%
-14 " 20 "	17.9%	0.29%
-20 " 28 "	14.5%	0.50
-28 " 35 "	10.2%	0.66
-35 " 48 "	9.6%	0.70
-48 " 65 "	9.2%	0.56
-65 " 100 "	6.4%	0.53
-100 " 150 "	6.0%	0.47
-150	13.0%	[illegible]

Deister Hindered Settling Hydraulic Sizer. The deslimed and sized spigot sands would go to tables and the slime overflow to a thickener. A diaphragm pump would deliver the thickened product to the flotation circuit. The thickener overflow plus the table tailing would go to waste dumps, together with the flotation tailing.

After hydraulic sizing the amount of table middling would be very small compared to the middling result from an unsized table feed and would be returned to the rod mill.

Nevada Massachusetts laboratory reports 0.7% total sulfur and 0.02% water soluble sulfur in Stormy Day mill feed. The sulfide sulfut combined with iron would mean a 1.3% Pyrite or 1.83% Pyrrhotite content. Not more than half of the above sulfur would go into the flotation feed and this might be depressed in the scheelite flotation step and go out in the tailing. Any residue could be burned in drying the flotation concentrate. That part of the sulfides recovered in the scheelite gravity concentrate would be partly burned in the drying step, and eliminated by the magnetic separator.

The gravity concentrate could, of course, be passed thru a simple flotation step which would eliminate any contained fine sulfides.

One advantage gained by hydraulic sizing would be the elimination of apatite (phosphorus) and fluorite from gravity concentrate. Scheelite has a specific gravity of 6.0; apatite and fluorite each have a Sp. Gr. of 3.2, a spread of 2.8 which is enough to assure those minerals landing in middling or tailing.

There never was a rejection or penalty imposed on account of phosphorus in Nevada Massachusetts gravity concentrate although we collected as much as 2.0% P in our flotation concentrate. Apatite and fluorite have a common calcium ion with scheelite and, therefore, are floated with the same reagents. Acid treatment eliminates the apatite but not the fluorite.

For acid treating a flotation concentrate at Toulon, I suggest a duplicate of the first acid treatment plant we had at Nevada Massachusetts as it was cheap to install and definitely more efficient than any of the modern plants I have seen. There was practically no soluble loss of WO_3 in the acid digestion step nor dust loss in the final drying step. Flotation concentrate was accumulated in bags and treated by two men in two days at the end of each month, then mixed into the gravity concentrate which it equalled in grade and purity.

I suggest the addition of two half-sized Deister Concentrator Co. slime tables to the flotation step. One table would retreat the flotation cleaner tailing and recover a large part of the contained scheelite as a gravity concentrate. This would reduce flotation tailing loss.

The other table would treat the scheelite cleaner concentrate. The operator would have a continuous check on his flotation efficiency. He would learn where to hold his depressants, such as quebracho, so as to get optimum grade of concentrate and recovery. The superintendent or manager could judge the efficiency of the flotation operation with one glance at the table. The tailing from this table could be returned to the conditioner at the head of the rougher flotation circuit and the concentrate would be acid treated. The flotation table tailing could also be accumulated, dried and sold to a refinery using high temperature steam and pressure in an autoclave charged with a solution of soda ash and a product such as the above but containing less than 10% WO_3.

Automatic density and automatic pH (alkalinity or acidity) control should be applied to the flotation feed in the flotation conditioner.

The concentrating tables could be worked over so as to yield a higher grade concentrate and recovery. Since there are no "platos" in the present Deister tables, the longest riffle should be slanted up about 2" from a line parallel to the direction of the head motion throw and all the other riffles laid parallel and at least 2 or 3 inches apart. The concentrate discharge end of the deck should be about 3/8" above the head motion end of the desk. That means the whole understructure including the head motion will have to be slanted up.

I suggest leaving one table as is and inserting a Weinig Wedge with a 5/16" lift under the deck covering. The Weinig Wedge has been used with definite advantage on Wilfley tables. Mine & Smelter Supply Company, Denver, Colorado, supply them.

If we assume the 1257.55 tons of Stormy Day ore contained 5% moisture there would be 62.87 tons of water and 1194.68 tons of dry ore. 1194.68 tons of dry ore at 0.47% WO_3 would contain 561.5 units of WO_3.

11915 lbs. of dry concentrate equals 5.9575 tons which @ 70% WO_3 would contain 417.0 units of WO_3.

561.5 - 417 - 144.5 units in the tailing
1194.68 - 5.9575 - weight of tailing - 1188.72 tons
144.5/1188.72 - 0.122% WO_3 in tailing or by
formula, 74.1% extraction.

A rough estimate of cost and time to bring the present Toulon plant up to date would be $20,000.00 and 4 months time, starting now.

Respectfully submitted,

George Crerar.

TREATMENT OF COMPLEX AND REFRACTORY METALLIC AND NON-METALLIC ORES & PRODUCTS

GEORGE CRERAR

MINERAL DRESSING ENGINEER

RICHLAND, CALIFORNIA

July 21, 1955

Nev-Tah Oil and Mining Company
Gazette Building
Reno, Nevada

Attention: Mr. A.L. Damon, General Manager

Dear Sir:

Subject: Report on Toulon Operation

Results of treating Stormy Day ore to date are as follows:

1257.55 wet tons of Stormy Day ore were milled during the month of January, 1955, but divided into two lots which assayed as follows:

#1 Heads	= 0.467% WO_3
#2 "	= 0.490% "
Equated average	= 0.47% "
810 wet tons milled during April Heads	= 0.44% "

The #2 lot milled in January averaged considerably higher in sulfide minerals and garnet than #1, and the April lot contained higher percentages of both sulfide minerals and garnet.

In the different scheelite ores I have seen, the scheelite in the high garnet and sulfide bearing ores is generally in small to minute crystals and requires finer grinding to liberate whereas scheelite in quartz veins and deposits predominating in epidote, wollastonite pyroxene, etc., and silicious pegmatites, is generally in coarser crystals.

Deducting 3.0% for moisture in above lot of Stormy Day ore leaves 1219.83 tons of dry ore which contained 1219.83 x 0.47 = 5.73 tons of WO_3.

Two lots of concentrate recovered from the January run weighed 11,915#, assaying an equated average of 71.7% WO_3, which amounted to (11,915/2,000) 0.717 = 4.27 tons of WO_3. 4.27/5.73 = 74.5% extraction of WO_3, which I consider a very good recovery on a low grade, high sulfide and garnet bearing ore.

The April mill run of 810 tons of wet ore, less 3% for moisture, left an estimated 785.8 dry tons which contained 0.46% WO_3 or 3.61 tons of WO_3. 7604# of concentrate which assayed 67.84% WO_3 = 2.58 tons of WO_3. 2.58/3.61 = 71.5% recovery on this lot of ore.

The reason for the lower recovery and grade of concentrates in the April run is the finer scheelite and heavier gangue.

Comparing Getchell and Toulon Economics on Stormy Day Ore:

Getchell

810 tons less 3.0% moisture = 810 - 24.3 = 785.7 tons dry ore; @ 80% of assay = 0.46 x 0.8 = 0.368 units per ton.

@ $47.00 per unit = $47.00 x 0.368 x 785.7 =		$13,592.60
Less treatment 810 tons @ $6.00 =	$4,860.00	
Freight Est. $5.00 =810T @$5.00 =	4,050.00	8,910.00
Net Mill Returns	=	$ 4,682.60

Toulon

785.7 dry tons @ 71.7% extraction = 0.715 x 0.46 x 810 = 266.41 Units @ $62 ($63 - $1 Phos. Penalty)	=	$16,517.42
Less treatment @$8.00 = 810x$8.00=	$6,480.00	
" freight @$3.50 = 810x$3.50=	2,835.00	9,315.00
Net Mill Returns		$ 7,202.42
Deducting Getchell Net Mill Returns		$ 4,482.60
		$ 2,619.82 in

favor of Stormy Day shipping to Toulon.

Crushing and Grinding at Toulon:

I estimate one 3'x8' Marcy rod mill would have a capacity of approximately 125 tons daily from -1/2" to - 28 mesh if the Gyrasphere thru product should be fed directly to the rod mill. However, screen tests have shown that approximately 50% of the Gyrasphere thru product is already minus 28 mesh; therefore, if the Gyrasphere screen oversize only should be fed to the 3'x8' rod mill, the total mill capacity would be approximately 200 tons per 24 hour day or 125 tons in 2 - 8 hour shifts.

On the basis of operating the mill with two shifts and treating 125-130 tons, there should be approximately 50 tons of -200 mesh solids in the hydraulic sizer overflow which would constitute feed for the flotation step.

Deducting 50 tons of flotation feed from the 130 tons would leave about 80 tons of sands to be sized and distributed between 5 or 7 tables, which would be a moderate load for the tables.

The Deister Launder Type Hydraulic Sizer installation will consume 50 - 60 Gpm of water, as the ore here is heavier than normal ore and each of the 5 sorting bowls should be supplied with a flow meter in the hydraulic water pipes -- the control valve to be in easy reach of the operator on the table floor. This may be done by running the main along the floor and verticals up to the sizer bowls with the floats in the graduated glass tubes in front of the operator. The graduations would be in gallons per minute from 5 Gpm to 20 Gpm. With this set-up, there is no danger of a competent operator turning on too much water or too little. The table will be a constant indicator of the right amount of hydraulic water.

The Deister hydraulic sizer is equipped with a disc spigot discharge valve with graduated openings. This makes it possible to change the spigot openings from the table floor by means of an extension valve handle. In the event of an increase in any spigot product that threatens to plug the spigot the operator turns the disc to a larger spigot opening, or if slime appears on the table the disc is turned to a smaller opening.

Regarding flow meters, it is my belief that flow meters on table wash water lines would very soon pay for themselves in higher recoveries. There is no question but that there is an optimum flow of wash water for each table treating a sized product. An uncontrolled wash water generally means more scheelite in the muddling.

The spigot products from the hydraulic sizer will be of comparatively high density and water will have to be added to the table feed. The amount of water to be added to the spigot products will vary; for example, a -20 mesh table feed should carry 2-1/2 - 1 and finer feeds about 3 - 1. Wash water should be 8 - 12 Gpm for 20M feed and about 7 Gpm for -60M.

Since gravity concentration yields the highest grade and definitely the cheapest tungsten mineral concentrates, effort should be made to make the operation as near precision as possible. Flotation, in my opinion, should be considered as a scavenging step.

There is a spread of 2.8 in specific gravity between scheelite and flourite and apatite, and a spread of 2.5 between scheelite and garnet. With efficient classification and tabling nearly all of the above three minerals should go into the tailing.

Garnet ending up in primary table concentrate and middling is a source of loss as middling and magnetic separator discard go to regrind, and that means that part of the carried scheelite is certain to be ground too fine to be recovered even by flotation. Therefore, the higher grade concentrate and the definitely lesser amount of middling made from a hydraulically sized table feed, the lower the overall tailing loss will be.

Hydraulic sizing is definitely improved by eliminating the slimes and passing the sands only thru the hydraulic sizer. In discussing this matter with Mr. Kellogg Krebs, I learn we can get a 4" Krebs cyclone that will serve the Toulon operation (at present tonnage) for $750.00, which is $250.00 less than the regular price. Delivery could not be made, however, before the end of July.

By eliminating the slime ahead of sizing, it is probable that the tailing from at least 3 of the tables could be sent directly to waste.

If the cyclone is installed, the cyclone overflow will go directly to the thickener to constitute the flotation feed. The hydraulic sizer overflow - which will carry a material amount of fine solids - could be pumped up to serve a pulp diluting water ahead of screening and as wash water on the screen.

The 4" Krebs cyclone will take 5 tons per hour and water up to 60 Gpm. If the 5 t.p.h. should go thru the screen at a density of 3.1, the water would amount to 60 Gpm. The hydraulic sizer could use 60 Gpm, but part of this water would go to the tables thru the sizer spigot. The pump taking the classifier overflow would be equipped with a branched discharge. One branch would end in a float valve in cyclone feed pump and the other branch would carry the surplus water to pulp dilution ahead of grinding and screening.

There should be flowmeters in this line and in any fresh water line delivering water to this circuit.

After the installation of a rod mill and one set of rolls, the hourly tonnage could go up to 8 or 9 tons per hour which would probably require the addition of another 4" cyclone.

The cyclone overflow would carry but little scheelite that might be recovered on a table; in this respect it would differ materially from the overflow of a rake or spiral classifier. More very fine scheelite would be recovered in the gravity concentrates.

Jigging:

There is a Richards Pulsator Jig on the top floor of the Toulon mill. I believe that jig can be fixed up to deliver a nearly finished hutch concentrate. A fine jig screen, 14 mesh, would have to be installed and a bed of 10 or 12 mesh shot one inch or possibly more on top of the screen. The screen would have to be divided into compartments, say 2" apart, to keep the bed from shifting. There is no suction on this jig except when the hutch is opened to discharge concentrate.

If the #1 set of rolls should be maintained in the circuit, the jig should do a nice job on the thru product. The pulsator tailing could constitute the feed to the screen, which could be 35 mesh instead of 28, and the oversize from the screen would be the rod mill one pass feed.

The 35 mesh undersize together with the rod mill thru product would be the feed for the hydraulic sizer. The sizer overflow, which should be 95% - 200 mesh, would go to a thickener to become flotation feed.

The flotation steps are outlined in the submitted flow sheet. There is, however, one step that was omitted in the flow sheet and that would consist of automatic reagent controls.

An automatic density controller would be the first step in automatic reagent feed control. This step will require further study and could come later if the life of the operation warrants.

Flotation:

Ions of elements exposed on the surfaces of the crystal latices of the minerals combine with certain reagents used in flotation practice to form water repellent or water avid surfaces. The rate at which these reactions take place is related to the concentration and temperature of the pulp, which is the main reason for constant density control.

Also, certain reactions are complete or incomplete varying with the degree of alkalinity or acidity of the pulp, which is the reason for automatic pH control. At pH 9.5, using soda ash as the source of alkalinity, the highest amount of scheelite would be recovered by flotation but it would probably be low grade - around 25 or 30% WO_3; however, if you increased the alkalinity up to 11.5 pH, the recovery of scheelite would be very low, if any. If you raised the alkalinity from 9.5 - 10.5, the degree of alkalinity would be 10 times what it was at 9.5. It is practically impossible to hold the degree of alkalinity at any fixed point manually. That is the reason for automatic pH control. There must be a point where recovery and cost lines cross -- a point where tailing losses and reagent costs would offset the gain from a higher grade product.

Since apatite and flourite have the same calcium ion as scheelite, they tend to float with the scheelite. Apatite is a phosphorus bearing mineral and therefore must be removed by digesting the flotation concentrates with diluted hydrochloric acid. Apatite is soluble in a dilute solution of hydrochloric acid of pH 2.0 but flourite is not. Sulfuric or nitric acids must be used if it is necessary to remove flourite.

Acid Treatment of Flotation Concentrate:

Equipment:

1 Digestor Tank, wood stave 2-1/2" thick, kiln dried. A good dimension for a small operation would be 4' or 5' inside diameter x 5' or 6' deep. Acid proof the digestor by applying 2 or 3 coats of a comparatively soft asphaltum that can be supplied by Standard Oil Company. In starting the acid proofing, go over the inside of the tank with a blow torch ahead of applying the hot asphaltum. After the first coat has set, repeat with blow torch or electric heater to warm the inside of the tank before applying the second and third coats.

A heavy cast iron 3" centrifugal pump will serve as an agitation. The suction will consist of a piece of 4" pipe passed thru a tight hole or gland arrangement in the lower side of the tank, 12" or 15" above the bottom. After a 4" ell is screwed onto the end of the 4" suction pipe, the open end of the ell should be centered on the center of the bottom. A nipple on the end of this ell should extend down to within 2" of the bottom. The ell should be on a loose thread on the suction pipe so that the nipple can be turned up or down by a chain or rod attached to the nipple and extending above the top of the tank.

A 3" discharge pipe from the pump connects into a 3" header that extends across the top center of the tank. From the header, 2 - 2" pipes terminating in street ells extend to the bottom of the tank. The street ells rest on the bottom and point in opposite directions so that when the pump is operating, the contents of the tank go into a swirling mass. Tees are used instead of ells in the header pipe connections. A 3" valve is connected into the pump discharge column and 2 - 2" valves are connected into the drop agitator pipes. A 2" nipple, valve and hose connection is connected into one end of the header so that the finished charge can be pumped to the filter tank, or water or compressed air can be used to clear either the suction or agitator pipes.

Filter Tank:

The filter tank can be 6, 7, or 8 feet in diameter x 2-1/2' deep inside. Staves and bottom can be of 3" lumber treated with asphaltum of the same grade as that used on the digestor and applied in the same way but on both inside and outside of tank and bottom, as this tank must stand a vacuum.

The vacuum can be made by a 3, 4, or 5 inch triplex pump connected into the bottom of the tank by a 2" suction pipe and nipple into 1/2 of a flange union bolted onto the bottom, plus heavy cast washers on the inside bottom of the tank.

The filter tile are about 12" square x 1-1/2" thick and made to the specifications of the tank dimensions, so that the outside tile conform to the arc of the inside circumference.

Hard burned common or fire brick are set on edge on the bottom of the tank so that the top edge of the brick completely closes the bottom of the joints between the tile. Around the inside circle of the tank, the edges of the tile may have to rest on a 1/2" strip of lumber to hold the hot asphaltum seal. Hot asphaltum or sulfur is used to seal between the tile.

Remember this: Do not charge the pulp into the filtering tank until water that has been flowed into the vacuum space under the tile has risen up thru the tile and stands at least an inch above the tile. This procedure has prevented blinding of the pores of the tile.

Operating the Acid Treatment Step:

Assuming the digestor tank is 5' or 6' deep, fill it half full of water, then start the pump and charge in the weighed quantity of concentrate.

Follow by adding acid carefully. If the concentrate is low grade, it will be high in calcite and the froth can rise so fast that it will boil over. Acid should be added with ALL DOORS AND WINDOWS OPEN as the gas is heavy and 10 or 12% can be deadly.

A small pump, pumping a steady stream of pulp up into a 500 ml beaker in which the electrodes of a pH meter have been immersed, will give a continuous reading of the acidity as the calcite is dissolved. When the pH stands at 4.0, the acid may be shut off as all the calcite is in solution as calcium chloride.

The beaker is so arranged that the overflow runs back into the tank.

Turn pump suction nipple up and allow contents of tank to settle clear. The addition of a small quantity of a 2% solution of Dow Chemical Company Separam #2610 helps to speed up the settling. Add water to tank half full, agitate, settle and decant. Repeat this step a third time.

To the concentrate in the digestor tank add an equal volume of water, turn suction nipple down and start agitation again. Add strong muriatic acid - 32-34% hydrochloric acid (HCl) until the pH reads 2.0, and hold it at 2 or down to pH 1.5 until the blue ditungsten pentoxide shows in the circulating pulp. The blue pentoxide indicates the apatite has been broken down and the phosphorus is in solution. Do not allow the pH value to drop to a pH 2.5, as phosphoric acid will begin to recombine with lime and other elements and reprecipitate again.

On the pH meter scale #7 is neutral - neither acid nor alkaline. On adding an acid as the acid concentration increases, the numbers decrease. For example, pH 2.0 is 10 x the acid strength of pH 3.0. On adding alkalis (bases) the numbers increase with the addition of a base; for example, pH 9.0 is 10 x the alkalinity of pH 8.0.

When the blue ditungsten pentoxide (W_2O_5) shows in the pulp in agitation, add a gallon or two (or three) of commercial Pyridine #2 to the charge and start pumping into the filtering tank thru the hose fastened to the nipple extending beyond the gate or plug valve on the far end of the horizontal part of the pump discharge line.

When the charge has been transferred to the filter tank and the digestor tank is fairly well cleaned out, the charge in the filter tank should be leveled off if necessary. Ordinarily, by proper use of the hose while charging the filter tank, leveling will not be necessary.

The pH value should still be 2.0 or an acid concentration above 2.5 to be sure that no phosphorus reprecipitates. The blue pentoxide solution should be replaced by a yellow-brown Pyridine-tungsten complex. Stop agitator pump and close valve on suction line.

Open the valve on the triplex pump suction line and start the pump. The filtrate can be discarded. As soon as the filter cake in the tank is dry, turn water plus acid to pH 2.0 to a depth of 4" - 6" over the filter cake. Open suction line valve and start pump again.

Repeat with one more (second) acid wash as before.

The third wash will be water alone equal to the thickness of the filter cake and this wash is made for the purpose of replacing the acid residue in the filter cake.

The two acid washes are necessary to prevent the Pyridine-tungsten complex from hydrolyzing and going out with the filtrate.

If a pH meter is not available, or the one that is available happens to be out of order, a solution composed of 7 parts water and 1 part standard muriatic acid will make a 4% hydrochloric acid solution that will have a pH value of approximately 2.0 = 4.0% Sol. HCl.

By doubling the strength of the phosphorus leach acid to 1/4 standard muriatic acid in 3/4 water, 1 hour to 1-1/2 hours agitation will reduce the phosphorus below penalty limits.

After the final wash (the water wash) has been drawn thru the filter cake, the cake is ready for drying and mixing into the gravity concentrate. This can be done by feeding the dried flotation concentrate into the stream of gravity concentrate as it is elevated to the bagging bin.

There are several different ways flotation concentrate can be dried without dusting. One is by use of an outside fired tube of 12" or more diameter with a 4" pipe, sealed at both ends, full length inside to roll with the large tube and flatten wet nodules of concentrate -- substantially the identical set-up we had at the Nevada Mass. Co. pilot mill. That tube was equipped with stacks at each end so as to balance the draft to just enough to carry the water vapor and volatized reagents out. The installation at Nevada Mass. Co. pilot mill was practically dustless. The tube in the tailing mill at Toulon can be converted into a duplicate of the foregoing Nevada Mass. Co. installation.

A steel trough in a half circle 14"-16" diameter x 20' long, equipped with a screw to fit could with the screw be connected to a low V-high amp. current that would raise the temperature tube to the boiling point of water, which could be installed in a close fitting insulated housing and stack at the discharge end. Natural draft would carry the water vapor out as formed. The temperature would not be high enough to cause distortion of the parts and the draft would remove the water saturated atmosphere, so that drying would take place rapidly and without dusting. The outside of the tube could be insulated so as to prevent the waste of heat and the housing dispensed with. A gentle current of air from a fan would speed up the drying.

A wide steel shallow launder heated electrically and driven by a vibrator would be dustless. The same equipment could be heated with oil inside of an insulated housing which would be practically dustless. This is the set-up used to dry light, fluffy molybdenite concentrate at Utah Copper.

Miscellaneous:

Screening:

The present Hummer screen installation is, in my opinion, neither economical nor efficient. Checking back for a year, I find the average life of 588 and 833 screens has been about 10 days -- approximately 1,000 tons each. When the screens are removed because of failure, approximately 30% of the screen area has been blinded by fuse and wood fiber.

It seems to me now that I computed a life of approximately 18,000 tons for Callow Screen belts at the Nevada Mass. Co.

Taggart reported each of 3 units in the old Magma mill were equipped with a 14 mesh Callow Duplex screen operating in closed circuit with a Marcy ball mill. Each duplex passed 250 tons daily and carried 125 tons circulating load. Feed contained 40% solids and 360 gallons per hour. Wash water = 6.0 Gpm.

Life of a bronze Ton-Cap screen was 90 days = 22,500 tons. One man can replace a Callow screen in 5 minutes. No down time as feed diverted to other screen of the duplex. Approximately 45 minutes and two men required to replace screen on Hummer and mill down during period.

Efficiency of Magma Callow screens in removal of undersize = 89%.

Estimated Effect of Flotation Concentrate Grade on Economics of Acid Treatment.

Assuming an 0.5% WO_3 mill feed yielded a 12.5% WO_3 flotation rougher concentrate and an 0.025% WO_3 tailing. The following treatment is based on the evidence that grading up a flotation scheelite concentrate by cleaning the rougher froth in a separate step simply builds up the grade of the tailing in proportion to the grade of the cleaner concentrate, as little or no scheelite is recovered from a cleaner tailing returned to the rougher circuit. There must be a point where the cost of acid treatment will more than offset the value of WO_3 in the tailing.

On the basis of a 12.5% WO_3 rougher concentrate (15.6% $CaWO_4$) containing 5.0% impurities -- mainly insoluble -- there would be (100 - (15.6 + 5.0) = 79% calcite and apatite which would have to be eliminated by acid treatment.

Approximately 2.6# of commercial (33% HCl) muriatic acid would be required to dissolve the calcite-apatite minerals and maintain the acid soluble elements in solution. Therefore, 2.6 x 20 x 79.4 = 4230# acid = 2.115 tons of muriatic acid per ton of concentrate, which, @ $50.00 p/ton delivered, would amount of $105.75 p/ton of rougher flotation concentrate acid treated.

On the basis of a 12.5% WO_3 rougher concentrate and a 0.025% WO_3 tailing, the results would be:

$$\frac{12.5 - 0.025}{0.5 - 0.025} = \frac{12.475}{0.475} = 26.2\text{: 1 ratio of concentration}$$

$$\frac{12.5 \times 100}{26.2 \times 0.5} = \frac{1250}{13.1} = 95.5\% \text{ extraction}$$

0.5 x 0.955 = 0.4775 units WO_3 recovered from a 0.5% WO_3 mill heads in the form of a 12.5% WO_3 concentrate.

0.4775 x $63.00 = $30.10 gross mill value of heads.

$\frac{\$105.75}{26.2}$ = $4.15 cost of acid p/t mill heads.

$30.10 = $4.15 = 25.95 = net mill value p/t heads after acid treatment.

After acid treatment the concentrate should consist of approximately:

74.74% scheelite = 6 0.20% WO_3 + 25.25% insoluble.

This product could be graded up by tabling or a separate batch flotation step. Also, it is probable that the insoluble content of the rougher froth could be reduced in practice.

Assuming a rougher flotation concentrate to contain 25.0% WO_3 and yielding a tailing averaging 0.05% WO_3, the concentrate would consist of 25.0x 1.24 = 31.0% $CaWO_4$, 5.0% insoluble and other impurities and 64.0% calcite and apatite, which would have to be dissolved by commercial muriatic acid (33% HCl).

$\frac{64 \times 52}{2000}$ = 1.66 tons of acid @ $50.00 = $83.00 = cost of acid p/ton of concentrate.

Ratio of concentration on the above would be 55.44:1 and extraction 90.2% = 0.5 x 0.90.2 = 0.451 units of WO_3 p/ton of ore. 0.451 x $63.00 = $28.41 = gross mill value recovered per ton of ore.

$\frac{83.0}{55.44}$ = $1.50 = cost of acid p/ton of ore.

$28.41 - $1.50 = $26.91 gross value of ore after deducting the cost of acid, a difference of 26.91 - 26.10 - $0.81 p/ton of ore in favor of the higher grade concentrates.

Assuming the production of a 50.0% WO_3 rougher flotation concentrate and a 0.10% WO_3 tailing and assuming the concentrate to consist of 62.0% scheelite (50.0% WO_3):

5.0% Insoluble and other impurities - 100- (62 + 5.0) = 33.0% calcite and apatite.

0.858 tons of acid would be required p/ton of concentrate, which, at $50.00 p/ton = $42.90 for acid per ton of concentrate.

Ratio of concentration = 124.75:1

Extraction = 80.16% = 0.40% WO_3 p/ton of ore recovered; 0.4 x $63.0 = $25.20 gross recovered value p/ton ore.

$0.35 = cost of acid p/ton of ore when producing a 50% WO_3 concentrate and a 0.10% WO_3 tailing.

$25.20 - 0.35 = $24.80 gross value less cost of acid.

$26.91 - $24.80 = $2.11 p/ton of ore in favor of producing the 25.0% WO_3 rougher concentrate and acid treating it.

There is no doubt but that there is a point where the cost of recovery crosses the line of financial returns.

In order to attain that point there must be an automatic control of pulp density and reagents in addition to automatic pH control, which is an established and successful procedure. It is certain a lower grade flotation concentrate will result from feeding 2# of oleic acid-elastoil mix when 1# would be ample; also, if 1# of above mix is being fed but 2# are needed to effect recovery, the tailing loss will go up.

Although the mill feed can be fairly constant, there can be considerable variation in the amount of flotation feed (slimes) in the ore as delivered from the mine, as well as that made in the grind.

As there is no practical way of continuously determining the solids directly in a pulp flowing in a launder or pipe, it must be done indirectly by measuring the added water to maintain a constant density.

If the flotation feed (slimed solids) increase in quantity, the density controller opens the water valve to admit more water to the pulp to maintain the constant ratio of water to solids. If the ore going thru the mill happens to be coarse and hard and less flotation feed (slime) is made, the density controller acts to cut down the added water - thereby maintaining a constant density.

I believe it is entirely feasible to connect the motor driven reagent feeders thru a Variac rheostat attached to the density controlled water valve which is opening and closing as the solids vary.

In an all flotation mill, the reagent feeders would be activated from the weightometer feeder.

Possible Improvements in Gravity Concentration.

Assuming the concentrating tables are supplied by a hydraulically sized feed, there are still further possible improvements to be considered.

For example, I believe groove riffles would recover more scheelite than the standard practice of using raised riffles. This would be particularly true in the finer sizes which in standard practice are maintained in a continuous boil from one riffle to another with no opportunity to settle onto the table deck. The pulp passing over the table is a high density mass that retards the settling out of any particles - differences in specific gravity are reduced as the density of the moving mass increases.

If the pulp moved smoothly across the table deck the heavy particles, large or small, would certainly contact the deck where extraneous forces such as lighter minerals and wash water would have but one surface on which to act. If in suspension in a moving mass of pulp, forces are acting on all surfaces. The mineral must be on the deck surface in order for the table head motion to push it into the concentrate band and from there to recovery.

If minerals moving across a smooth table deck slide into a groove, the contents of which are mobile due to the action of the deck, the heavy particles will settle to the bottom of the groove and crowd the lighter particles to the surface where they will be washed into middling and tailing. This is an application of the basic principle of sink-float concentration to finely ground mineral.

Most of the scheelite recovered by gravity is recovered by the first few riffles. Whatever is caught by lower riffles has been washed over the upper riffles and has settled in the lower riffles by accident. In present day practice there is no material change in the spacing or size of the riffles. Even using raised riffles, it seems to me there would be a gain in recovery by increasing the distance between riffles from the top down so as to give the pulp a chance to level off between riffles and allow fine heavy material to settle thru the pulp film to the deck and move along the deck to the nearest groove or riffle.

There is a 2'x4' laboratory size table at Toulon that could be set up to take a cut out of the feed to any of the mill tables. Experimental decks could be tested on the laboratory table and the results checked against full size table by sampling and assaying the products of both. It would take some rigging up to do this at Toulon, but it could be done.

There is the possibility that by two-staging the overflow from a Krebs cyclone that two separate flotation feeds might be made; one of very fine crystalline matter, -325 mesh sands, and the other of mainly colloidal matter which would require a different treatment.

There are calcareous accretions on the Toulon tables along the contact of the riffles and deck. These rough areas could cause losses by retarding the movement of concentrate along the riffles and, also, causing local boils in the pulp - stirring fine concentrate back up into the pulp on the way into the tailing.

Careful scraping, so as not to rough up the riffles, and the use of a 10% solution of muriatic acid would remove the above-mentioned matter.

Respectfully submitted,

George Crerar

INVENTORY TOULON WORKS

July 1955

GRAVITY MILL

1 Wemco Table No. 4 Ser. No. 42-3043F
1 Wemco Table No. 4 " " 42-3043B
1 Table Motor GE 1-1/2 HP 220-440 Ser. No. 5578902
1 Table Motor GE 1-1/2 HP 220-440 " " 5618023
5 Plat-O Tables Belt driven
5 Plat-O Self Oiling Heads
1 Magnetic Separator
2 Wheelbarrows for separator rejects
1 12 x 16 Roaster Tube with oil burner
2 12 inch elevators 40 feet
1 8 inch Elevator 30 feet
2 6 inch Elevators 30 feet
1 Gallagher 2 cell Flotation Machine
Motor Float Drive GE 5 HP 220-440 RPM 1735 Ser. No. BR11561
1 16 x 36 Rod Mill (concentrate regrind)
1 20 inch x 8 feet Screw Classifier
Allis-Chalmers Motor Gear Reduction 3 ..
1 Small Belt Conveyor conc. dewatering classifier to waste
3 Pairs Allis-ChalmerskRolls 14 x 24
1 Pair Stearns-Rogers Rolls 14 x 24
Roll Drive GE Motor 75 HP, RPM 900, Ser. No. 68-351 Extra paper pulley, 1 used - 1 new
1 Krogh 1-1/4" pump Allis Chalmers motor 1-1/2 HP 220-440 RPM 1740 Ser. No. 67794K
1 Table Drive Westinghouse Motor 10 HP, RPM 1140 Ser. No. 2319872
1 Wet Trommell Screen
1 Single Callow Screen
1 10-foot Cone
1 6-cell Sizer
1 Curtis Compressor, Wagner Motor 1/2HP, RPM 1725, Ser. No. IU
1 Ingersoll Rand Motor Pump Ser. No. 2PVH10. Motor 10 HP, 220-440 V. RPM 3470, Ser. No. EL1677
2 2-ton Chain Blocks
1 3-ton Chain Blocks
1 Braun Pulverizer Type UA, GE Motor 1-1/2 HP, 220-440 V. RPM 1740, Ser. No. 5404140
1 Denver Fire Clay Co. Crusher No. 2, GE Motor 1-1/2 HP, 220-440, RPM 1740, Ser. No. 5431106
1 Watson Generator KW5 Amps. 40 Volt 125 RPM 1140, Type AW8P, No. 32442 Westinghouse Motor 7½ HP, RPM 1725, Ser. No. 2330558
10 GE Magnetic Switches 7½ HP, 440 V., No. GR7006D40H
3 Square D Motor Switches 440 7½ HP Type KXR1
1 GE Starting Compensator Type 1, Form K, 100 HP
1 De-ion Linestarter Size 2 Style 999207A Westinghouse
1 GE Motor 2 HP RPM 1150 Ser. No. 3999489
1 Power Hacksaw

1 Sterling Grinding Stand No. 1
1 9 x 16 Steel Thickener Tank
1 8 x 30 Wood Thickener Tank
1 Toledo Platform Scale Style 501 No. 717583, Cap. 2850
3 Primary Transformers 6600 V. to 440 Volt
2 Lighting Transformers on pole adjacent to substation
1 Tyler Hummer Electric Screen 4' x 5' 2-surface Ser. No. 11542
1 Outside Ore Bin for odd lot ores, together with conveyor belt
from crushing plant
3 Clarkson Reagent Feeders mounted near concentrate flotation section
1 Lennox Furnace (for mill equipment with oil burner, electrically driven)
1 P&H Electric Welder, Type H263, 440 V. with leads for same
1 Thor Electric 2-wheel grinder complete with stand for same
1 Acetylene Welding & Cutting Unit complete with Igniter
1 Small Syntron Feeder located near pump sump
1 Bendarari Jiggs 24" Ser. No. 380
1 440 to 110 Volt Transformer located near Tyler screen
1 Oil Storage Tank, tank car type, located near spur track and
marked "Rare Metals Corporation".
1 10" x 20" Farrell Jaw Crusher driven by 40 HP GE Motor with
multiplex Vbelt drive to crusher
1 Conveyor located above Telsmith secondary crusher approximately
18" x 3'
1 Acme Root Blower with electric motor - Vbelt drive
1 Main Elevator Drive Falk Reducer, motor unit
1 3" Wilfley pump driven by U. S. Electric Motor, 10 HP
in main pump sump

Crushing Plant

1 Telsmith Crusher, Westinghouse Motor 25 HP, RPM 865, Ser. No. 2318673
Fairbanks-Morse Induction Motor Starter No. 56284 30 to 40 HP
1 3 foot x 6 foot Pan Conveyor
Rite-Lo Speed Gear Reduction Motor 1-1/2 HP, RPM 60, Ser. No. 4927
1 16 inch 60 feet Conveyor Belt
1 12 inch Elevator Belt 40 ft. centers
1 5/16 Punched Trommel Screen, GE Motor, 10 HP, RPM 870, Ser. No. 4830669
Westinghouse Auto-Start 5 to 10 HP, No. 521035A
1 12 x 20 Fine Ore Bin, wood
1 Pan Conveyor (spare for main ore grizzly) no power

INVENTORY TOULON WORKS

July 1955

TAILINGS PLANT SECTION

5 Plat-O Tables
5 Self-Oiling Heads, Westinghouse Motor 1-1/2 HP, 220-440 RPM 1134
Ser. No. 117EM877 Reeves Drive No Pulley
GE Motor 1-1/2 HP, 220-440 RPM 1125, Ser. No. 5346803 Reeves
Drive No pulley
Westinghouse Motor 2 HP, 440 RPM 1140 Ser. No. 1332409 Reeves
Drive Pulley
Westinghouse Motor 2 HP, 220-440 RPM 1150 Ser. No. 43237
Reeves Drive Pulley
1 EIMCO Ball Mill Complete with liners and balls and U drive GE
Motor 100 HP. RPM 720, Ser. No. 252006 GE AC Drum Switch
2 Pairs Double Callow Screens Incomplete - some parts in gravity plant
1 Roper Gear Pump No. N2172 Westinghouse Motor - 1-1/2 HP, 220-440,
RPM 1730 Ser. No. 8106106
1 Wilfley 2" Pump (no motor)
1 U.S. Motor 3 HP, 220-440 RPM 600 Ser. No. 182448
1 Gardner Vertical Air Compressor with motor and motor circulating pump,
4" 150 Lb. Pressure No. 73174
1 5-ton Block on 40 ft. overhead trolley and beam
1 40 ft. x 10 ft. Thickener Tank. U. S. Syncrogear Gear Reduction
Motor 2 HP, 220-440 RPM 79 Ser. No. 179989
1 30 foot x 20 foot Fine Bin
1 Oliver Filter Ser. No. 4279R 5' 4" x 4' GE Motor 220-440 1 HP,
RPM 1720 Ser. No. 5376959 Reeves Drive
Sterling Motor 1 HP, 220-440 RPM 1800 Ser. No. 51048
6 Gallagher Flotation Cells No. 40
GE Motor 7-1/2 HP, 220-440 RPM 1450 Ser. No. 5774635
GE Motor " " " " " " " GY14441
GE Motor " " " " " " " GY16974
1 Gallagher 4 cell Reagent Feeder
Westinghouse Motor 1/4 HP, 220 RPM 1725 Ser. No. HH
Transformer HP5EVA RPM 440 Volts 110-220 No. 3557
1 Dorr Diaphragm Pump
1 Conditioner Tank 8 x 8 Wood
GE Motor 5 HP 220-440 RPM 1735 Ser. No. BR9947
1 Conditioner Tank 5 x 5 Wood
Allis Chalmers Motor 3 HP, 220-440 RPM 1740 Ser. No. N5063-355
Westinghouse Gear Reduction Motor 1-1/2 HP, 220-440 RPM 258,
Ser. No. 17235
2 Dorr Diaphragm Pumps
1 Thickener Tank 4 x 4 Wood
1 Thickener Tank 8 x 12 Wood
Diaphragm Pump
Pacific Gear Reduction Motor 2 HP, RPM 115, Ser. No. 5MR817
2 Reagent Conditioner Tanks 6 x 8 Wood
1 Electric Eye
1 16 inch Roaster S.T. Johnson Oil Burner Type BH-O Ser. No. 117736
Oil Burner Motor 1/6 HP Ser. No. ED
Sterling Gear Reduction Motor 220-440 RPM 30 3 HP Ser. No. 21289
1 Ingersoll Rand Vacuum Pump 10 x 4 Class ER-2, No. 63942 GE Motor 5 HP
440 RPM 1155 Ser. No. 4071453

1 Small Root Blower (no plate)
1 Motor Pump Ingersoll Rand Size 1-1/2 inch. RV5, RPM 3470 Ser. No. 103617
GE Motor Pump 5 HP, 220-440 RPM 3470 Ser. No. HM2439
1 Ingersoll Rand Motor Pump Size 1RVF-2 RPM 3470 No. A38656
GE Motor 2 HP, 220-440 RPM 3470, Ser. No. HS837
2 5 KVA Transformers at tailings plant substation
3 75 KVA GE Transformers - same location
1 Small Roots Blower
Partial barrels of following flotation reagents:
Aero X301,- Cresylic Acid,-Soldim Silicate,-Emulsol X20,-Short Wooden barrel of black reagent, probably BTO,- one oil-fired space heater mounted on two wheels and with electric-driven oil burner.

Gravity Mill Shop

1 1/8" Electric Drill
1 Van Dorn Electric Grinder
1 1/2" Thor Electric Drill
1 3/4" Thor Electric Drill
1 8" Thor Electric Saw

General Classification

1 75 HP Electric Motor (spare motor for main drive in gravity plant)
1 HD 14 Allis Chalmers Tractor powered with General Motors Diesel engine equipped with bulldozer.
1 Hough Payloader with cable for same
1 Dodge 4x4 Service Truck equipped with front winch and front boom
New Elevator Buckets, Size AA
1 New movable head for 10 x 20 Farrell Jaw Crusher
1 Stationery Jaw for same
1 New pair Roll Sheels for Stearns-Rogers rolls
1 Ingersoll Rand motor pump on mill circulating water (located near Tyler screen
1 Bank of 4 Kraut Flotation Cells
1 Half-size Wilfley Table less head motion
1 Hevi-Duty Electric Muffle complete with thermocouple and temperature recording scale
Assorted Glass Funnels with metallurgical reagents

Telephone 9-0962

CERTIFICATE OF ASSAY

DEASON & NICHOLS

ASSAYERS & CHEMISTS

C. Ivan Nichols — Wm. J. Deason

160 So. West Temple Street

Salt Lake City 1, Utah

Nev-Tah Oil & Mining Company
430 Gazette Bldg.
Reno, Nevada

July 14, 1955

ASSAY PER TON OF 2000 POUNDS

DESCRIPTION	NO.	GOLD OUNCES	SILVER OUNCES	WET LEAD %	COPPER %	ZINC %	INSOL %	Tungsten WO_3	%	%	%
Jagged Top - Lot 3											
391								0.13			
Lot 4											
392								0.08			
Blair - Lot 1											
393								0.08			
394								0.07			
Blacksmith Tunnell											
395								0.20			
396								0.39			
Prospect Pit											
397								0.12			
398								0.12			
Alpine Lot 2											
399								0.18			
400								0.36			
401								0.31			
Stormy Day Lot 1											
402								0.30			
Lot 2											
403								0.33			
Stormy Day											
404								0.10			
405 - Tommy Kent								0.34			
Clyde Morrison											
406								0.48			

C. Ivan Nichols

CHARGES $

Telephone 9-9962

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DEASON & NICHOLS
ASSAYERS & CHEMISTS
C. Ivan Nichols Wm. J. Deason
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Salt Lake City 1, Utah

Nev-Tah Oil & Mining Co.
Gazette Bldg.
Reno, Nevada

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DESCRIPTION	NO.	GOLD OUNCES	SILVER OUNCES	WET LEAD %	COPPER %	ZINC %	INSOL %	~~Tungsten~~ % WO3	%	%	%
Temple 407								0.30			
Chester 408								0.30			
Gail Pier 409								0.42			
Reid & Mayfield 410								0.37			
Graig 411								0.40			
Modoc 412								0.30			
Dennison 413								0.33			
Paradise 414								0.29			
D D EOT 415								0.30			
Composite Ending								0.30			
D D Tails 417								0.31			

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C. Ivan Nichols — Wm. J. Deason

160 So. West Temple Street

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Reno, Nevada

- 2 -

July 14, 1955

ASSAY PER TON OF 2000 POUNDS

DESCRIPTION	NO.	GOLD OUNCES	SILVER OUNCES	WET LEAD %	COPPER %	ZINC %	INSOL %	Tungsten % WO_3	%	%	%
Nightingale											
373								0.07			
374								0.48			
375								0.22			
376								0.08			
Joe Uisco-Star											
377								0.15			
378								0.54			
379								0.79			
380								0.76			
381								0.18			
382								0.11			
383								0.11			
Delamare-Lot 3											
384								0.17			
385								0.33			
386								0.12			
Lot 7											
387								0.30			
Lot 5											
388								0.21			
389								0.34			
Ragged Top-Lot R2											
390								0.13			

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DEASON & NICHOLS

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Salt Lake City 1, Utah

Nev-Utah Oil & Mining Co.
430 Gazette Bldg.
Reno, Nevada

July 14, 1955

ASSAY PER TON OF 2000 POUNDS

DESCRIPTION	NO.	GOLD OUNCES	SILVER OUNCES	WET LEAD %	COPPER %	ZINC %	INSOL %	Tungsten WO3 %	%	%
Nightingale										
# 351								0.16		
352								0.16		
353								0.17		
354								0.43		
355								0.13		
356								0.11		
357								0.10		
358								0.10		
359								0.105		
360								0.14		
361								0.12		
362								0.12		
363								0.08		
364								0.07		
365								0.07		
366								0.07		
367								0.07		
368								0.21		
369								0.08		
370								0.07		
371								0.07		
372								0.075		

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