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Region I, Alaska District

SPECIAL REPORT
ON
LODE TIN MINING AND MILLING OPERATIONS
LOST RIVER MINE, SEWARD PENINSULA, ALASKA

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APPENDIX A
LABORATORY BENEFICIATION

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APPENDIX A - LABORATORY BENEFICIATION

INTRODUCTION

To present a clear picture of the nature of the ore and the problem of concentration involved, the description of laboratory testing conducted on Lost River ore has been summarized. For the sake of clarity, discussion of laboratory studies has been divided into four parts:

1. Lode, ore, Bureau of Mines, Rolla, Missouri.
2. Placer concentrate, Bureau of Mines, Albany, Oregon.
3. Lode ore, Bureau of Mines, Juneau, Alaska.
4. Lode ore, A. W. Fahrenwald, Moscow, Idaho.

LABORATORY BENEFICIATION STUDIES, LODE ORE, ROLLA LABORATORY

Ore Tested

During the course of the Bureau of Mines drilling campaign at Lost River 1942 to 1944, a number of samples were obtained from existing adits and surface trenches; these were submitted to the Rolla, Missouri station of the Bureau of Mines for evaluation and beneficiation testing^{1/}. The ore submitted comprised samples from the Cassiterite, Ida Bell, and Quartz-Porphry dikes and from the Greenstone Lode. A composite sample was prepared to represent the type of ore available for possible production.

Physical Character

A microscopic examination of the composite sample revealed that the minerals of economic value, listed in order of abundance were: cassiterite, wolframite, scheelite, arsenopyrite, sphalerite, galena, and molybdenite. The gangue was composed chiefly of quartz, muscovite, chlorite, beryl, pyrite, oxides of iron, and clay minerals.

The cassiterite occurred mainly in amber to dark transparent and translucent crystal fragments. Most of it was freed by minus 14-mesh grinding, but a portion of the cassiterite was intimately associated with wolframite and iron oxides.

The wolframite occurred largely as dark brown to black platy grains. Some was associated with arsenopyrite, pyrite, scheelite, and calcite. Grinding to minus 14-mesh was estimated to be sufficient to liberate most of the wolframite.

^{1/} H. E. Heide, Investigation of the Lost River Tin Deposit, Seward Peninsula, Alaska: Bureau of Mines Rpt. of Investigations 3902, 1945, pp. 43-55.

Chemical Character

A partial chemical analysis of a representative portion of the ore tested is shown in table A-1.

TABLE A-1. - Chemical analysis

Assay, percent							
Sn	WO ₃	Fe	Zn	As	Pb	Mo	
1.02	0.29	2.79	0.18	0.10	0.10	0.03	

Treatment Methods

The laboratory investigation included a study of table concentration methods and flotation of sulfides from table concentrate, and a preliminary study of separation of tin and tungsten by magnetic and chemical separation procedures.

Table Concentration

A sample of ore was crushed to pass 14-mesh and hydraulically classified into a slime product and 4 sand products estimated to approximate 14- to 35-mesh, 35- to 65-mesh, 65- to 150-mesh, and 150- to 200-mesh. The two coarsest sand products were tabled to produce finished concentrates and tailings with high tin content. These tailings were combined and crushed to pass 65-mesh then hydraulically classified into the next 2 sizes of sand and slime.

The sand products were tabled on a laboratory shaking table to produce finished concentrates, middlings, and tailings. The middlings were combined, ground to minus 200-mesh, and mixed with the slime fraction. The combined middling and slime was tabled to yield concentrate, middling, and tailing.

Results of tabling treatment are shown in table A-2.

TABLE A-2. - Tabling Lost River ore

Product	Weight, percent	Assay, percent		Distribution, percent	
		Sn	WO ₃	Sn	WO ₃
Comb. conc.	1.91	37.14	14.80	75.7	85.9
Middling	3.04	2.16	0.59	7.0	5.4
Tailing	95.05	0.17	0.03	17.3	8.7
Calc. head	100.00	0.94	0.33	100.0	100.0

By tabling ore crushed to minus 14-mesh and hydraulically sized with middling re-treatment, 75.7 percent of the total tin and 85.9 percent of the tungsten were recovered in a combined concentrate that assayed 37.14 percent Sn and 14.80 percent WO_3 . An additional 5 to 7 percent of the tungsten and tin reported in a final middling product. In normal plant operation the final middling would be recirculated; the circuit would eventually be equalized and the middling would pass into either the concentrate or the tailing.

The gangue mineral in the concentrate consisted largely of free particles of topaz, but much of the sulfide content of the ore was also concentrated. The concentrate contained 6.2 percent Fe, 3.1 percent As, 1.1 percent Pb, 0.3 percent Zn, and 0.10 percent Mo in addition to the tin and tungsten shown in table A-2.

Flotation of Sulfides

Removal of the bulk of the sulfides from the gravity concentrate was effected by flotation. The concentrate was stage-ground to minus 65-mesh and the pulp was conditioned at 25 percent solids in a laboratory flotation cell. Sulfides were floated with a xanthate collector and were cleaned once. Results are shown in table A-3.

TABLE A-3. - Flotation of sulfides

Product	Weight, percent	Assay, percent						Distribution percent	
		Sn	WO_3	As	Fe	Pb	Zn	Sn	WO_3
Sulfide conc.	11.9	0.42	0.95	30.7	30.4	4.1	1.5	0.1	0.8
Tin-Tungsten tail	88.1	42.15	16.70	0.05	2.9	0.7	0.2	99.9	99.2
Calculated feed	100.0	37.2	14.8	3.7	6.1	1.1	0.35	100.0	100.0

By flotation, most of the sulfide minerals were removed from the gravity concentrate with only slight tin and tungsten loss. Insufficient sample was tested to attempt recovery of separate lead and zinc concentrate.

Magnetic Separation of Tin and Tungsten

A series of magnetic separation tests was made on concentrates both with the sulfides remaining and with the sulfides removed by flotation. Pre-treatment of the concentrate by oxidation roasting also was investigated. All tests gave unsatisfactory results. Best results were obtained by using a high-intensity ferro-filter type separator on concentrate without prior removal of sulfides or roasting pre-treatment. By this method, 96.5 percent of the tin in the feed was recovered in a product that assayed 42.7 percent Sn and 9.8 percent WO_3 . Only 44 percent of the tungsten was present in the magnetic product at a grade of 44.1 percent WO_3 and 8.8 percent Sn.

Chemical Separation of Tin and Tungsten

Preliminary investigation was made of two chemical methods of tungsten extraction from the tin-tungsten concentrate; these two methods were, (1) sintering with soda ash and solution with water and (2) leaching with sodium hydroxide solutions without previous treatment. Results indicated that, properly controlled, either method will extract virtually all of the tungsten present with extraction of only a minor amount of tin. Costs would be high by either method because an excess of sodium carbonate or sodium hydroxide is required and is lost on subsequent acidification treatments.

Suggested Flowsheet

Based on the laboratory investigation, the metallurgical staff of the Rolla station suggested the flowsheet shown in figures A-1 to A-4 inclusive, for the treatment of Lost River ore.

It will be noted that the flowsheet was designed for treatment of 500 tons of ore per day, and included refinements that would not be economic for a plant of smaller capacity. The suggested slimes plant was similar to that in operation at the Sullivan concentrator of the Consolidated Mining and Smelting Company of Canada.

Product	Weight percent	Assay percent	Loss percent				
Slimes conc.	11.0	20.0	30.0	4.0	1.0	0.5	0.2
Slimes tail	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Flotation tail	0.5	0.5	0.5	0.5	0.5	0.5	0.5

The loss of tin and tungsten in the slimes tail is estimated to be 0.5 percent of the total tin and tungsten in the concentrate. This loss is due to the fact that the slimes tail is not treated with sodium hydroxide solution.

Chemical Separation of Tin and Tungsten

A series of laboratory experiments were made on concentrates from the Rolla station with the following results: (1) The loss of tin and tungsten in the slimes tail is estimated to be 0.5 percent of the total tin and tungsten in the concentrate. This loss is due to the fact that the slimes tail is not treated with sodium hydroxide solution. (2) The loss of tin and tungsten in the flotation tail is estimated to be 0.5 percent of the total tin and tungsten in the concentrate. This loss is due to the fact that the flotation tail is not treated with sodium hydroxide solution. (3) The loss of tin and tungsten in the slimes conc. is estimated to be 11.0 percent of the total tin and tungsten in the concentrate. This loss is due to the fact that the slimes conc. is not treated with sodium hydroxide solution.

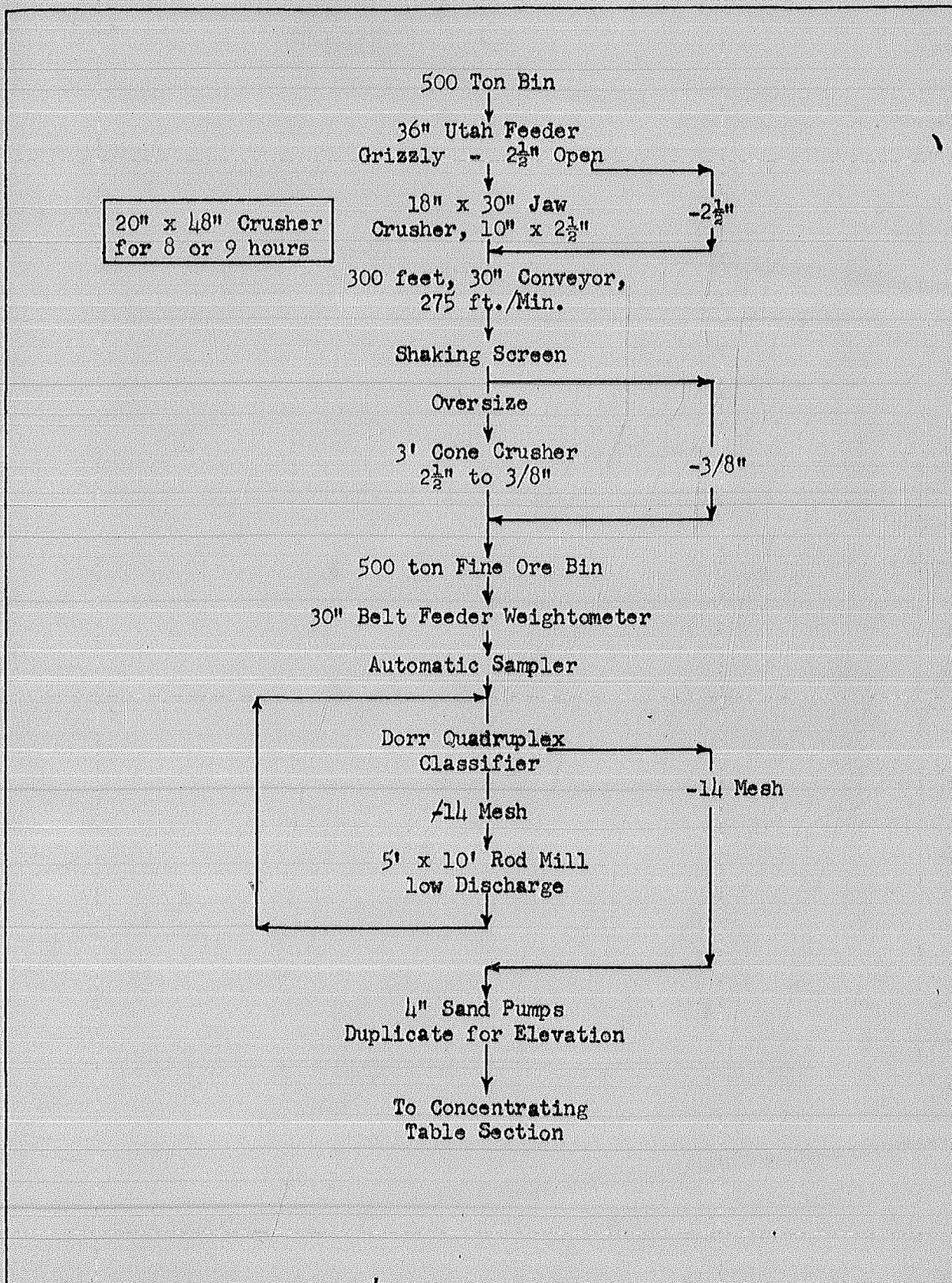


Figure A-1. - Flowsheet for crushing section

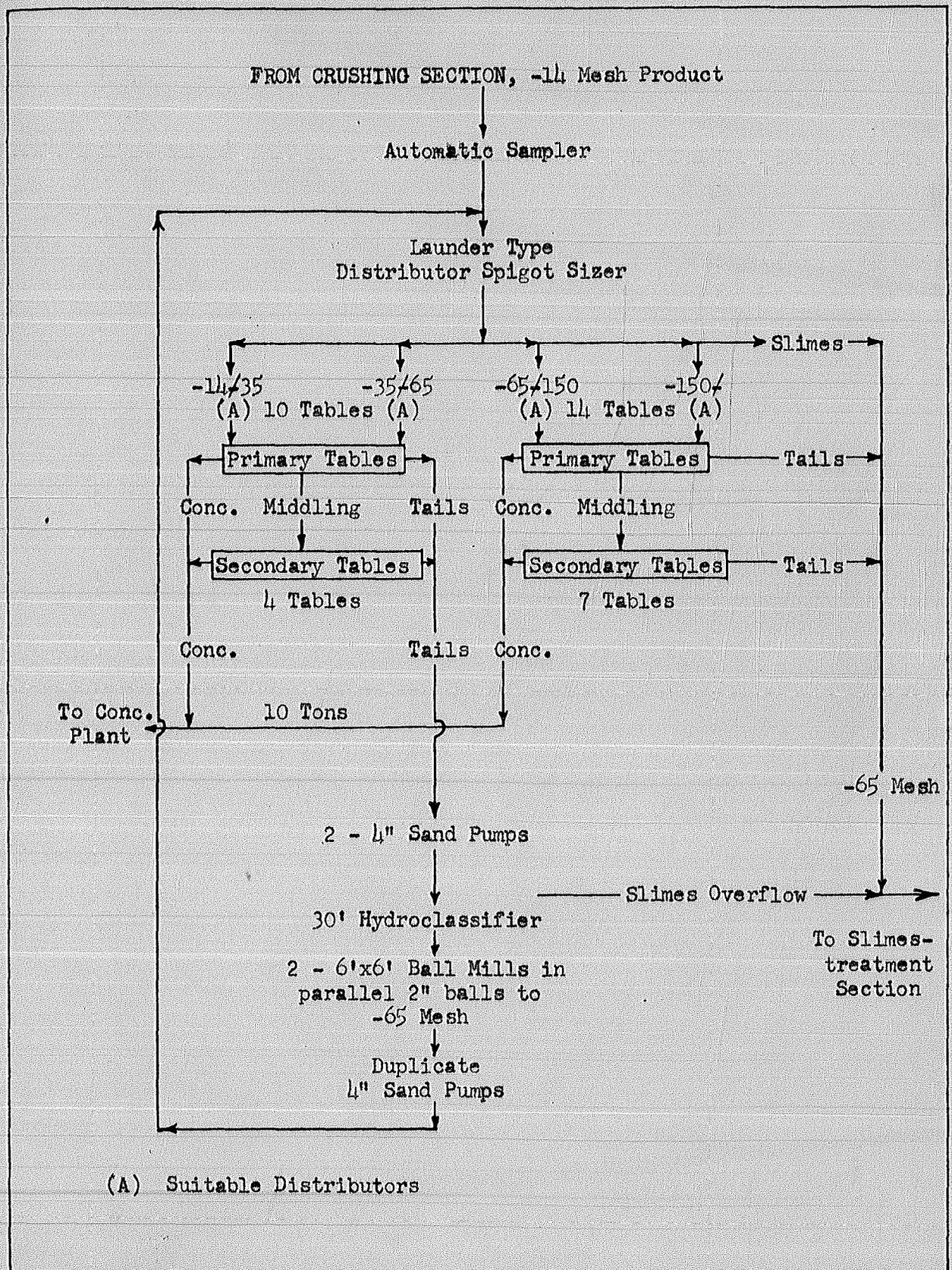


Figure A-2. - Flowsheet for concentrating-table section

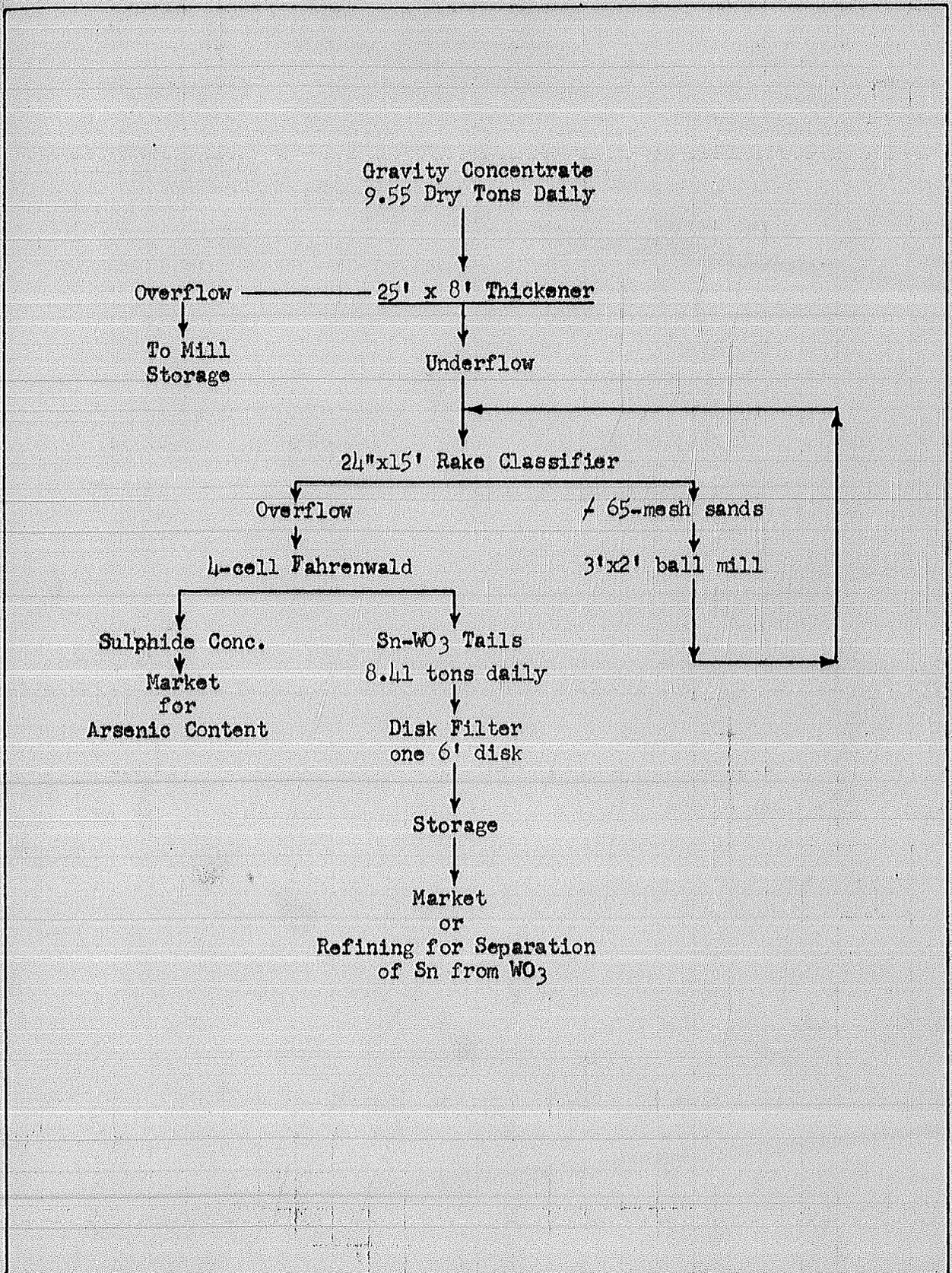


Fig.A-4.0 - Flowsheet for flotation of gravity concentrate

LABORATORY BENEFICIATION STUDIES, PLACER CONCENTRATE, ALBANY LABORATORY

Ore Tested

Very little information is available concerning the methods, concentration techniques, and recoveries of the placer mining operations at Lost River in 1949. In May 1950, the United States Tin Corporation submitted a sample of sluice box concentrates to the Bureau of Mines laboratory, Albany, Oregon for tests to determine the feasibility of further concentration by jigging and spiral methods^{2/}.

Physical Character

The sample submitted contained cassiterite, wolframite, and scheelite in a gangue dominantly composed of magnesium-bearing calcite, some amblygonite, topaz, limonite pseudomorphs, a small amount of titaniferous magnetite, and a trace of fluorite and tourmaline.

Most of the cassiterite and wolframite in the sample was present in the plus 48-mesh fractions. The bulk of the scheelite was determined to be in the minus 65-mesh material; much of it occurred as tiny inclusions in cassiterite grains.

Chemical Character

Chemical analysis showed that the product contained 5.4 percent Sn and 0.77 percent WO₃.

Spectrographic analysis revealed the presence and approximate quantities of the metals indicated in table A-4.

TABLE A-4. - Spectrographic analysis

Al	Bi	Ca	Cu	Pb	Mg	Sn	W	Cr	Fe	Mn	Si	Ti	V	B	Na	Zr
C	E	A	G	D	C	A	D	E	C	D	C	E	E	E	E	E

Legend: A - more than 10 percent E - 0.01 to 0.1 percent
 B - 5 to 10 percent F - 0.001 to 0.01 percent
 C - 1 to 5 percent G - less than 0.001 percent
 D - 0.1 to 1 percent

Treatment Methods

The testing investigation conducted on this sample was limited to sizing, jigging of coarse material, and table and spiral concentration of the minus 9-mesh fraction.

^{2/} Memo R. R. Wells to J. D. Bardill: Mineral Dressing Report, Beneficiation of U. S. Tin Corp. tin-tungsten sluice box concentrate, Unpublished, June 22, 1950.

Sizing

The ore as received was screened on a 9-mesh shaking screen and portions of the undersize and oversize were prepared for analysis. Results are shown in table A-5.

TABLE A-5. - Sizing

Product	Weight, percent	Assay, percent		Distribution, percent	
		Sn	WO ₃	Sn	WO ₃
Oversize	57.5	6.1	0.51	65.0	37.7
Undersize	42.5	4.45	1.14	35.0	62.3
Calculated head	100.0	5.4	0.77	100.0	100.0

Jigging

The plus 9-mesh portion of the sample was crushed to minus $\frac{1}{2}$ -inch and screened on a 9-mesh sieve. The oversize was treated in a laboratory, Harz-type hand jig. No treatment was given the screen undersize. Results are shown in table A-6.

TABLE A-6. - Jigging plus 9-mesh fraction

Product	Weight, percent	Assay, percent		Distribution, percent	
		Sn	WO ₃	Sn	WO ₃
Concentrate	19.95	16.0	1.5	63.3	58.7
Tailing	52.29	1/0.1	.1	1.0	10.2
Screen undersize	27.76	6.5	.57	35.7	31.1
Calculated head	100.00	5.04	0.51	100.0	100.0

1/ Less than.

By jigging, excellent recovery of tin was made; over 98 percent of the tin in the fraction treated was recovered in a concentrate assaying 16.0 percent Sn. The treatment recovered 85.2 percent of the tungsten in the jig feed at 1.5 percent WO₃ grade. The low grade of concentrate was attributed to treatment of insufficient sample to properly build a bed in the jig. In plant practice, crushing to minus $\frac{1}{2}$ -inch probably would be unnecessary.

Spiral Concentration

The original minus 9-mesh material was treated, without further crushing, in the laboratory Humphreys spiral concentrator. By this method 71 percent of the tin and 79 percent of the tungsten in the feed were recovered in a concentrate that assayed 15.0 percent Sn and 3.7 percent WO₃.

A second spiral test was run on the minus 9-mesh ore crushed to pass a 20-mesh sieve. Results are shown in table A-7.

TABLE A-7. - Spiral concentration, minus 20-mesh

Product	Weight, percent	Assay, percent		Distribution, percent	
		Sn	WO ₃	Sn	WO ₃
Concentrate	27.9	12.6	2.1	94.5	81.5
Middling	3.8	.64	.08	.6	.4
Tailing	57.9	.13	.016	2.0	12.9
Slime	10.4	1.03	.36	2.9	5.2
Calculated feed	100.0	3.72	0.72	100.0	100.0

Table Concentration

Table tests were run to parallel those made with the spiral. Table treatment of uncrushed minus 9-mesh material recovered 84.5 percent of the tin and 93.9 percent of the tungsten in a concentrate that assayed 31.1 percent Sn. Results of shaking table treatment of the material crushed to minus 20-mesh are shown in table A-8.

TABLE A-8. - Tabling, minus 20-mesh

Product	Weight, percent	Assay, percent		Distribution, percent	
		Sn	WO ₃	Sn	WO ₃
Concentrate	6.5	51.6	8.6	78.9	81.7
Middling	3.3	13.9	1.9	10.8	9.3
Tailing	80.4	.39	.02	7.4	2.3
Slime	9.8	1.29	.47	2.9	6.7
Calculated feed	100.0	4.25	0.68	100.0	100.0
Combined concen- trate & middling	9.8	38.9	6.3	89.7	91.0

Summary of Results

The sluice box concentrate sample submitted for testing was found to be amenable to up-grading by gravity methods with high recoveries of tin and tungsten. Tabling produced cleaner concentrates than spiral treatment. It was assumed, however, that a single or double re-treatment of the spiral rougher concentrates would produce a product of suitable grade. Such treatment was not performed in the laboratory because of insufficient sample.

As the result of these tests and of a similar series of tests made by the Denver Equipment Company, the U. S. Tin Corporation installed two double compartment jigs and three Humphreys spiral concentrators on their placer operation at Lost River. During the 1950 and 1951 seasons, using this equipment, a total of 128.5 tons of concentrate was produced at an average grade of 54.6 percent Sn and 9.3 percent WO₃.

LABORATORY BENEFICIATION STUDIES, LODE ORE, JUNEAU LABORATORY

Ore Tested

A total of eight ore samples, as well as several samples of mill products were submitted to the Juneau laboratory for beneficiation testing from the period April 1952 to October 1954. For the sake of brevity, this report summarizes the results of investigations conducted on the two samples most nearly representative as to grade and character of the feed to the mill during the same period. They have been designated samples 1 and 2 for simplicity of presentation.

Physical Character

Sample 1:

This sample essentially contains kaolinite, white topaz, sericite, less fluorite and a magnesian-bearing calcite; some quartz, fine-grained cassiterite, chlorite, limonite, altered feldspar, muscovite; and small amounts of pyroxene, amphibole, wolframite, apatite, pyromorphite, and scheelite. Also present are trace amounts of arsenopyrite, pyrite, malachite, cerussite, and tourmaline.

The cassiterite ranges in color from nearly colorless or light amber to nearly black. The small amount of pyromorphite occurs as resinous, greenish aggregates associated with cerussite.

Maximum liberation of the cassiterite and tungsten minerals occurs in the plus 100-mesh fraction. Some of the coarse wolframite is liberated in the plus 48-mesh fraction; the coarser cassiterite grains are freed in fractions up to plus 20-mesh. The bulk of the cassiterite appears to occur in the minus 48-mesh 100-mesh size range.

Sample 2:

This sample contains quartz, kaolinite, sericite, altered feldspar, fluorite, a magnesian-bearing calcite; some altered orthopyroxene and amphibole, chlorite, fine-grained cassiterite, limonite; and small to trace amounts of wolframite, pyrolusite, scheelite, cerussite, galena, apatite, altered pyrite, arsenopyrite, smithsonite, chalcopyrite, malachite, and bismuthinite.

Maximum liberation occurs in the minus 200-mesh fraction, although there is only a small amount of locking of the cassiterite in the minus 100-plus 200-mesh fraction.

3/ Unpublished memorandum, H. D. Hess to R. R. Wells, Jan. 16, 1953.

4/ Unpublished memorandum, H. D. Hess to R. R. Wells, May 20, 1952.

Chemical Character

Representative head samples prepared from the ores were submitted for assay. Results are shown in table A-9.

TABLE A-9. - Chemical analysis

Sample	Assay, percent			
	Sn	WO ₃	CaF ₂	SiO ₂
1	1.8	0.28	12.0	11.5
2	1.0	.18	6.7	31.3

A semi-quantitative spectrographic analysis, made of representative portions of the ores, revealed the presence and approximate quantities of the metals listed in table A-10. Any other elements, if present, are in amounts lower than the minimum detectable by the routine technique employed.

TABLE A-10. - Spectrographic analyses

Sample	Al	As	Be	Bi	Sn	Ca	Cu	Pb	Mg	Fe	Mn	Si	W	B	Ba	Li
1	A	E	F	E	C	C	E	D	D	C	E	B	D	E	E	E
2	A	D	E	E	C	A	E	D	B	C	D	A	D	E	E	E

Legend: A - more than 10 percent E - 0.01 to 0.1 percent
 B - 5 to 10 percent F - 0.001 to 0.01 percent
 C - 1 to 5 percent G - less than 0.001 percent
 D - 0.1 to 1 percent

Treatment Methods

Good gravity concentration practice on ores requires removal of the valuable mineral constituents from the circuit as soon as liberation is accomplished to prevent the chance of the minerals being ground to slime and ultimately lost. This is particularly vital when the minerals to be recovered have a tendency to slime readily. During the laboratory investigations conducted on Lost River ore, utmost consideration was given to the study of treatment methods minimizing the slime loss of tin and tungsten.

The wide differences in specific gravities of the gangue and ore minerals indicated that beneficiation by gravity methods would be applicable. The intimacy of association of a portion of the cassiterite with the gangue, however, precluded treatment at coarse sizes.

For laboratory tabling tests, ore was screen-sized and each fraction was tabled separately. This method of sizing table feed is admitted to be inferior to hydraulic sizing but was the best substitute available at this laboratory.

Sizing

During the course of the laboratory studies, numerous sizing tests were made to determine the distribution of ore minerals in various fractions. The screen analyses summarized in tables A-11 and A-12 show typical results. For these tests, ore was broken in a jaw crusher and roll-crushed in stages to pass a 3-mesh screen. The samples were screen-sized wet, using standard Tyler sieves, and the minus 200-mesh fraction was deslimed by decantation. Each fraction was analyzed for tin and tungsten. Results are shown in tables A-11 and A-12.

TABLE A-11. - Screen analysis, Sample 1

Product	Weight, percent	Assay, percent		Distribution percent	
		Sn	WO ₃	Sn	WO ₃
Plus 6-mesh	34.07	1.4	0.16	25.6	22.5
Plus 10-mesh	15.20	2.0	.36	16.3	22.1
Plus 20-mesh	13.87	2.3	.32	17.2	18.0
Plus 35-mesh	8.81	2.7	.32	12.8	11.5
Plus 48-mesh	3.84	3.0	.29	6.2	4.5
Plus 65-mesh	2.74	3.6	.32	5.3	3.3
Plus 100-mesh	2.62	3.2	.31	4.5	3.3
Plus 200-mesh	2.98	3.3	.35	5.3	4.1
Minus 200-mesh sand	5.67	2.0	.29	6.1	6.6
Slime	10.20	.13	.10	.7	4.1
Calculated head	100.00	1.9	0.24	100.0	100.0

TABLE A-12. - Screen analysis, Sample 2

Product	Weight, percent	Assay, percent		Distribution, percent	
		Sn	WO ₃	Sn	WO ₃
Plus 6-mesh	24.20	0.63	0.13	17.1	18.5
Plus 10-mesh	18.66	.85	.17	17.9	19.1
Plus 20-mesh	14.49	1.3	.18	21.2	15.6
Plus 35-mesh	9.87	1.1	.24	12.3	14.4
Plus 48-mesh	2.85	1.9	.30	6.1	5.4
Plus 65-mesh	3.72	1.5	.22	6.3	4.8
Plus 100-mesh	2.99	1.4	.22	4.7	4.2
Plus 200-mesh	2.57	2.1	.26	6.1	4.2
Minus 200-mesh sand	8.37	.8	.19	7.6	9.6
Slime	12.28	.05	.06	0.7	4.2
Calculated head	100.00	0.90	0.17	100.0	100.0

Crushing to minus 3-mesh ($\frac{1}{4}$ -inch) reduced approximately 50 percent of the ore to minus 10-mesh. The undersize contained 55 to 65 percent of the total tin and tungsten.

Ten to twelve percent of the total weight of the ore was present in the slime fraction (approximately minus 400-mesh). Approximately 0.7 percent of the tin and 4 percent of the tungsten were present in this fraction. In these and subsequent tests it was noted that desliming must be accomplished at extremely low pulp density to minimize tin and tungsten losses.

The various sized fractions were studied, using a binocular microscope, and the following observations were made:

1. Plus 6- and plus 10-mesh: No free concentrate grains. Considerable free gangue. It was noted that numerous gangue grains had minute mineral inclusions. These inclusions would increase the apparent specific gravity of the grains to no marked degree so that the grains probably would report in a gravity tailing.

2. Plus 20-mesh: A few free concentrate grains. More free gangue. Bulk of material as locked particles.

3. Plus 35-, 48-, and 65-mesh: Some free concentrate grains. Much free gangue. Bulk of material still locked.

4. Plus 100-mesh: About half of concentrate grains free, balance locked in aggregate grains.

5. Plus 200-mesh: Some locking still apparent. Most mineral grains free.

6. Minus 200-mesh sand: Nearly all of mineral liberated.

Gravity Concentration Treatment

Numerous tests were run on Lost River ore samples using various crushing and grinding techniques and several gravity separation methods.

Jigging was employed satisfactorily on minus 6-plus 14-mesh fractions of those ores which contained cassiterite grains liberated in that size range. Preliminary tests, on ore crushed to minus 20-mesh, made with the Humphreys spiral concentrator gave non-conclusive results. The test operation was hampered and adjustments were made difficult by the masking of the concentrate-tailing separation line by the slime-forming constituents of the ore. In no case, however, were results obtained superior to those made by tabling. It was decided that since a low-mineral tailing cannot be produced by tabling minus 20-plus 35-mesh ore, it is doubtful that it can be achieved with a spiral concentrator.

Shaking table treatment of the slime fraction gave recovery of cassiterite superior to recovery by treatment with a laboratory launder setup that simulated treatment with a Buckman tilting table. Insufficient data were obtained on this phase of the investigation, however, to draw a definite conclusion on this point.

The preliminary tabling tests also indicated that:

1. Concentration is readily effected from material coarser than 200-mesh. In some tests up to 95 percent of the tin and 82 percent of the tungsten in the plus 200-mesh fraction were recovered in the concentrate. Concentration of the material finer than 200-mesh, however, was more difficult and less efficient.

2. Of all the crushing and grinding techniques tried, a combination of roll-crushing to minus 3/4-inch and attrition grinding appeared to be most satisfactory. By this method the soft loosely cemented agglomerates in the ore were broken and the ore minerals liberated with very little breakage of the minerals themselves. Thus, up to 80 percent of the tin and 65 percent of the tungsten were retained in the easily treated plus 200-mesh fractions. In comparison, a test that employed conventional ball mill grinding of ore to minus 20-mesh resulted in the retention of only 58 percent of the tin and 53 percent of the tungsten in fractions coarser than 200-mesh.

Subsequent testing indicated that roll-crushing as fine as minus 1/4-inch followed by attrition grinding resulted in only a negligible increase in tin and tungsten slime loss.

3. By careful decantation or mechanical separation in dilute pulp, a slime product (approximately minus 400-mesh) low in tin content was removed from the fine sand, thus reducing the amount of fines to be tabled. When attrition grinding was employed, this slime product contained up to 50 percent of the original weight of the ore and contained only about 5 percent of the tin and 10 percent of the tungsten.

4. Tabling of sized fractions of ore gave better recoveries than tabling unsized feed.

5. Tabling of minus 10- plus 20-mesh and minus 20- plus 35-mesh material yielded concentrate, but no tailing low enough for rejection was made in these size ranges. The minus 35- plus 48- and minus 48- plus 65-mesh material produced not only a concentrate and low metal tailing but also a middling product that required further grinding for adequate liberation.

6. Although some true middling was produced from the minus 65-mesh fractions, regrinding of this material resulted in over-sliding of tin and tungsten minerals.

Based on the information gained from the preliminary tests, ore was crushed to minus 3/4-inch and attrition ground in a cement mixer, (no large ball mill was available), using a few 1 1/2-inch balls as grinding media. The material was periodically screened on 10-mesh and the oversize returned for further grinding until all of the ore passed 10-mesh. The material was screen-sized on 20-, 35-, 48-, 65-, 100-, and 200-mesh sieves. The minus 200-mesh fraction was carefully deslimed by decantation and the slime was discarded without treatment. Each sized fraction was tabled separately. The 10/20-mesh portion yielded a concentrate and a tailing which was attrition-ground to pass 20-mesh, sized and added to the corresponding fractions of original ore. The 20/35-mesh fraction was treated similarly. The 35/48 and 48/65 were tabled to produce concentrate, middling, and tailing with only the middling being reground and re-treated. The middlings from the 65/100 and 100/200-mesh fractions were added to the next finer size without regrinding. The minus 200-mesh sand was tabled to produce only concentrate and tailing. Results are shown in tables A-13 and A-14.

TABLE A-13. - Table concentration, Sample 1

Product	Weight, percent	Assay, percent		Distribution, percent	
		Sn	WO ₃	Sn	WO ₃
10/20 Conc.	0.43	61.6	6.0	14.0	12.3
20/35 Conc.	.47	61.6	5.1	15.4	11.8
35/48 Conc.	.27	57.7	5.0	8.3	6.9
48/65 Conc.	.26	60.7	5.1	8.4	6.4
65/100 Conc.	.35	56.6	5.4	10.5	9.3
100/200 Conc.	.54	52.6	5.4	15.1	14.3
Minus 200 Conc.	.64	47.2	5.9	16.0	18.2
35/48 tail	7.97	.30	.03	1.3	1.0
48/65 tail	6.69	.23	.04	.8	1.5
65/100 tail	6.47	.16	.04	.5	1.5
100/200 tail	10.06	.16	.03	.8	1.5
Minus 200 tail	20.06	.30	.085	3.3	8.4
Slime	45.79	.23	.03	5.6	6.9
Calculated head	100.00	1.9	0.20	100.0	100.0
Combined Conc.	2.96	55.8	5.4	87.7	79.2

TABLE A-14. - Table concentration, Sample 2

Product	Weight, percent	Assay, percent		Distribution, percent	
		Sn	WO ₃	Sn	WO ₃
10/20 Conc.	0.25	62.2	3.1	15.2	4.7
20/35 Conc.	.32	54.0	4.5	16.9	8.2
35/48 Conc.	.19	54.4	5.8	10.0	6.5
48/65 Conc.	.18	55.3	6.7	9.7	7.1
65/100 Conc.	.15	58.8	6.5	8.6	5.9
100/200 Conc.	.27	56.1	8.0	14.7	12.9
Minus 200 Conc.	.31	41.2	7.0	12.5	12.9
35/48 tail	10.74	.20	.08	2.0	5.3
48/65 tail	8.58	.20	.06	1.7	2.9
65/100 tail	5.94	.10	.03	.6	1.2
100/200 tail	11.15	.10	.07	1.1	4.7
Minus 200 tail	17.75	.16	.14	2.7	14.7
Slime	44.17	.10	.05	4.3	13.0
Calculated head	100.00	1.0	0.17	100.0	100.0
Combined Conc.	1.67	53.8	5.9	87.6	58.2

By attrition grinding and tabling sized feed, over 85 percent of the tin and an average of about 65 percent of the tungsten were recovered in a gravity concentrate that assayed approximately 55 percent Sn and 5.5 percent WO_3 .

Gravity Concentration, Conclusions

Based on the results of laboratory investigations, the following general conclusions were drawn:

1. Removal of the ore minerals from the grinding circuit as soon as liberation is effected is imperative to prevent excessive losses of tin and tungsten as slime.

2. Stage crushing and attrition grinding prior to tabling resulted in higher recoveries than normal grinding methods. By attrition grinding treatment, a minimum amount of the tin and tungsten minerals was reduced to the minus 200-mesh fraction from which it is difficult to effect good mineral recovery.

3. Up to 50 percent of the weight of the ore can be rejected as slime of low tin content if separation of slime and fine sand is effected efficiently in a dilute pulp.

4. Shaking table concentration of sized fractions of the ore gave most consistent results and highest recoveries of all the concentration methods tried. For ores containing relatively coarse, free cassiterite particles, jigging of the coarser fractions probably would be applicable.

5. Plant concentration of the sand fractions (plus 200-mesh) should be effected readily with good recovery of tin and tungsten minerals. The sand tailing should average about 0.2 percent Sn for heads containing 1 to 2 percent Sn, and concentrates assaying more than 50 percent Sn can be expected.

6. Plant recovery from the more refractory fine material will be dependent both on the efficiency of the concentrator and efficiency of the desliming operation.

7. Over-all plant recovery, therefore, will be dependent upon the ability of the grinding circuit to produce liberated ore minerals in the coarser sand fractions.

A suggested flowsheet based on the results of laboratory investigation, is shown in figure A-5.

Flotation of Sulfides

Preliminary studies indicated that the plant gravity concentrate is amenable to flotation treatment for the removal of sulfides. In a typical test, a sample of plant concentrate assaying 53.1 percent Sn, 2.3 percent WO_3 , 3.2 percent S, and 2.7 percent As was ground to pass a 65-mesh sieve

and the sulfides were floated in a slightly acid circuit using potassium amyl xanthate and pine oil as collector and frother. Removal was made of 94 percent of the sulfur and 90.5 percent of the arsenic with a loss of only 0.5 percent of the tin and 1.0 percent of the tungsten. The tailing (tin-tungsten product) assayed 59.5 percent Sn, 2.6 percent WO_3 , 0.2 percent S, and 0.3 percent As.

Tin-Tungsten Separation Tests

Only a cursory study was made of methods for the separation of tin and tungsten in the plant concentrate. Flotation of cassiterite, flotation of scheelite and wolframite, and magnetic separation methods were tried briefly. These preliminary tests were not satisfactory. A thorough investigation of this phase of the problem should be made but was considered to be secondary to the problem of producing a good grade gravity concentrate with high recovery.

LABORATORY BENEFICIATION STUDIES, LODE ORE, FAHRENWALD

Concurrent with the testing program of the Bureau of Mines and the construction and initial operation of the mill, A. W. Fahrenwald, well-known metallurgical consultant, conducted laboratory studies on samples of ore submitted by the company. These studies were supplemented by a visit to the mill to study the conditions and problems of plant operation.

In a report^{5/} based on the laboratory studies and his visit to the plant, Mr. Fahrenwald urged that the mill be rearranged to include a minimum of six shaking tables, hydraulic classification of table feed, and operation of the grinding circuit in a manner to minimize over-grinding. In an earlier report^{6/} he stated that grinding should be effected by rod milling in a relatively thin pulp. He submitted the proposed flowsheet shown in figure A-6. It will be noted that this flowsheet is similar to that suggested by the Bureau of Mines laboratory results, (fig. A-5).

^{5/} Report by A. W. Fahrenwald to K. J. Kadow, U. S. Tin Corp.: Operation at Lost River, Alaska, June 18, 1952. Correspondence.

^{6/} Report by A. W. Fahrenwald to K. J. Kadow, April 16, 1952. Correspondence.

PROPOSED FLOWSHEET (FAHRENWALD)
 UNITED STATES TIN CORPORATION MILL
 LOST RIVER, ALASKA

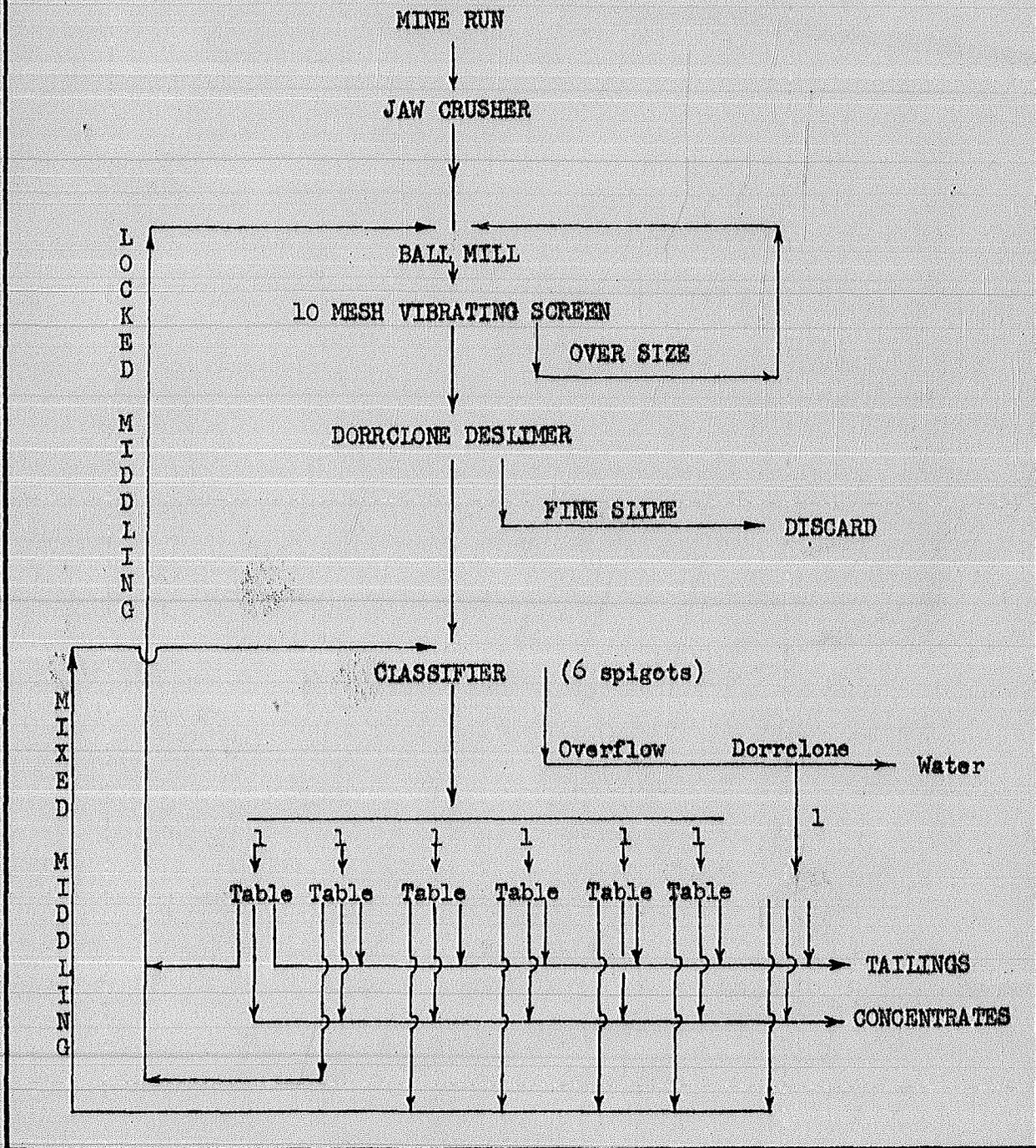


Figure A-6. — Fahrenwald flowsheet