Performance of a Shaken Helicoid as a Gravity Concentrator*

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SYNOPSIS

The enrichment and recovery of valuable minerals from fine slimes is a universal problem in the field of mineral dressing; various machines have been designed for operation on these minerals, with differing degrees of success.

The shaken helicoid was designed and developed by Dr. C. R. Burch of Bristol University for operation with slime, or near slime, feeds. This report describes a short programme of test work carried out to ascertain its capabilities and its operational characteristics. The examinations reported, though not extensive, indicate the effects of pulp density, frequency of vibration, rate of pulp flow and size of feed material.

The performance of the helicoid is compared with that of a Frue vanner and round frame operating on a Cornish tin slime. In general, the overall performance of the helicoid appears to be superior to that of the vanner and round frame and it is estimated that the use of the helicoid should give recoveries up to 40 per cent with enrichment ratios between 6 and 22.

THE SHAKEN HELICOID[1], which was described in a lecture by Dr. Burch[2] to the Cornish Institute of Engineers, was developed as a result of his interest in the Cornish tin mining industry, particularly in the relatively large quantities of cassiterite normally associated with the slimes which are rejected in the mine tailings. To recover the whole or a proportion of the values might have a considerable effect on the present economy of this industry; the helicoid was, therefore, designed with this object in view. Various types and sizes of helicoid were built and tested by Dr. Burch and his assistants, Dr. R. H. J. Sellin and Mr. R. H. Mozley, and eventually continuous tests were carried out on a 10-in diameter machine at the Hawk's Wood tin mine in Cornwall, where, according to Dr. Burch, an enrichment ratio‡ of about 5 was obtained. Having proved the effectiveness of his machine, he continued with laboratory-scale development work. These experiments culminated in the manufacture of a large fibreglass model of the helicoid, upon which Warren Spring Laboratory agreed to carry out a limited test programme, in order to determine its operating characteristics and also to provide an independent opinion of its effectiveness as a concentrating device.

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

‡Enrichment ratio $= \dfrac{\%\text{ values in concentrate}}{\%\text{ values in feed}} =$ recovery × concentration ratio.

DESCRIPTION OF HELICOID

Photographs of the helicoid are reproduced in Figs. 1 and 2 (Plate I). This particular model is moulded in the form of a 3-ft diameter spiral having a 2-in pitch and an overflow trough running down the entire length of the periphery. A vertical steel spindle is fitted down the central axis of the spiral and two 'out-of-balance' weights, which rotate at controlled speeds about this spindle, impose horizontal circular vibrations on the system. These vibrations are facilitated by the suspension, which comprises three thin vertical wires, each equally spaced and attached to the helicoid and to an upper triangular steel supporting frame.

In operation, the pulp is fed from the header tank to a fantail distributor situated at the top of the spiral and the rate of flow is controlled either by an automatic metering valve, or by inserting a calibrated orifice in the tank outlet and adjusting the head as required. The pulp is cut into fractions at the discharge end of the spiral by seven launders disposed across the radius and feeding into the separate collection vessels shown in Fig. 3 (Plate II). For the purpose of these tests the launders have been numbered 1–7 in an outward direction across the radius of the helicoid.

According to the theory propounded by Dr. Burch, the concentration of heavy minerals on the helicoid is achieved by a combination of two well-known physical effects; the radial flows which are imposed by the 'river bend' action, and the forces exerted on particulate matter contained in a fluid when it is subjected to shear forces.

As the pulp flows down the flat low-pitched vibrating spiral, shear forces are imparted to the fluid and, according to the action described by Bagnold,[3] these impose vertical forces on the solids, tending to move them upwards towards the surface. The effects of these forces are related to particle density, consequently the lighter materials tend to separate and move in the direction of these forces, i.e. to the surface and away from the bed of the spiral. Because of the centrifugal force and the unbalanced pressures which occur when a fluid is confined to a horizontal circular flow-path, transverse radial currents are superimposed on the main flow; these move radially inwards along the bottom of the channel and radially outwards along the surface of the fluid.

In combination, these two effects cause the lighter particles in a pulp to rise to the surface where the outward-flowing currents transport them to the periphery of the helicoid, while the heavier particles on the bed of the spiral will be concentrated, by the inward-flowing currents, towards the centre of the spiral.

OPERATIONAL CHARACTERISTICS OF THE SHAKEN HELICOID

Tests and Procedures

Although this machine had already undergone a number of tests at Bristol University, where the products had been examined and their assays compared, no data relating to the conditions of operation had been recorded. This report presents the results of additional limited tests which were carried out under controlled conditions to verify the operating characteristics of the shaken helicoid. The effects of variations in the

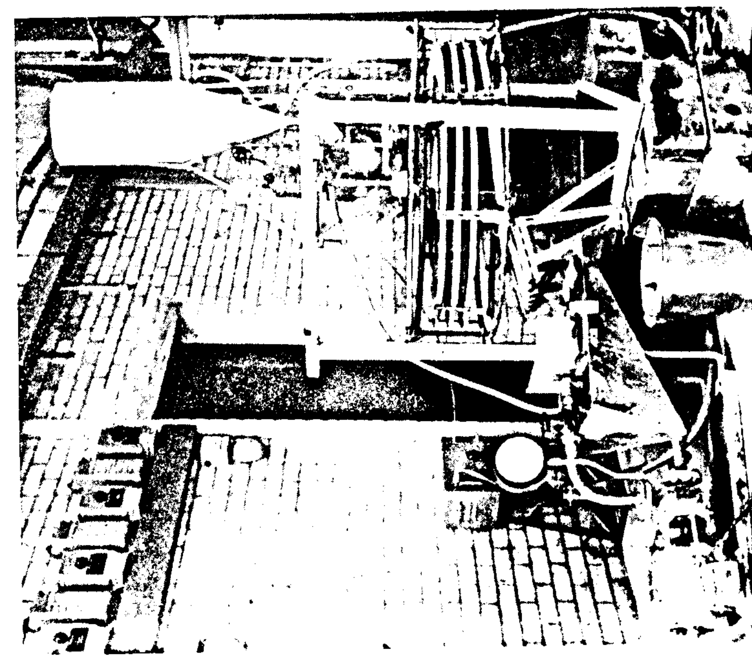

Fig. 1.—Shaken helicoid.

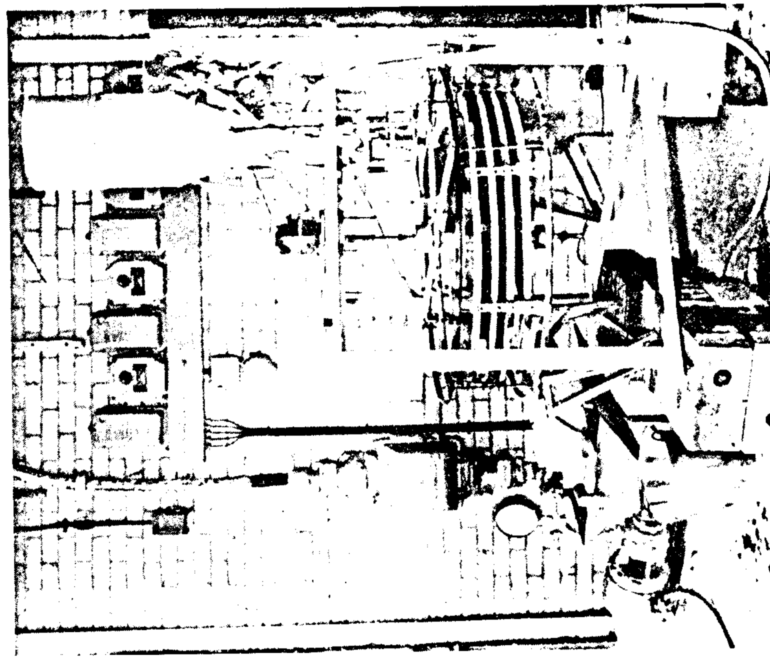

Fig. 2. —Shaken helicoid.

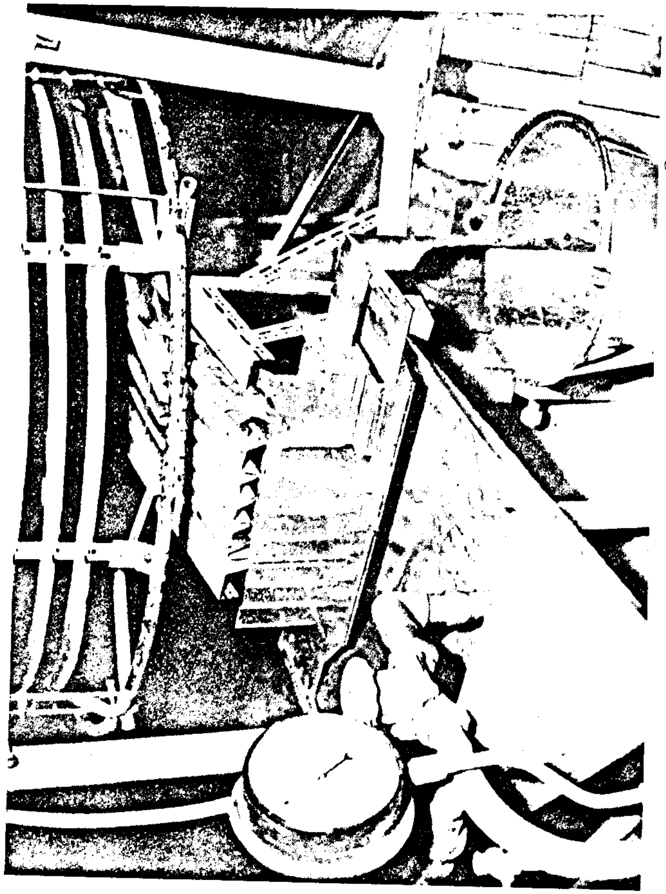

Fig. 3.—Product discharge and sampling arrangements.

frequency of vibration, pulp density, rate of feed and size of feed were investigated and the results are reported.

The helicoid was operated in closed circuit, as indicated in the diagram shown in Fig. 4, using a standardized procedure for all the tests. After adjusting the controllable variables, operations were allowed to continue for a period of approximately $\frac{3}{4}$ h until equilibrium conditions were attained.

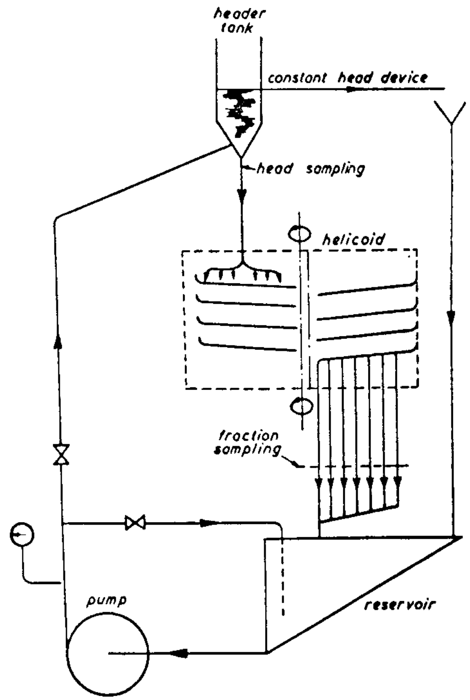

Fig. 4.—Diagram showing closed circuit flowsheet of test rig.

During each test, in addition to head samples, seven separate products were collected by sliding the trough-type splitter into the effluent flow. These samples were filtered, dried and analysed for heavy mineral; in certain instances, size assay analyses were made on the product using beaker decantation for sub-sieve sizing.

The materials used for these investigations were Hawk's Wood wolframite slimes, 74 per cent of which were finer than 300 mesh, and various classified fractions of a South Crofty tin ore.

Mineralogical examination of such finely-divided materials proved difficult and time-consuming. However, cursory examinations showed that approximately 70 per cent of the wolframite in the Hawk's Wood material was completely liberated and that practically all the cassiterite in South

Crofty vanner and round frame feeds examined later appeared to be unlocked.

Analyses

The relative usefulness of chemical and vanning assays is a subject of considerable controversy, each possessing certain advantages. Throughout these tests, knowing the limitations involved, chemical assays have been used to determine the values in the various samples.

As an immediate analytical service was not available during the initial investigations, the conditions for each test point were selected without previous knowledge of the characteristics being explored. For this reason, certain ranges, which now apparently require more detailed investigations, were not fully examined, although the approximate characteristic shapes of response curves were determined.

Performance Characteristics

During the initial stages of each test a rapid build-up of solids occurred on the bed of the helicoid and the rates at which these travelled towards the discharge were considerably less than the velocity of the fluid flow. Consequently, with the recirculating system employed, the solids content in the feed was higher than, but varied in sympathy with, that in the effluent, until the deposit on the bed reached a condition of equilibrium.

Figs. 5, 6 and 7 indicate typical results obtained during tests with a tin ore, and are presented to introduce the several characteristics of the helicoid's performance. The concentration of heavy mineral, represented by the ratio of enrichment in each fraction produced under three sets of operating conditions, is demonstrated in Fig. 5. This confirms the trends expected from the theoretical surveys. It can be seen that the enrichment ratio exceeds unity in fractions 1 and 2 and only in the former is a worthwhile beneficiation achieved. However, to produce a balanced assessment of such results they must be considered in conjunction with their respective recoveries. In Fig. 6 these are shown to vary from 33 to 39 per cent and 23 to 38 per cent in fractions 1 and 7 respectively and to less than 16 per cent in fractions 2 to 6.

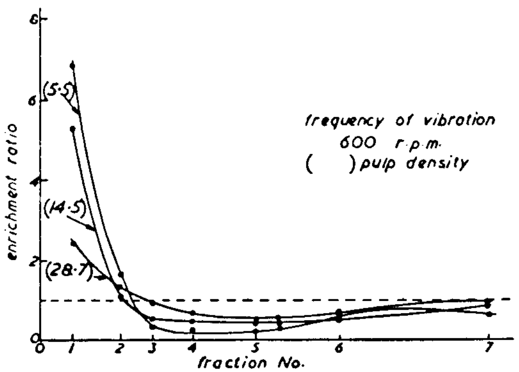

Fig. 5.—Enrichment ratio and fraction number.

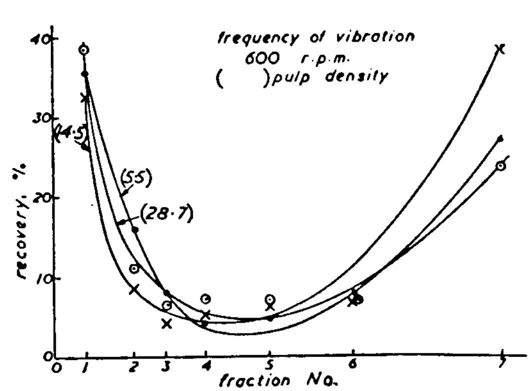

Fig. 6.—Recovery and fraction number.

These results have been combined in a slightly different form and plotted in Fig. 7, which indicates the respective enrichment ratios and recoveries which could be obtained from the cumulative combination of fractions 1 to 7. Approximately 35 per cent of the values are shown to report in fraction 1, where the enrichment ratio varies from 2·4 to 7, depending upon the conditions of operation. It would appear from these results that the collection of fraction 1 as concentrate might result in optimum beneficiation, although the particular role of the helicoid would have to be considered, and a combination of fractions 1 and 2 might provide optimum conditions for a rougher operation.

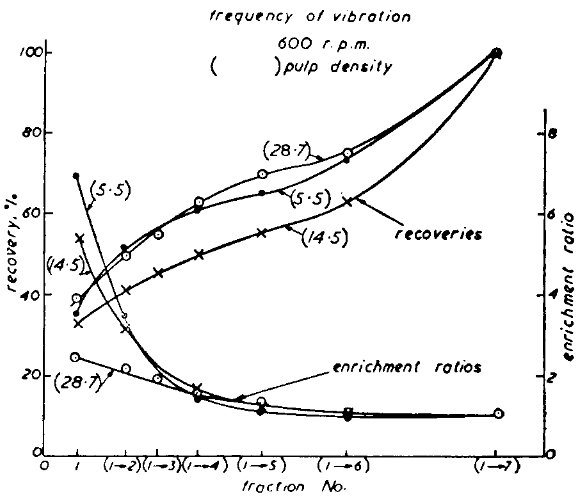

Fig. 7.—Cumulative enrichment and recovery values for combined fractions.

In addition to exploring the concentrating capabilities of the helicoid, one set of results relating to a tungsten ore is presented to demonstrate the degree of classification achieved in a radial direction across the outlet (Fig. 8). That classification occurs, particularly in sizes greater than 20μ, is readily apparent from the relative size characteristics of samples 1–7, with the coarser products reporting towards the centre of the helicoid.

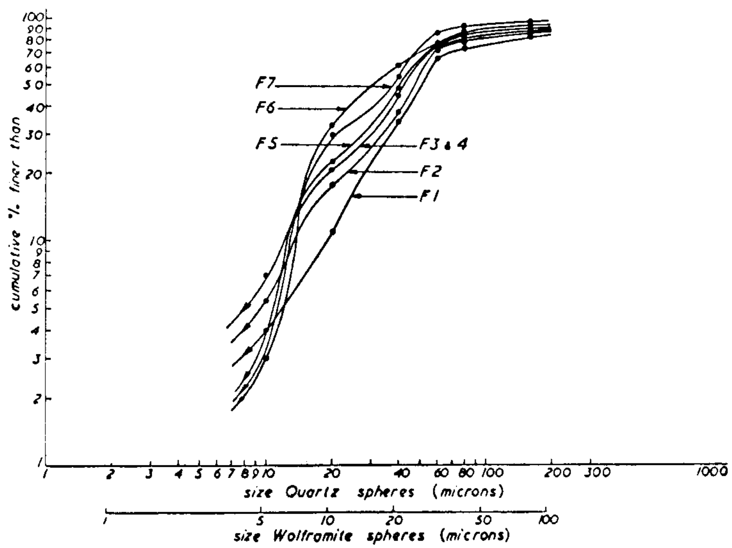

Fig. 8.—Classifying action.

Additional information is given in Fig. 9, where eight characteristics are presented; numbers 1–7 refer to the individual samples extracted from the helicoid while the remaining one refers to the head. These curves are the first differentials of the size recovery curves, their enclosed areas being proportional to the total wolframite content of the head. In addition they indicate a relative amount of wolframite contained in each micron subdivision of each of the eight samples.

An examination of the characteristic F1, and a comparison between this and the head characteristic, show that a large proportion of the values in the coarser sizes, above 20μ, are concentrated in fraction 1. It is also clear that the finer fractions in this sample contain more heavy mineral than those in samples 2, 3, 4, 5 and 6. However, the distribution graph for sample F7 indicates the occurrence of a reverse concentration in respect of the fine material, and a large preponderance of values contained in the finer fractions of the head congregate in the lower size ranges of the sample extracted at the periphery of the machine. For example, more than half of the tin contained in the 5μ material is concentrated in fraction 7.

The results of this particular investigation are not sufficiently detailed to elucidate completely the mechanisms involved in producing such a distribution of values. However, it is interesting to note the double-nodal form of the head characteristic and to examine this in relation to the mono-mineralic single-node characteristics which represent fractions 1–7. From this it would appear that the heavy mineral is present in at least two forms, one of which, being more readily amenable to this type of separation, reports largely in fraction 1, while the other is distributed between fractions 2–7.

The recovery of values in relation to their size is clearly illustrated in Fig. 10, in which size recovery curves are plotted for two conditions of operation. With the higher pulp density the recoveries at sizes in excess of 32μ are greater than 80 per cent, while at the lower feed density the recovery of 10μ material exceeds 30 per cent.

Mineralogical reports have indicated that 70 per cent of the values in this material are released, and, assuming this applies mainly to the smaller size ranges, it is estimated from size distribution curves that composites occur only in sizes greater than 50μ. Consequently the characteristics shown in Fig. 10 would be expected to reach a maximum at approximately this size.

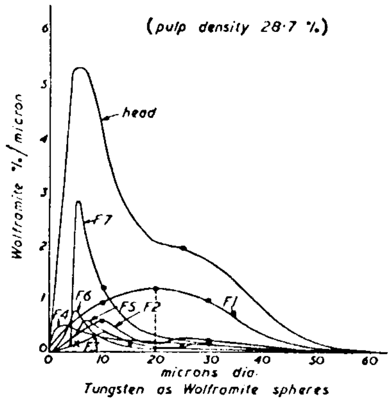

Fig. 9.—Helicoid performance.

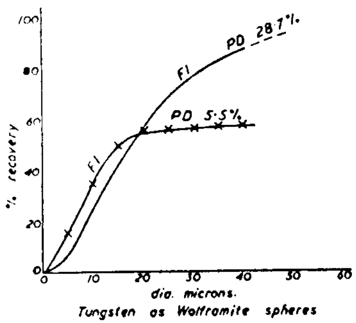

Fig. 10.—Size recovery curves.

The results confirm this when operating with high pulp densities, but with a solids content of only 5·5 per cent in the feed, the maximum recovery occurs, and becomes constant, at a size of 20μ and above. It is suggested that the losses can be attributed to a reduction in the depth of silt on the bed of the spiral. This, in conjunction with the contour of the bed around the area from which the concentrate is extracted, results in a proportional decrease of silt collected in this fraction; consequently there is a relative deficiency of coarser particles, resulting in a reducing recovery above a size of 20μ.

Pulp Density

The results of some of the tests in which wolframite slimes were processed are shown plotted in Fig. 11; they demonstrate the correlation between the ratio of enrichment in the first fraction and the pulp density for two different frequencies of vibration of the helicoid. Within practical limits, the effectiveness of the machine varied in an inverse relationship to the pulp density, and maximum enrichment ratios of between 6 and 7 resulted when pulp densities of 5–10 per cent were employed. With pulps containing less than 5 per cent solids the operating characteristics of the equipment appeared to undergo a considerable change; the quantity of product reporting in fractions 1 and 2 decreased, in some instances to zero in fraction 1, and the resulting enrichment ratios experienced a considerable 'spread' (between 2 and 14), with corresponding recoveries around 2 per cent.

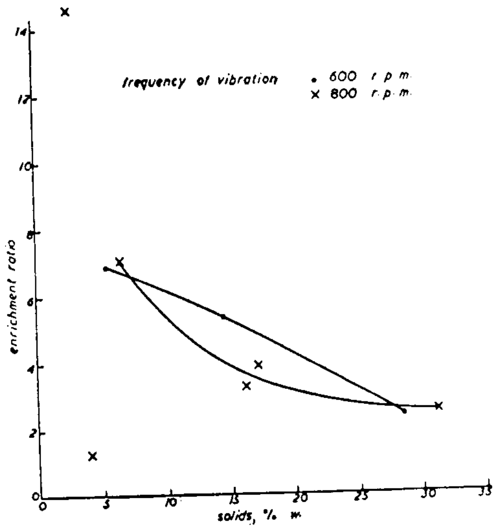

Fig. 11.—Solids in pulp and enrichment ratio.

The respective recovery figures obtained from the tests are indicated in Fig. 12, where values in excess of 30 per cent are reported over a wide range of pulp densities. This result would appear to be reasonable when allowances are made for the fineness of feed and the fact that the concentrate was collected from only one of the seven offtakes.

Frequency of Vibration

The existence of a critical and somewhat restricted range of operating speeds is apparent when the effects produced by variations in the frequency of vibration upon enrichment ratios and recoveries are examined. The

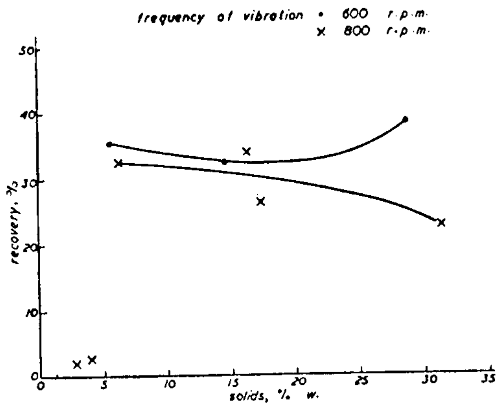

Fig. 12.—Recovery and solids in pulp.

relationships which exist between these variants are shown in Figs. 13 and 14 where parameters depicting the influence of changes in pulp density are also included. Optimum conditions occur between 600 and 700 c/min and further investigations within this range might produce results superior to those indicated by the extrapolated characteristics in these diagrams. Poor results can be expected if operating speeds below 600 c/min are employed and similarly this may be true at speeds in excess of 800 c/min.

It is difficult without further investigation to postulate a convincing theory to account for these limitations. However, it is of interest to note that the velocity of rotation of the bed at the optimum frequency coincides

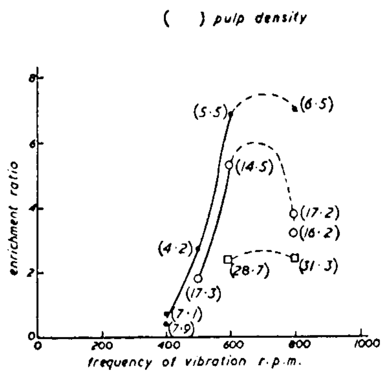

Fig. 13.—Frequency of vibration and enrichment ratio.

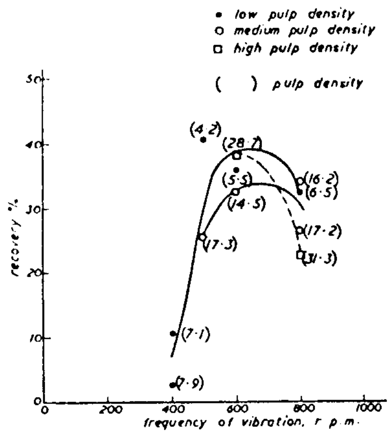

Fig. 14.—Frequency of vibration and recovery.

with the mean velocity of flow of the pulp. It is recognized under these conditions that no shear occurs between the pulp and the bed when the latter moves in the direction of the flow and that maximum shear is imposed when the movement of the bed is in the opposite direction. It is suggested that this alternating shearing action, from zero to maximum shear, may induce a concentration of the heavy values towards the centre of the helicoid.

Rate of Pulp Flow

Preliminary tests were made to examine the effects which different flow-rates had upon the performance of the helicoid, but were not pursued when it became obvious that the range over which the flow could be varied was limited. It was difficult to wet the complete surface of the spiral at pulp flow rates below 7 l./min, while at rates exceeding 9 l./min the level of the pulp built up and overflowed the peripheral trough. Hence, a flow rate of 8 l./min was maintained throughout these tests.

Amplitude of Vibration

In the limited time available it was not possible to investigate the effects of variations in the amplitude of vibration; consequently the machine was operated with the settings used by Dr. Burch in his tests. However, an optimum setting, particular to each set of conditions would be expected, especially frequency of vibration, and related to size, shape and density of the minerals involved.

Size of Feed

Cursory explorations were carried out using two different sizes of wolframite (Hawk's Wood slimes, of which 74 per cent was finer than 300 mesh, and Hawk's Wood sands, having 6 per cent finer than 300 mesh) to indicate the operational trends when the size of feed was varied. There was surprisingly little difference in the results, which are presented in Figs. 15 and 16. It may be concluded generally that the results from the coarser material were slightly inferior to those obtained when slimes were treated, but this conclusion should be qualified, as improved results might be obtained if the conditions of operation were adjusted to accommodate the beneficiation of coarser feeds.

It is of interest to note the concentration which occurs in fraction 7; from Fig. 9, it may be deduced that this is largely due to the presence of released fine heavy mineral (5–7μ) together with locked particles in the 'sand'. This trend suggests that a classifying process might be profitably employed before concentration, particularly when coarser feed materials are encountered.

A further variation in performance was noted and attributed to changes in the size of feed. In certain tests the position of the flow on the spiral changed and no material reported in fraction 1 at the helicoid offtake. In such instances the concentrate had to be taken from fraction 2 and often its efficiency and recovery was inferior to that expected, indicating that the available spiral capacity was not being fully utilized.

This condition occurred when extremely fine materials were treated; this fact and the contour of the bed of the spiral are considered to be the

main factors which affect it. When fine slimes are being concentrated, a greater proportion of solid material moves with the main flow, but with coarser feeds more material silts up on the bed. The area available to the main flow is thereby reduced and the level of the fluid rises and extends

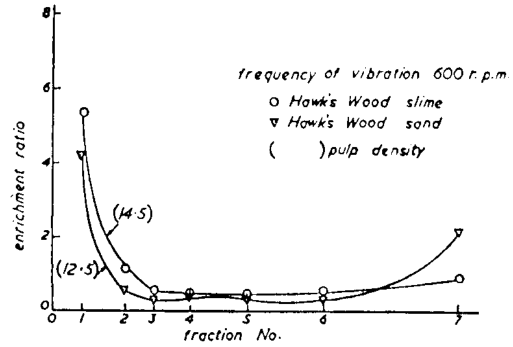

Fig. 15.—Ratio of enrichment and fraction number.

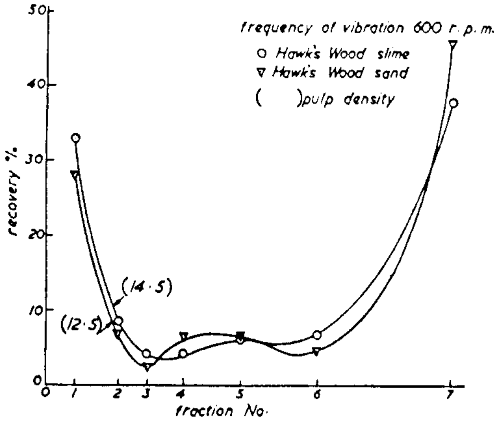

Fig. 16.—Recovery and fraction number.

accordingly. The improved results obtained under 'silted up' conditions suggest that an even flatter contour than that already provided might improve operations still further. Also, in practice a variable splitter would obviously be required to achieve the most suitable conditions of concentration.

Location of Values

The increased concentration of the values towards the centre of the helicoid has already been demonstrated in Fig. 5, and further examinations of the products have been carried out to determine the exact division of the heavy mineral between the less fluid coarser fractions, which silt out on the bed of the machine and the free-flowing pulp above.

Twelve samples were obtained; six were extracted under operating conditions and six were taken from the silt remaining on the deck after operations had ceased. These were analysed for tin; the results are presented

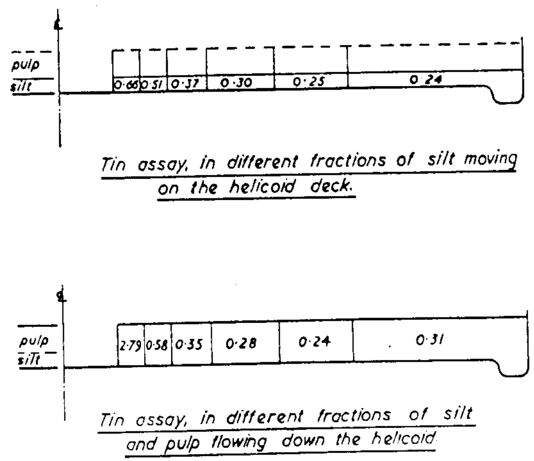

Fig. 17.—Radial distribution of values at discharge.

in Fig. 17. This figure represents diagrammatic cross-sectional views of the helicoid on which the positions and tin assays of the various offtake samples are indicated. From this it appears that most of the values in the higher-grade inner concentrate remain in the free-flowing pulp, which immediately suggests a means by which the efficiency of the helicoid might be improved. For example, it might be possible to position a horizontal splitter to deflect the high-grade pulp from the concentrate fraction or, alternatively, a specially constructed sieve bottom could be employed at this offtake.

Conclusions

The performance of the helicoid was found to vary considerably, according to the conditions under which it was operated, though its general characteristics were in accord with the theory, advanced by Dr. Burch, that a concentration of heavy mineral tends to build up towards the centre of the spiral.

Two main factors were found to be of prime importance to the operation of the machine; these were the pulp density of the feed and the frequency of vibration. The optimum value of pulp density was found to be approximately 10 per cent solids and variations from this figure produced reductions in the ratios of enrichment and/or in the recoveries. Optimum

frequency of vibration, when operating with an amplitude of about ⅛ in., occurs within the range 600–700 c/min; beyond these limits the effectiveness of the machine was found to fall rapidly.

The operation of the machine was influenced noticeably by the size and quantity of material in the feed. In particular, the radial position of the products leaving the machine was affected by these variants, and with increasingly finer or less dense feeds the quantity of pulp collected from the inner sampling position decreased, in some instances to zero.

Results produced on two differently sized feeds (6 per cent and 74 per cent finer than 300 mesh) were shown to be similar, although slightly higher ratios of enrichment and recoveries resulted from tests made with the finer material. However, this result cannot be considered as indicative of the general characteristics of this machine, and further investigations, in which the amplitude of vibration is adjusted to the size of feed, are recommended.

The size distribution across the radius of the helicoid does not vary continuously to produce a completely classified product, although a tendency for coarser fractions to form towards the central axis was noted.

A single helicoid, of the size and type tested, fed with a pulp containing about 10 per cent solids and vibrating at 600–700 c/min, will process about 110 lb feed per hour. The specific power consumption would be approximately 0·65 h.p. per ton and recoveries of about 30 per cent, related to enrichment ratios of 6–7, might be anticipated.

The investigation was restricted to tests made with tin and tungsten ores having a siliceous gangue; the specific gravities of cassiterite and wolframite were similar and the effect of changes in the gravity differential were not investigated.

Scale-up factors for the helicoid remain unexplored and without test data to indicate the interdependence of the many design features involved (i.e. pitch of spiral, shape of bed, diameter of spiral, frequency of vibration, etc.) it is difficult to predict their effects on performance. On reflexion, however, it would appear that with a constant pitch, shape and flow depth, relationships might exist in which throughput and diameter are directly proportional.

COMPARISON OF THE HELICOID WITH COMMERCIAL MACHINES

Comparison between the general operating characteristics of the helicoid and other machines is essential if it is to be considered commercially and its most suitable role determined.

As the helicoid was developed for the beneficiation of tin ores, comparison was made, in this series of tests, with the performance of a vanner and round frame operating in a Cornish tin mine. The materials for these tests were collected from suitable positions in the South Crofty mill circuit; they represent the feed to and tails from a vanner, the feed and tail from a rougher round frame and the tails from a cleaner frame. In addition, a quantity of the feed to a slime table (concentrating coarse, classified,

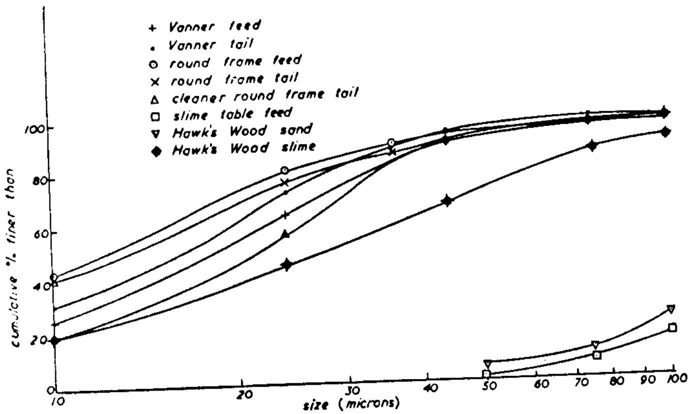

Fig. 18.—Size distribution of products collected from South Crofty and Hawk's Wood mines.

reground table middlings) was collected for investigation. The size analyses of these products are shown in graphical form in Fig. 18.

Vanner

Vanner 13 in the South Crofty flowsheet was selected as the reference machine. It occupies an area of approximately 15 sq. yd and is operated as one of a block of five, the whole being actuated from a common power supply estimated at 2½ h.p. This machine treats approximately 500 lb solids per hour in a pulp containing approximately 25 per cent solids. Typical tin assays of the feed, tails and concentrate were 0·58, 0·48 and 4·95 per cent, respectively, and these represent a recovery of 19 per cent with a corresponding enrichment ratio of 8·5.

Size assay analyses of the feed and tails, which were collected from the vanner for these investigations, are presented in Table I. These results

TABLE I

Size of material, μ	Vanner feed			Vanner tail		
	Wt., %	Tin assay, %	Tin distrib., %	Wt., %	Tin assay, %	Tin distrib., %
> 36 . . .	16·0	0·46	16·8	11·5	0·29	8·9
< 36 > 24 . .	20·0	0·48	21·8	16·5	0·40	17·9
< 24 > 12 . .	31·5	0·52	37·3	37·0	0·43	43·1
< 12 > 6 . .	19·5	0·36	15·9	16·0	0·35	15·2
< 6 . . .	13·0	0·28	8·2	19·0	0·28	14·9
Calc. head . .		(0·44)			0·37	
Assay head . .		0·41			0·33	

have been used in constructing Fig. 19 which shows the percentage weight of tin, of the feed and the concentrate, in sized fractions of 1μ spread. The concentrating action of the vanner does not appear to be effective over the whole size range of the material presented, but is specific to two smaller ranges, the optimum size in each being 5 and 14μ.

Comparative tests were made on the helicoid; the products were treated and the results reported in the same way as those from the vanner (see Table II).

TABLE II

Size of material, μ	Helicoid feed			Helicoid tail		
	Wt., %	Tin assay, %	Tin distrib., %	Wt., %	Tin assay, %	Tin distrib., %
> 36 . . .	16·0	0·46	16·8	15·5	0·12	6·4
< 36 > 24 . .	20·0	0·48	21·8	15·5	0·21	11·4
< 24 > 12 . .	31·5	0·52	37·3	42·5	0·36	53·4
< 12 > 6 . .	19·5	0·36	15·9	12·0	0·35	14·7
< 6 . . .	13·0	0·28	8·2	14·5	0·28	14·1
Calc. head . .		(0·44)			0·29	
Assay head . .		0·41			0·30	

The characteristics for recovery and enrichment ratios are shown in Fig. 20, which demonstrates that a concentrate represented by fraction 2 would produce a recovery of about 30 per cent and an enrichment ratio of 6 to 7. This compares favourably with the 19 per cent recovery produced by the vanner, although it is slightly inferior in enrichment ratio. However, it is considered that, by adjusting the cutoff point for the helicoid concentrate, its enrichment ratio could be increased at the expense of its recovery and a performance superior to that of the vanner in both respects would result. The validity of this assumption is supported by extrapolating the graphs in Fig. 20.

The size ranges in which the helicoid is most effective are shown in Fig. 21; these are in relatively close agreement with those already presented in Fig. 19 for the vanner. The apparent similarity in the selectivity of the most effective size ranges is probably due to a combination of factors common to both operations. For example, the fundamental processes in each machine are similar and the ore may be such that the liberation in these particular fractions, or the shape factors at these sizes, is most conducive to the techniques which are being compared. The area enclosed by the curve illustrating the performance of the helicoid is considerably larger than that of the vanner and represents another method of assessing and comparing the individual recoveries. Obviously, straight comparisons of the results of the chemical analyses of extracted samples are inadequate and further factors must be considered before a practical assessment can be reached.

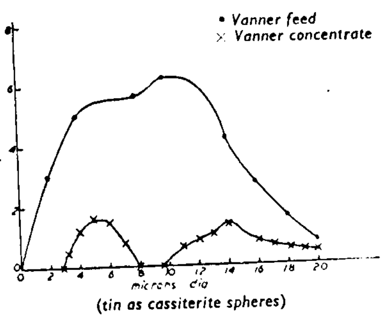

Fig. 19.—Vanner performance.

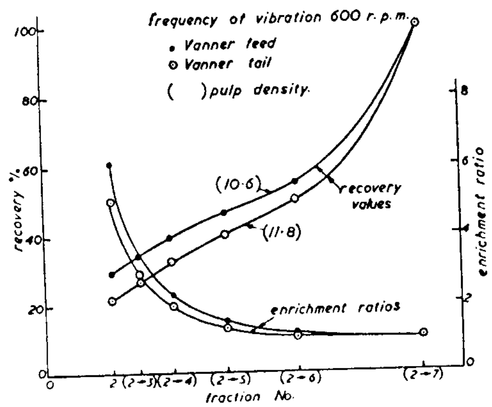

Fig. 20.—Cumulative enrichment and recovery values for combined fractions.

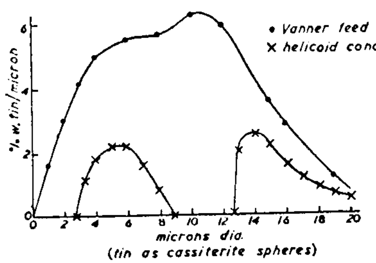

Fig. 21.—Helicoid performance.

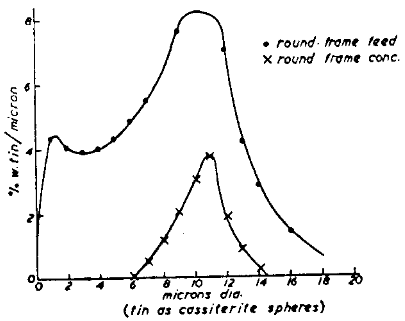

Fig. 22.—Round frame performance.

Space requirement is an important consideration, and to place it in its true perspective it must be related to throughput. The vanner occupies about 15 sq. yd and treats 500 lb solids per hour. In comparison a single 36-in diameter helicoid occupies an area of about 1 sq. yd and will handle 110 lb/h. Hence a helicoid installation will process about three times the throughput of the vanner per unit of machine area.

With respect to power consumption, the vanner operated as one of a batch of five and, although its actual consumption was not determined, ½ h.p. has been estimated as a working figure,* representing a specific consumption of 2 h.p. per ton of solids. For comparison, the recorded input to the helicoid approximated to 26 W, which represents a consumption of 0·65 h.p. per ton of solids.

The relatively low pulp density of the helicoid operation is synonymous with high rates of water consumption, which may be a critical factor in arid regions. However, as wash-water is not employed, this disadvantage is offset and the resulting total consumption is similar to that of the vanner.

Round Frame

Round frames are employed in the South Crofty flowsheet for the final extraction of values from mixed fines originating from a regrind circuit and

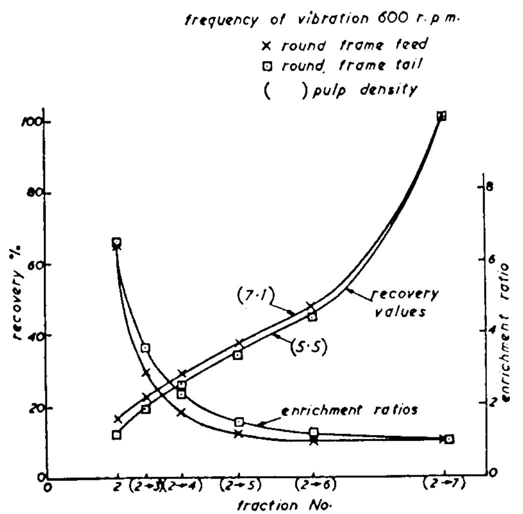

Fig. 23.—Cumulative enrichment and recovery values for combined fractions.

from the middlings produced by a bank of slimes tables. The size distribution and tin assays of the various fractions in this feed and in the tails from a rougher frame are shown in Table III. These results have been used to plot tin distribution curves (Fig. 22) for the feed and concentrate from this

* 1-h.p. machine quoted by the mine.

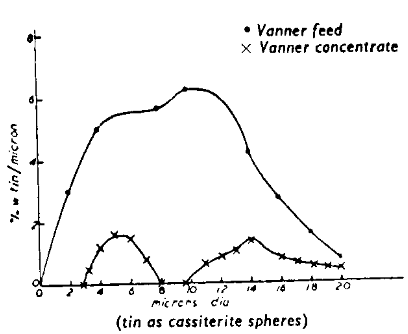

Fig. 19.—Vanner performance.

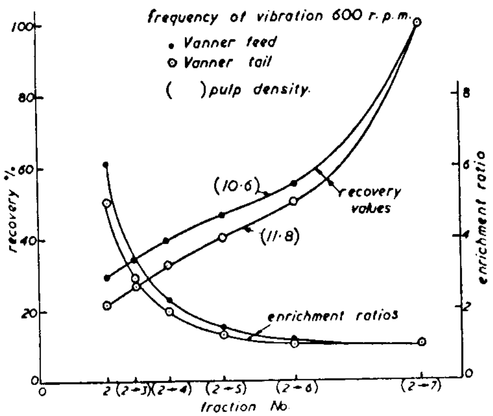

Fig. 20.—Cumulative enrichment and recovery values for combined fractions.

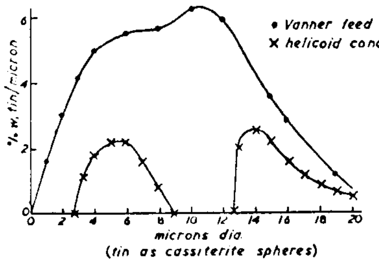

Fig. 21.—Helicoid performance.

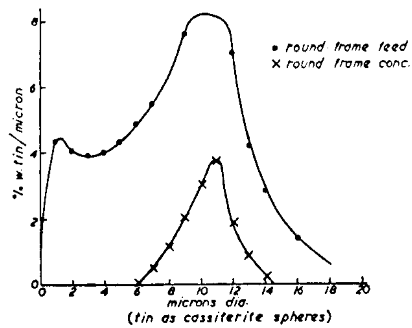

Fig. 22.—Round frame performance.

Space requirement is an important consideration, and to place it in its true perspective it must be related to throughput. The vanner occupies about 15 sq. yd and treats 500 lb solids per hour. In comparison a single 36-in diameter helicoid occupies an area of about 1 sq. yd and will handle 110 lb/h. Hence a helicoid installation will process about three times the throughput of the vanner per unit of machine area.

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The relatively low pulp density of the helicoid operation is synonymous with high rates of water consumption, which may be a critical factor in arid regions. However, as wash-water is not employed, this disadvantage is offset and the resulting total consumption is similar to that of the vanner.

Round Frame

Round frames are employed in the South Crofty flowsheet for the final extraction of values from mixed fines originating from a regrind circuit and

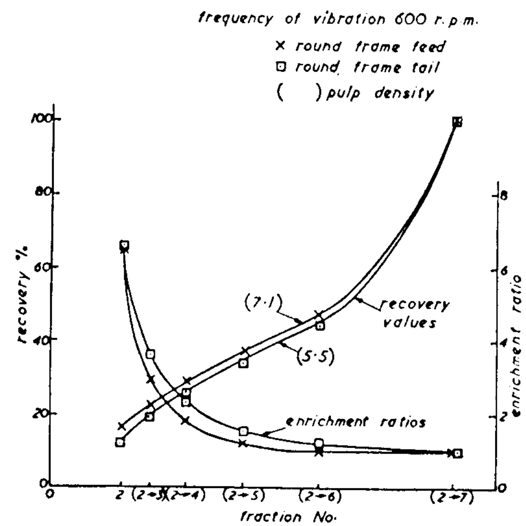

Fig. 23.—Cumulative enrichment and recovery values for combined fractions.

from the middlings produced by a bank of slimes tables. The size distribution and tin assays of the various fractions in this feed and in the tails from a rougher frame are shown in Table III. These results have been used to plot tin distribution curves (Fig. 22) for the feed and concentrate from this

* 1-h.p. machine quoted by the mine.

machine. A comparison of these curves with those in Fig. 19 shows that the frame extracts the tin from a size fraction which has not been affected by the vanner.

TABLE III

Size of material, μ	Round frame feed			Round frame tail		
	Wt., %	Tin assay, %	Tin distrib., %	Wt., %	Tin assay, %	Tin distrib., %
> 36 . . .	10·0	0·62	15·7	14·0	0·65	23·4
< 36 > 24 . .	10·5	0·45	11·9	10·5	0·57	15·4
< 24 > 12 . .	31·5	0·57	45·7	28·0	0·41	29·6
< 12 > 6 . .	24·0	0·21	12·7	20·0	0·34	17·5
< 6 . . .	24·0	0·23	14·0	27·5	0·20	14·1
Calc. head . .		0·39			0·39	
Assay head . .		0·44			0·38	

The recovery attained with this equipment, treating 440 lb dry solids/h, was calculated from figures supplied by the mine to be approximately 20 per cent. The concentrate showed a grade of 1·4 per cent tin and, in relation to the quoted feed assay of 0·59 per cent tin, an enrichment ratio of 2·4 resulted. The power involved was quoted by the mine as approximately 1·5 h.p./frame, which gives an estimated specific power consumption of 6·8 h.p./ton feed. This appears to be an excessive figure and it is considered that, with normal conditions a specific consumption of 1·1 h.p./ton might be achieved.

Results produced with the helicoid, operating on the same feed material and also on the round frame tail, are shown in the graphs of Fig. 23, where it is seen that enrichment ratios of 6·5 were obtained in both instances and respective recovery values of 17 per cent and 13 per cent resulted. The throughput of the helicoid during this part of the investigation amounted to approximately 70 lb/h, with a specific power consumption of 1 h.p./ton at a pulp density of 6 per cent. The latter figure is not now considered to be the optimum and it is probable that the results reported here can be considerably improved by increasing the solids content of the feed to about 10 per cent. Obviously, under these conditions, the throughput of the helicoid would increase accordingly and a specific power consumption of 0·65 h.p./ton would result.

Examination of the space requirements of each machine demonstrates the considerable advantage held by the helicoid in this respect. It operated at a specific feed rate of 70 lb/h/sq.yd, compared with 15·5 lb/h/sq.yd on the 18-ft diameter frame.

The performance of the helicoid, therefore, shows a marked improvement on that of the round frame; its ratio of enrichment is 2 or 3 times as great and this result is produced with a specific power consumption only a half and a space requirement only one quarter of that required by the frame.

The recovery of the frame is 3 per cent superior to that of the helicoid, although it is considered that this deficiency can be offset, at the expense of the grade, by adjustments to the position of the concentrate splitter.

OPERATIONS WITH COMBINATIONS OF HELICOIDS

Rougher and cleaner operations were then undertaken to examine the possibilities of enriching the concentrates still further. The South Crofty vanner feed was again used and three 'straight-through' tests, comprising a rougher, a scavenger and a cleaner operation on the two combined concentrates, were carried out.

Before the start of each test the helicoid was operated in closed circuit until a condition of equilibrium was reached; at this stage the circuit was adjusted for straight-through flow and concentrates and tailings were collected in separate containers for physical and chemical examinations. The stage recoveries and tin assays are shown in Table IV and represent the individual performances resulting from batch test work, while an overall balance for the tests is given in Table V. The final product, containing 9·2 per cent tin and representing an overall recovery of 13 per cent with an enrichment ratio of 22·4, is extremely encouraging.

TABLE IV.—*Batch treatment*

Stage	Products	Wt., %	Sn		
			Assay %	Units	Stage recovery, %
3	Cleaner tail	12·36	1·2	0·148	71·5
	Cleaner concentrate . . .	0·64	9·2	0·059	28·5
3	Calculated cleaner feed . .	13·00	(1·59)	0·207	100·0
3	Combined rougher and scavenger concentrates assay . . .		1·6		
2	Scavenger tail	87·0	0·28	0·243	74·7
	Scavenger concentrate . .	7·7	1·07	0·083	25·3
2	Calculated rougher tail . .	94·7	(0·33)	0·326	100·0
1	Rougher tail	94·7	0·34	0·322	71·6
	Rougher concentrate . . .	5·3	2·41	0·128	28·4
1	Calculated rougher head . .	100·0	(0·45)	0·450	100·0
	Rougher head assay . . .		0·41		

With recirculation of the cleaner tails and assuming that the final grades of the concentrate and tailings are the same as those already achieved in the batch tests, then from the overall estimated performance indicated in Table VI a recovery of approximately 39 per cent could be expected.

TABLE V.—*Overall balance for batch treatment*

Products	Wt., %	Sn		
		Assay, %	Units	Recovery, %
Cleaner concentrate	0·64	9·2	0·59	13·1
Cleaner tail . .	12·36	1·2	0·148	32·9
Scavenger tail .	87·0	0·28	0·243	54·0
Calculated head .	100·0	(0·45)	0·450	100·0
Assayed head .		0·41		

TABLE VI.—*Estimated overall balance—recirculation of cleaner tail*

Products	Wt., %	Sn		
		Assay, %	Units	Recovery, %
Cleaner concentrate	1·91	9·2	0·175	38·9
Scavenger tail .	98·09	0·28	0·275	61·1
Head . .	100·00	0·45	0·450	100·0

For such a system it is estimated that eight rougher and eight scavenger machines would be required to feed one cleaner helicoid. This represents a seventeen-machine unit, capable of handling about 900 lb of feed per hour with a power input of 0·6 h.p. For an equivalent size of machine, operating with a similar recovery, the throughput of the helicoid system would be almost twice as great and the enrichment ratio three times as high as that of the vanner. The specific power consumed in achieving this result is estimated at approximately 1·4 h.p./ton of feed, as compared with 2 h.p./ton in a single vanner and 0·65 h.p./ton for one helicoid.

The overall performance specified in each of the two systems and in the estimated arrangement is superior to that of the basic vanner performance used for this investigation. However, it was noted that the more advantageous results are achieved with helicoids at power ratings over 100 per cent higher than those of a single-stage system but 30 per cent lower than the requirements in a single vanner.

Conclusions

A vanner operating on tin slimes was used as a basic measure of comparison for the tests carried out in the laboratory on the shaken helicoid. The overall performance of the helicoid was superior to that of the vanner; its recovery was 50 per cent higher and its space requirements were considerably lower, with an equivalent power only 32 per cent of that required by the vanner. In the test, the enrichment ratio was 2 points

lower than the 8·5 produced by the vanner, but it should not be difficult to adjust this figure at the expense of recovery by regulating the cutoff position of the concentrate deflector until both the recovery and the enrichment ratio exceeds that of the vanner.

Both machines are effective over almost identical size ranges between 2μ and 20μ. This may be accounted for by the close similarity of their concentrating actions or, which is more probable, to the particular distribution of released value in the feed.

The tests indicate that, by employing a combined helicoid unit comprising eight roughers, eight scavengers and a cleaner machine, enrichment ratios as high as 22 might be expected in conjunction with recoveries of approximately 39 per cent. The power input for this operation is estimated at 1·4 h.p. per ton of feed.

A number of alternative arrangements can be envisaged and designed which would achieve results between the two extremes already reported. For example, a unit containing four roughers, three scavengers and one cleaner could be operated to produce a final product giving a recovery of 45 per cent and an enrichment ratio of 9. Again the power consumption would be of the order of 1·4 h.p./ton, but the excess power consumed would be more than compensated for by increased recoveries and grades.

The conclusions resulting from a comparison between the performance of a helicoid and that of a round frame are similar to those already reported in relation to the helicoid and vanner investigation, the overall performance of the helicoid being superior to that of the round frame, particularly in specific power consumption, enrichment ratios, and space requirements.

Acknowledgements.—This work formed part of the research programme of the Warren Spring Laboratory and is published by permission of the Director.

The authors wish to acknowledge the ready co-operation of South Crofty, Ltd., for arranging to supply feed materials and data relating to their plant operations.

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2. Burch, C. R., and Mozley, R. H. Some experiments in gravity concentration. *Trans. Cornish Inst. Engrs*, **12**, 1956-57, 24-40.
3. Bagnold, R. A. Experiments on a gravity-free dispersion of large solid spheres in a Newtonian fluid under shear. *Proc. roy. Soc.*, **225A**, 1954, 49-63.

Mr. F. A. Williams said that the author had presented a large amount of information but the investigation described was noticeably incomplete. The association of part of the uranium with the pyrite appeared to have been established, and the author had now added proposals for a flowsheet.

If the investigation was to be continued he would suggest that particular attention should be paid to three aspects:

(1) the need to use the combined size-density-assay method of sample analysis whether the assay was radiometric, chemical or mineralogical;
(2) quantitative assessment of the degree of overgrinding of the uranium-bearing pyrite; and
(3) investigation of means of minimizing that overgrinding.

The flowsheet in Fig. 2 (p. 341) indicated that there were three intermediate products to which the size-density-assay method of sample analysis should be applied—(1) the output of the jaw crusher; (2) the classifier sands; and (3) the table middlings. Those three products together constituted the feed to the grinding mill where appreciable overgrinding of fully released and partly released pyrite might be taking place.

The two well established methods of reducing overgrinding in such circuits were flotation and gravity concentration in jigs.

The speaker said he had recently been instrumental in having size-density-assay analyses made on samples from a similar flow of a low-grade wolframite ore assaying less than 0 2 per cent WO_3 where overgrinding was suspected and thus confirmed. It should be practicable to apply that method to samples of the comminuted Jacobina ore which was shown on Table VIII (p. 349) as containing 2–5 per cent of pyrite.

In the case of the wolframite ore he had mentioned each sample was screen-sized down to 80 μm, each size fraction was then density fractionated first in bromoform and then in methylene iodide and finally each size-density fraction was chemically assayed. Overgrinding having been indicated, a test jig had been installed in the circuit and a performance analysis made by means of the size-density-assay method applied to time/weight samples. Not only was a good recovery of fully released wolframite obtained but there had also been worthwhile beneficiation in terms of intergrown grains of even the 2·8–3·3 density fraction down to the fine size of 80 μm. A similar result in terms of size and density could be expected with the Jacobina ore.

The size-density-assay method of sample analysis was frequently used by economic geologists in assessing new prospects of metalliferous ores with which they were concerned. He thought it should be used more frequently by mineral processing engineers and research workers.

The author appeared well aware of the need to avoid overgrinding of the pyrite, but to have lacked the time in which to carry out a more detailed investigation. On page 348 he stated 'It has not been possible, in the time available, to determine the optimum conditions in regard to grain size distribution, but it is clear that the removal of materials from the grinding circuit as soon as they have been reduced below about 35 mesh is highly desirable'. He would like to ask the author whether he fixed the critical size at 35 mesh because that was normally about the upper limit of effective flotation or whether it happened to be the coarsest size at which the pyrite was present in appreciable amount in the ore before comminution.

The President closed the meeting with a vote of thanks to the author for the paper, and to Mr. Elkan for introducing it. He invited contributors to send further remarks in writing for publication later.

Performance of a Shaken Helicoid as a Gravity Concentrator

E. DOUGLAS, B.Sc., A.M.I.Mech.E., ASSOCIATE MEMBER, and
D. L. R. BAILEY, B.Sc.

Authors' reply to further contributed remarks on paper published in August, 1961* (Transactions, *vol. 70, 1960–61*), *pp. 637–57, and on joint discussion published in April and June, 1962* (Transactions, *vol. 71, 1961–62*), *pp. 397–436 and 547–50*

Messrs. E. Douglas and D. L. R. Bailey: Mr. Gill has stressed the inaccuracies involved in obtaining 'snap' sampling from an operational circuit. These were discussed and admitted in our reply to the discussion.† Such inaccuracies will automatically be reflected in Figs. 19 and 22 (p. 652) of the original paper, and this also was agreed in discussion.

Without further, more detailed, work on the mill and in the laboratory, it is difficult to assess the exact extent of any resulting discrepancies. However, we find it difficult to accept the considerable (up to 50 per cent) errors which would be necessary to eliminate the characteristics (perhaps we should describe them as 'trends') we have reported, particularly as almost identical trends resulted from the helicoid tests.

From the details given in Mr. Gill's Table A and from a further examination of our Table II of the paper, we must agree with Mr. Gill's conclusion that Table II does not supply reliable evidence for the helicoid's failure to recover tin in the 8–13 range and consequently the conclusions arising from this set of results are not valid.

In answer to 'Appeal to concentrates', having of necessity employed a feed-tailings difference to assess the results of the vanner and the round frame, it appeared to be more consistent to repeat this procedure for the helicoid test. Owing to the considerable lapse of time (approximately three years) since these tests were carried out samples have been discarded and we regret that we are not able to undertake the additional examinations necessary to re-assess the data given in Table II of our paper.

*Vol. 71, pp. 684–6.
†pp. 547–50.

$Q_i = \pi r_i^2 V_i \sim r_i^2\sqrt{P/\gamma_t}$. According to Chaston's formula[7] in the case where the value of $\left(\frac{r_o}{r_i}\right)^2$ does not lie between 1 and 2, a more exact result will be given by the formula:

$$Q_i \sim r_i^2 \sqrt{\frac{P}{\gamma_t}\left(\frac{r_o}{r_i}\right)^2} \sim r_i r_o \sqrt{P/\gamma_t}.$$

(According to Dahlstrom's formula[8] $Q_o \sim |r_i r_o|^{0 \cdot 9}\sqrt{P}$.)

The specific gravity of heavy medium greatly changes in both radial and axial directions. Measurements of researchers in Russia,[1] illustrated in my Fig. B, show that it is not correct to use in the cone force equation the specific gravity γ_t and viscosity ν of the *entering* heavy media.[6,3]

According to page 330 of Professor Lilge's paper, the rate of shear of medium at the intersection of the envelope of zero vertical velocity and maximum tangential velocity is $S = -1477$ sec^{-1}. This result is obtained on differentiating either eq. 6: $V_T r^n = \boldsymbol{K}$, or eq. 7: $V_T = \boldsymbol{m} r_T + \boldsymbol{b}$. Note that $\frac{dV_T}{dr} = -nV_T/r$ ft/min per in. $= \frac{-nV_T}{r_T} \cdot \frac{12}{60}$ sec^{-1} $= -\frac{0 \cdot 8.4626}{0 \cdot 501} \cdot \frac{12}{60}$ $= -1473$ sec^{-1}: alternatively, $\frac{dV_T}{dr} = +\boldsymbol{m}$ ft/min per in. $= +\boldsymbol{m}.12/60$ sec^{-1} $= +7500.12/60 = +1500$ sec^{-1}. According to the evidence of Figs. 4 and 5, however, it is $S = 0$ at this locus. (At the maximum point of the curves the tangent is horizontal.)

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Performance of a Shaken Helicoid as a Gravity Concentrator

E. DOUGLAS, B.Sc., A.M.I.Mech.E., ASSOCIATE MEMBER, and D. L. R. BAILEY, B.Sc.,

and

Gravity Concentration of Fine Cassiterite

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

Authors' replies to joint discussion on papers published in August, 1961* (Transactions, *vol. 70) pp. 637–57, and January, 1962, pp. 215–25, respectively*

Messrs. E. Douglas and D. L. R. Bailey: Among the objects in presenting this paper were those of drawing the attention of the mining industries to the existence of the shaken helicoid, of indicating its operational characteristics and of assessing the opinions of the technical experts concerned regarding its practical potential. We are therefore grateful for the extensive written discussion which has resulted and for the comments, suggestions and criticisms which have been raised. In particular we are indebted to Dr. Burch for his detailed mathematical treatment of helicoid flow, and to Mr. Gill for the additional points of qualification he has introduced.

A general concensus of opinion appears to be that the prototype helicoid gives slightly better results than the vanner and has space and power advantages. The main criticisms of the tests carried out relate to those from which the performances of machines operating in the field were calculated. For practical reasons the sampling procedures employed at the mine were not sufficiently accurate, in view of the low-grade materials and the resulting low metallurgical efficiency, to provide accurate comparisons. Further, it is pointed out that numerous variables are unavoidably introduced when laboratory and field operations are compared and consequently 'likes' are not compared. We agree with these objections which give added weight to the opinion that any developments subsequent to the tests reported should include investigations using suitably designed apparatus in an operating mill flowsheet.

Sampling

The main objective of the sampling procedure adopted at the mill was to obtain working materials. The procedure was of necessity limited to a minimum disruption of the mill, coupled with transport facilities and the delays involved in settling—all samples being dewatered to facilitate 'trucking' back to the laboratory.

*pp. 397–436.

The samples extracted varied in pulp weight from approximately 400 lb to approximately 1 ton, depending on the point of extraction and on the material concerned. Automatic samplers were not available and consequently sampling points had to be chosen in convenient sections of launders or pipe runs. In the case of the vanner, about 450 lb of pulp (100 lb + of solids) were collected (full bore) from the tails over a period of 10–15 min, after which a feed sample was immediately taken in a similar manner. Concentrate samples were not collected in bulk owing to inaccessibility (they fell at irregular intervals into a pipe cemented into the ground) and, further, at the low belt speeds (possibly not more than 3 ft/min) they would have to be associated with lengthy stage-sampling of the feed and tails to be reasonably representative. For test purposes these samples were sub-divided, using a rotary pulp sampler, prior to dilution with Stevenage tap water (total hardness approximately 250 ppm $CaCO_3$).

Test feed and product samples were vacuum-filtered, oven-dried, and agglomerates broken down by shaking the container (rolling not necessary), then riffled into analytical and sizing samples. Sizing was by beaker decantation, using a 5-l. beaker, the pulps being effectively dispersed by vigorous agitations (using stirrer) and by introducing quantities of sodium silicate. Supernatant liquids were syphoned off and four washes were employed.

Replicate size assays of the samples associated with the test results were not obtained although repeat decantations of another tin feed are shown in Table A and indicate the inaccuracies involved in this particular sizing procedure.

TABLE A.—*Triplicate size analyses*

Size fraction BSS, μ	*Test 1 wt., %*	*Test 2 wt., %*	*Test 3 wt., %*
+100	3·9	3·6	3·3
+200	9·9	9·9	9·6
+300	9·3	9·2	8·9
+36μ	14·4	14·5	14·9
+24μ	10·0	8·9	8·9
+12μ	10·7	11·5	12·2
−12μ	41·8	42·4	42·2

The tin content of each fraction was assessed by X-ray fluorescent spectrometry, the results being repeatable within ±5 per cent. The overall accuracies of the methods adopted are reflected in the differences between calculated and assayed head values.

We agree with Mr. Gill that the relative coarseness of the vanner feed, compared with that of the tail, cannot be wholly accounted for by concentrate weight and it is suggested that the out-of-balance (approximately 6 per cent) may be attributed to feed variation over the two 10-min sampling periods. The sample of feed to the round frame, which was a concrete fixed-bed machine, had to be extracted from a launder serving four frames in parallel, while the tailing was sampled from the product of one of these. The former was extracted in about 1 hour (1200 lb pulp at 7½ per cent solids) and the latter over a period of 4 hours (2200 lb pulp at 4½ per cent solids). This one, of many variations from an ideal procedure, could account for the discrepancies indicated in the size distributions of

the round frame feed and tails (Table III, p. 654) and we agree that such discrepancies would affect the accuracy of the graphs presented. It is pointed out that, of Tables I, II (pp. 650, 651) and III, only Table II represents laboratory operations.

Derivation of Figs. 19, 21 and 22 (p. 652)

The particular size distributions shown in Figs. 19, 21 and 22 were obtained by plotting 'cumulative per cent finer' tin values—size curves for feed and tails (i.e. Table II modified to a cassiterite sphere basis). The tangents to these curves, at any specific particle size, represent tin contents, per micron increment in size. These tangents, related to feed, when plotted against size, give the feed characteristic shown in Fig. 21. Similarly, a tailing characteristic can be produced and, by difference, the approximate concentrate representation can be obtained.

Fig. A.

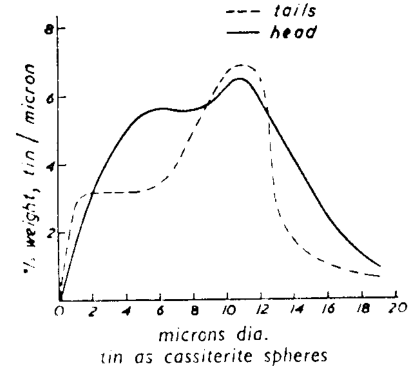

Fig. A shown above is introduced to demonstrate the intermediate stage prior to the Fig. 21 plot; it also indicates the inaccuracies accruing from the numerous operations performed. However, these by no means obliterate the tin depletion areas between 2 μ–9 μ and 13 μ–18 μ.

Batch Treatment

In the first two (rougher and scavenger) of the three stages which comprise the batch tests, the helicoid was operated for approximately 45 min, under recirculating conditions before the 'straight-through' operations commenced. During this period solids (pulp) were added until the feed became constant and of an equal pulp density to that of the bulk feed which was subsequently introduced when the circuit was switched to straight-through conditions. Approximately 30–40 lb of dry solids were treated in each of these two stages. After each operation the deck was cleaned of silt which had built up to a natural operating thickness.

In rougher operations the silt was depleted in values and, as suggested by Mr. Gill, this would account for differences between the calculated and assayed head figures quoted in the paper. In subsequent treatments these differences do not appear to obtain, suggesting that the silt is of similar grade to the feed. The weight of solids involved in the cleaning operations was limited by the working sample available at this stage to only 5 lb and this necessitated a final operation under closed-circuit conditions. This would affect the result, relative to steady open-circuit conditions, although in view of the low silt weight and the agreement between the feed assays, calculated both on a cleaner product basis and on a rougher and scavenger concentrate basis, we do not consider that more extensive operations would be unduly misrepresented by the figures quoted in Table IV (p. 655).

Throughout the tests only visual accounts were taken of the deposit build-up on the helicoid deck. Generally, this varied in accordance with the pulp density of the feed; in some cases, with fine feeds and low pulp densities, it was reduced to what appeared to be film thickness. As Mr. Chaston suggests, such variations will affect performance and may contribute to the anomalies which were noted between low-density feeds and those containing more than 5 per cent solids.

We are grateful to Mr. Michell for correcting our statement that, per lb of solids, the vanner and the helicoid consume similar amounts of water. In arriving at this conclusion a vanner wash-water figure of 3 g/min* was assumed—at 1·25 g/min, the specific water requirements of a vanner will be some 50 per cent less than that of the helicoid.

Chemical/Physical Assays

We agree with Mr. F. A. Williams that more detailed investigations into the degree of locking in the various products would provide additional important data concerning the results recorded. However, such operations as heavy liquid centrifuging and mineralogical assays on extremely fine materials are very time-consuming and could not be considered for this preliminary test work. Again, a chemical assay basis of assessment was chosen to provide reasonably quick, relative, but not necessarily absolute, comparisons. Physical methods of comparison, particularly when locked values are present, introduce numerous difficulties and inconsistencies.

We cannot add to Mr. Williams's presumptions regarding economic cut-off levels for recovery and grade in vanning operations—they will be influenced by local conditions such as throughput, transport difficulties and market conditions. With respect to the latter it is interesting to note that extremely low grades, down to at least 5 per cent Sn, are acceptable for chemical treatment.

Mr. I. R. M. Chaston: Dr. Robinson and Mr. Douglas expect recovery and enrichment ratio to vary inversely. This is generally the case with an unchanged feed, but in my experience—and here I am supported by Mr. Hutchin's contribution—desliming usually has the effect of increasing both recovery and enrichment ratio. The tilting table results show the effect of desliming, while the shaking table figures show the inverse effect, because both the feeds are of a similar deslimed nature. The only true method of comparison between two machines is to operate both from the same feed source at the same time. This is not always possible and in making any other comparison possible differences in the feed nature must be borne in mind. This is brought out by Mr. Denyer, Dr. Pryor and Mr. Michell in the discussion of the paper by Douglas and Bailey. In my own paper comparison is deliberately kept to a broad generalization and is only intended to hold in that all the feeds considered are fines produced in the milling of lode tin ores and have the same order of size range and are therefore broadly similar.

It is difficult to accept Mr. Jones's diagram as proving that 30-μ cassiterite is too small for efficient gravity concentration, the more so since the diagram is incorrect on several counts. The alluvial tin mining industry is now taking a close interest in fine cassiterite since recent work has shown that it is economically possible to recover it. Volatilization has received a lot of attention for many years, but has, as yet, proved too expensive a process except in special cases. Tin flotation has actually been carried out on run-of-mine ores. A tin mine in Germany ran from 1938 to 1945 recovering tin by flotation from an ore containing about 0·5 per cent Sn and producing 10 per cent Sn concentrates with a recovery of about 80 per cent. These concentrates were then treated by a Waelz kiln volatilization process to recover the tin. The plant was dismantled and removed to Russia after the war. The milling and flotation alone cost the equivalent of 11s./ton at a time when tin was worth £200–£250 a ton.

This brings out the main reason for pursuing investigations in gravity concentration. For all its drawbacks gravity concentration plant is generally easy and cheap to operate. In dealing with the general run of tin ores, which by most mining standards are of low grade, this is of the greatest importance.

In reply to Professor Rey, it is clear from page 220 that the desliming took place before and not after the flotation. The actual assay of the cyclone overflow was 0·32 per cent Sn and of the float 0·21 per cent Sn. The calculation is further complicated by the return of all by-pass feed and cleaner tailings to the head of the flotation circuit. Total tailings—that is, float and tilting table tailings—assayed 0·49 per cent Sn with a feed to the circuit of 0·70 per cent Sn in Test 1; and total tailings, including the cyclone overflow, assayed 0·48 per cent Sn in Test 2 with a feed to the circuit assaying 0·84 per cent Sn. Table IV on page 223 does give the assays of the shaking table tailings which were 1·06 and 1·04 per cent Sn, respectively. Professor Rey's calculation in his Table B (p. 433) appears to be incorrect. The calculated tailing assays shown in that table would indicate recoveries of 72·2 and 95·5 per cent respectively and not the 75·5 and 94·0 per cent given in the paper.

Mr. Douglas's suggested explanation of the double recovery effect found when the tilting tables were fed with a cycloned product is most ingenious. However, it would also be expected to hold for table concentration with cycloned feeds and the example quoted in the introduction to the paper does not support this. Also the double recovery effect could be clearly seen in the results published in reference 2 (p. 225) in my paper, when the feed to the tilting tables was not deslimed.

In answer to Mr. Moir, the table used was a fully riffled full-size James

*Taggart, A. F. *Handbook of mineral dressing*. (New York: John Wiley and Sons, Inc., 1944), 11:92.

table. As was stated in the introduction, table efficiency was still found to be very high with properly prepared feeds at feed rates of up to 1000 lb/h on full-size slime or Plat-O tables.

Mr. Collins will find that desliming is rarely employed in sulphide flotation because the resulting improvement does not usually compensate for the cost of the operation and for the loss of values in the slime, whereas in non-sulphide flotation it often does.

Finally I would like to take this opportunity of thanking my company, Beralt Tin and Wolfram, Ltd., for their encouragement of my work and for their permission to publish the results obtained.

Determination of Silver in Lead Sulphide Concentrate by Atomic Absorption Spectroscopy

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Authors' reply to discussion on paper published in October, 1961, pp. 15–26*

Messrs. Rawling, Amos and Greaves: We should like to thank Mr. Morgan for his presentation of our paper and all who contributed to the discussion.

Mr. Wright mentioned the likelihood of spectral interference when the resonance lines of other elements are close to the analytical line. This could only happen when the interfering line is emitted by the hollow cathode and the interfering element is in the flame. In fact, tin is present in approximately the same amount as silver in our lead concentrate; but, as Mr. Hames pointed out, the tin resonance line is not emitted by the lamp. Each new lamp should be examined for interference of this nature.

Another type of interference arises if the monochromator is unable to resolve the line undergoing absorption from other lines. In this condition sensitivity will be reduced and calibration curves will deviate from linearity. This can happen in the case of nickel, where the resonance line at 2320·1 Å is very close to the emission line at 2319·8 Å.

Use of the least squares method has since proved to be an unnecessary refinement and, as Mr. Edwards points out, with careful work only two calibration points are necessary, provided the calibration is linear—as in fact it is at 6mA lamp current over the range 10 to 15 ppm of silver.

We were interested to note Mr. Hames's remarks on the use of propane. We have now used this gas for some time and find that it tends to give more stable flame conditions for elements such as lead and zinc. Currently we are using a 10-cm burner having a slit $\frac{1}{16}$ in. wide, this width being necessary to hold the flame on the burner by lowering the gas velocity.

*pp. 227–31.

Dr. James requested further details of the sensitivity of the method. Since publishing the paper sensitivity has been increased to the extent that, using a 10-cm burner, a 0·05-ppm silver solution gives 1 per cent absorption. Much of this increase has come from improvement in optical alignment and the use of a longer burner. Like Allan,* using a magnesium lamp we have also noted an apparent increase in the sensitivity of the silver lamp with aging. This sensitivity can be further increased, with some loss in stability, by removing the baffles from the spray-chamber. The use of a scale expansion unit† and organic solvents are further techniques for increasing sensitivity. Using a 50 per cent ethyl alcohol solution of silver, 1 ppm gives 42 per cent absorption. Current work using solvent extraction for the determination of gold gives a five-fold increase in sensitivity compared with that in aqueous solution.

There is no point in increasing amplifier gain, as it is not the intensity of the emitted line that is of concern, but rather the extent to which the line undergoes absorption. We are at present studying the application of atomic absorption to geochemical prospecting.

In reply to Dr. Chaston, the paper states that the silver lamp is very stable after 10 min. Actual lamp changing and optical alignment take only 2 min. The use of two power supplies, or power supply modification to take two lamps, could prove advantageous.

At Sulphide Corporation we are using atomic absorption in what could be described as a routine but intermittent way. We do not generally have large numbers of any particular determination, but cover a fairly wide variety. Elements determined regularly by this method are zinc, lead, cadmium, copper, silver and iron, while developments are taking place for manganese, magnesium, calcium, antimony and bismuth. We have found that of the lamps in routine use only cadmium needs a 'warm-up' period in excess of 5 min. This may be a question of manufacturing technique in the case of our two cadmium lamps rather than an inherent defect for this element.

Like Mr. Edwards, we have found no particulate interference in the flame. This source of interference should, however, be considered when working with very low concentrations at maximum sensitivity.‡

Dr. Marriott refers to the use of a leaded brass hollow cathode for lead, copper and zinc. While the convenience of such an arrangement is obvious, difficulties can arise, and a relatively short working life for the lamp could result, from preferential sputtering of the more volatile element.§

Mr. W. R. Barnes raised the question of interference when determining trace amounts of elements having the same valency as a major constituent. Theory predicts that this would be of no moment. In fact we determine cadmium in zinc concentrates at Broken Hill and cadmium in zinc metal and zinc in cadmium metal at Cockle Creek with no prior separation.

*See list of references on p. 25 of the paper.

†David, D. J. The determination of molybdenum by atomic-absorption spectrophotometry. *Analyst*, 86, Nov. 1961, 730–40.

‡Willis, J. B., C.S.I.R.O., Melbourne, private communication.

§Walsh, A., C.S.I.R.O., Melbourne, private communication.

Professor F. D. Richardson, replying to the discussion, said that, in answer to Mr. Wright's first question, FeO must be counted as CaO. As regards the second, if the electrons were not localized in the equilibrium

$$[Pb] = (Pb^{2+}) + 2(e),$$

the concentration of lead in the slag should be proportional to $\sqrt[3]{a[Pb]}$ and not to a[Pb] as found. If the electrons were localized adjacent to the (Pb^{2+}), the solubility might be proportional to a[Pb], but it was questionable whether it was meaningful to make a distinction between (Pb^{2+} 2e) and (Pb^{0}).

Dr. Lumsden was right in stating that it was difficult to understand how the $\Delta \bar{S}$ values could be so great. Perhaps the variation of solubility with temperature (as distinct from the individual solubility values) was not as accurate as was supposed and data on aPb in Pb–Au alloys was not as good as was thought. At present they had no satisfactory explanation.

He agreed with Dr. Richards and considered 1500° C as a transient slag temperature only. He had taken the highest likely temperature attainable, so that their estimate of the amount of lead dissolved as metal and subsequently precipitated might be a maximum.

On Dr. Chaston's point, the purpose of the work was to measure neutral metal solubilities. In the blast-furnace there was lead present in slag as oxide and metal droplets. The work showed that lead dissolved as neutral metal was at all times much less than either of them. The PbO content could in principle be minimized by lowering the oxygen potential and the droplets by lowering slag viscosity.

He agreed with Dr. Mitchell about slag structure, but had not stressed the complexity as it was not germane to the treatment, although it was referred to in the discussion on Fig. 3 (p. 211). There were virtually no dimers in the vapours of Cu, Au, Ag and Pb.

Dr. Littlewood asked how they could be sure that they were measuring Pb^{0} solubility. First, p_{O_2} must be very low indeed as the graphite was not attacked, while with lead at an activity of about 0·5 a_{PbO} could hardly have exceeded 10^{-6} and wt. per cent PbO could hardly have exceeded 0·0004 per cent. If the gas purification had only brought the partial pressure of oxygen down to a steady value, p_{O_2} and the graphite test had been invalid, the lead dissolved as PbO might have been much greater than 0·0004 per cent. It would, however, have decreased with temperature, because the reaction—

$$[Pb] + O_2 = (PbO)_{slag}$$

was strongly exothermic. In actual fact the solubility doubled for a 50° C rise. He considered that no useful purpose was served by discussing the dissolution of PbO in terms of ion activities, as distinct from the compound activities. The latter were perfectly clear and should admit of no misconstruction.

The President expressed his thanks to Professor Richardson. He wished to convey members' appreciation also to Dr. Meyer who unfortunately was not able to be present at the meeting.

Performance of a Shaken Helicoid as a Gravity Concentrator

E. DOUGLAS, B.Sc., A.M.I.Mech.E., ASSOCIATE MEMBER, and D. L. R. BAILEY, B.Sc.,

and

Gravity Concentration of Fine Cassiterite

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

Report of joint discussion at February, 1962, General Meeting (Chairman: Mr. A. R. O. Williams, President). Papers published in August, 1961 (Transactions, *vol. 70) pp. 637–57, and January, 1962, pp. 215–25, respectively*

Mr. E. Douglas, in presenting the first paper, said that in many mineral dressing operations the presence of slime or near-slime materials was a source of embarrassment which became increasingly irritating and often frustrating as their valuable mineral content increased. As greater efficiencies were sought and richer deposits became less plentiful there was a natural trend to additional operations involving the working of low-grade supplies, including old tailings dumps, with a consequent increase in the slime problem. The extraction of values from such materials was a recognized but extremely difficult problem, which had been partially solved by the use of slime and Buckman tables, round frames and vanners, and it was that which confronted Dr. Burch during a visit he made to Cornwall. That, in association with subsequent discussions with Brigadier Bagnold, caused Dr. Burch to produce what he described as the shaken helicoid which had been developed and tested by him and his colleagues at Bristol University. To obtain an independent assessment of its capabilities, Warren Spring Laboratory was asked to test the machine under controlled conditions and the results of those tests were presented in the paper.

The helicoid which had been tested had three to four turns, each approximately 3 ft in diameter with a 2-in pitch, and had, in cross-section, a flat, almost horizontal, deck. In operation the spiral vibrated in sympathy with two rotating out-of-balance weights, so that every point on its surface described a circle of approximately $\frac{1}{8}$ in. in diameter. As the pulp flowed down and around the spiral it was influenced by the well-known river-bend action, in which out-of-balance pressures, induced by velocity gradients in the pulp, caused transverse flows to occur. Those flowed radially outwards across the surface of the pulp and radially inwards across the deck. Obviously, for continuity there was a downward flow at the periphery and an upward flow at the inner radii where the smaller and lighter particles were elutriated preferentially towards the surface of the pulp. At the same time the rapid vibrations of the spiral induced shear forces into the pulp and Bagnold had shown that under such conditions pressures were developed, perpendicular to the planes of shear, the resulting forces acting upon the particulate matter being proportional to the particle diameters, squared. Therefore the larger particles moved away, more positively, from the areas of greater shear.

In combination the effects of Bagnold forces and the river-bend action produced conditions under which both large and small particles were pushed towards the surface of the pulp under separately controllable influences, thereby providing separations, independent of size but directly related to density of material. As the lighter materials approached the upper strata of the pulp they were transported, according to the river-bend action towards the periphery, and conversely the heavier particles which had remained in the proximity of the deck were concentrated progressively towards the centre of the helicoid as the pulp moved down the machine.

He wished to correct a mistake in the paper. On page 640 it was stated that the tests, the results of which were presented in Figs. 5, 6 and 7, had been carried out on a tin ore; in fact a *tungsten* ore had been used.

Both pulp density and frequency of vibration appeared to be extremely important in controlling the performance of the helicoid. Within limits, the former appeared to be inversely proportional to the enrichment ratio while the setting of the latter was critical, and their best conditions were obtained at frequencies in the range of 600 to 700 c/min.

Having obtained a brief insight into the characteristics of the helicoid, facilities had been very kindly provided by Mr. A. L. Thomas and his associates at the South Crofty mine, so that direct comparisons could be made between its performances and those of a vanner and two round frames operating in the South Crofty flowsheet. A typical vanner operation on a feed 80 per cent of which passed 350 mesh produced recoveries of something like 20 per cent, with an enrichment ratio of approximately 8·5. That appeared slightly inferior to the result produced on the helicoid, in which they had obtained 30 per cent recoveries and an enrichment ratio of between 6 and 7. When it was realized that only one-third of the space and power required by the vanner was needed for helicoid operation, at that stage of the helicoid's development, it appeared to have quite an attraction for fine feed concentration. By comparison, the helicoid was superior to the round frame in performance and in space and power requirements; it appeared that those advantages could be put to good use by building multi-unit equipments, possibly similar in size to a vanner and probably having greater throughputs, and comprising a rougher, scavenger, and cleaning decks.

Such a system had been batch-tested and had resulted in an enrichment ratio of 22, coupled with a recovery (based on middling recirculation) of approximately 38 per cent.

To summarize, he pointed out that the material used for the tests was unclassified to allow direct comparisons with practical operations on the vanner and the round frames. The helicoid appeared to be superior to both vanners and round frames to varying degrees, but where multi-units were concerned the helicoid appeared to have extraordinary capabilities and should be considered further in that respect.

Anticipating a possible query on the paper concerning the apparent discrepancy in classification between Fig. 19 (p. 652) and the related Table I (p. 650), he explained that Table I was based on quartz sphere

equivalent diameters while Fig. 19 had been converted to cassiterite sphere sizes. The right end of the feed characteristic should have been a little flatter than that shown in Fig. 19.

Mr. I. R. M. Chaston said that the 50th anniversary of froth flotation was recently celebrated in Denver, Colorado. The impressive list of contributors to that meeting was evidence of the enormous amount of thought and work that had gone into developing the theory and practice of flotation to its present stage over the last 50 years. He thought it had been unnoticed that it was only just over 60 years since the first introduction of the Wilfley shaking table. That had so revolutionized gravity concentration that it caused Professor Richards in the introduction to the first edition of his classic book *Ore dressing* to say: 'The appearance of the Wilfley table is an event of such importance that the book should either have been put on the market in 1896, before the first Wilfley table appeared, or have waited until 1905, when the adaptation of the mills to the newcomer would be complete '. Richards went on to say of that development that 'the improved design was continually finding new applications and had not yet prospected its whole field of usefulness '.

That had been in 1903 but in 1962 he believed they had still far to go. Richards's basic laboratory investigations began to develop possibilities for the shaking table, but the new work on flotation quickly overshadowed the gravity concentration field and began to occupy most of the attention of investigators. There were now good financial reasons for interest in gravity concentration to increase. Mining costs were usually greater than milling costs and the constant search for economies was leading more and more to the introduction of larger-scale and less selective mining, which resulted in more waste being extracted with the ore. Such changes meant that the continuing trend was for milling plants to handle more and more ore at a lower and lower grade. Gravity concentration was fairly non-selective but where it was applicable in the coarser sizes—as it often was—large tonnages could be handled very cheaply. He saw a big future for gravity concentration as a means of making a preliminary cheap rejection of waste at a coarser size than flotation could operate, thereby reducing the amount of material requiring fine grinding and flotation.

For preliminary concentration the tabling of the extreme fines was not likely to be of value, because of the limited table capacity in the finer sizes. For fundamental studies, however, that limitation, resulting in a general slowing down of the action on the table, might be of great value, since it allowed table operation to be studied closely with reasonably small samples. Richards's basic work on tabling showed excellent results in sizes below 300 mesh but much of the later work of other experimenters in the field showed poor recoveries in the finer sizes and that had led to the general belief that in sizes much below 300 mesh tables efficiency was low. Those poor results were probably due to an inability to prepare the finer feed correctly. As was suggested in the paper, cyclones now permitted an improvement in the feed classification to the point where plant operation