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Gravity Concentration of Fine Cassiterite*

I. R. M. CHASTON,† A.R.S.M., B.Sc., ASSOCIATE MEMBER

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SYNOPSIS

Details are given of operating results for an installation of tilting concentrators used to recover fine cassiterite, these results being compared with those reported for other special slime concentrators. An explanation is proposed for the presence and absence of the double recovery peaks which appear to be a characteristic of all the special slime concentrators considered with the exception of the round frame, and a plea made for the re-definition of 'slime', to take into account the change in technology brought about by the general use of the hydrocyclone.

Examples are given of fine cassiterite recovery on full riffled sand tables and it is concluded that the shaking table gives the best recovery of cassiterite in sizes down to 10 μ, provided that the feed is sufficiently slime free. It is suggested that the best recovery of fine cassiterite will be made by continued cyclone classification and tabling for the $+10$-μ cassiterite, with treatment of the true slime product on one of the special slime concentrators, possibly adding reagents to give preferential flocculation of the cassiterite.

The efficiency of various machines for recovering fine cassiterite from the so-called slimes produced during tin-dressing operations is a problem which has perhaps gained more attention from research workers in mineral dressing than it deserves when the considerable number of fundamental processes in gravity milling which have little more than an empirical basis for their operation is taken into account.

To the actual operator in this field, however, the question is of immediate and often pressing importance, since it is usually in the fine fractions that the biggest and most obvious loss of value occurs. A recent paper[1] discussed the recoveries made by a vanner, a round frame and a helicoid from 'slime' material taken from a Cornish lode tin mine and it is therefore perhaps timely to fill out the picture of fine cassiterite recovery by discussing some figures and details of operation for another form of concentrator developed for this purpose, the tilting concentrator. By comparing the results achieved with the various types of concentrators, it may be possible to obtain some insight into the fundamental processes at work in fine gravity concentration.

Description of the Denver-Buckman Tilting Concentrator

Each unit of the Denver-Buckman tilting concentrator consists of five decks each 6 ft square set one above the other in a frame. The feed is split between the decks, each of which is fed evenly across its full width. During feeding the decks slope at an angle which is adjustable but is usually about 10° from the horizontal. After a predetermined period the

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[1]etc. See list of references at the end of the paper.

feed is cut off and the decks are then tilted backwards to an angle of 45°, the concentrates lodged on the surface being automatically washed off by water jets. The decks are covered by rubber sheet with a honeycomb pattern impressed on its surface, each depression acting as an individual pocket to collect concentrate. The feed cut-off, table tilting and wash water cycle are all controlled from the drive motor. A 1-h.p. motor will serve two units.

Background to the Work

The experimental work described in this paper was carried out during the brief life of a lode tin mine in Thailand. The tilting concentrator had been installed with the mill and at first had been fed with the entire mill tailings. Owing to the coarse gangue material (up to 20 mesh) in these tailings, which effectively scrubbed all the fines off the decks, very little cassiterite was recovered in the concentrates and it was obvious that the feed would have to be restricted to the finer fractions of the tailings. All the fine overflows from the mill were then collected in a 54-ft thickener and the underflow from this thickener fed to the tilting concentrators. It was found that the recovery was still not at all good and this was ascribed to the high proportion of sulphides (up to 30 per cent) in the feed. The thickener underflow was therefore pumped to a conditioner where ethyl xanthate, amyl xanthate and cresylic acid were added and the sulphides were then floated in a bank of eight flotation cells. The final underflow was then fed to the tilting concentrators. From this point the flowsheet took the form shown in Fig. 1.

Timing Cycle and Feed Surges

Investigations were then undertaken into the actual operation of the tilting concentrators, ~~attention at first being paid to the time cycle of the~~ roughers, since it appeared to be of prime importance. This was done by taking tailing samples at regular intervals during the feed cycle of one of the roughers.

These samples were then assayed and gave the results shown in Fig. 2. They show that the losses are not at all progressive during the feed cycle, but that there was a definite increase in tailing assay after 4 min. The feed cycle was therefore reduced to 4 min and the tailings again sampled and assayed, giving the results shown in Fig. 3. Again there is an unexplained peak in the tailings cycle. Measurements of feed rates showed that, despite the surge cone, the portion of feed returned to the circuit during the washing cycle of the other rougher unit was increasing the flow of feed to the first unit and this change in flow rate over the surface of the decks was obviously sufficient to wash off some of the settled material. The flow of return tailings from the cleaner unit did not appear to be sufficient to affect either the feed flow or the feed assay. It was therefore obvious that the recycled products would have to be returned to some point where they would not affect the feed rate to the concentrators. In this instance it was convenient to return this material to the conditioner in the flotation circuit, which effectively eliminated the surge effect.

Tailings samples from the tilting concentrator then gave the results shown in Fig. 4. The tailing assay increased regularly throughout the feed

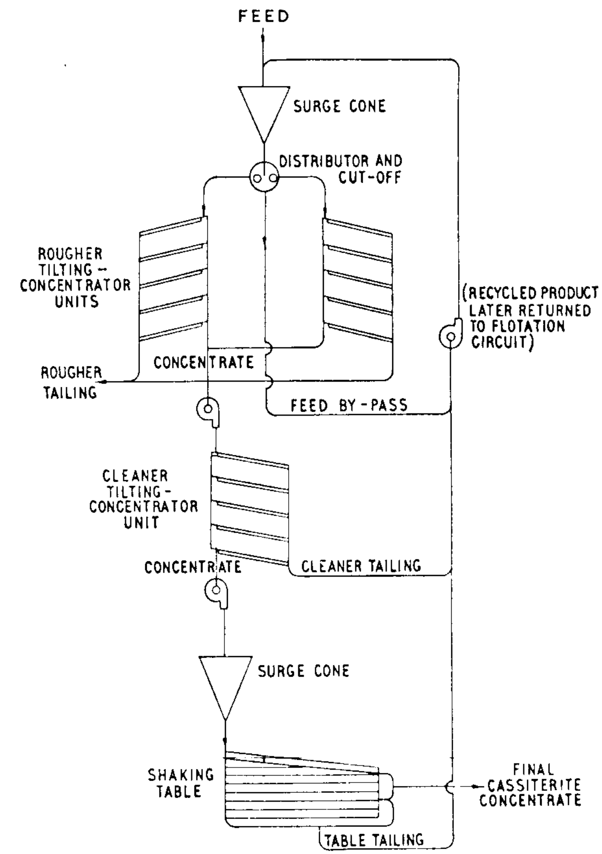
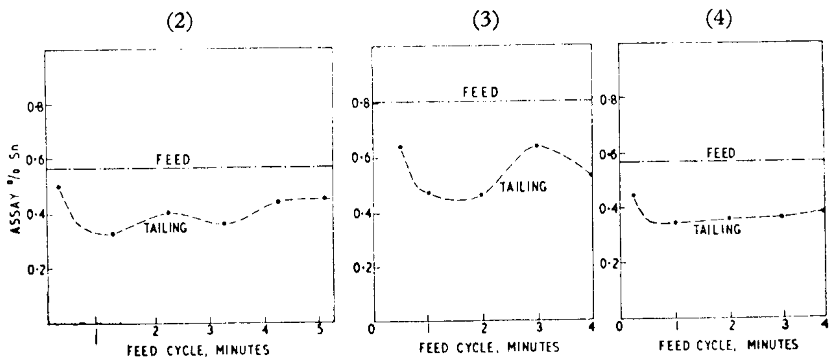


Fig. 1.—Early tilting concentrator circuit.



Figs. 2, 3 and 4—Feed and tailings assays before and after eliminating surge effects.

cycle after a high initial figure caused by the fact that a small amount of the settled material was not removed during the washing cycle but was disturbed sufficiently to be washed off with the initial flow of the new feed.

It is possible that improved results might be obtained from these machines if it were arranged to recycle the tailings for, say, the first ½ min of the feed flow. This could be worked in conjunction with the feed-timing device and should present no great mechanical problems.

Notes on the Operation of the Tilting Concentrator

Two small operating points which may interest potential users of this type of equipment are that the sides of the frame need to be boarded up to reduce losses from splashing during the washing cycle and that the rubber covering, which is held down by side straps, has a tendency in the tropics to stretch and buckle after several months operation. It would probably be an improvement to have the rubber covering bonded to the backing as the slightest channelling of the pulp will obviously reduce the performance of the unit.

Sizing Analysis

The sizing tests given here were carried out by beaker decantation using four stages of washing. The results are quoted for sizes 10 per cent below the nominal size used for calculation of the settling period to allow for incomplete washing. Unfortunately facilities were not available in the mine in Thailand for microscopic confirmation of the actual sizing achieved, but for this type of work exact knowledge of the sizes present is not as important as the consistency of the sizing achieved. For this reason duplicate sizing tests were made on some of the samples to determine the extent to which the method is reproducible. The results quoted in Table I illustrate that with a practised operator the consistency of sizing is remarkably good.

TABLE I.—*Duplicate sizing by beaker decantation*

Sizing equivalent cassiterite spheres, μ	Duplicate samples			
	Wt. %	Assay tin %	Wt. %	Assay tin %
> 36	8·3	0·60	8·7	0·55
< 36 > 27	11·6	0·40	12·0	0·42
< 27 > 18	16·9	0·40	16·2	0·50
< 18 > 13	9·1	0·57	9·6	0·59
< 13 > 9	10·1	0·53	10·2	0·63
< 9	44·0	0·28	43·3	0·25
	100·0	0·39	100·0	0·41

Recovery of Cassiterite by the Tilting Concentrator

Figs. 5, 6 and 7 show the weight distribution of the feed, the tin distribution of the feed and the recovery of cassiterite in individual size

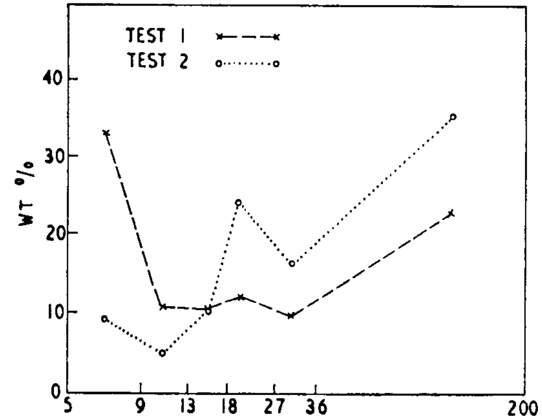


Fig. 5.—Size distribution of the feed to the tilting concentrators.

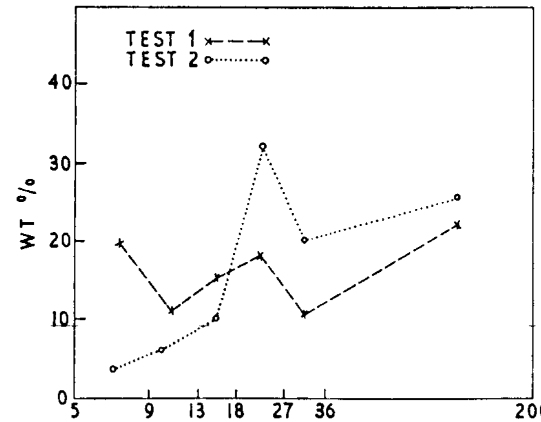


Fig. 6.—Distribution of tin in the feed to the tilting concentrators.

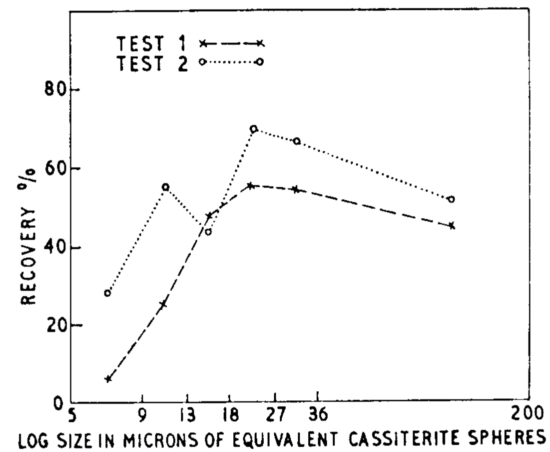


Fig. 7.—Recovery of cassiterite by the rougher stage of the tilting concentrators.

ranges for two tests on the rougher units. Test 1 represents conditions after the adjustments to the timing cycle and the elimination of feed surges. The very low recovery of the very finest sizes of cassiterite in test 1 shown in Fig. 7 suggested that if the fairly large proportion of relatively low-grade material in the very finest sizes could be removed, the recovery in the next coarser sizes might be improved. A 6-in cyclone was therefore built and the thickener underflow pumped through it. The underflow from the cyclone was fed to the conditioner before the flotation cells and tilting concentrators and the cyclone overflow was sent direct to the tailings. The cyclone, although of primitive design, made a recovery of nearly 95 per cent of the $+9$-μ cassiterite. The removal of the slime had a markedly beneficial effect on the flotation, cutting reagent consumption by half and resulting in almost complete sulphide flotation. The total effect was to double the grade of feed to the tilting concentrator and to halve the quantity treated. Table II gives assays and flow rates for the two tests.

TABLE II.—*Assays and flow rates for two tests with the tilting concentrator*

	Test 1	*Test 2*
Assay of thickener underflow, % Sn	0·70	0·84
Assay of feed to tilting concentrators, % Sn	0·81	1·82
Assay of tailings from tilting concentrators, % Sn	0·53	0·89
Assay of rougher concentrates, % Sn	3·11	7·55
Assay of cleaner concentrates, % Sn	4·01	13·05
Underflow from thickener, tons/h	2·4	2·1
Feed to tilting concentrator, tons/h	2·0	0·8
Pulp density of feed to tilting concentrator, %	15	12

Table III compares the recovery and ratio of concentration of the tilting concentrator with some reported figures for other machines.[1] There ~~having been different types of feed, comparison can only be made very~~ generally, but it would seem that the tilting concentrator is at least as good as the other machines in the field. It must be admitted that the tilting concentrator described is expensive (the price of a single unit was quoted in 1953 as being equivalent to £1400 ex factory).

TABLE III.—*Comparison of the tilting concentrator and other fine recovery machines*

	Tilting concentrator		Helicoid	Vanner	Round frame
Feed, % Sn	0·81	1·82	0·41	0·58	0·59
Concentrate, % Sn	4·01	13·05	2·4	4·95	1·4
Recovery, %	40	55	30	19	20
Ratio of concentration	12·4	13	22	44	8

Double Recovery

The recoveries for test 2, as shown in Fig. 7 are all considerably greater than in test 1, except for the 13- and 18-μ size range, where recovery appears to be slightly depressed. There would seem to be a definite double recovery effect with the deslimed feed, with recovery in the 13- and 18-μ size range being appreciably less than in the size range between 9- and 13-μ and in the size ranges above 18 μ. A similar effect has been noted in the work on vanner and helicoid,[1] with the difference that in both these cases there was apparently no recovery at all in the middle size range. In the case of the vanner this middle range was between 8- and 10-μ and in the case of the helicoid it was between 9- and 13-μ cassiterite. This double recovery effect could also be seen in results quoted in the discussion of an early paper on the tilting concentrator[2] but was not commented upon at the time. Here the reduced recovery was in cassiterite sizes around 18 μ.

Tests by Douglas and Bailey[1] have indicated that the double recovery effect is not found in the case of a round frame concentrator, which suggests that the mechanics of concentration in the use of such units is different from the mechanics of concentration for the tilting concentrator, the vanner and the helicoid. All that these three seem to have in common is that concentration is effected with a disturbed bed, although the mechanism of disturbing the bed is different in each case. For the tilting concentrator the disturbance is caused by the eddies in the flow caused by the depressions in the surface of the rubber covering the decks. In the vanner the disturbance is caused by the vibration of the bed and by the feed and wash water falling on to the settled bed, and in the helicoid the disturbance is applied in the form of vibration. On the round frame the bed settles out from the feed flow on to a relatively smooth surface and is thereafter only disturbed by the surface flow of material and wash water rolling and sliding over the top of the settled bed.

A major exception to this double recovery effect appears to be the shaking table. On the face of it, concentration takes place on a shaking table in a disturbed bed similar to those discussed above, but, as is demons~~trated later, does not give rise to double recovery. Either the proposed~~ theory is at fault, therefore, or, as seems very possible under the complex conditions of concentration on the shaking table, secondary concentration effects combine to overcome the middle range loss at the expense of some of the finest recovery.

If further investigation supports this hypothesis for the presence and absence of double recovery it would seem possible to effect both forms of concentration with the tilting concentrator, the double recovery being effected by the normal decks and the single recovery being effected with smooth-surfaced decks. In the latter case, operation might be improved by using longer leaves and by supplying a wash-water stage prior to tilting. In both cases there would still be the advantage of high throughput for the floor area occupied as compared to other fine-treatment equipment.

Thus far only the conditions required to produce the double recovery effect have been considered without suggesting how it is brought about. It seems very possible that the basic mechanism leading to double recovery in these cases is the same as that suggested by the author to account for the double recovery found in jig and pan concentration.[3] Briefly, this supposes that the high recovery in the finest sizes is made by sub-interstitial sizes of heavy particles collecting under free settling conditions, while the reduced recovery is made in the finest size range of the particles collecting much more slowly under hindered settling conditions, i.e. those particles which are the same or slightly larger in diameter than the spacings existing between particles forming the bed under the conditions of concentration.

The work of Lill and Smith[4] seems to confirm this hypothesis. In a bed of $\frac{1}{4}$-in glass beads they found that steel cubes and spheres had a minimum penetration velocity at a size of about 5 mesh B.S. Below this size the penetration velocity increased rapidly. In a rather wider size range bed containing glass chips from $\frac{1}{8}$ to $\frac{1}{6}$ in. in size, the minimum penetration velocity was shown to occur at about 10 mesh. As the size range of the bed extends downwards so interstitial sizing will obviously become finer, and the work of the author in Malaya showed that a depressed recovery occurred between 30 to 100 mesh in jigs fed with a very long range feed of $-\frac{1}{2}$-in material. It is similarly obvious that a bed formed from a feed substantially below 200 mesh would have a very small interstitial spacing, even when the disturbance of the bed was sufficient to keep the very finest material from obstructing the interstices. If the hypothesis outlined is true, the results suggest that this spacing will be of the order of 10 to 15 μ for the feeds to the slime concentrators in the tests considered. This hypothesis also suggests that the reason why the double recovery effect is not found in round frame treatment is that the bed is formed by particles settling from a slow-moving body of pulp. Under these conditions the interstices of the bed will almost certainly be blocked with very fine unconcentrated material and unless this is distributed by agitation of the bed, the very fine heavy material may not be able to settle into the bed and will therefore be lost to the tailing.

Fine Recovery on Shaking Tables

All the machines referred to—the tilting concentrator, the vanner, the ~~helicoid and the round frame~~—are designed to make a recovery of fine cassiterite from the 'slimes' produced in milling which it has been shown cannot be treated effectively on the shaking table. The word slime is, however, an extremely vague description. In gravity milling it is usually used to describe the thickened final overflows from the various classifiers in the milling process. This product will often have a size analysis similar to some of the feeds in flotation milling and here the term slime is generally reserved for single micron or sub-micron sizes of material which interfere in flotation by consuming reagent or coating coarser particles.

In this sense, slime is used to denote fine particles which interfere with the recovery of the coarser values and it is suggested that this meaning can usefully be extended to the gravity milling field. Here, the slime treated by the special machines already mentioned consists in fact of two parts—a coarser which can readily be concentrated on the average fine sand table, and a finer which interferes with this type of concentration by increasing the density and viscosity of the pulp on the table and interfering with the normal concentrating process.

The detrimental effect of this finest material on normal tabling is well known and much of the slime produced in normal gravity concentration is the product of efforts to remove such material from the sand tables. It does not seem to be generally recognized, however, that material down to about 20 μ or even finer can easily and efficiently be treated on the normal fine-sand table provided that the extremely fine material is absent.

The improvement in the recovery of the finer sizes on the tilting concentrator made by reducing the quantity of fines in the feed has been demonstrated in the work already described. Fig. 8 shows the recoveries made in treating the final concentrates from the tilting concentrator, which contain virtually no material less than 5 μ, on a fully-riffled sand table. If this is compared with the recoveries shown in Fig. 7, it is evident that the recoveries made in the different size ranges in tabling the two different grades of concentrates from the tilting concentrator are all very much higher than any of the recoveries made in these same size ranges by the tilting concentrator.

The respective grades of feed, tailings and concentrates from the shaking table are given in Table IV, the results indicating an overall recovery of 75·5 and 94·0 per cent, respectively.

TABLE IV.—*Operation of a shaking table*

	A	*B*
Feed, % Sn	4·01	13·05
Tailing, % Sn	1·06	1·04
Concentrate, % Sn	41·4	49·29
Ratio of concentration	13·7	4·0

The size and assay distributions of the two respective concentrates are given in Table V.

TABLE V.—*Shaking table concentrates*

Size equivalent cassiterite spheres, μ	*A*			*B*		
	Wt. %	Tin assay %	Tin dist. %	Wt. %	Tin assay %	Tin dist. %
> 36	48·9	32·44	38·4	44·4	48·23	43·5
< 36 > 27	14·8	50·52	18·1	23·8	50·39	24·3
< 27 > 18	24·7	53·04	31·7	21·6	52·19	22·9
< 18 > 13	7·0	50·26	8·5	5·1	51·37	5·3
< 13	4·6	31·82	3·3	5·1	39·75	4·0
	100·0	41·41	100·0	100·0	49·29	100·0

The very high recoveries made in all sizes from 13 to 36 μ shown in Fig. 8 are particularly impressive and are far better than anything achieved by the normal run of slime concentrators for an equivalent ratio of concentration. It must be admitted that the feed rates to the tables in these cases were only of the order of 100 kg/h, which means that to treat any tonnage of material the number of tables required would be considerable, but this must be set against the much higher recovery attainable.

The low recovery in the +36-μ size range of cassiterite is interesting, but unfortunately it cannot be said definitely whether this loss is due to

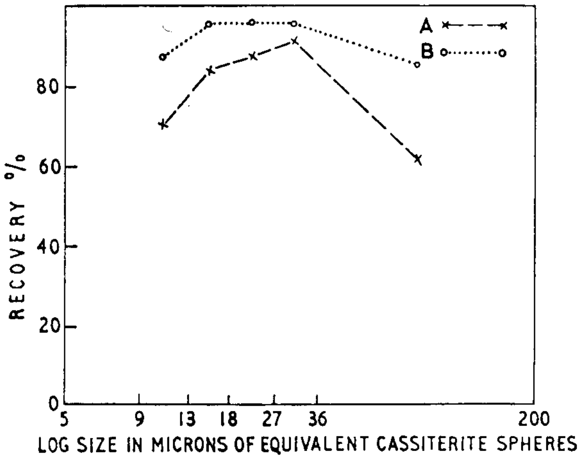


Fig. 8.—Recovery of cassiterite on a shaking table.

the mechanics of the operation tending to reject coarse cassiterite or, as seems more likely, to some of the +36-μ cassiterite being in the form of locked grains with insufficient density differential to be collected with the concentrate. In practice it would obviously be preferable to ensure that the coarsest material (in this case the +36-μ cassiterite range) were treated separately and this table concentration stage restricted to the −36+10μ cassiterite.

Until recently the whole argument presented here would have appeared academic in character, since there was no convenient method of classifying this type of slime in a way which would give the separation required. However, with the general use of hydrocyclones it is now quite a simple matter to separate dilute pulp, with cyclones correctly chosen and operated, fairly sharply in the region equivalent to 20-μ quartz. Indeed much finer splits can be made with specially selected equipment. This cyclone overflow then becomes the true slime product produced in the milling. The cyclone underflow, which, if correctly produced, will contain very little of the true slime size range of material, can properly be called a fine sand and with correct treatment should give a high recovery of values. The slime product could be thickened and it would be interesting to see which of the many machines for treating the present 'slimes' would make the best recovery on this much finer product. In treating this micron-size material the surface charges on the particles will begin to play a predominant part in the recovery and it may be found that much of the work on zeta-potential measurements, which until lately has been of interest mainly in flotation and slime flocculation and dispersion, will become of practical interest in gravity concentration. By correct reagent additions it may be found possible to reduce the surface charges on the cassiterite to the point when van der Waals attractive forces overcome the repulsion of the individually charged particles. If this could be done selectively without affecting the surface charge of the gangue particles to this point, the cassiterite particles would flocculate and greatly improve the possibility of their recovery.

Conclusions

The foregoing discussion suggests that a better recovery of fine cassiterite would be made from the slime produced in milling lode-tin ores if the dilute pulp were first treated in medium-sized cyclones at low pressure to remove the cassiterite coarser than, say, 40 μ. The underflow from this stage would be tabled normally to recover free cassiterite, but it would appear that in some cases there is still a considerable quantity of locked cassiterite in these sizes and it might be necessary to regrind the tailings from this tabling stage. The overflow from the first cyclones could then be recycloned at higher pressures in small units to give a split in the region of 10-μ cassiterite. At times it might be necessary to dilute the first underflow with clean water and recylone to remove nearly all the finest slime and give the highest table recoveries. The underflow from these cyclones would best be treated on fine sand-tables to give the highest recovery possible, but if the tonnage were excessive some other form of concentrator which gives a higher throughput for a given floor area might be preferred. The overflow from this second stage of cycloning, which should contain little +10-μ cassiterite, and for which there exists at present no convenient way of further sizing, can now be considered a true slime product. This could be thickened and treated on one or other of the slime concentrating machines, with, perhaps, chemical additions to influence the surface charges of the mineral particles present and increase the possibility of cassiterite recovery by selectively flocculating the cassiterite.

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I have only one minor criticism of this excellent paper. The section on performance would have been improved by a comparison between results obtained on several widely differing soil types by the authors' field determination and by one of the lengthy but thoroughly reliable methods for small quantities of cobalt, such as the nitroso-R salt procedure.

Gravity Concentration of Fine Cassiterite

I. R. M. CHASTON, A.R.S.M., B.Sc., ASSOCIATE MEMBER

Author's reply to further discussion on paper published in January, 1962* (Transactions, *vol. 71, 1961–62*), *pp. 215–25*

Mr. I. R. M. Chaston: In his further contributed remarks, Mr. Williams suggests that screening at 50 to 100 mesh would offer attractive scope for tabling the small quantity of fines from jig tailing which contains the bulk of the loss of heavy minerals. The writer installed one such system in Malaya where the cleaner-jig tailing was screened with a sieve-bend with 0·7-mm apertures and the undersize pumped through a small cyclone with the cyclone underflow feeding a Wilfley table. The results are shown in Table I. About 70 per cent of the tin in the table feed was recovered in the concentrates. Obviously a finer split would have been better since there was hardly any tin in the table feed coarser than 200 mesh. The screen was changed to one with an aperture of 0·5 mm, and later to one with an aperture of 0·25 mm with the results on the undersize as shown in Table II.

TABLE II.—*Size range of undersize with sieve-bends of different apertures*

Size mesh B.S.	*0·5-mm aperture undersize Wt. %*	*0·25-mm aperture undersize Wt. %*
+52	8·5	2·3
−52 +100	18·8	6·7
−100 +200	39·9	38·8
−200	32·8	52·2
	100·0	100·0

These sieve-bends were made with 5-mm stainless steel bars and lasted about 600 hours, being turned daily.

Some operators of sieve-bends have expressed disappointment at their performance, but this can usually be traced to incorrect methods of application. In particular, needless use is often made of the potentially very high

*Vol. 71, p. 686.

TABLE I.—*Screening and tabling the tailing from a jig*

Size mesh B.S.	Feed to sieve-bend			Sieve-bend oversize			Feed to table (sieve-bend undersize after cycloning)			Table tailings			Table concentrates		
	Wt. %	% Sn	Dist. Sn %	Wt. %	% Sn	Dist. Sn %	Wt. %	% Sn	Dist. Sn %	Wt. %	% Sn	Dist. Sn %	Wt. %	% Sn	Dist. Sn %
+10	63·3	0·04	23	73·3	0·05	62	—	—	—	—	—	—	—	—	—
−10 +22	9·5	0·04	3	10·8	0·05	9	—	—	—	—	—	—	—	—	—
−22 +52	8·3	0·07	5	12·6	0·07	15	19·1	0·04	2	16·5	0·05	9	—	—	—
−52 +100	2·8	0·22	6	1·8	0·14	4	18·7	0·04	2	29·7	0·04	14	12·8	0·90	—
−100 +200	4·4	0·10	4	0·8	0·20	2	19·9	0·07	4	24·2	0·04	11	29·9	2·96	3
−200 +300	5·7	0·13	7	0·3	0·15	1	19·5	0·24	13	15·4	0·05	9	23·3	23·37	20
−300	6·0	0·97	52	0·4	1·05	7	22·8	1·23	79	15·2	0·32	57	34·0	62·25	77
	100·0	0·11	100	100·0	0·06	100	100·0	0·36	100	100·0	0·09	100	100·0	27·61	100

capacity of the sieve-bend which has to be paid for in a high rate of screen wear. Poor design often leads to only part of the screen being used and this gives uneven wear to the screen bars. This is often the case when too long a screen is used; screening then is completed before the end of the screen is reached and it becomes blocked. This affects the feed flow over the rest of the screen and often eventually causes blockage over the entire screen. Increasing the feed velocity over part of the screen overcomes this trouble, but leads to uneven and accelerated wear.

Where the amount of feed is not great it is preferable to use a low-velocity feed spread over a wide but very short curved screen. With the low feed-velocity, wear rates are low and ordinary woven-wire screening can be used. Table III shows results using this type of screen on a scavenger jig

TABLE III.—*Low-velocity screening on a curved 40-mesh woven-wire stationary screen*

Size mesh B.S.	*Screen oversize*	*Screen undersize*
+30	0·4	—
−30 +52	40·9	1·6
−52 +60	23·1	4·0
−60 +72	17·0	12·3
−72 +100	12·0	30·9
−100 +150	5·3	35·6
−150	1·3	15·6
	100·0	100·0

concentrate where most of the cassiterite was finer than 100 mesh. The screen oversize was returned to the primary jig for retreatment and the greatly reduced amount of concentrate collected in the screen undersize was tabled. The screen was made of brass 40-mesh woven-wire and lasted over a month.

This type of screen can be used for finer splits down to 200–300 mesh, but both for this, and for the conventional type of sieve-bend, several stages of dilution and re-screening are needed to give a clean oversize, however dilute the initial feed pulp may have been. To avoid excessive and expensive use of fresh water it is possible, where the undersize is subsequently cycloned, to circulate a portion of the cyclone overflow for the dilution of the oversize between screening stages.

OBITUARY

Ryno de Beer, O.B.E., died on 2nd October, 1961, at the age of 71.

Born in South Africa, Mr. de Beer joined Glen Deep Gold Mining Co., Ltd., Johannesburg, in 1906 as learner, and by 1909 had held the posts of assistant assayer and chief assistant assayer. He then worked for a year at Geldenhuis Deep, Ltd., Johannesburg, as chief assistant assayer. He left South Africa in 1910 for Southern Rhodesia, where he spent the remainder of his life.

Mr. de Beer was employed for four years as chief assayer, smelter and sampler by Eldorado Gold Mining Co., Ltd., and in 1914 transferred to Shamva Mines, Ltd., as chief assayer. Promoted cyanide manager later that year, he remained in charge of metallurgical work at Shamva until 1929 and during that period was also consulting metallurgist to Goldfields Rhodesian Development Co., Ltd. He did metallurgical work for the Mayfair and Wanderer Gold Mining Companies for nine months in 1929, also serving as reduction officer to the latter company.

In October, 1929, he was appointed metallurgist to the Southern Rhodesian Government and established the metallurgical laboratories at Salisbury. He managed Antelope mine during 1931, and later the same year became manager and part owner of Red Hill mine, Selukwe. Five years later he took over the management of Kamchatka mine, at the same time carrying out consulting metallurgical work on behalf of Thistle Etna Gold Mining Co., Q.Q. Mines, Ltd., and other small mines.

Mr. de Beer was invited to join the Southern Rhodesian Government's Roasting Plant Board on its formation in 1937 and remained a member until his death. He was appointed manager of the Government Roasting Plant, Que Que, in 1956, and at the time of his death was consultant.

In 1945 Mr. de Beer returned to Shamva mine and worked on his own account until it closed in 1956. He also served on the board of Eastern Smelting Co. for some eleven years, and was consultant to Globe and Phoenix Gold Mining Co., Ltd.

He was awarded the O.B.E. in 1955 for his services to the mining industry.

Mr. de Beer was elected to Associate Membership of the Institution in 1914 and transferred to Membership in 1938. He was also a Member of the South African Institute of Mining and Metallurgy.

Albert Huddleston died in hospital at Jinja, Uganda, on 11th July, 1961, at the age of 46.

He was born in Millom, Cumberland, and studied at Hatfield College, Durham University, from 1933 to 1937, gaining a first-class B.Sc. honours degree in geology. ~~He worked during the vacations in the laboratories of Millom and Askam Haematite~~ Iron Co., and took part in the National Union of Students three-month expedition to Spitzbergen in 1936.

Dr. Huddleston was awarded the degree of M.Sc. (Geology) by the University of Durham in 1946, and in 1952 that of Ph.D. (Geology) by the University of South Africa.

His first engagement after graduation in 1937 was with Gold Coast Selection Trust, Ltd., as junior mining geologist, and two years later he joined the Gold Coast Geological Survey Department. A few months later he was commissioned in the 53rd Gold Coast Field Coy., and was attached to various commands, his military service being mainly on geological and geophysical investigations connected with underground water supplies in Kenya, Tanganyika, Abyssinia and Somalia. He was demobilized in February, 1944, with the rank of captain.

Dr. Huddleston resumed work with the Gold Coast Geological Survey Department until his transfer in 1946 to the Mines and Geological Department, Kenya Colony. There he was largely engaged in regional geological survey of Kisii and Kakamega goldfields, and was the author of published reports on those areas. In 1952 he returned to the Gold Coast to take up the appointment of Deputy Director of the Geological Survey there, and held that post until his retirement in 1956.

He went back to Kenya and worked for a time as administrative secretary to the East African Veterinary Research Organization, but in 1959 joined the Uganda Geological Survey Department where he was serving as geologist at the time of his death.