Recovery of Cassiterite at the Sungei Besi Mines, Selangor, Malaya*

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SYNOPSIS

Post-war advances in the field of earth-moving equipment have so reduced mining costs on the Sungei Besi opencast tin mine in Malaya that it is now possible to treat much lower-grade ground at a profit than was previously possible. The lower-grade ore, however, contains much fine cassiterite and a new dressing plant has become a necessity both to handle the increased throughput due to mechanization and to ensure efficient recovery of fine mineral.

A plant has now been designed and is in the erection stage. Jigging remains the main concentrating method, increased throughput having been achieved by removing, with two stages of cyclones, the greater part of the slime and material under 200 mesh B.S.‡ ahead of the primary jigs for treatment by spirals. Fine cassiterite not overflowed by the primary cyclones is recovered by screening jig tailings and re-jigging the undersize in scavenger jigs.

THE SUNGEI BESI MINES, LTD., IS A TIN-MINING COMPANY operating two adjacent opencast mines in Selangor and a dredge in Johore. The present paper concerns only the two opencasts and their common treatment plant.

The orebody being worked at Sungei Besi is a deep alluvial-eluvial deposit on a limestone-granite contact. The floor levels of the two opencasts are at present approximately 300 ft below that of the treatment plant and boring has proved the mixed clay and sand to extend considerably deeper in places. Values are highly erratic, those contained in clay, normally of very coarse partly-water-worn cassiterite, rising occasionally to over 100 lb/cu. yd, while the sand is much lower in grade, varying anywhere between 0 and 3 lb of finer more angular cassiterite per cu. yd. With the exception of some pyrite and arsenopyrite there are virtually no other heavy minerals in the deposit.

Operations began in the South opencast about 50 years ago, when all the excavating was done by hand and the spoil hoisted to the treatment plant in trucks up an inclined haulage-way, the treatment plant consisting of a series of puddlers discharging to sluice-boxes. Power was supplied by steam and it was not long before a steam-shovel was put into operation.

The company then built a hydroelectric plant in the surrounding hills and, shortly after the power line was brought in, electrically-operated

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‡All mesh sizes referred to are British Standard, and the long ton of 2240 lb is used in this paper.

excavators, still in use, replaced the steam shovel, and jigs replaced the sluice-boxes.

When the North opencast was opened the treatment plant was enlarged and moved to a point between the two mines, being fed by two endless haulages from the North and a double-drum haulage from the South. This was the position when the Japanese overran the Peninsula in 1942. During the occupation the Japanese worked all the rich pockets in both the mines until further development was necessary and then allowed them to flood. When the company again took over in late 1945 many years of pumping out and repairing slopes were necessary and it was not until 1952 that all the silt which had been deposited on the floors of the opencasts by the flooding was finally cleared.

By this time it was evident that considerable economies could be made by replacing the inclined haulage-ways with conveyor belts, one having been in use for some time for dumping purposes and another being on order to hoist ground from the floor of the North opencast, feeding direct to the puddlers. This belt was installed in 1953 in conjunction with a clay-cutter designed on the mine. The cutter had become necessary because very hard cassiterite-bearing clay is broken by the excavators into lumps too large for the belt. By 1956 the system had been extended to the excavators, which loaded directly on to the belt through a mobile slow feeder.

The mechanization of the mining side reduced costs immediately in two ways:

(*a*) The contract trucking gangs previously on piece work at $8\frac{1}{2}$d./cu. yd could be reduced in number and paid at only $1\frac{3}{4}$d./cu. yd without reducing their overall wage;

~~(*b*) Excavators previously averaging 15 000 cu. yd/month doubled their~~ yardage by being able to operate continuously.

The limit of payability up to that time had been in the region of 3·3 lb/cu. yd but, the yardage per excavator being doubled and the piece-work rates being reduced, this limit was lowered to 0·85 lb/cu. yd, thereby making profitable almost all the lower-grade sandy material which previously had to be dumped during development. To maintain this lower limit of payability, however, it was necessary to continue mining with the same number of excavators, all at the increased yardage, and a further complete conveyor system was ordered.

When trucking, three excavators were normally in operation supplying pay ground at a rate of approximately 45 000 cu. yd/month to the plant, of which the maximum capacity was 60 000 cu. yd. With the increased yield from the three excavators approaching 90 000 cu. yd/month the plant had then to be extended. In addition, as the cassiterite in the lower-grade sandy material was much finer than that previously treated, provision had to be made for fine mineral recovery, although probably no extension to the puddler plant would be required.

Treatment water is supplied from the mine's own reservoir, the capacity of which was just sufficient when treating 45 000 cu. yd/month. There is no possibility of enlarging the reservoir catchment area and it was thus a further complication that no more water could be used for treating the

bigger yardage. It was decided, therefore, that a new, modern plant was required, designed to overcome the three problems: (i) fine cassiterite recovery, (ii) return of maximum amount of clean water to the reservoir, (iii) increased throughput (up to 240 tons/h).

Before beginning tests in 1956 it was agreed that to overcome the first two problems efficient desliming at as early a stage as possible was required. At the same time it was suggested that the jigging capacity should be increased by adding further jigs. It would then be necessary, in order to recover the fine tin, to screen the jig tailings at a suitable mesh and re-treat the undersize. For these reasons the flowsheet shown in Fig. 1 was suggested.

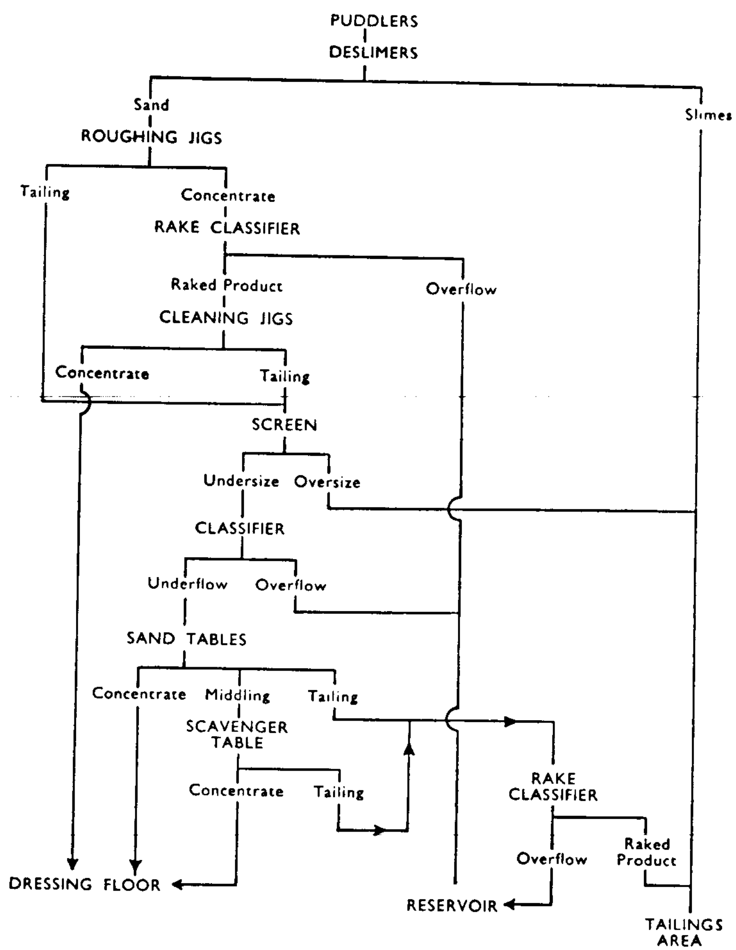

Fig. 1.—Preliminary flowsheet.

TEST WORK

(a) *Rake Classifiers*

At first a rake classifier (one of several already on the mine) was installed between the puddlers and jigs for desliming. Preliminary tests, however, showed that one of the difficulties to be dealt with was the erratic nature of the feed. The quantity being fed to the puddlers was uncertain and the size distribution of cassiterite and material extremely irregular. Screen analyses of the feed taken over a number of months showed that in the rich clays as much as 89 per cent of the total feed could be −200 mesh while in the more sandy ground this might be reduced to 49 per cent with over 40 per cent of +85 mesh. Some screen analyses are given in Table I. To reduce fluctuations in feed large ore bins were contemplated, provided with feeders, but owing to the sticky nature of the material after rain they were not considered practical.

TABLE I.—*Typically Divergent Screen Analyses of Mine Ore*

Mesh	Progressive totals (per cent)	
	Clay	Sand
+ 10	0·2	2·5
+ 22	1·2	8·0
+ 36	2·7	20·0
+ 44	3·7	25·0
+ 52	4·2	28·5
+ 60	4·7	33·0
+ 72	5·7	36·5
+ 85	6·5	40·0
+100	7·5	43·0
+120	9·0	46·5
+150	9·5	48·5
+200	11·0	51·0
−200	89·0	49·0

TABLE II.—*Rake Classifier Test on Puddler Discharge*

Machine used in this particular test was a 4-ft double-acting Stokes rake classifier run at 16 strokes/min at a slope of 3 in. in 1 ft.

Classifier overflow = 70·72 tons/h. Raked product = 3·5 tons/h.

Mesh	Feed Cum. %	SnO_2 in feed Cum. %	Overflow Cum. %	SnO_2 in overflow lb/h	Raked product Cum. %	SnO_2 in raked product Cum. %
+ 10	3·42	18·64	—	—	27·05	9·10
+ 22	11·72	55·38	0·73	—	52·59	51·20
+ 36	25·23	72·00	5·02	—	72·22	75·77
+ 44	35·89	77·32	7·93	—	81·40	84·66
+ 52	39·20	80·86	9·96	10·93	86·02	88·04
+ 60	45·65	85·30	13·72	10·73	91·65	92·26
+ 85	51·26	91·20	18·06	17·87	96·47	96·48
+100	54·72	93·04	20·29	15·90	97·55	97·43
+120	57·30	96·96	23·14	39·27	98·70	98·71
+150	58·64	98·26	23·65	23·75	99·02	99·14
+200	59·80	99·46	24·26	15·84	99·67	99·57
−200	40·20	0·54	75·74	7·19	0·33	0·43

Total: 141·48

Many tests were carried out with both 6-ft Dorr single-acting and 4-ft Stokes double-acting classifiers but no acceptable results were obtained (Table II). This was due to the erratic feed and to the high slime content which prevented reasonable settling. Diluting the pulp and using more machines would have entailed the use of too much water, the recovery and cleaning of which would have been expensive. In the end it was decided that rake classifiers alone could not at this stage be used for desliming.

(b) *Screens*

During the tests described thought had been given to the idea of screening the jig tailings for fine cassiterite recovery. When it became apparent that rake classifiers alone could not be used to deslime efficiently it was suggested that if the puddler product were first screened at 48 mesh and the oversize fed to a rake the pulp density in that machine could quite easily be diluted sufficiently to overflow all the fine material showing with the oversize through adherence to the larger particles. Thus all +48-mesh material would be jigged and only the −48-mesh material from the screen and rake overflow would require desliming before further treatment. Fewer primary jigs would therefore be needed.

Tests were started on screening the puddler discharge (Table III). Available on the mine was a 4-ft by 6-ft vibrating screen and this was put in the ~~circuit between the puddlers~~ and jigs. At the same time another screen designed on the mine was also put in circuit in order to compare results with identical feeds. The mine screen, originally designed to work under water, had a horizontal bed shaken by means of an eccentric drive so that the material was moved along the screen cloth. This was found impracticable and finally it was tested with half the stroke under water and the other half above, the water level being maintained by use of suitable weir overflows. This screen proved satisfactory, the oversize 'climbing' well, and by comparison with the first vibrating screen it appeared to fulfil its purpose extremely well. There was in fact little to choose on performance alone between the two screens but the mine horizontal shaking screen had only half the screening area of the vibrating screen, used much less water and was cheaper.

Tests were continued to get some idea of running and replacement costs on both types of screen, since it appeared that screening was in fact a practical solution to part of the problem at least. However, about that time a paper on work utilizing a curved bar screen was published* and the capacities quoted so exceeded the capacities of the two screens under test that it was decided to put another screen of the sieve bend type in circuit.

*STAVENGER, P. L., and REYNOLDS, V. R. Application of the DSM screen. *Min. Congr. J., Wash.*, **44**, July 1958, 48 51.

TABLE III.—*Screening Test on Puddler Discharge*

	Mesh	Cumulative %			
		Bulk		SnO_2 content	
Feed	+ 10	4		4	
	+ 36	23		48	
	+ 72	48		73	
	+120	77		90	
	+200	86		97	
	−200	14		3	
		Horizontal shaking screen	Vibrating screen	Horizontal shaking screen	Vibrating screen
Undersize	+ 10	Nil	Nil	Nil	Nil
	+ 36	6	2	19	10
	+ 72	31	22	50	35
	+120	60	60	70	69
	+200	81	75	91	92
	−200	19	25	9	8
Oversize	+ 10	20	9	3	9
	+ 36	72	80	81	80
	+ 72	82	91	96	93
	+120	89	96	99	97
	+200	94	99	100	100
	−200	6	1	Nil	Trace

It was not possible to obtain bars of a suitable size in Malaya so an old slotted jig screen was used bent into a suitable arc with the slots at right-angles to the flow. No screens with slots smaller than $\frac{1}{16}$ in. were available; the tests could not be absolutely conclusive since slots of approximately $\frac{1}{32}$ in. were required to give a separation at 48 mesh. The results of these tests are shown in Table IV. As can be seen this screen gave a good separation at 36 mesh except that far too much −200-mesh material appeared in the oversize. This material, which adhered to the larger particles, would in any case be dealt with by the desliming rake classifier

TABLE IV.—*Sieve Bend Tests on Puddler Discharge*

Amounts given as a progressive total percentage.
Curved Bar Screen constructed from slotted jig screen with $\frac{1}{16}$-in slots.

Mesh	Feed	Undersize	Oversize
+ 10	1·7	0·0	2·8
+ 36	23·3	3·4	37·4
+ 72	40·9	15·3	60·3
+120	50·9	26·8	68·2
+200	55·4	32·7	72·0
−200	44·6	67·3	28·0

TABLE V.—*Sieve Bend Tests carried out by Geevor Tin Mines, Ltd.*

GROUP 2

Screen Opening 1 mm.
Feed: sp. gr. 1·45 = 50% solids
Underflow: sp. gr. 1·4 = 46% solids
Overflow: sp. gr. 1·6 = 60% solids

Screening Analysis

Feed	Cum. %
+ 18 mesh	21·00
+ 25 ,,	29·70
+ 52 ,,	52·20
+ 72 ,,	60·25
+100 ,,	65·85
+150 ,,	74·95
−150 ,,	25·05
Underflow	
+ 18 mesh	—
+ 25 ,,	4·95
+ 52 ,,	29·50
+ 72 ,,	41·95
+100 ,,	50·35
+150 ,,	63·85
−150 ,,	36·15
Overflow	
+ 18 mesh	62·65
+ 25 ,,	74·00
+ 52 ,,	89·00
+ 72 ,,	92·25
+100 ,,	94·70
+150 ,,	96·05
−150 ,,	3·95

GROUP 3

Feed: sp. gr. 1·4 = 46% solids
Underflow: sp. gr. 1·3 = 37% solids
Overflow: sp. gr. 1·65 = 64% solids

Screening Analysis

Feed	Cum. %
+ 18 mesh	25·50
+ 25 ,,	33·10
+ 52 ,,	53·05
+ 72 ,,	62·00
+100 ,,	68·15
+150 ,,	76·90
−150 ,,	23·10
Underflow	
+ 18 mesh	—
+ 25 ,,	2·4
+ 52 ,,	20·9
+ 72 ,,	34·3
+100 ,,	44·8
+150 ,,	60·0
−150 ,,	40·0
Overflow	
+ 18 mesh	42·35
+ 25 ,,	56·00
+ 52 ,,	81·90
+ 72 ,,	88·80
+100 ,,	91·95
+150 ,,	94·75
−150 ,,	5·25

following the screen. The curved screen, the surface area of which was the same as that of the horizontal screen and half that of the vibrating screen, handled during these tests approximately three times the volume of material dealt with efficiently by the other two screens. Thus with the advantages of high capacity, no moving parts, no water consumption and probably longer screen life further screening tests were abandoned and the sieve bend was decided upon. Results of tests carried out in the United Kingdom with sieve bends are given in Table V. Two small screens made up of square section welding rods have now been in use on the mine for some months giving adequate screening both in a small closed-circuit grinding operation and in a pilot plant constructed to test out the main plant as a whole.

(c) *Cyclones*

There remained the problem of desliming the −48-mesh material from the screens and classifiers. Following up the claim† that the Phœnix hydrocyclone could handle fluctuating loads without loss of efficiency, a 27-in

†RABONE, P. The Phœnix hydrocyclone. *J. S. Afr. Inst. Min. Metall.*, **57**, 1956–57, 724–32.

hydrocyclone was constructed and tests were carried out on the puddler discharge passed through a trash screen but containing all the +48-mesh material. It was immediately obvious that an excellent jig feed could be obtained direct from the cyclone spigot but that a considerable quantity of the finer sand sizes reported in the overflow. The pulp density of the feed was, of course, much higher than is normal in cyclone practice, being merely a puddled product, and owing to the water shortage it was not possible to dilute the pulp sufficiently to get efficient desliming in one operation. When a pilot plant was installed later, a 12-in hydrocyclone was constructed to take the overflow from the 27-in cyclone, and from tests it appeared that efficient desliming was possible using two stages of cyclones.

At this stage of the investigation it was suggested by the Research Division of the Department of Mines, Federation of Malaya, that the advantage claimed for the Phœnix hydrocyclone, namely capability of handling fluctuating loads, might in this case be outweighed by its low capacity. A conventional cyclone was therefore made and used on the puddler discharge. It was found that this type, operating at a higher pressure, did in fact give equally good results but had a much higher capacity. The Research Division were at that time developing their high-capacity low-pressure hydrocyclone, which is cheaper to manufacture and requires the high velocity preferable for a finer overflow. It was decided, therefore, to use the conventional type in conjunction with a large pump sump designed to even out the fluctuating load.

(d) *Humphreys Spirals*

Investigations in England with Humphreys spirals on samples from the mine containing fine cassiterite gave encouraging results and a five-turn

TABLE VI.—*Fan-shaped Launder Test*

Value of feed = 3·60 lb/cu. yd Value of middling = 2·60 lb/cu. yd
Value of concentrate = 5·33 lb/cu. yd Value of tailing = 1·00 lb/cu. yd

Mesh	Feed Cum. %		Concentrates Cum. %		Middling Cum. %		Tailing Cum. %	
	Bulk	SnO_2	Bulk	SnO_2	Bulk	SnO_2	Bulk	SnO_2
+ 10 .	—	—	—	—	—	—	0·1	—
+ 22 .	3·0	11·85	0·3	3·00	1·0	6·15	8·1	—
+ 36 .	17·5	38·89	4·3	22·25	14·0	25·12	33·1	—
+ 44 .	27·7	50·00	10·8	34·73	27·0	35·38	46·1	8·00
+ 52 .	35·7	56·30	18·3	42·75	38·0	44·10	55·6	16·00
+ 60 .	46·7	62·97	31·3	49·75	51·5	51·79	67·1	26·00
+ 72 .	56·2	68·90	45·3	54·75	64·0	56·92	76·6	34·00
+ 85 .	64·8	72·60	63·3	61·00	74·5	61·02	85·1	38·00
+100 .	73·1	77·04	73·8	65·25	83·0	66·15	90·6	44·00
+120 .	83·1	81·48	87·3	72·75	92·0	74·87	95·6	52·00
+150 .	87·6	87·03	93·8	80·75	96·0	84·10	97·6	64·00
+200 .	93·5	96·29	98·0	93·25	98·5	96·41	98·5	82·00
−200 .	6·5	3·71	2·0	6·75	1·5	3·59	1·5	18·00

TABLE VII.—*Concentrating Spiral Tests*

	Mesh	Concentrate Cum. %		Middling Cum. %		Tailing Cum. %	
		Bulk	SnO_2	Bulk	SnO_2	Bulk	SnO_2
Scavenger	+ 10 .	—	—	—	—	—	—
Jig Tailing	+ 22 .	—	—	—	—	—	—
Overflowed	+ 36 .	—	—	—	—	—	—
From Rake	+ 44 .	—	—	—	—	—	—
Classifier	+ 52 .	—	—	—	—	—	—
(see Note 1)	+ 60 .	—	—	—	—	—	—
	+ 72 .	0·6	—	0·5	—	1·5	—
	+ 85 .	1·4	—	1·3	—	2·3	—
	+100 .	4·5	—	4·3	—	3·2	—
	+120 .	11·6	—	18·5	—	5·4	—
	+150 .	29·9	8·2	48·2	—	13·8	—
	+200 .	71·3	41·0	84·7	—	61·1	—
	−200 .	28·7	59·0	15·3	100·0	38·9	100·0
Secondary	+ 10 .	—	—	—	—	—	—
Cyclone	+ 22 .	—	—	—	—	—	—
Spigot	+ 36 .	—	—	—	—	—	—
Product	+ 44 .	—	—	—	—	—	—
(see Note 2)	+ 52 .	—	—	—	—	—	—
	+ 60 .	—	—	—	—	—	—
	+ 72 .	15·2	—	2·2	—	2·0	—
	+ 85 .	22·4	—	3·2	—	2·5	—
	+100 .	32·4	—	5·1	—	3·3	—
	+120 .	46·9	—	9·6	—	4·5	—
	+150 .	65·7	10·0	21·1	Trace	8·8	—
	+200 .	86·7	37·0	67·1	Trace	26·0	—
	−200 .	13·3	53·0	32·9	100·0	74·0	100·0

Note 1

Value of feed = 0·08 lb/cu. yd
Value of concentrate = 0·85 „ „
Value of middling = 0·07 „ „
Value of tailing = 0·03 „ „

Note 2

Value of feed = 0·08 lb/cu. yd
Value of concentrate = 1·33 „ „
Value of middling = 0·08 „ „
Value of tailing = 0·03 „ „

spiral lent to the mine by the Department of Mines Research Division was installed in a pilot plant. Tests with this machine have proved that fine tin recovery is good and orders have been placed for the necessary spirals. The results of these tests are discussed later with the pilot plant and are shown in Table VII. These tests have justified the recommended use of spirals as roughers and sand tables as cleaning concentrators.

(e) *Cannon Concentrators*

Preliminary experiments on a number of fan-shaped short sluices were also carried out in order to test the possibility of using the Cannon concentrator in place of the spirals. The tests, results of which are shown in Table VI, proved unsatisfactory and testing was discontinued.

THE PILOT PLANT

Tests on all the machines having been completed, it was apparent that the correct answer to the problem was to return to the simple flowsheet originally proposed but to replace by cyclones the classifiers originally suggested for desliming and to use scavenger jigs. A possible flowsheet on these lines was therefore drawn up (Fig. 2).

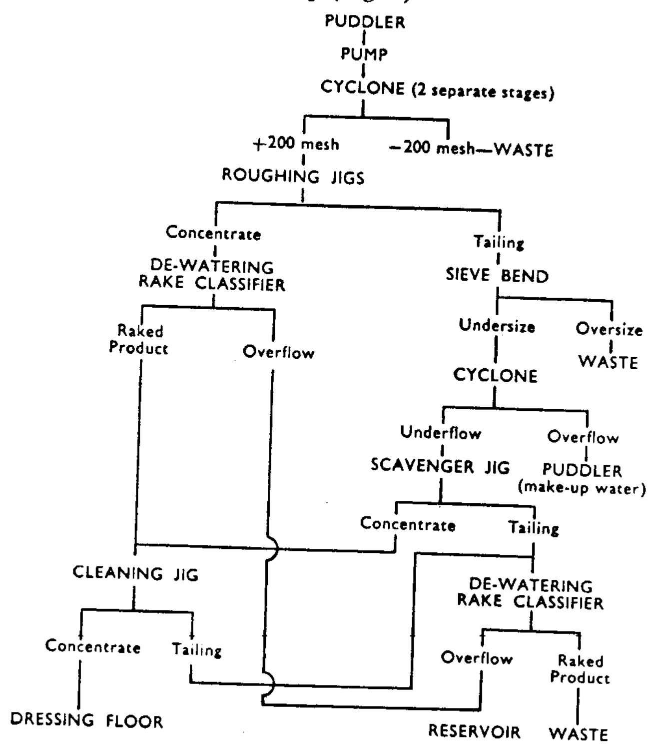

Fig. 2.

In order that each part of the plant could be tested while running continuously under normal conditions a pilot plant incorporating all the equipment found suitable was installed, the mechanical details of which were as follows.

Discharge from puddlers was pumped by an 8-in/6-in Warman sand pump to the primary cyclone (a 24-in conventional type with a 30-sq.in inlet, 8-in vortex finder and 3-in apex). The overflow from the cyclone went to waste and the underflow to a roughing Pan-American jig, the concentrate from the jig being fed by rake classifier to a cleaning Pan-American jig. The curved screens ordered had still not arrived on the mine so tailings from the primary jig were screened over a sieve bend of $\frac{1}{8}$-in square section rods with $\frac{1}{8}$-in aperture, the undersize from this screen being pumped, together with the overflow from the rake classifier, by a 6-in/4-in Warman pump to a 27-in Phœnix-type hydrocyclone (considered adequate in capacity for the pilot plant). The overflow from this hydrocyclone was found to be too dirty to return to the reservoir, which is also used for domestic purposes, but it could be returned to the puddler service tank. The underflow from the cyclone fed the Pan-American scavenger jig, the concentrate from which was pumped to a 12-in dewatering cyclone feeding on to the secondary jig. The tailings from the cleaning jig were originally allowed to run to waste in the pilot plant, but screen analyses of samples were taken and the possibility of re-treating them on spirals was borne in mind and later proved. This pilot plant was in operation by the beginning of August, 1958, and several tests were carried out.

The information gained from the tests is summarized as follows.

The feed investigated contained a high clay content and it was quickly determined that a higher pressure was needed to keep coarser sands and cassiterite out of the overflow. It was decided to use 9 lb/sq.in, since anything higher would cause heavy wear on the cyclone. It was also shown that the split with the conventional-type cyclone was just as good, if not better, than that obtainable with the Phœnix model.

It was then found that at 9 lb/sq.in very fine cassiterite reported in the overflow and that fluctuations in slime content seemed to be responsible in the main for such losses. That the overflow would therefore have to be treated on a secondary cyclone at a slightly higher pressure was proved by pumping the overflow to two 12-in cyclones at 10 lb/sq.in and making an excellent fine sand underflow with cassiterite up-graded to a stage at which it could probably be further up-graded by spirals prior to sand tables.

With a feed higher in sand content it was also proved that a slightly larger amount of +200-mesh sand will report in the overflow from the primary cyclones unless there is dilution of the pulp. This is not desirable and once again secondary cyclones are required to treat the overflow from the primary. It was not possible to determine the quantity limits of the secondary cyclones' underflows, but the maximum amount will be quite small and will not require excessive equipment to up-grade. Almost all the cassiterite reporting is −150 mesh with about 70 per cent −200 mesh.

A 16-in conventional cyclone was then made to replace the 27-in hydrocyclone on the scavenging circuit, and better results were obtained, but again it was found necessary to re-treat the overflow in smaller cyclones, so ensuring that no recoverable cassiterite will be lost in cyclone overflows and a fine cassiterite product for treatment by spirals will have been concentrated.

The jigs were not as efficient as required but that could not be rectified until suitable motors were obtained and the eccentrics altered to suit the change of speed. However, all rougher jig tailings were re-treated in the pilot plant and the secondary tailings will also be re-treated in the final plant.

Tests so far have thus proved that the cyclones are desliming efficiently without losses of recoverable mineral in the overflows, while jig tailing losses from the cleaning and scavenger jigs average about 0·11 lb/cu.yd of tailing. In addition, as already mentioned, jigging efficiency can be improved and the eccentrics of the scavenger jig are to be altered to give $\frac{1}{2}$-in stroke, and motors of the correct speed are on order. Finer jig screens are

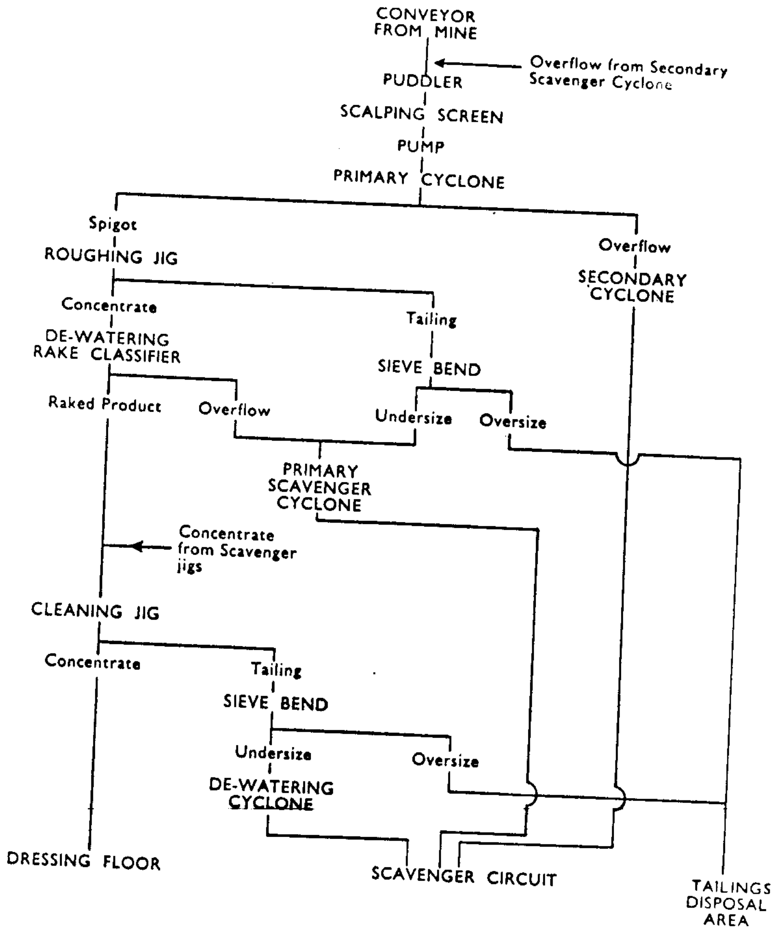

Fig. 3.—Final flowsheet (one primary circuit).

also to be fitted and the hematite bed altered accordingly in order to give the fine cassiterite a better chance to settle. The concentrate recovered from the scavenger jig is fine but that from the roughing jigs is coarse. Thus the cleaning jig must be set for coarse mineral recovery and some of the losses from this jig are due to feeding the scavenger jig concentrate to it; but, as later tests have shown, it is economical to re-treat the tailings by spirals after screening, it is not expected that undue losses will occur. If they do, a separate cleaner for the scavenger circuit may prove necessary.

The five-turn Humphreys spiral borrowed from the Department of Mines was put into circuit and tests were carried out to ascertain the possibilities of recovering fine mineral from the secondary cyclone spigot product and from the scavenger jigs tailings.

In the first test, results of which are shown in Table VII, the scavenger

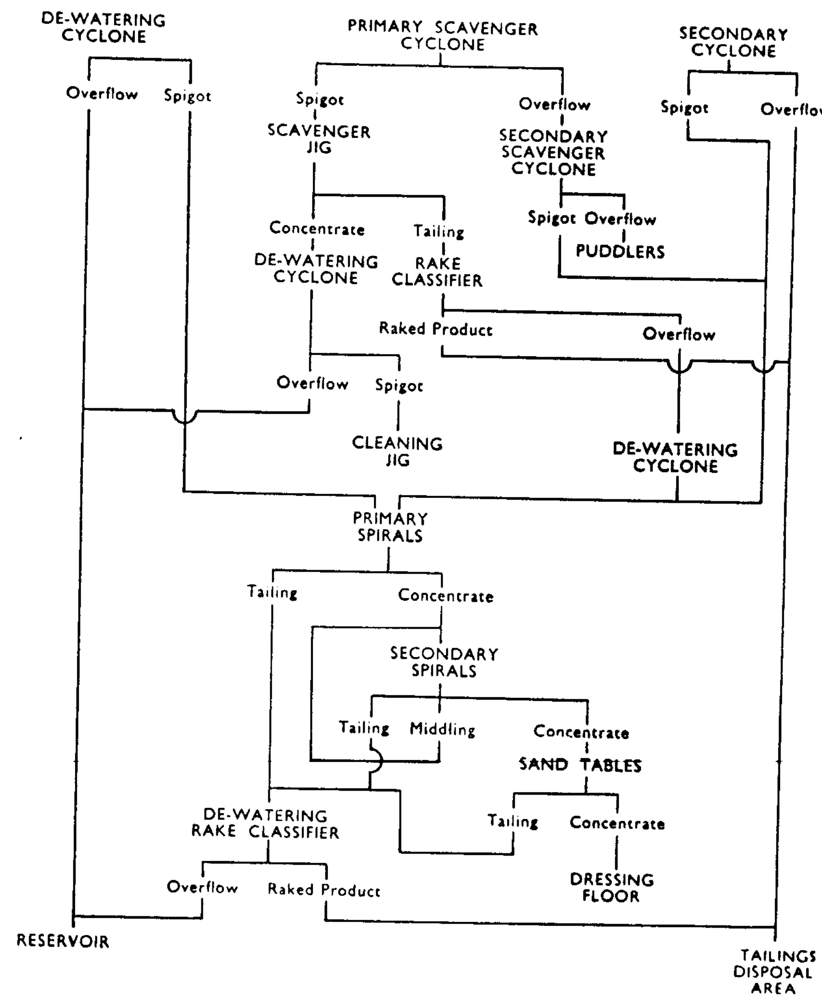

Fig. 4.

jig tailings were fed into a rake classifier and the overflow containing the fine tin losses was pumped to a 14-in cyclone, the spigot product of which was fed to a spiral. The value of the rake overflow was 0·08 lb/cu. yd, and the test showed that the value can be up-graded very considerably by cyclones and rougher spirals, with negligible losses from the latter. Only a small middling product was taken and it was not considered that re-treatment of it would be justified.

In the second test, results of which are given in the lower half of Table VII, the underflow from the secondary cyclone was led to the spiral and results were similar to those in the first test. It was not possible in either test to carry out a continuous rougher and cleaner test but there is no doubt that the values can be up-graded to 4 or 5 per cent by spiral cleaner operation, to be followed by table treatment of the spiral concentrate.

The final flowsheet is shown in Figs. 3 and 4. All that remained was to estimate the number and sizes of the various parts of the plant.

FINAL PLANT

(a) *Jigs*

From the screen analyses of the puddler discharge taken over a number of months it was apparent that the maximum feed to the roughing jigs would be in the region of 160 tons/h, so that as eight jigs were already installed in the plant, they are to remain to take a maximum load of 20 tons/h each. The only modification required will be to reduce the old 4-cell Harz jigs to 2-cell machines as it has been found that with a deslimed feed 2-cell operation is sufficient.

The pilot plant and screen analyses of the roughing jig tailings indicated that four 2-cell jigs were required for the scavenging operation and that two 2-cell cleaning jigs would take the concentrates from both roughing and scavenging jigs.

(b) *Cyclones*

Tests have shown that with the increase in throughput the total puddler discharge will be in the region of 2100 gal/min (at about 30 per cent solids). The ideal arrangement for the primary cyclones appeared to be to have four, each cyclone feeding two roughing jigs. Units of fairly large capacity were therefore required and to deal adequately with the wide size range it was decided to install 24-in cyclones with an inlet area of 25 sq. in to operate at a pressure of 9 lb/sq. in. Using Chaston's formula,* the capacity should be in the region of 750 gal/min ± 20 per cent per unit, proved correct by actual tests. Pumps with a total capacity of 2500 gal/min at the required head are being installed.

It is estimated that the overflows from the primary cyclones will total some 1900 gal/min with mainly −200-mesh solids. Much smaller secondary cyclones are therefore required, and after tests with various sizes 14-in units were decided upon for this duty. They have an inlet of 8 sq. in and eight are required.

The estimated feed to the primary scavenger cyclones is 1400 gal/min. A wide screen range will again be encountered and 16-in cyclones with a 12-sq. in inlet appear the most suitable. They should have a capacity of about 360 gal/min, and four are required.

The secondary scavenger cyclones will have to take some 1100 gal/min from the primary units and separate very fine sizes, so that 12-in cyclones with 8-sq. in inlets are to be installed and a higher pressure will be used.

*CHASTON, I. R. M. A simple formula for calculating the approximate capacity of a hydrocyclone. *Trans. Instn Min. Metall., Lond.*, **67**, 1957 58 (*Bull. Instn Min. Metall., Lond.*, no. 615, Feb. 1958), 203 8.

The capacity per cyclone will be about 280 gal/min, and four are required.

The rake classifiers taking the scavenger jig tailing overflow about 740 gal/min. For adequate removal of solids from this overflow tests have

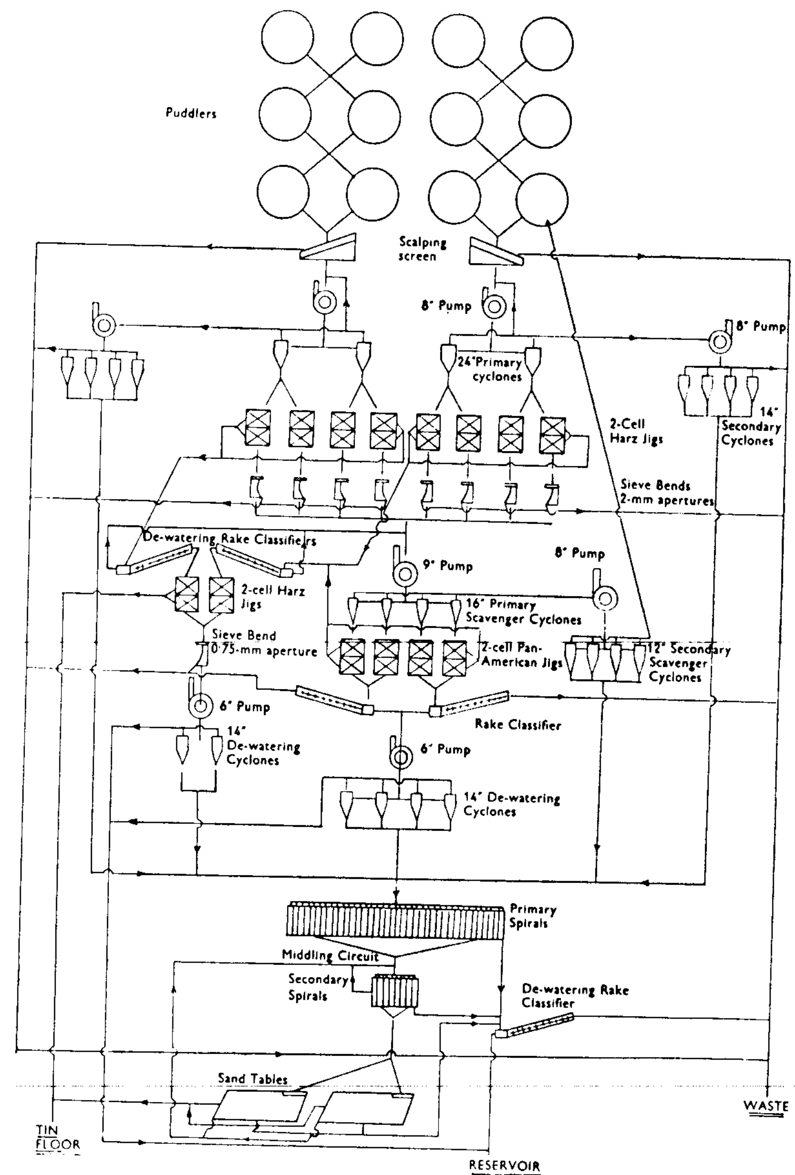

Fig. 5.—New treatment plant, final layout.