TABLE IIA.—*Recovery of Heavy Minerals in the Primary Jigs of Plant B*

B.S. Sieve No.	Distribution: Percentages reporting in hutch products — Hutch number 1	2	3	4	Total	Percentage lost
	Cassiterite sp. gr. 7·0					
	%	%	%	%	%	%
6	100·0	—	—	—	100·0	—
6/8	98·3	1·7	—	—	100·0	—
8/10	97·2	2·4	0·4	—	100·0	—
10/12	98·0	1·8	0·2	—	100·0	—
12/16	96·4	2·9	0·1	0·1	99·5	0·5
16/25	95·6	3·3	0·1	0·1	99·1	0·9
25/52	92·3	6·5	0·4	0·1	99·3	0·7
52/72	87·3	8·9	1·8	0·4	98·4	1·6
72/100	70·4	18·7	4·3	3·4	96·8	3·2
100/120	55·6	18·4	11·8	10·7	96·5	3·5
120/150	46·1	25·0	9·3	14·7	95·1	4·9
150/170	35·9	33·5	10·5	18·0	97·0	2·1
170/240	24·6	21·8	9·1	22·3	77·8	22·2
240/300	9·6	8·6	5·3	12·6	36·1	63·9
	Columbite sp. gr. 5·5					
	%	%	%	%	%	%
12	100·0	—	—	—	100·0	—
12/16	84·1	15·9	—	—	100·0	—
16/25	88·8	9·3	0·6	0·4	99·1	0·9
25/52	85·8	11·1	1·6	0·4	98·9	1·1
52/72	80·8	12·9	3·3	0·8	97·8	2·2
72/100	64·8	18·2	8·1	5·1	96·2	3·8
100/120	50·4	20·9	15·1	11·8	98·2	1·8
120/150	36·7	29·4	12·3	17·8	96·2	3·8
150/170	29·9	29·7	13·6	21·9	95·1	4·9
170/240	22·6	24·7	13·2	27·3	87·8	12·2
240/300	9·2	9·8	5·4	10·8	35·2	64·8

Ragging — ¾″ + ½″ hematite sp. gr. 4·4
Speed 125 strokes/min
Stroke 1½″

in the table middling and 2 per cent in the gate product. Concentration of some of that low-grade material in jigs had given encouraging results. After drying and screen sizing an excellent separation of conductors from non-conductors could be achieved in the Carpco model HT 460 high-tension separator.

Tables I and II of the paper had had perforce to be based on timed

samples of the hutch products only. Owing to the construction of the tailing launders a representative tailing sample could not even be obtained, let alone timed. The accompanying Tables IA and IIA showing recovery in a primary jig on another plant provided a useful check. The four hutch products and the tailing discharge were all sampled at half-hourly intervals over a period of four days. But whereas the hutch samples could be timed, the tailing sample could not be timed. Only a couple of months later, when the tailings pipes had been altered, did it become possible to make approximate determinations of the rate of tailing discharge. That emphasized the need for adequate sampling facilities to be built into jig plants.

Table IA was of particular interest as it again showed how a ragging of hematite sp. gr. 4·4 tended to exclude the coarsest fractions of topaz sp. gr. 3·5, a fact which did not show up in Table I in the original paper. Exclusion of topaz happened to be their objective, but for the recovery of valuable semi-heavy minerals a lighter ragging than hematite would obviously have to be used.

Allowing for the degree of irregularity, there was no significant difference between the recovery of cassiterite and columbite shown in Tables II and IIA for different makes of diaphragm jig. That supported the author's contention in the paper that the results given, although only a first approximation, should apply to jigs in general. The choice between different makes of jig was more a matter of engineering detail, which was of overriding importance in terms of ease of transport, convenience of access, running time and maintenance costs. In the field of recovery there still remained plenty of scope for more research.

TABLE IIIA.—*Heavy and Semi-heavy Minerals in Total Primary Jig Hutch Product*

Mineral	Relative percentages: Lead 1	Lead 2	Lead 3
	%	%	%
Topaz	42·90	39·55	36·40
Zircon	31·97	33·37	4·28
Ilmenite and magnetite	11·60	6·93	2·10
Cassiterite	3·65 } 9·32	9·24 } 18·48	44·31 } 55·79
Columbite	5·67 }	9·24 }	11·48 }
Anatase	3·37	1·67	0·25
Monazite	0·81	0·20	0·11
Xenotime	0·03	Tr	0·08
Orangite	Tr	—	—
Cemented concentrate	—	—	0·99
Totals	100·00	100·00	100·00

Table III (p. 170 of the paper) indicated what a large quantity of various other minerals was recovered in the primary jigs along with the cassiterite and columbite, and he had mentioned the considerable variation between the mineral content of various leads. That variation, illustrated by the accompanying Table IIIA, must influence plant design and practice and particularly subsequent mineral dressing of the concentrates. As stated in the paper one of its objects was to draw attention to the remarkable efficiency of jigs for recovering semi-heavy minerals and its impact on subsequent mineral dressing practice.

Mr. J. B. Braithwaite* said that when he came to Nigeria ten years ago he had been shocked by the losses from sluice boxes in the field. They might remember that in 1953 the Government had brought Mr. F. Bice Michell out to Nigeria to report on mineral dressing, but even before that Mr. Williams was experimenting with jigs which were replacing sluice boxes.

He was particularly interested in the amount of columbite recovered from hutches 3 and 4. Table II showed that to be nearly 30 per cent of the total columbite recovered, and as that result was obtained with the flow of 10 cu. yd/hour, very low for the size of jig, he thought it merited the attention of those operators running jigs with only two hutches in series. Mr. Williams's figures indicated that with the two hutches in series, serious losses of columbite might occur.

Mr. R. N. Hammon† paid tribute to the work Mr. Williams had done in stimulating interest throughout the Plateau minesfield in more efficient recovery of minerals during alluvial mining.

He queried the author's preference for increasing the length of jig capacity rather than the width. In the Bisichi company plants 4-hutch (42 in. by 42 in.) jigs were fed about 250 gallons of pulp per minute, and about 40 gallons of water per minute per hutch were added. When the four hutches were arranged in line the volume of feed to the fourth hutch was therefore about 370 gal/min, an increase of 48 per cent over the original feed. When the hutches were arranged two by two the feed volume to the second hutch was increased to 165 from the 125 gal/min feed to the first hutch, or an increase in volume of 32 per cent. That indicated an advantage in the second arrangement.

A number of jigs on the market, when supplied as single hutches in line, had a drop of several inches after every second hutch. That was undoubtedly a mistake, as the turbulence caused by the drop destroyed the stratification of minerals which had been taking place; in effect separation had to start all over again and at an increased rate of flow.

Mr. D. Hinton said he was particularly interested in Table IIA (cassiterite and columbite), and asked for particulars of the screen analysis

*Acting Deputy Chief Inspector of Mines, Nigeria.
†General Manager, The Bisichi Tin Co. (Nigeria), Ltd.

and overall recovery. In the 240–300 range the cassiterite loss was 63·9 per cent, but that fraction probably constituted only a very small part of the feed.

Mr. J. Vitoria referred to p. 169 where, in Table II more columbite in the B.S. sieve size − 300 + 325 was saved than cassiterite, in spite of the difference of 1·5 in sp. gr. He asked for the reason.

Mr. F. A. Garner said that the difficulties of sampling and the method of analysis, i.e. grain counting, made it impossible to obtain a full metallurgical balance and in his opinion the figures quoted should be regarded as giving a general picture rather than the accurate analysis. Geologists had told him that grain counting gave results which varied from chemical analysis by up to 3 per cent with high-grade material in the coarser sieve sizes and up to 20 per cent with low-grade material in the finer sieve sizes. He wondered what effect that would have on results.

Mr. J. A. Bain* said Mr. Garner had questioned the implied accuracy in the grain-counting techniques and preferred chemical methods of assaying. In fact the degree of accuracy depended on the number of mineral grains counted, and to maintain a high standard of accuracy the size of the count in the speaker's laboratory was related to the frequency, or amount of the required mineral present. For a frequency of 80 per cent a count of only 300 grains would give an accuracy of $\pm$ 6·5 per cent, and the result should lie between 9·4 and 10·6 per cent. That already brought the operation to the first decimal place. A mechanical counter greatly facilitated such work.

Contrary to general belief a mineral analysis was seldom simply a direct visual counting of the amounts of the various minerals present. The basis of accurate physical analysis was the separation of a sample into a number of fractions, in such a way that each contained a high percentage of one of the minerals under analysis. After screening, the samples were therefore subjected to various methods of concentration. With a plant sample containing cassiterite and columbite, for example, screening was followed by concentration of each screen size on a superpanner to reject the sand and obtain a heavy mineral concentrate (at least 98 per cent recovery). That was succeeded by acid leaching, Franz magnetic separation (almost 100 per cent recovery) and perhaps heavy liquid separation (virtually 100 per cent recovery). In most cases that gave two final products—a magnetic product containing as much as 95 per cent columbite, or even more, and a non-magnetic product containing up to 90 per cent cassiterite. Those high-grade products were weighed, grain-counted and the results corrected for differences in specific gravity of the constituent minerals.

Assuming the original sample to contain 10 per cent columbite, a magnetic fraction containing after concentration 95 per cent columbite would be subjected to a count of, say, 300 grains. That was accurate

*Mineralogist, A.O. Nigeria, Ltd.

to ± 0·3 per cent. It would be appreciated that when back-calculated to the original weight of sample, the physical analysis would give a result between 9·97 and 10·03 per cent. A direct chemical assay for columbite gave a result accurate only to ± 0·5 per cent, that is, between 9·5 and 10·5 per cent. The chemical assay might be even more unreliable because of such irregularities as intergrowth of two minerals, inclusions (e.g. columbite in quartz) and the occurrence of the chemically analysed element in more than one mineral.

In their laboratory a mineral analysis was carried out for as many as ten minerals simultaneously, in twelve screen sizes. For some of them, e.g. zircon, thorite, xenotime monazite, and topaz, a chemical assay, if not impossible, was highly impractical. Their physical assaying had an additional advantage towards accuracy in that 250 g of material was used for analysis, whereas the chemical assay employed a 2-g sample only.

In ore-dressing research physical analysis for minerals should be used whenever possible. Chemical assaying for elements was a less informative substitute which could be seriously misleading.

Mr. D. Foord said that over the past 30 years, by maintaining intensive prospecting programmes, the mining companies on the tinfield had been able to prove fresh reserves ahead about as fast as the known reserves had been depleted. The finding of such reserves was becoming increasingly difficult as the years went by, but the replacement of sluice boxes by jig plants had created reserves out of ground containing fine cassiterite or columbite not formerly known to exist.

Little attention had been paid to fine cassiterite until recent years. Prospecting methods of the past, involving washing a drill sample in a calabash, resulted in a recovery approximating that of a sluice box. However, that fine cassiterite did exist there could be no doubt. He had recently been working on a large greisen vein in which most of the cassiterite was extremely fine. Upon weathering and removal by normal erosion processes the bulk of the cassiterite from such a greisen vein would be too fine to be deposited in an ordinary alluvial wash. He wondered what happened to such cassiterite. Similarly, most of the alluvial cassiterite now mined was round and water-worn, whereas its original form in the greisens was crystalline and rough. Cassiterite was a fairly insoluble mineral and the fines must still be in existence somewhere. Until the columbite boom and the research which went into the recovery of fine columbite, both in the laboratory and in the field, little was known in Nigeria either of the existence or the recovery of fines. He felt that if the life of the tinfield was going to be extended it would be by the location and recovery of such fines.

Mr. T. W. Bennetts asked for information on two points worthy of further discussion. The first was the case for extra jigs in series, as proposed by the author, as compared with the use of jigs in parallel, as described by Mr. Hammon. The author had said that the extra cost of installing two

more cells per line of jigs would be comparatively small, but there was the increased cost of pumping to consider. The flowsheet showed five gravel pumps and one sand pump. There were also a nozzle pump capable of delivering 2200 gal/min to the monitor and two other pumps which were not shown—one 3000 gal/min hutch water pump and one 12-in gravel pump for disposal of waste.

By adding two extra cells to each line of primary and secondary jigs the only pumps which would remain the same were the two 10-in gravel pumps feeding the 'rod deck' screen and the 10-in gravel pump feeding the primary cyclones. All the other pumps would have to be larger. The primary jigs would then be 4 by 6 cells—an increase of eight hutches. As 100 gal/min per cell was allowed for hutch water, the hutch water pump capacity would have to be increased by 800 gal/min. The increased pulp from those extra eight hutches would be in the region of 300–400 gal/min—half as much again as the 6-in gravel pump could handle. Thus an 8-in gravel pump with a capacity of 1100 gal/min would be required. At the same time the two 10-in cyclones feeding the secondary jigs would have to be increased to three to cope with the increased feed from the 8-in gravel pump. That type of increase would continue throughout the unit, while, owing to the extra hutch water, the 12-in gravel pump used for the disposal of waste would have to be increased to a 14-in unit.

It was very doubtful if the extra recovery would pay for all the increased costs.

His second point referred to the author's earlier paper, where the method of mining was by D.W. 21 scraper loaders bringing the decomposed granite to the jig unit. Then approximately 1400 gal/min of water was added by a monitor and the 10-in gravel pump was able to handle 124 cu. yd/hour, against the 60 cu. yd/hour using 2200 gal/min water in the alluvial plant under discussion. The jig unit only saved 30 per cent of the columbite when being fed at 124 cu. yd/hour. It was thought that the high slime content of the feed (about 50 per cent of the feed was slime) was causing some of the loss. The feed rate was chopped to 60 cu. yd/hour with the same water content, i.e. 1400 gal/min. The recovery of columbite then increased to 60 per cent. It was then decided to introduce cyclones. The feed rate was not increased but the extra de-sliming increased the recovery of columbite to 78 per cent.

In the case of alluvial wash there was very little slime, and if belt conveyors or earth-moving equipment such as D.W. 21 loaders or Euclid dump trucks were used to remove all the overburden, which had a high slime content, and the remaining wash transported to the jig unit dry a small quantity of water could then be added and the 10-in gravel pump could handle the feed at a low water : solid ratio.

He felt that the use of pumps had to be cut down to a minimum if minerals in the £300–£400 per ton range were to be saved economically. Gravel pumps were not an efficient way of feeding jig units; they required about 600 tons of water to pump, say, 60 tons of solids. At least half of that water had to be removed before the mixture was fed on to the jigs.

That 600 tons of water plus all the hutch water had to be pumped to waste or back into a return water pond in Nigeria during the dry season.

Mr. J. L. Farrington drew attention to the economic limitations to working stream deposits containing only a little extremely fine mineral and to the need to use reliable methods of valuation.

Mr. A. M. Coskinas* said recovery of the semi-heavy minerals associated with columbite and cassiterite could become economically important to operators on the Jos Plateau, but the paper covered the recovery in jigs only and did not deal with procedures in the dry mill where the finished products would be produced. The author mentioned the Carpco high-tension separator very briefly. Was it proposed to place that machine at the head of the dry-mill flowsheet—that is, immediately after the screening of the feed—and did he propose to pre-heat the feed to that separator?

Mr. E. K. Furze replied that one series of test runs had been made using a jig and tables as the preliminary concentrating unit, while a parallel test using screens and the high-tension separator had also been made. Although assay results were not available the latter method, in the opinion of the speaker, gave a more positive separation between the limits of 30 mesh and 100 mesh. On grain sizes larger than 30 mesh, high-tension separation was not found to be particularly effective, while in the minus 100-mesh fraction it was found that ultra-fine conductor minerals tended to report with the non-conductor product. It was therefore probable that both approaches would have to be combined to achieve the most efficient results.

AUTHOR'S REPLY TO DISCUSSION: NIGERIA

Mr. F. A. Williams: Mr. J. Braithwaite's queries about the relative advantages of four cells in series and two sets of two cells have been ably dealt with by Mr. Hammon, but the fact remains that more detailed research work is needed to determine reliably differences in percentage recovery in relation to variation in the rate of sand feed and the degree of dilution.

Mr. D. Hinton asked what was the overall loss of cassiterite and columbite from the jig for which a performance analysis was presented in Table IIA. Actually there is very little really fine cassiterite and columbite in this lead. The overall loss of cassiterite was only 2 per cent and that of columbite 5 per cent. This is a dragline washing plant. As only wash is treated the grade of the feed to the jig plant is very high—averaging about 10–15 lb of cassiterite and 1–2 lb of columbite per cu. yd, but sometimes

*Managing Director, Tin and Associated Minerals, Ltd.

very much richer. Even such small percentage losses from a four-cell jig might justify the use of two more cells.

Mr. F. A. Garner's questions about the accuracy of grain counting have been comprehensively answered by my colleague, Mr. J. A. Bain, who made the mineral analyses on which were based the additional jig performance tables submitted to bring the record up to date.

The irregularities in the tables to which Mr. J. Vitoria has drawn attention arise from variations in both the rate and the grade of feed to the jigs and the consequent difficulty of obtaining truly representative samples. Automatic sampling, more particularly of the tailing, over a longer period, say a month, should give more regular results.

Mr. D. Foord asked what happens to all the fine cassiterite shed from primary deposits. It is of interest to record that when a superpanner was used to value an alluvial bore 65 ft deep very fine cassiterite and columbite was found throughout the overburden. This overburden would represent a more sluggish stage of the stream. Very fine cassiterite and columbite probably travels a long way downstream, but values would become progressively diluted by barren tributaries.

I support Mr. Bennetts's remarks about the economic need to pump less water for more yardage and the desirability of discussing this in more detail at some future date. If less money is wasted by pumping excessively large volumes of very dilute pulp more could be spent on increasing the percentage recovery of columbite and cassiterite as by using six cells in series. Then of course even more semi-heavy mineral would be recovered at the same time.

From the reply by Mr. Furze to Mr. Coskinas it would appear that solutions are within sight to the problems arising in the separation of cassiterite and columbite from the large tonnages of semi-heavy minerals sent to the mill.

As a later written contribution to the discussion I now submit Table VI

TABLE VI.—*Percentage Recovery Efficiency of Individual Cells*

Jig Type	Mineral	Cell No.			
		1	2	3	4
Pan-American	Cassiterite	57·9	64·7	53·9	83·1
	Columbite	49·9	53·4	39·6	44·4
	Xenotime	49·2	53·8	33·1	52·3
	Average	52·3	57·3	42·2	61·6
Yuba-Richards	Cassiterite	72·0	63·7	39·8	41·8
	Columbite	59·4	55·4	46·3	52·9
	Zircon	49·6	43·8	43·7	60·6
	Average	60·3	61·0	43·3	55·1
	Overall average	56·3	59·1	42·7	58·3

and some comments thereon, both having a bearing on points raised by Mr. Hammon, who queried my preference for increasing the length of jig capacity rather than the width and also drew attention to the deleterious effect of excessive drops between cells on recovery. Table VI confirms the deleterious effect of excessive drops between hutches but supports my preference for more cells in series.

Mr. Hammon cited figures for the progressive dilution of the load passing over the jig, occasioned by the amount of added hutch water, part of which rises through the bedding. That such dilution takes place is irrefutable. Where Mr. Hammon has gone astray is in inferring without proof by sampling that this dilution would adversely affect recovery. Table VI shows that actually it does not adversely affect recovery. One can only guess at the reason for this. Possibly the beneficial effect of diluting the slime in suspension about balances the adverse effect of the greater speed with which the diluted bed of sand moves down the length of the jig.

For Table VI the figures for Pan-American jigs were calculated from those recorded in Table VI of my earlier paper.* Those for Yuba-Richards jigs were calculated from Tables IA and IIA included in my introduction (pp. 441, 442). Actually Yuba-Richards jigs as supplied have only a small uniform drop between each of four cells in series and even this drop could be evened out with the addition of riffles graded in height. At the time of sampling, however, some experimental riffles in use had produced an excessive drop between the second and third cells. This fault was later rectified but we have not yet experimented with eliminating drops between cells altogether. It would be more difficult to eliminate them from Pan-American jigs.

AUTHOR'S WRITTEN REPLY: LONDON

Mr. F. A. Williams: I wish to thank Mr. G. C. Ackroyd for his very comprehensive introduction of the paper and also all those who contributed to the discussion.

Several speakers wanted to know more about the method used to estimate the percentage tailing losses when the only data available were screen and mineral analyses of representative timed samples of the spigot discharges from the hutches. Actually the method is quite simple and straightforward. Under ideal conditions it would, theoretically, give perfectly accurate results, but because of the considerable degree of irregularity in jig performance in a commercial plant the results were in practice only an approximation as is clearly stated in the paper.

Study of a number of operational analyses of jig performance, all made in terms of closely sized fractions of individual minerals, revealed that, *on the average*, the percentage recovery efficiency of each cell is independent of the grade of the feed to it, i.e. the amounts of any specific mineral of a particular grain size recovered in successive cells approximates to a regular diminishing series. This is a very useful relationship to know, particularly for generalizing rather irregular or incomplete sampling results.

For example, in the case of a four-cell jig with each cell pulsating at the same speed and length of stroke with the same type, size and thickness of ragging and with the same upward flow of hutch water, if, say, 40 per cent of a specific mineral of a particular grain size is recovered in the first cell leaving 60 per cent to pass to the second cell, the amount recovered in the latter would be $60 \times \frac{40}{100} = 24$ per cent, leaving 36 per cent to pass to the third cell; there the amount recovered would be $36 \times \frac{40}{100} = 14 \cdot 4$ per cent, leaving $21 \cdot 6$ per cent to pass to the last cell; and there the amount recovered would be $21 \cdot 6 \times \frac{40}{100} = 8 \cdot 64$ per cent, leaving $12 \cdot 96$ per cent to pass to the tailing. If the recoveries only are now equated to 100 per cent the percentage represented by the last cell is $9 \cdot 93$. Therefore, if sampling of only the spigots of the hutches shows $9 \cdot 93$ per cent of the recovery to be from the fourth cell, then the percentage of the original feed lost on the tailings is $12 \cdot 96$. In this way a conversion graph can be constructed to give the tailing losses when only the spigots are sampled. Theoretically the progressive dilution of the feed by that portion of the hutch water which rises through the bed would reduce the percentage recovery efficiency of successive cells. In actual practice no such reduction in efficiency takes place. (See my reply to Mr. Hammon, opposite.)

Tables I and II in the paper were compiled from spigot sampling only and using the above method for estimating the tailing losses. Tables IA and IIA (pp. 441 and 442) were compiled from spigot *and* tailing samplings. The results show considerable irregularity, owing to the great variations in the rate and value of the feed, but are substantially in agreement. This vindicates using the above method of estimation when the value and rate of flow of the tailing cannot be directly determined with reasonable accuracy.

In commercial plants it is nearly always possible to sample the spigot discharges individually by taking the total flow of each for a uniform measured period of time and repeating this procedure at regular intervals, over several shifts if necessary, in order to get reasonably representative results. But, unfortunately, the positioning of the tailing launders most commonly precludes comparable sampling of the tailing discharge of individual jigs, particularly for determining the rate of discharge, for which purpose it is desirable to take the whole flow for measured periods of time. This difficulty applies both to shore-based jig plants and to dredges—hence the necessity to estimate the tailing losses in many cases. (See also my remarks on p. 436.)

Dr. Mackay raised the question of the recovery of composite grains from comminuted hard ores. Tables I and II, since supported by Tables

*Trans. Instn. Min. Metall., Lond., 67, 1957–58, 100.