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How South-West Wisconsin tailing piles can be worked at a profit

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HOW SOUTH-WEST WISCONSIN TAILING PILES
CAN BE WORKED AT A PROFIT

by

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A

THESIS

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Professor of Mining.

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G. H. P.

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INTRODUCTION

It was not until the Flotation Process was introduced in this S.W. Wisconsin zinc district that the writer was convinced that some of the tailing piles here could be worked at a profit.

In 1928, the Badger Zinc Company built a 120 ton Flotation Plant about two miles north-east of Linden. It was after the second visit to the plant that the writer learned the above company was making a profit on a 5% zinc ore with very little lead and with a low market price for the finished product, namely $30.00 per ton for a 60% zinc concentrate.

The total mining and milling costs per ton of dirt (run of mine ore), were said to be about $1.75. Upon further investigation, it was found that the milling cost was about one-third if this, namely about 60¢ per ton of dirt. Now, if this company, with their flotation plant, could make a profit with 5% ore, and a low market the writer was struck forcibly with the idea that some of the tailing piles, running as high as 2% zinc, could also be worked at a profit. True, a 2% tailing pile is only slightly more that one-third as rich as a 5% run of mine ore, but the cost of working a tailing pile is also only about one-fifth as much as mining and milling a 5% ore, as was already pointed out above. Especially would this idea be more convincing with a higher market.

In order to profitably work a pile assaying 2%, or slightly lower, a low grade concentrate must be cheaply made, and a simple, low cost flotation scheme used. With these two facts in mind, the writer conducted various tests on laboratory machines as to the best way this could be brought about.

The Badger Zinc Company continued to operate for approximately three years, but due to further drops in the market price for zinc, and increased mining costs, they were forced to shut down. They operated the wet mill (gravity concentration), and Flotation Plant as a custom mill for a short time, but there was not enough custom ore coming in for continuous operation; so the plant stood idle for about three years. It was then taken over by the Pioneer Flotation Company, and it was for this company that the writer did considerable consulting work, coupled with a report on "How This Plant Could Operate as a Custom Mill aided by Low Grade Concentrate from Nearby Tailing Piles".

It is from this above mentioned report that some of the following material is drawn, but the body of this thesis is the result of actual experience, deductions, and laboratory tests, as the writer practically "grew up" with flotation in this district.

GENERAL GEOLOGY [1]

The Wisconsin zinc district embraces an area about 60 miles long north and south, and about 30 miles wide east and west. It is almost wholly in the southwest corner of Wisconsin, extending but short distances into northern Illinois and eastern Iowa.

The strata of the district consist of a series of Silurian and Ordivician limestones, shale, and sandstones, underlain by Cambrian sandstones and pre-Cambrian crystalline rocks.

Silurian	Niagara Limestone	50 feet
	Mequoketa Shale	160 feet
	Galena Limestone	230 feet
Ordivician	Platteville Limestone	55 feet
	St. Peter Sandstone	70 feet
	Lower Magnesian Limestone	200 feet

Nearly all of the Niagara and Mequoketa formations have been eroded away. The Platteville, Belmont, and the Sinsinawa Mounds in southwest Wisconsin, and the numerous mounds in northwestern Illinois are all that remain of this formation in the district.

In the northern part of the district, there is an east and west unbroken ridge. Extending from this ridge there is a north and south ridge running through the middle of the district in a southerly direction for a distance of about 30 miles. The lower part of the Mequoketa Shale and the upper part of the Galena Limestone are the formations exposed on these ridges. The ground from these two ridges slopes to drainage basins.

The rock formations of the district dip to the south-southwest, at a slope of about 20 feet per mile. A series of anticlines, synclines, and structural basins have been produced by slight compression and folding of the strata. A north and south lateral pressure produced the main east and west axis of most of the basins.

The secondary axis of the basins running north and south was produced by an east and west lateral pressure. Frequently, one side of these basins will dip more steeply or show changes in slope. At these points, the ore bodies usually occur located in either the lower part of the Galena formation, or the upper part of the Platteville formation, or both.

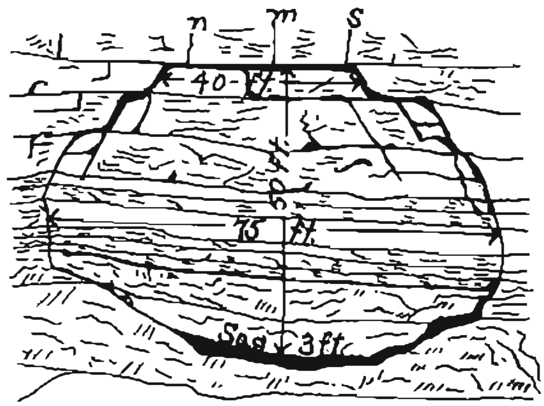
"The Galena formation consists of heavily bedded dolomites, interbedded with chert beds and nodules and thin bedded clay shales. The base of the formation consists of thin bedded, brown carbonaceous shales, from 1 to 15 feet in thickness. These shales are known as the "oil rock". The bottom of this shale marks the base of the Galena limestone formation. A blue clay shale usually occurs immediately below the oil rock. This shale varies from a few inches to 6 or 7 feet in thickness and is known as the "clay bed". Immediately below the clay bed is the "glass rock", a fine, compact limestone, usually gray or brown in color, which breaks with a conchoidal fracture from which characteristic it derives its name. At some places there are shale beds resembling the oil rock, between the clay bed and the glass rock. At other places there is practically no oil rock, and at still other places there is no clay bed".

The base of the glass rock, which is the upper part of the Platteville limestone formation, marks the base of the ore bearing formations, although there are a few cases in which ore has been found below the glass rock.

ORE DEPOSITS OF SOUTHWEST WISCONSIN [2]

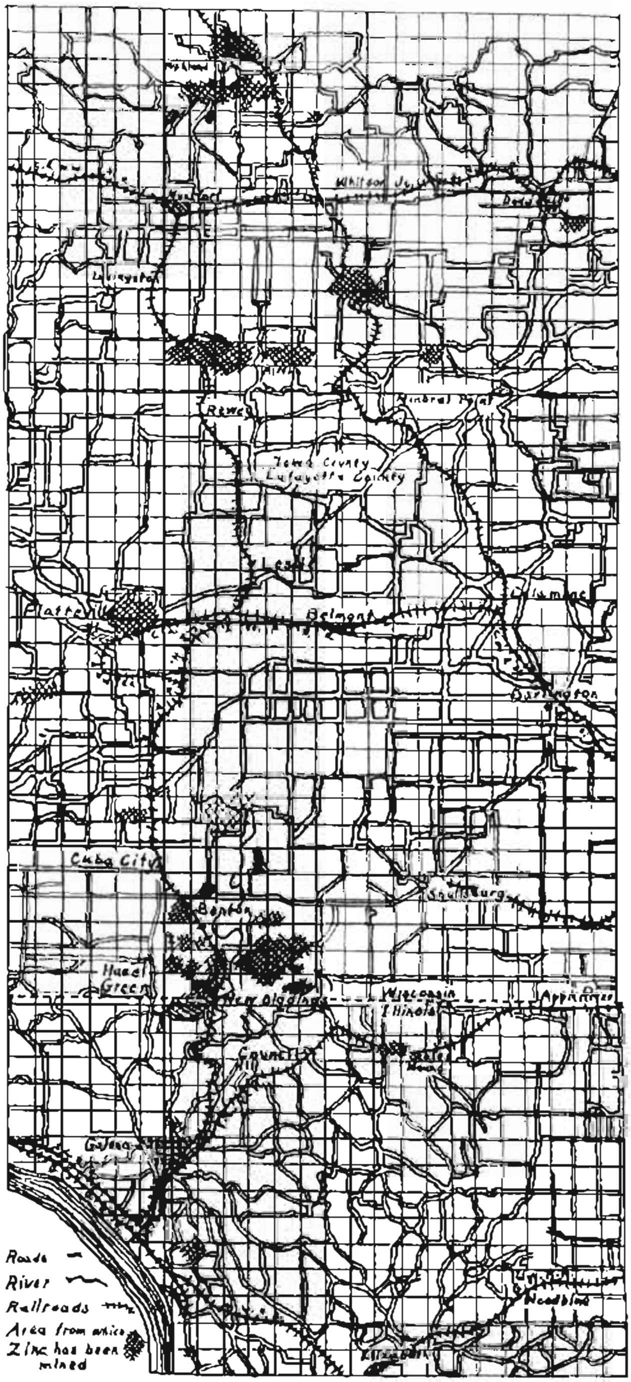
"The Wisconsin orebodies fill partial openings in the limestone made by the subsidence of large prism-like masses in the bottom portion of the limestone. The limestone stratum in which the ore occurs is about 150 feet thick, (given as 250 feet under Geology of this report), and is underlaid by a persistent bed of clay shale. It looks as if the limestone might have been dissolved out for a foot or two above the shale along certain channels to such an extent that finally a large irregular prism of limestone detached itself from the solid mass and fell down a distance of perhaps two feet. The result of this subsidence being that the interior of the prism is cracked up to a certain extent and certain openings are made along the top and sides. These openings have served for the deposition of the ores.

Fig. 1 Section of flats and pitches of Robert's Mine, Linden, Wis. N, M, S are north, middle, and south crevices. (After Chamberlain) 3



The openings thus formed in the cross-section have the shape of a rude arch, usually quite flat at the top, and breaking down in irregular steps along the sides. In the local phraseology, the ores deposited in the level at the top of the arch are called "flats", and those occuring along the sides are called "pitches". The slightly broken up interior of the prism is called the "core". See Fig. 1, above.

The mineralization of the prism is irregular, and in some cases these prismatic orebodies have been proven to be of considerable length. (About half a mile). Their sizes also vary to the extensiveness of the dissolution of the lime, or whatever it was that caused the subsidence.



WISCONSIN - ILLINOIS ZINC DISTRICT 3

Fig. 2

DESCRIPTION OF TAILING PILES

The tailing piles of S.W. Wisconsin can be divided into two districts, namely the northern and southern.

In the northern district, with Mifflin as a central point, there are, at a conservative estimate, 2 million tons. One pile alone contains 1 million tons. Another pile about ¾ of a mile away contains 500,000 tons. The remaining 500,000 tons are scattered within a radius of about 5 miles in a few piles, ranging in size from 50,000 tons to 150,000 tons.

The southern district is centered around New Diggings. Here can be found, at a conservative estimate, also 2 million tons.

The above mentioned 4 million tons do not include all the tailings in S.W. Wisconsin districts, but are those running higher than average in zinc.

There is very little difference between the tailing piles in the southern and northern districts, all averaging about 1.5% Zn. Some piles run over 2% Zn while others run slightly under 1.5% Zn, but on the whole, they will all average 1.5% Zn.

About 97% of the tailings in the above mentioned piles will pass a 3/8" screen.

ORIGIN OF TAILING PILES

The tailing piles are the result of the milling of an ore composed of galena, sphalerite, marcasite, and pyrite in a gangue of dolomite, calcite, and barite. The chief commercial mineral in the original ore is sphalerite averaging about 8% zinc. Galena is also recovered, but will average only about 1% Pb in the original mill feed.

The zinc and lead minerals were recovered in a gravity concentration process of which a 2-jig mill is typical. This type of mill recovered on an average 70% of the zinc and about 80% of the Pb.

The 30% loss of zinc was found to be distributed as follows: 10% of the original zinc in fines or sludges, 2% in rock piles (waste from grizzly over bin), and the remaining 18% in the tailing piles. About 75% of this 18% loss in the tailing piles occurs in the fine sand range, namely between 10 and 80 mesh, which is 37% of the weight of the tailings.

From several tests, it was found that practically all minus 65-mesh material can be considered as slimes, or sludge, and according to Coghill and Anderson[4] these slimes will assay approximately the same as the original ore from which these piles originated.

WEIGHT PER CU. FT.; CU. FT. PER TON; SPECIFIC GRAVITY; % VOIDS; and % MOISTURE

The specific gravity of the heads just as they come from the pile is found to be 2.40, but this is the gravity considering these tailings as one solid mass, without voids; so on running a test for % voids on the tailings (original heads), the voids were found to be 32.5. This then brings the specific gravity of the original tailings down to 1.60

The weight of one cu. ft. of tailings from specific gravity computations is found to be 106 lbs. On checking the above figure, a cu. ft. of tailings was actually weighed and found to be 100 lbs. in the natural state.

The % moisture on an actual test was found to be 5%. The following table gives the data under both wet and dry conditions:

DATA ON HEADS (natural)

Sp. Gr. natural	% Voids natural	% Moisture natural	Wt/Cu. Ft. moist
1.60	32.5	5.0	100

Wt/Cu. Ft. dry	Cu. Ft./Ton moist	Cu. Ft./Ton dry
95.0	20.0	21.0

These figures agree closely with those of the owners of these tailing piles, whose figures are 100 lbs per cu. ft, or 2,700 lbs. per cu. yd.

SCREEN ANALYSIS OF TAILINGS

Product	Percent of Total Weight	ZINC Assay per-cent	ZINC Percent of Total Zinc	IRON Assay per-cent	IRON Percent of Total Iron
Composite	100.0	1.88	100.0	4.60	100.0
Thru 3/8 on 3/16 inch	29.07	0.96	14.8	4.3	27.1
Thru 3/16 on 10 mesh	26.34	0.64	8.9	4.5	25.7
Thru 10 on 14 mesh	14.24				
Thru 14 on 20 mesh	6.1				
Thru 20 on 28 mesh	5.86				
Thru 28 on 35 mesh	5.35	2.97	58.9	4.63	37.5
Thru 35 on 48 mesh	3.36				
Thru 48 on 65 mesh	2.37				
minus 65 mesh	7.31	4.49	17.4	6.13	9.7

The sample used for this analysis was representative of two million tons. The weight of the sample was 31,302 grams so as to make this analysis as representative and accurate as possible. This agreed closely with three other analyses using 1,000 grams for the sample weight.

A typical analysis of a pile lower than the average in zinc content is as follows:

* After W. C. Trewartha.

Zn	1.3 percent
S	3.5 percent
CaO	26.6 percent
$CaMg(CO_3)_2$	77.53 percent
Fe	3.7 percent
SiO_2	11.8 percent
MgO	14.3 percent

In order to be absolutely certain of the critical point,(size at which to reject the uneconomical portion), the minus 10 plus 20 mesh product was assayed. It assayed a trifle short of 4½ percent zinc.

The above table shows that all of the plus 10 mesh product could not be profitably concentrated.

Since the minus 10 plus 20 mesh is a trifle coarse for a fine sand table, it is also the most valuable portion of the feed, but the riffle arrangement of the table shown on page 26, makes it unnecessary for finer crushing.

PROFITABILITY IN TAILING PILES DUE TO

COARSE SAND CONCENTRATION OF ORIGINAL ORE

As can be seen from the screen analysis on page 11, the jig is an efficient concentrator in the coarse sand range, namely above 10 mesh, but in fine sand concentration and in concentration of slimes, its efficiency ends; therefore, the recoverable zinc in the tailing piles. Since the coarse tailings run so low in zinc, it can be seen that the high recovery in the coarse range is not due so much to the mechanical efficiency of the jig as to the fact that a maximum unlocking of valuable minerals from the gangue occurs in the coarse range at about 3/8 inch and in the fine range again at 65 mesh.

Incidently, this is also the larger size of slimes. The reasons for this coarse sand concentration were: (1) maximum unlocking occurs at 3/8 inch in the coarse range; therefore, no unnecessary expense of crushing it finer, (2) to crush as coarse as possible in order to avoid production of more fines, since before the advent of flotation there was no efficient method of recovering these, (3) fine sand concentrates from tables too high in lime, (4) recovery on tables no higher than with jigs. Another very important disadvantage for table concentrates arises in the grading up process. Both jig and table zinc concentrates must be graded up by removing nearly all the marcasite with which both are contaminated due to the small difference in gravity between marcasite and sphalerite. The grading up is brought about by giving the concentrates a slight or skin roast after which the marcasite can be removed by means of a magnetic seperator.

Even in grading up a coarse jig concentrate, there is approximately an 8% loss in fines in the roasting process; therefore, in the roasting of finer concentrates from tables a considerably greater loss in fines would result. Hence, a table concentrate was looked upon as an unfavorable product by ore buyers, both from a standpoint of high loss of fines in the grading up process and in the high lime content after grading up, because this lime could not be removed. This high lime in table concentrates exists because the limestone gangue contaminates the zinc band on a table deck, and if a reasonable recovery is to be effected, this contamination must be left unmolested.

A typical 2 jig zinc concentrate will assay on an average 37% zinc and after the removal of the marcasite this is graded up to 59 and 61% zinc.

WHY CHOOSE MINUS 10 MESH AS CRUCIAL SIZE

Most writers who have written about S.W. Wisconsin tailings were of the opinion that only the minus 20 mesh material should be considered from a standpoint of additional recovery of zinc from these tailings. This has been found to be not quite true for the representative sample the writer is working on.

After several screen analyses of a sample representing approximately two million tons of tailings from the northern district, it was found that the minus 10 plus 20 mesh material contained considerable sphalerite, and the zinc assay was almost as high as that of the slimes, namely over 4%. This material was first examined closely with a magnifying glass.

The minus 3/16 inch plus 10 mesh material was then examined closely, but nothing of much value could be seen in this range. A subsequent zinc assay proved this fact for the zinc content was found to be very low.

The minus 3/8 inch plus 3/16 inch material showed some valuable zinc mineral, but not enough for profitable extraction. About every third handful of this material picked up at random showed a large piece of pure sphalerite about 5/16 inch in size, in addition to a few middlings, which were found in each handful. As a result, this size range assayed nearly 1% zinc.

The fact that there was not much valuable material above the 10 mesh screen simplifies the gravity concentration flow sheet in this project in that no crushing is necessary, and the fact that there was valuable material in the minus 10 plus 20 mesh range complicates the problem a trifle because this material is somewhat coarse for tabling, thereby causing a wide size range of table feed, namely minus 10 plus 65 mesh. But this slight complication can be mostly alleviated by using the type of riffling shown in the diagram on page 26.

This projected riffling will allow for a stronger cross current of water in order to separate the concentrate in the coarser size range at the same time affording protection for the fine concentrate which otherwise would be washed into the tailing zone, were it not for this added riffle protection.

With reference to the screen analysis on page 11, the minus 10 plus 20 mesh material accounts for 20% of the total weight of tailings, and since this material assayed a little more than 4% zinc, this size range includes 45% of the total zinc content. Again referring to the screen analysis on page 11, it can be seen that 76% of the total zinc is found in 44% of the total weight, i.e., in the minus 10 mesh material.

In the tailings discarded by the average mill in this district 20% by weight is usually considered as sludge, but in the tailing piles under discussion here, only 7% of their weight is sludge, the remaining 13% will be found in what was formerly a sludge pond made while the mill was in operation, this pond being entirely separate from the tailing piles. This sludge can also be worked at a profit (actually has been done mostly on account of a previous report by the writer), but the recovery is not as high as with regular tailings. The table recovery is lower and the flotation recovery is also lower due to the oxidized condition of the zinc sulphide.

The recovery of the zinc minerals in the regular tailing piles is higher because they will have to undergo additional grinding in a ball mill after which there will be fresh surfaces exposed. This is the answer to the exceptionally high recovery obtained in our laboratory flotation tests, even though some of the zinc sulphide minerals were in a slightly oxidized condition.

WET VS DRY SCREENING

The moisture content in the tailings, being about 5% makes it almost compulsory for wet screening if a reasonable efficiency is desired. The original tailings in the pile being damp, have a tendency to be slightly lumpy and sticky. It can be readily seen that on screening them in their natural state the screening capacity would be low.

Another important reason for wet screening is that even in a dry state the coarse, rejected pieces have fines clinging to them. These fines can not be removed by vibration alone, hence a forceful stream of water which will not only increase screening efficiency, but will also recover the fines clinging to the coarse rejected pieces. These clinging fines, from an actual test amount to 10% of the weight on the rejected portion retained on the 3/16 inch and 10 mesh screens. This rejected portion accounts for 56% of the total weight of the tailings. As an illustration, suppose 1,700 tons per day are being screened, then 56% of 1,700 tons, or 952 tons will be rejected by the 3/16 inch and 10 mesh screens, and 10% of 952 tons, or 95.2 tons, of valuable fines will be saved in one days operation due to wet screening alone.

TYPE OF SCREENS TO USE

It is proposed to screen about 1,500 tons of original tailings daily. This amount is necessary in order to obtain 100 tons of table concentrates assaying 12-15% Zn.

With a screen efficiency of 95%, it was found from several tests that 55.2% of the tailings are rejected by the 3/16" and 10 mesh (.07") screens. (See screen analysis). But to be on the side of safety, 57.5% will be assumed as rejected, leaving 42.5% as a safe percentage passing through the 10 mesh screen. Thus it can be seen that slightly more than 600 tons of minus 10 mesh is table feed which will give at least 100 tons of flotation feed assaying 12-15% Zn.

The proverbial screen in this district is the trommel, but in this case the writer finds it more practical to use the Leahy Vibrator. It is proposed to use two double-deck vibrators, namely a 3/16" and .07" screen in each. The reason for this is to avoid the purchase of three Leahy vibrators. According to the above tonnage, each screening unit will handle 750 tons of original tailings. Each 3/16" screen will reject approximately 30% of 750 tons, leaving 525 tons for each of the 10 mesh screens.

Why this particular method of screening is to be used can best be answered by observing the following page of data sent by the Deister Concentrator Company.

DATA OF LEAHY NO - BLIND SCREEN

	Dry Screen	Wet Screen
Screening Area - - - - - - -	5' X 3' - -	5' X 3'
Percent undersize in feed -	65% - - - -	50%
Tons per hour, under-size -5/16 -	30	
Tons per hour, "" -10 mesh	- - - - -	12
Screening Efficiency - - -	90% - - -	90%
Life of screen cloth, mill days - - -	30 - - -	15
Power consumption - - -	½ HP - - -	¼ HP
Cost per ton of undersize produced - -	.08 cents	.6 cents

TESTS ON LEAHY NO-BLIND SCREEN

Condition of Feed	Tons per 24 Hours Feed	Oversize	Undersize
Wet	900	510	390
Wet	550	250	300
Dry	1085	365	720

* *

% U.S. in feed	% U.S. in Oversize	Percent Efficiency	Screen Size
50.4	12.5	86.0	.07" open.
56.8	5.0	96.0	.07" open.
67.9	4.7	97.7	.263" open.

COMPARISON OF SCREENING COSTS

One Year's Operations - Dry Screening

Comparisons of Trommels with Leahy No-Blind Screens

Thirteen Trommels Replaced by Four No-Blind Screens

	Trommel	No-Blind Screen
Product - - -	-3 mesh (.263" opening)	3 mesh (.263" opening)
Tons per day each - -	170 - - - - - - -	720
Size - - - - - - - -	3' dia. X 9' long -	3' X 5'
Screening area - -	81 sq. ft. - - - -	15 sq. ft.
Power - - - - - - - -	3-½ HP - - - -	½ HP

Screening Costs per Ton Screened

	Trommel	No-Blind Screen
Power at 1¢ per KW hr. -	$ .0037 - - -	$ .00015
Attendance at $7.50 per day	.0034 - - -	.00038
Maintenance (Labor and Material)	.0101 - - -	.0027
	$ .0172 - - -	$ .00323

AVERAGE TABLE TEST ON QUARTER-SIZE BUTCHART TABLE

PRODUCT	Percent of weight	ZINC Assay percent	ZINC Percent of feed content	IRON Assay percent	IRON Percent of feed content
Feed (minus 10 mesh deslimed)	100.00	3.00	100.00	4.52	100.00
Zinc Concentrate	16.9	12.8	72.1	8.2	29.2
Table Tailing	83.1	1.0	27.7	4.05	70.9

(The percent of zinc feed content and the percent of iron feed content columns do not quite add up to 100 percent because the weights were taken to the nearest tenth of a pound).

From the above table it can be seen that the zinc extraction for the average table test is 72.1 percent. The ratio of concentration is $\frac{c - t}{h - t} = \frac{12.8 - 1}{3.0 - 1} = \frac{11.8}{2} = 5.9$

The reason a higher recovery is expected on a full sized table, over that on a quarter-size, is that some valuable fines can be recovered which had been washed into the tailing zone on the quarter-size table, and a steady feed(not easily attainable in these laboratory tests) will also undoubtedly be a large contributing factor to this expected increased recovery.

PURPOSE OF FOREGOING TABLE TEST

Whether or not a profit can be made by working these South-West Wisconsin tailings depends upon producing as cheaply as possible, at least a 10 percent zinc concentrate by some method cheaper than all flotation. The net return for the zinc in one ton of these tailings at the present price would be about one-half the cost it would take to float them; therefore, the all flotation method would be a losing proposition.

Large capacity concentrating methods such as sluicing and jigging (was actually tried), were thought of, but everything reverts to tabling. Tables are preferred for four reasons, namely: (1) Operating costs are low; (2) The feed (fine sand) to be sent to these tables is the ideal field for them; (3) Everything is in the open i.e., operator can tell at a glance what adjustments are to be made, and can make them very quickly when necessary for best results, which all practically amounts to finger-tip control; and (4) Recovery by tabling is higher in this particular case with the exception of flotation.

It has been proven in laboratory tests that a 15 percent zinc table concentrate can be made on a quarter-size table with a 72 percent recovery; therefore it is not expecting too much to assume a 15 percent concentrate with an 80 percent recovery on a full-size table.

RESULTS OF LABORATORY TABLE TEST

After the final arrangement of riffling was decided upon, as shown on page 26 , 48 pounds of deslimed minus 10 mesh feed was passed over a quarter-size Butchart table. This feed assayed 3 percent zinc and the resulting products were 8 pounds of 12.8 percent zinc cincentrate and 40 pounds of tailing assaying 1 percent zinc.

Two other tests were made as a check. In one case both concentrate and tailing ran slightly higher in zinc and in the other case, both concentrate and tailing ran lower. Since the above figures are the result of an average test, it will be taken as an example.

In these tests the feed was passed over the table once, and the resulting products, namely concentrate and tailing were then sampled after the total 48 pounds had passed over the table. This means the samples were obtained from the table when it was performing at its best and at its worst, and should be a good average. Had samples been taken from the concentrate and tailing zones at about the half-way mark, while the table was in motion, better results would have been obtained, namely higher concentrate, and lower tailing. But the idea was not to get results from performances at their best, but only under good average conditions.

With reference to the table p. 22, a very noticeable feature is that the iron in the tailings is only about one-half of 1 percent lower than the iron in the feed, and that the iron only doubled itself in concentration, while the concentration of the zinc was more than quadrupled. As the writer sees it, there could be two reasons for the behavior of the iron, namely; part of the iron being in a fine sand state, thereby being washed into the tailing zone (most of the zinc was in coarse sand range), and secondly, that the iron occured as a middling but in the form of small specks on the dolomitic gangue, which of course would then be found in the tailing zone.

Another noticeable feature was the effect of table speed on grade of concentrate with no appreciable effect on the grade of tailing. At 280 strokes per minute, a 15.5 percent zinc concentrate was produced, while at 220 strokes per minute a 12.8 and a 11.3 percent zinc concentrates were obtained. The assay of the tailings only varied a few hundredths of 1 percent in each case. This increase in grade may not have been all due to increase in speed, but other factors such as condition of feeding may have had something to do with this also, for the deslimed tailings in these tests were fed to the tables by means of hand held scoops, resulting in periods of steady feed, rushes and lulls.

RECOMMENDED DECK PLAN OF BUTCHART SAND TABLE

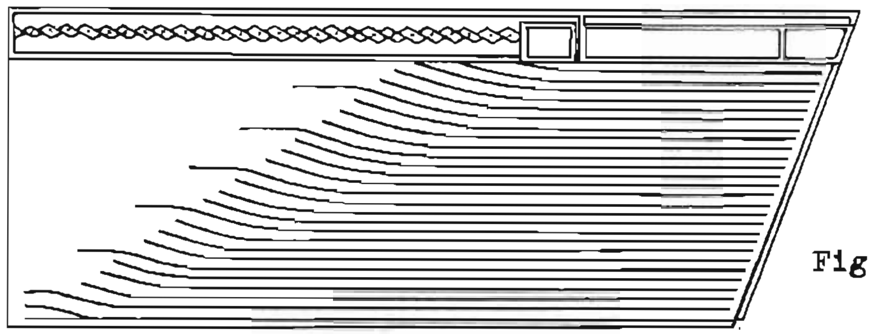


Fig. 3

The above figure is the plan of the deck of one of the eight Butchart tables which will concentrate approximately 600 tons of minus 10 mesh deslimed material.

It should be noted that every fourth riffle should extend 24 inches horizontally beyond a diagonal line through the ends of the remaining shorter riffles. This is not an original design of the writer, but was borrowed from a photograph left at the Wisconsin Mining School by a representative of the manufacturers of Butchart tables.

The above design was adopted after several different riffle arrangements had been tried with actual tests, in order to cope with the wide size range of table feed. The results from a test with the above plan showed not only a higher recovery but also a higher grade concentrate on a quarter-size table.

TABLING OF SLIMES

The laboratory slime table test did not turn out quite as well as expected. The test was made on a quarter-size Deister slime table with a pool riffle design such as shown in figure 4, on page 28.

The result was a 15.5 percent zinc concentrate and a 1.8 percent tailing, from a 4.5 percent zinc slime feed. The percent extraction figures out to be 68 percent.

The weight of the slimes which were fed to the table was only 10 pounds. (All that was available at that time since over 30 pounds saved for this test were destroyed while laboratory was undergoing repairs, and it takes 150 pounds of tailings to give 10 pounds of slime, because there are only 7.5 percent of slimes in the tailings) and the result was that it took nearly all of the 10 pounds to make a good bed and then there was no more feed.

This test is not at all discouraging because two slime tables in actual operation in the southern district were running a 4 percent zinc slime from a sludge pond getting a concentrate assaying a little better than 18 percent, and the tailings assayed 1 percent zinc. This gave an extraction of 80 percent.

Coghill and Anderson made several slime table tests on a quarter-size Butchart table running a typical South-west Wisconsin sludge assaying 5 percent zinc. Their results were even better than the above actual operation results.

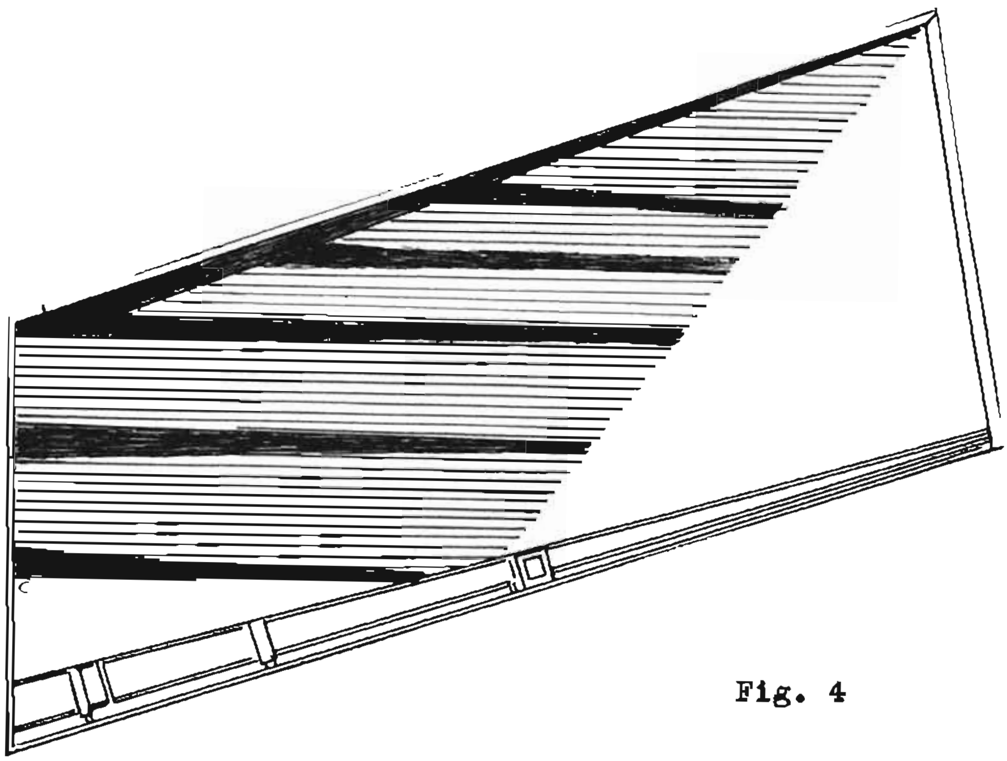


Fig. 4

RECOMMENDED DECK PLAN OF DEISTER SLIME TABLES

DESCRIPTION OF DEISTER SLIME TABLES 5

According to several tests, 7 percent of the original tailings will be slimes. This means that 105 tons of minus 80 mesh material will be concentrated on three slime tables. A plan of the deck of each table is shown in the above figure.

The wide tapering dark marks designate pool riffles. These have a triangular shape, being about three-fourths of an inch high at the head end and tapering to a feather edge. The purpose of these riffles is to form a more or less quiet pool allowing finer concentrates to freely settle, which otherwise would not have this protection.

Proposed Flow Sheet For Production of Low-Grade Concentrate by Tabling

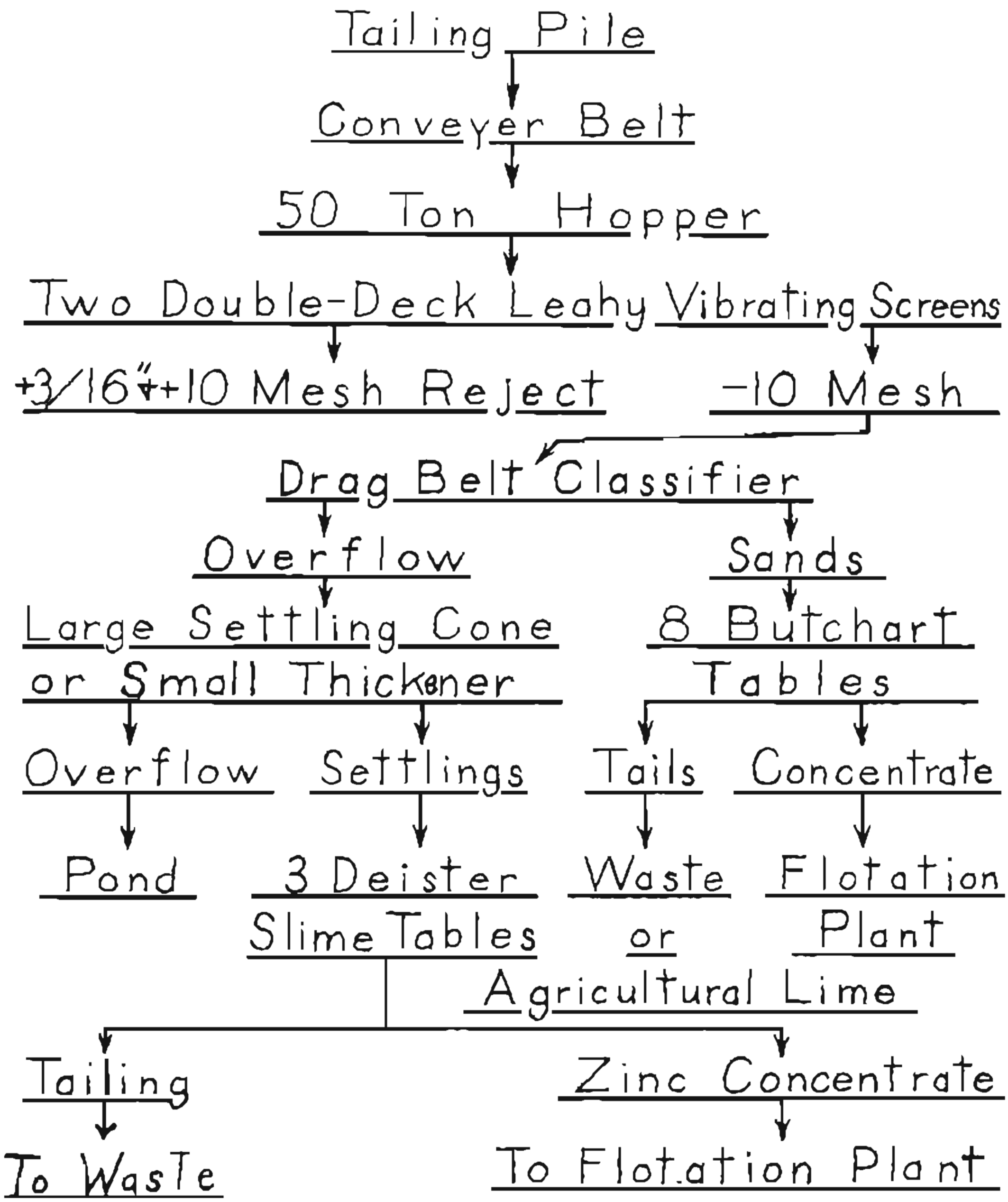


Fig. 5

DAILY OPERATING COSTS OF GRAVITY MILL

Gravity Concentration	Per Ton	Total
Purchase of 1,500 tons tailings	\$ 0.10	\$ 150.00
Handling and Loading	0.025	37.50
Plus 10 mesh and plus 3/16 inch reject disposal (900 tons)	0.013	19.50
Screening	0.009	13.50
Tabling	0.017	25.50
Table Tailing Disposal (500 tons)	0.007	10.50
Water Supply	0.008	12.00
Depreciation	0.008	12.00
Oil, Grease, Waste, and Repairs	0.003	4.50
Miscellaneous	0.003	4.50
Reserve for Future Expense	0.007	10.50
Total Gravity Mill Expense	0.20	\$300.00

The above "Reserve for Future Expense" is to build up a fund for the moving of the gravity mill for a distance of $\frac{3}{4}$ of a mile, at the end of the first three years. Then, not quite two years later, the gravity mill must be moved twice more at one year intervals. This reserve fund will be ample to not only defray the moving expense (three movings), but also the cost of hauling the 100 tons of gravity mill concentrate to the original flotation plant which will not be moved from its first location.

FLOTATION TEST ON ORIGINAL TAILINGS

(Not Previously Concentrated)

Type of ore - - - - - - low grade tails (see following explanation sheet).

Wt. of sample - - - - 2,000 grams

Primary grind - - - - Rolls all thru 20 mesh

Final grind - - - - All thru 65 mesh

Water - - - - - - - Tap

Pulp density - - - - - Three to one

Temperature - - - - - Room

Machine - - - - - - - Denver Sub A

Reagent	To Ball Mill (5 min.)	To Conditioner
H_2O	1,000 lbs. per ton	5,000 lbs. (to shale cell)
Soda Ash	3 lbs. per ton	
Aerofloat	.15 lbs. per ton	

Floated 6 min. to remove shale and lead.

(zinc circuit).

Lime	5 lbs. per ton 2 min.
$CuSO_4$	1.2 lbs. per ton 5 min.
Xanthate (Sodium)	.10 lbs.
Yarmor Pine Oil	.15 lbs. per ton

1st conc. 2 min. 1st mids 4 min. and 2nd mids 8 min.

EXPLANATION OF FOREGOING FLOTATION TEST

The purpose of this test was to try out a more or less standard flotation scheme on an exceedingly low grade ore (original tailings just as from the pile).

As can be seen from the results, this scheme worked almost as well as on a higher grade ore for which it was developed. Practically a 60% zinc concentrate was obtained without cleaning. Had this concentrate been cleaned, even once, it would have assayed 62% or more, as this would have removed some of the iron which ran 2.5%.

The conditioner referred to is the flotation cell itself, with the air-cock closed and impeller speed about one-half of the regular flotation speed. When froth is removed, the machine is run at full speed with air-cock open.

The shale referred to is the oil rock, some of which was not unlocked from the dolomitic gangue. This shale comes off with what little lead there is, in a dirty looking froth in the lead circuit at the beginning of the process. Another fact brought out in this test was that even in spite of such a large proportion of gangue to such a small proportion of zinc sulphide, a dispersing agent such as sodium silicate was unnecessary.

These tailings were from a pile known as Rule Mine Tailings, which is a pile the writer worked on, on other tests.

As can be seen from the results below, a very important point brought out in this test is that there is not much zinc carbonate in the tailings of this northern district (low assay of flotation tailings). Too much of this carbonate would have made this project a failure, since Smithsonite, the zinc carbonate, cannot be recovered by flotation, to my knowledge.

This test was conducted slightly differently from the main flotation tests in this report. The idea was to find out the grades of concentrates in different periods.

FLOTATION RESULTS OF LOW GRADE FEED

(NATURAL TAILINGS)

Product	% Zn	% Fe
Heads	1.60	5.2
Conc.	59.30	3.5
1st mids	36.30	10.5
2nd mids	4.85	15.0
Tails	.12	2.02
Shale	.74	7.1

FLOTATION TEST ON AVERAGE FLOTATION FEED

Type of ore Mixture of sand and slime table concentrates (13.5 percent zinc)

Weight of sample - - - - - - - - - - 2,000 grams

Final grind - - - - - - - - - - - All thru 80 mesh

Water - Tap

Pulp density - - - - - - - - - - - - - - Four to one

Temperature - - - - - - - - - - - - - - - Room

Machine - - - - - - - - - - - - - Denver Sub A

Reagent To Conditioner

Lead and Oil Rock Circuit

Water - - - - - - - - - - - - - 8,000 pounds per ton

Lime - - - - - - - - - - 4 pounds per ton, 5 minutes

Aerofloat (25 percent)- - 0.1 pounds per ton, 2 minutes

Floated 5 minutes to remove shale and lead

Zinc Circuit

Lime - - - - - - - - - - - 5 pounds per ton, 7 minutes

Copper sulphate - - - 1.8 pounds per ton, 3 minutes

Xanthate (sodium ethyl)- 0.15 pounds per ton, 1.5 minutes

Yarmor pine oil - - - - 0.1 pounds per ton, 0.5 minutes

Rougher froth removed for 4 minutes

Scavenger froth removed for 6 minutes.

FLOTATION EXPERIMENT NUMBER ONE

	Weight	Per-cent weight	Per-cent zinc	Percent of total zinc in feed	
Composite	2,000	100	13.53	100	
Lead froth	56	2.8	7.24	1.4	
Cleaner froth	139	6.95	64.78	33.3	rougher froth
Cleaner tails	210	10.5	60.43	47.0	
Scavenger froth	212	10.6	18.6	14.5	
Rougher and scavenger tails	1383	69.1	.74	3.8	

The percent of zinc extraction is 96.2. (This is including that which went over in the lead froth, but this zinc in the lead froth really cannot be considered a total loss because this froth is run over a table, and the lead from this table is a low grade finished product, while this zinc which will be in the middling zone of the table would be returned to the circuit. Further, this percent zinc in the lead froth was high, as four subsequent tests cut this percentage down to an average of 4 percent, which amounts to a zinc content of .6% of the total zinc).

The pH of the rougher and scavenger discarded tailings was 10.4 in this test .

EXPLANATION, RESULTS, AND DISCUSSION OF THE

FOREGOING LABORATORY FLOTATION TEST NUMBER 1

A sample of 2,000 grams consisting of 1,640 grams of 12.8 percent zinc sand table concentrate and 360 grams of 15 percent zinc slime table concentrate all thru 80 mesh, was introduced into a 2,000 gram Denver Sub A Laboratory Flotation machine with 4,000 c.c. of water (4,000 pounds per ton).

The machine was turned on at half speed and the pulp conditioned for five minutes. At the beginning, four grams of lime (4 pounds per ton) were added, and three minutes later .1 gram of twenty-five percent aerofloat (.1 pound per ton). The air cock was then opened and the machine speeded up. The froth, which was now removed for five minutes, had a dirty gray color and contained a small amount of lead and considerable oil rock.

The machine was then retarded, the air cock closed, and the following reagents were added: five grams of lime, four minutes later 1.4 grams of copper sulphate, one and one-half minute after this .1 gram of sodium ethyl xanthate and one minute after this .1 gram of yarmor pine oil. This was all done during a seven minute time interval. The air cock was then opened, the machine speeded up, and the zinc rougher froth was removed for four minutes. This was saved to be cleaned later. This rougher froth consisted of small light gray bubbles, neither tough not brittle.

At the beginning of this period (scavenger froth removal), .4 grams of additional copper sulphate were added. While the machine was running at full speed, the froth was being removed with a paddle. About a minute after the copper sulphate addition, .05 gram of sodium ethyl xanthate were added, the froth being removed uninterruptedly. This scavenger froth was removed for six minutes more. In practise, this froth would return to the head end of the zinc circuit. The bubbles here were lighter in color.

The machine was then drained and a sample of the tailings was taken for pH determination, which was found to be 10.4.

After the machine was cleaned, the rougher froth was added with additional water (pulp density 10 to 1). No additional reagents were added here. The air cock was opened and this cleaner froth was removed for nearly two minutes. The bubbles were exceedingly heavily laden with zinc concentrate and were golden brown in color. The tailings from this cleaner cell would also go back to the head of the zinc circuit.

On subsequent assays, it was found that the cleaning of this rougher froth was unnecessary, since it assayed over 62 percent zinc.

FURTHER DISCUSSION OF FLOTATION TESTS

These tests were conducted so as to conform as nearly as possible to actual plant operating conditions. In actual operation, there would be a mixture of approximately 1,640 pounds of sand table concentrate and 360 pounds of slime table concentrate for every ton of flotation feed; therefore, for every 2,000 gram sample for these tests there were 1,640 grams of sand table concentrate and 360 grams of slime table concentrate.

The timeing in these tests was proportioned, as though there were two rougher, one cleaner, and four scavenger cells in operation.

The high pH (10.4) of the tailings, test number 1, was not harmful but unnecessary. Just as good results were later obtained with a lower pH of 9.4, but the amount of lime used in this test (9 pounds per ton) would have to be used in operation, if not slightly more, because the water used in plant operation taken from an old mine shaft would be slightly more acidic than the tap water used in this test.

In four subsequent tests, the amount of zinc carried over in the lead froth was about one-half of that carried over in test number 1. This was brought about by removing the lead froth only three minutes (2 minutes less) by adding weaker aerofloat, namely 15 percent, instead of

25 percent, and by using a smaller amount of aerofloat.

In a total of five flotation tests with a feed consisting of a mixture of sand and slime table concentrates (13.5 percent zinc), all thru 80 mesh, an average recovery of 95.5 percent was obtained. The highest concentrate assayed 65.14 percent zinc and the lowest ran 63.6 percent zinc. From previous experience with laboratory flotation tests and actual operation, the writer feels that the above results can be almost duplicated. In fact actual plant operation has usually proven out better, but in this case that could hardly be possible.

One thing that has been very noticeable is that an average flotation feed of about 14 percent zinc seems to give better all around results for ores in this district, from both a standpoint of recovery and grade of concentrate.

With reference to the table of results from Flotation Test Number One, it should be remembered that the cleaner concentrate and cleaner tailing together make up the rougher froth. Although it was not necessary to clean the rougher froth from test number one since it assayed 62 percent, it was found that the average assay of the rougher froth from four other tests was 57 percent zinc. This shows that it is imperative to send the rougher froth through one cleaner cell which will then give a cleaner concentrate averaging 64 percent zinc, and the cleaner tailings (average assay

50 percent zinc), of course go back to the head of the zinc circuit to recirculate.

In the event an ore should be encountered which contained more lead than that contained in the feed for these flotation tests, what should be done in order to recover all possible lead and yet keep it practically free from zinc?

The remedy is to add approximately .6 pounds of zinc sulphate and .2 pounds of sodium or potassium cyanide for every ton of ore. These two reagents should be added to the ball mill, and together they will act as zinc and iron depressants in the lead circuit.

A complete analysis of a mixture of flotation concentrates from Wisconsin jig concentrates and table concentrates follows:[4]

Zn	62.83%	Cu	None
Pb	0.76	Bi	"
Fe	2.40	As	"
Cd	0.21	Sb	"
CaO	0.15	Co	"
MgO	0.22	Ni	"
SiO_2	0.34	Al	"
S	32.61	Mn	"
Insol(incl.SiO_2)	0.49	V	"
		Cl	"

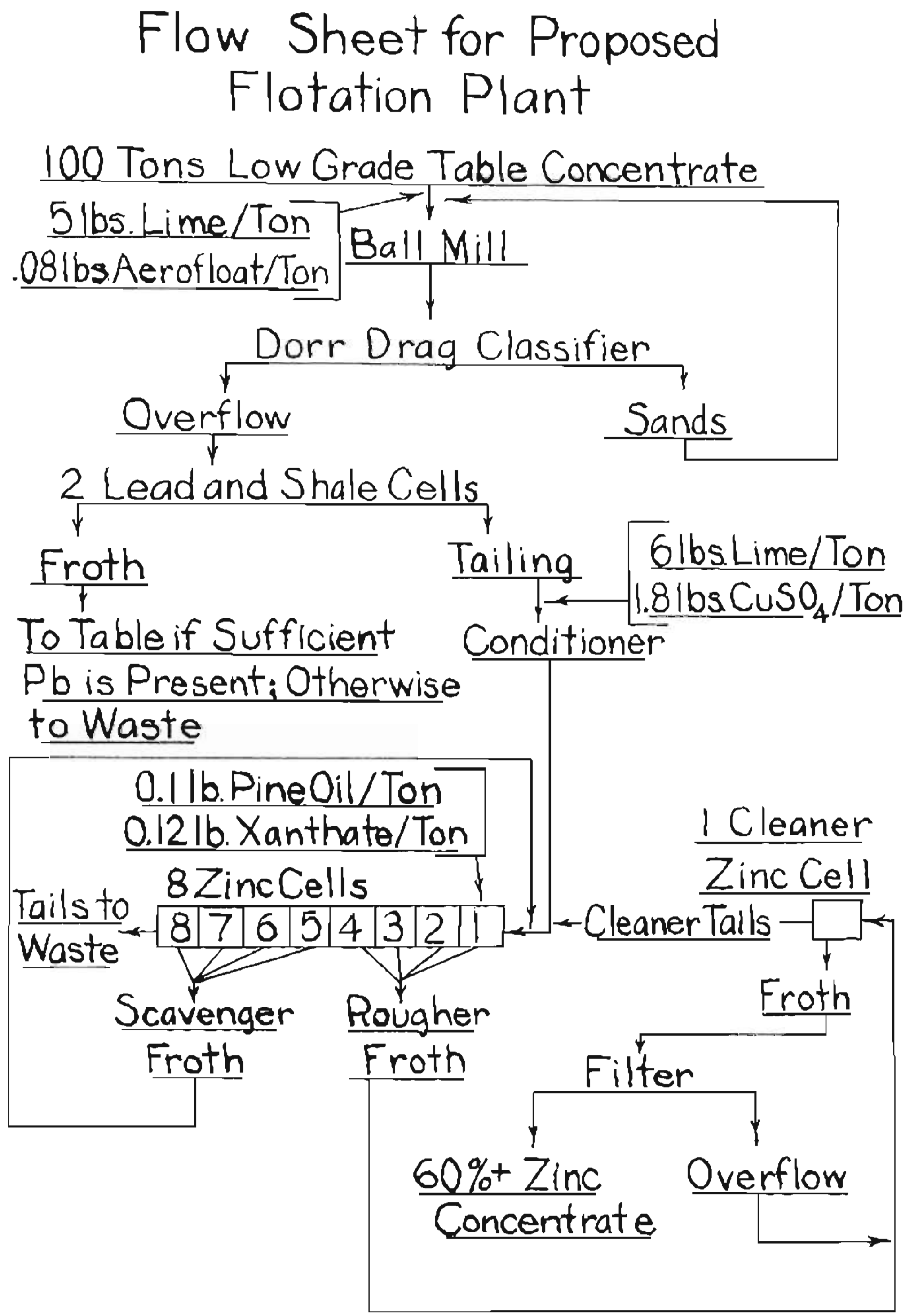

Flow Sheet for Proposed
Flotation Plant
100 Tons Low Grade Table Concentrate
5 lbs. Lime/Ton
.08 lbs Aerofloat/Ton
Ball Mill
Dorr Drag Classifier
Overflow
Sands
2 Lead and Shale Cells
Froth
Tailing
6 lbs Lime/Ton
1.8 lbs CuSO4/Ton
To Table if Sufficient
Pb is Present; Otherwise
to Waste
Conditioner
0.1 lb. Pine Oil/Ton
0.12 lb. Xanthate/Ton
1 Cleaner
Zinc Cell
8 Zinc Cells
Tails to
Waste
8 7 6 5 4 3 2 1
Cleaner Tails
Froth
Scavenger
Froth
Rougher
Froth
Filter
60%+ Zinc
Concentrate
Overflow

Fig. 6

FLOTATION REAGENTS

Two of the noticable facts on observing the Flotation Flow sheet, page 41 , are the large amount of lime used, and that lime is also used to neutralize the water instead of the usual soda ash (Na_2Co_3).

In the past it was thought that if lime were used for neutralizing water before the removal of lead sulphide, it would interfere with the flotation of galena, but several laboratory tests were made with ores containing galena, and it was found that lime did not interfere. This is quite an advantage for this project since lime is much cheaper than soda ash.

This small item alone involves a saving of approximately 8¢ per ton for flotation reagents. This idea of using lime instead of soda ash originated after the study of "Flotation Reagents" in a pamphlet obtained from the Denver Equipment Company , from which a small amount of this data is drawn. Quoting from the above pamphlet, "Lime interferes with the flotation of galena in the presence of ethyl xanthate, thiocarbanilid, or 25% aerofloat. In the above laboratory tests 25% aerofloat was used both as the frother and collector in the lead circuit.

The large amount of lime is necessary to obtain a pH of 9.6 in the zinc flotation tailing. This required high lime, in the writer's opinion, is necessary because of the presence of marcasite, some of which is in a slightly oxidized state, causing formation of H_2So_4 after the addition of water. Were it not for the high lime content of the tailings themselves, still more lime would have to be added in the zinc circuit.

That the dolomitic limestone in the tailings furnishes additional alkalinity can be proved by the fact that one operator in the district attempting the flotation of sludge without any previous concentration cut his lime bill almost in half on the addition of high lime tailings. Lime is used for two reasons, namely, to furnish the necessary alkalinity in both the lead and zinc circuits, and to depress the marcasite in the zinc circuit.

The 15% aerofloat added to the ball mill acts as a frother and a collector in the lead circuit. In this circuit the lead and oil rock are removed in the form of a dirty froth.

Copper sulphate is known as a reactivating agent and is added in the conditioner before the zinc circuit. It precipitates films of copper sulphide on the zinc minerals and thus makes them amenable to flotation.

Sodium ethyl xanthate is used as the collector in the zinc circuit. It is the cheapest xanthate, and it is found to be just as effective as the more expensive xanthates, such as amyl and pentasol. The xanthate is added in the first flotation cell, because this gives it sufficient time to act, and besides, if it were added sooner it would be apt to collect more marcasite than the small amount which it already does.

Pine oil, the frother for the zinc circuit is also added to the first zinc cell. It is unnecessary to add it sooner because it acts fast, It seems as though most operators prefer cresylic acid as a frother, but in laboratory tests, and in actual operation, the writer found it to be less effective than pine oil, and further pine oil is considerably cheaper, and much more pleasant to handle.

It can be noted that the reagents chosen are not only effective but the cheapest in their respective class. This is a decided advantage because the idea behind this project is low operating costs.

FLOTATION PLANT OPERATION

With reference to the flow sheet of the proposed flotation plant, it is seen that the lead froth from the lead cells is not cleaned or filtered. The amount of lead even in flotation feed from run of mine ores is so small (mostly removed in the gravity mill), that it does not warrant this extra expenditure.

This lead froth, containing considerable oil rock and some iron, is very dirty and assays about 15 percent iron and 2.6 percent lead. The usual practise which was inaugurated by the first flotation plant in this district is to run this lead froth over a table, thereby grading it up to 60 percent or more, which is then a saleable product. The tailings from this table are then sent back to the first lead cell to recirculate.

Another noticeable feature is that no thickener for the cleaner zinc froth before going to the filter is needed. This was found unnecessary from actual practise.

The first flotation plant in the district used nine rougher zinc cells and one cleaner, the froth from all nine rougher cells going to the one cleaner. The feed to the zinc circuit was not conditioned, but went from the classifier to the first zinc cell direct. This arrangement worked well for a fresh run of mine ore, but about a year ago, while this same plant was running some sludge with fair success, the writer suggested a quick and inexpensive change. This was to use a small thickener as a conditioner which worked well by keeping the pulp level fairly low, and a second change was to use the first four zinc cells as roughers, and the last five as scavengers. This change resulted in a slight increase in recovery and a higher grade of concentrate.

The conditioner was used to give the slightly oxidized zinc mineral more time to be acted upon by the reagents. By using the first four zinc cells as roughers, they cut down the load in the cleaner cell, besides giving it a much higher grade froth to work on.

The pulp density in the zinc circuit gives the best results at a four to one ratio, with a pH of the tailings at 9.5.

Although a total of nine cells for the zinc circuit are shown on the flow sheet, laboratory tests have shown that a 95 percent recovery and a 64 percent zinc concentrate can be obtained in a time interval of 10 minutes. According to the manufacturers a pulp of this density would take on an average of about two minutes per cell for circulation; therefore, it is the writers' opinion that this plant could operate with seven zinc cells, including one cleaner. It is recommended to use the first two cells as roughers

(since almost 80 percent of the total zinc in the feed would be removed here), and the last four cells as scavengers, leaving the one remaining cell as the cleaner.

In the laboratory flotation test described on another page, the lead and oil rock froth assayed 7.24 percent zinc. This was the highest zinc assay of all the five laboratory flotation tests made. This was because the lead and oil rock froth was removed for five full minutes which was too long. In later tests this time was cut to three minutes resulting in a 4 percent zinc assay. The amount of this lead and oil rock froth, however, is so small that even if it assays 4 percent zinc, this amounts to only 0.6 of 1 percent of the total zinc in the feed.

In order to cut down this amount of zinc going over with the lead froth, the writer recommends only two lead cells in actual practise, which will be equivalent to almost four minutes of time. This will be ample time to remove practically all of the lead, and from experience it is known that the zinc assay of this lead and oil rock froth will not be more than 2 percent in actual plant operation.

DAILY OPERATION COSTS FOR FLOTATION

OF 100 TONS OF TABLE CONCENTRATES

Labor and Superintendence	$ 36.00
Power	25.00
Depreciation and Amortization	27.50
Property Insurance	1.00
Workman's Compensation Insurance	1.00
Repairs and Replacements	3.00
Property Tax	.50
Income Tax	15.00
Flotation Reagents	29.00
Oil, Grease, and Waste	.50
Balls and Liners (Wear)	2.00
Miscellaneous Supplies	1.00
Disposal of Concentrates including Trucking and Trimming Cars	12.00
Assaying	3.00
Freight on Concentrate	54.00
Office Expense	1.00
Miscellaneous	3.50
Total	$215.00

It is to be understood that the above costs are for 100 tons of flotation feed which makes the flotation cost per ton of original tailing $195/1,500 tons, or $.1433

DISCUSSION AND RESULTS OF ONE DAYS' OPERATION

By means of a scraper system, 1,500 tons of tailings will be elevated into a 50 ton hopper. This hopper will then feed two double-deck vibrating screens which will reject 58 percent (870 tons), letting 42 percent (630 tons) pass through.

These 630 tons of minus 10 mesh material will then go to a mechanical classifier. The fine sand from the classifier will then be the feed to the eight sand tables, amounting to 520 tons, while the overflow from the classifier containing the slime (110 tons) will go to a large cone, or small thickener. It is assumed that about 5 tons of slimes will be lost in the overflow from the thickener or cone, leaving 105 tons of thickened slimes as the feed to three slime tables.

The 520 tons of feed to the eight sand tables assay 2.97 percent zinc. Assuming an 80 percent extraction (which is more of a fact than an assumption), the resulting product from the sand tables will be 82.3 tons of 15 percent zinc concentrate.

The 105 tons of feed (4.49 percent zinc) to the slime tables will give 19.5 tons of 18 percent zinc concentrate.

The feed to the flotation plant will then consist of 101.8 tons with an average assay of 15.59 percent zinc.

With a 93 percent recovery (actually is 95 percent but will be considered 93 percent as a safety factor), the finished product from the flotation plant will be 23.8 tons of 62 percent zinc concentrate. With a 60 percent concentrate valued at $34.00, a 62 percent concentrate will be given a bonus of $1.50 per ton; so the daily return will be 23.8 X 35.50 = $844.90. The total daily expense will be 1,500 X .3433 = $515.00. The net daily return at present price is $329.90.

It should not be forgotten that 870 tons rejected by the 3/16 inch and 10 mesh (.07 inch), will make an excellent product for road surfacing, bringing in additional income. Another small source of income would be the table tailings (500 tons daily), which should be in demand for agricultural lime, being the ideal size namely, minus 10 plus 65 mesh.

A brief summary of the above discussion for one days' operation is as follows:

Sale of 23.8 tons 62% zinc concentrate @ 35.50=		$844.90
Gravity mill expense 1,500 tons @ $0.20=	$300.00	
Flotation plant expense $215 daily equivalent to $0.1433 per ton of original 1,500 tons	= 215.00	
Total Expense		515.00
Net Daily Income		$329.90

MAXIMUM CAPITAL INVESTMENT FOR THIS PROJECT

The gravity concentration plant will not cost more than $15,000.00. This will include eleven tables, two double-deck vibrating screens, scraper system for handling tailings at piles, conveyor belt system for moving tailings from piles, pumping equipment, power installation (obtained from electric high line), elevators for tailing and concentrate, mechanical classifier, and the shelter or building.

The maximum cost for the flotation plant will not be over $35,000.00, including the building. This figure was arrived at after carefully checking costs given by the Denver Equipment Company, of Denver, Colorado.

So for a maximum investment of $50,000.00, a net average return of at least $250.00 a day can be expected for 200 days per year for 6.5 years.

The present set-up is even better than that. As an inducement for someone to begin a project such as this, the owners of one and one-half million tons of these tailings offer to throw in nearly a thousand tons of high grade ore which was cleaned up after the mines were shut down. The returns from this ore alone would be over $5,000.00.

Sufficient water for both gravity and flotation plants can be readily obtained from the old shaft near the one million ton pile.

AGRICULTURAL LIMESTONE AND ITS CONNECTION WITH THIS PROJECT

According to present indications, it appears that approximately 500 tons of dolomitic limestone will be produced daily by tabling 600 tons of minus 10 mesh tailings. This, according to various agricultural bulletins is the most desired and effective size.

The term magnesium limestone is applied to one that contains both carbonate of lime and carbonate of magnesia. The term dolomitic limestone is also applied to the same material. The value of this limestone for agricultural use is usually considered as equal to that of ordinary limestone, that is, one containing 85% of lime and magnesia carbonates would be equal in value to an ordinary limestone containing 85% of carbonate of lime.6

With reference to page12 , the $CaMg(CO_3)_2$ runs 77.5% but on tabling this material, most of the Zn, Fe, and S will be removed, hence the $CaMg(CO_3)_2$ would be graded up to at least 80%.

Since the table tailings almost approach the true agricultural lime in size and content, tney should be in considerable demand, especially if they would be sold for 25¢ per ton. But since this added income would be small and uncertain, it will not be considered as an additional source of profit in this venture.

IMPORTANCE OF SAMPLING

In view of the fact that a project of this kind requires a considerable initial investment, the method and carefullness of sampling cannot be stressed too strongly.

A sample consisting of several tons used throughout these tests was obtained from several test pits in different parts of each pile. Five piles are represented in this large sample, and the amount of sample from each pile was in proportion to the size of the pile.

None of the test pits were over four feet deep, but this is considered on the side of safety in that operators and former workers, alike, claim that the lower values of the larger piles are near the surface and that deeper down in the piles the tailings will assay higher in zinc. Different spots on the pile are also richer in zinc even near the surface.

It is said that in the early formation of two of these piles some of the night shifts were very inefficient andsome mornings on the checkup it was difficult to distinguish between concentrate and tailing. The latter statement may be somewhat exaggerated, but that the night shifts were frequently asleep on the job seems to be the general opinion.

In obtaining the above samples care was taken not to dig these pits in the richer parts of the piles, nor at places where tailing elevators were formerly located.

For a more truly representative sample, it would be adviseable to use one of the methods in use in the Tri-State Field, which involves churn drilling. In this method a heavy casing is kept not over twenty-four inches in advance of the drill bit. No water is used in order to avoid losing valuable fines. To remove the dry tailings from the inside of the casing, a tool similar to the horn-socket fishing tool is attached to the end of a sinking bar and worked up and down inside the casing, packing the dry tailings into the socket tool. The dry tailings pack well and can then be raised to the surface and removed by pounding on the socket.[7]

CONCLUSION

All the doubtful points as to the success of this project have been practically taken care of in this report.

One logical question which could still be asked is, "Is the sample from which these tests were made a representative one?" As was mentioned before, this sample came from various test pits near the surface of the several piles and should be on the side of safety, since it is almost a fact that the tailings are richer in zinc deeper down in the piles. Further, the writer checked the assays of this sample with several spout sample assays made while the formation of some of these same piles was in progress.

Another logical question is, "If this project looks so good, why didn't someone try it before?" During the long depression, it is reasonable to assume that no one cared to sink $50,000.00 into a waste product, and before the depression, small and large operators alike were successful in locating small ore bodies; so why bother with tailing piles?

Since the depression, only a few people besides the writer have thought about it, and these few who would like to begin this project according to the writers' scheme, cannot raise the necessary capital.

According to the figures on another page, a net daily profit of $330.00 can be obtained when the market price for a 60 percent zinc concentrate is $34.00 per ton, which is lower than the average.

Most of the flotation operating costs were obtained from actual operating conditions, and the freight charge on the flotation concentrates was obtained from the Vinegar Hill Zinc Company of Platteville, Wisconsin.

Some of the operating costs in the gravity mill were assumed, but most of these could be checked by data furnished by various manufacturing companies.

On the whole, the writer feels that all operating costs are figured sufficiently high to allow for any possible contingency, which may occur.

A quick check-up on recovery is as follows: 76.3 percent of the zinc is recovered in the screening operation, 80 percent in the gravity mill, and 93 percent will be assumed as the flotation recovery, although it actually was higher. This gives an average recovery of 56.7 percent.

A rapid check-up on this whole project for finding the minimum income per ton of natural tailings is to take 50 percent of the zinc content and multiply this by 52 percent of the E. St. Louis price per pound.

A still shorter check, giving the same results as above is to move the decimal point of the assay of natural tailings one place to the right. This gives the recoverable pounds of zinc per ton of tailings, then multiply this by 52 percent of the E. St. Louis price per pound.

The minimum net profit per day can then be found by deducting $0.34 from the above minimum income per ton figure.

As an example, these tailings assayed 1.88 percent zinc. By moving the decimal point one place to the right gives 18.8 pounds recoverable zinc. If the E. St. Louis price in 5 cents per pound, 52 percent of this is 2.6 cents. 18.8 X $0.026 = $0.4888. $0.4888 minus $0.3433 = $0.1455 as a minimum profit per ton of natural tailings at this price.

As an after-thought, it would be strongly advisable to have a set of high speed rolls handy shortly after this project is started, so as to divert the reject from the 3/16 inch screen or the reject from the 10 mesh, or both, when these aforementioned rich spots are encountered. These spots can be easily recognized when encountered, because of the larger amount of pure pieces of coarse sphalerite.

The writer is willing to wager that there will be at least 200 days of this coarse reject milling which means only about 600 tons of tailings daily need be handled instead of 1,500, with an increase of at least $70 daily profit even after considering the added cost of crushing.

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