shown a 14-in cyclone, having an 8-sq. in inlet and operated at moderate pressure, to be most satisfactory. The capacity of such a unit is about 210 gal/min, and four are required.

(c) Screens

Screen tests on the roughing jig tailings have indicated that a sieve bend having apertures of 2 mm is most suitable to give a good split of the coarser sands. This aperture is equivalent to a screen size of 16 mesh but practical tests have shown that a considerable amount of sands up to 40 mesh will report in the oversize. There should be no cassiterite coarser than 100 mesh in the feed to these screens.

The aperture size indicated for screening the cleaning jig tailings is 0·75 mm. This is equivalent to a screen size of 40 mesh but, again, smaller sizes will report with the oversize. The mineral in these tailings is normally all −150 mesh.

(d) Concentrating Spirals

Because of the great variations in feed sizes and values, and because of the necessarily limited scope of the tests, it is not possible to estimate at this stage, with any degree of accuracy, how many spirals will be required.

Assuming a maximum throughput of 240 tons/h with a maximum slimes content, tests have indicated that 70 per cent of the total feed will be in the primary cyclone overflow, i.e. 168 tons/h. Of this, something like 14 per cent will be recovered from the secondary cyclones, i.e. 23·5 tons.

It is probable that the scavenger jig tailings will amount to about 50 tons/h maximum and that the maximum amount of solids from the cyclones taking the rake overflow will be 10 tons/h. There will probably be a further 10 tons or so of screened cleaning jig tailings per hour.

Since maximum tonnages will not be obtained from the secondary cyclones and scavenger jig tailings at the same time it is quite likely that the total amount from these two sources will not exceed 28 tons/h. Together with screened tailings from the cleaning jig this means a maximum of 38 tons/h to the roughing spirals. It may therefore be necessary to have 38 roughing spirals and possibly eight cleaners, followed by two sand tables up-grading the spiral concentrate. The full details on this part of the plant will not be available until the plant is in operation.

The final layout is shown diagrammatically in Fig. 5.

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Recovery of Fine Alluvial Cassiterite: Correlation of Bore Valuations with Plant-Scale Recovery*

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SYNOPSIS

A new method for the recovery of fine cassiterite from alluvial bore and plant samples has been put into practice in the Plateau minesfield, Nigeria, and is thought suitable for use in Malaya and other alluvial tinfields. The method yields both valuations and size analyses down to 300 mesh and is fast and reliable. The field and laboratory equipment and procedures are described in the paper and applications discussed. The use of this method both for valuing the alluvial deposits and for ore-dressing research, leading to higher recoveries of fine cassiterite in plants incorporating hydrocyclones and jigs, may bring some ground of marginal value into the category of payable reserves and indicate the possibility of improving recoveries from some richer ground.

The paper is divided into three parts: valuation and analysis of alluvial bore samples, investigation of plant scale recovery, and flowsheets.

THE LOSS OF FINE CASSITERITE in gravel-pump mining and dredging is of concern in Malaya and other alluvial tinfields. The method of sample valuation and analysis described here, which has been thoroughly tested by use for check boring and plant sampling in Nigeria, can be used both for the valuation of reserves and for research on recovery.

Because the commercially recoverable cassiterite in alluvial deposits is substantially all free, the accepted practice, until quite recently, was always to value alluvial bore samples by the physical separation of the cassiterite. It is therefore important that a correlation should be established between bore valuations and plant-scale recovery in terms of the grain size of the cassiterite. Usually the amount of cassiterite recovered by hand puddling and decantation followed by hand panning of the clean washed sand was assumed to represent plant-scale recovery. Accepted procedures have been described by Harrison.[1]

The recent ore-dressing improvement of replacing sluice boxes on open-pit mines by plants containing hydrocyclones, sieve bends and jigs has, however, extended the commercial recovery of cassiterite to much finer grain sizes. The feasibility of improving recovery in the jig plants on dredges is also being studied and some experiments with sieve bends, hydrocyclones and spirals are being carried out. There has thus arisen

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

an acute need for a combination of valuation and screen analysis of both bore and plant samples. The method used by Harris[2], in which screen-sized sands and slimes are chemically assayed, is not entirely satisfactory. It is slow and does not distinguish between cassiterite which is free and that contained in grains of sand.

All practical methods of valuing alluvial bore and plant samples must be, in some degree, compromises between speed, cost, reliability and detail. The procedure for the valuation of alluvial bore and plant samples which the author devised and put into practice in the Plateau minesfield of Northern Nigeria is still such a compromise but more in keeping with present-day requirements than older accepted methods. This procedure should also be suitable for use under Malayan conditions and in other alluvial fields.

The procedure is briefly as follows. Any losses which occur in desliming each sample before panning are retrieved at the bore or sampling site by pumping the slimes through a small hydrocyclone. Any losses which occur in panning each sample are retrieved in a central laboratory by concentrating the sand tailing on a half-size Holman sand table, the table heads being then cleaned up on a Haultain Superpanner. The concentrate is dried and screen sized and each sized portion valued by grain counting under the microscope, usually after the removal of some of the minerals by magnetic separation and/or leaching with hydrochloric acid. This procedure yields both a valuation and a screen analysis of the free cassiterite content. In Nigeria the columbite content of the samples is also determined

In Nigeria this procedure is now being used by a leading tin mining company not only for valuing bore samples but also for valuing head and tails samples from plants to determine recoveries in relation to grain size. During the author's recent tour of Malaya, discussions of the problems of recovering fine cassiterite indicated that the procedure advocated might be useful there and in other alluvial minesfields.

A preliminary investigation of the amount of fine cassiterite (and columbite) in alluvium on the Jos Plateau, Nigeria, was first carried out by the author and his colleagues several years ago. (This was part of a more comprehensive programme of geological studies in which the techniques of quantitative fragmentary petrography were used in a more detailed investigation of the alluvial deposits than had been customary in valuation by boring.) A representative sample of each section of a bore was deslimed very carefully with an abundance of water, the clean sands were screen sized and each sized fraction was concentrated in a Haultain Superpanner. The concentrates were then physically assayed by grain counting. This method was too slow for routine use in alluvial bore valuation, but it yielded figures of great interest. The results for cassiterite are set out in Table I. Wash containing relatively coarse cassiterite was found at two horizons. In the original boring at this site the values of all the other sections obtained by ordinary panning had been reported as 'trace' or 'nil', but the superpanner revealed appreciable values from the surface to bedrock.

It was the additional amounts of fine cassiterite and columbite revealed by the use of the superpanner which led the author to devise and put into

TABLE I.—*Cassiterite in Alluvium, Plateau Minesfield, Nigeria*

Section ft	Cassiterite content							Clay content, %
	lb/cu. yd	Screen analysis %, B.S. sieve nos.						
		+25	25/52	52/72	72/100	100/150	−150	
0– 5 .	0·17	—	4·6	3·8	8·4	39·9	43·3	67
5–10 .	0·14	—	3·4	2·7	6·2	30·8	56·9	75
10–15 .	Tr	—	—	—	—	—	—	74
15–20 .	0·13	—	6·0	8·6	8·2	37·8	39·4	82
20–25 .	0·08	—	16·7	1·8	4·2	10·8	66·5	40
25–30 .	0·36	—	8·8	13·0	16·5	31·0	30·7	54
30–35 .	0·80	1·7	11·5	6·0	11·0	14·0	55·8	46
35–40 .	0·34	—	—	7·1	11·6	30·7	50·6	60
40–43 .	0·74	—	6·9	7·6	13·2	19·0	53·3	46
43–49 .	4·71	12·0	19·8	25·0	14·5	16·8	11·9	11
49–55 .	0·15	—	18·1	13·9	16·2	23·3	28·5	78
55–60 .	0·16	—	—	—	7·5	17·8	74·7	81
60–61½ .	1·39	—	3·6	2·3	5·2	17·9	71·0	46
61½–64½ .	7·04	2·3	3·0	70·7	17·2	4·6	2·2	15

practice a much more rapid procedure for valuing samples to include very fine cassiterite.

I. VALUATION AND ANALYSIS OF ALLUVIAL BORE SAMPLES

Field Equipment and Procedure

A Monopump (Model CD 40B4), which does not require priming, was used in the field to deliver 8 gal/min at a pressure of 8 lb/sq. in. to a hydrocyclone of 3-in diameter. This gave a split at approximately 20 μ for sand and about half that grain size for cassiterite. The prototype was installed for testing in a Land-Rover and driven from the rear power take-off (Fig. 1, Plate I). Later portable units were used (Fig. 2, Plate I), driven by Model Z Briggs and Stratton petrol engines developing about 3½ h.p. at 1500 rev/min, which are light enough for two men to carry. Lighter engines of less horsepower should suffice as, according to the suppliers of Monopumps, the model used requires only 1½ h.p. This version of the field equipment should be suitable for use under Malayan conditions and in other countries where it is not usually possible to take a Land-Rover to the bore sites.

Bore samples are deslimed at the bore sites in the usual way and the slimes collected in half oil drums, the slimes being then pumped through the hydrocyclone. The underflow, which still contains a little colloidal slime, is diluted with the water in which the clean sands have been panned and pumped through the hydrocyclone a second time. The sand tailings from panning are screened through a 12-mesh sieve, the oversize discarded and the undersize added to the final hydrocyclone underflow. This mixture is then sent to the central laboratory for the recovery of the remaining, mainly fine, cassiterite and columbite. The bore cores are washed up in sections of 5 or 10 ft and the products of each section are kept separate. A very important aspect of this field procedure is that it keeps pace with the rate of boring.

It is of interest that the mixture of sand and slime obtained from the periodic sampling of heads and tails at recovery plants in Nigeria is similarly dealt with. In this application also the rate at which samples can be cycloned and panned keeps pace with a suitable time interval for taking the samples and with the normal size of the samples. For dealing with samples on dredges an electrically-driven Monopump could be used.

Core Measurement

Methods of core measurement used in Malaya and elsewhere in valuing alluvial deposits by boring have been described by Harrison.[1] In Nigeria all alluvial bores were valued until recently by a 'shoe factor' approximating to a theoretical core of the diameter of the cutting edge of the shoe. As core recovery is usually very much less than the theoretical this practice alone resulted in undervaluation of the ground. In addition there were the losses in desliming and panning since shown also to be appreciable. The result was that statistics of recovery in relation to bore values conveyed a false impression and failed to draw attention to the poor recovery efficiency of sluice boxes for cassiterite. As in Malaya, great quantities of valuable mineral have been lost by the failure of the gravel pump mining industry to replace sluice boxes by jig plants in the early 1920s when jigs first began to replace sluice boxes on bucket dredges. Hydrocyclones ahead of jigs were first used in Nigeria in 1953 for treating decomposed columbite-bearing granite.

With the more effective methods of recovering fine cassiterite and columbite from bore samples, a method of core measurement to replace the shoe factor was also introduced in Nigeria. This had been devised by ~~the author and~~ used in Australia more than 20 years earlier and attempts to give the original volume of the core *in situ* as accurately as practicable. The last hidden factor leading to the undervaluation of alluvial bores was thus eliminated, and deductions for losses in plant-scale recovery would henceforth have to be made openly, however embarrassingly large such deductions might have to be until recovery plants were improved.

In Malaya it was formerly a common practice to base bore valuations on the measured volume of the wholly or partly disintegrated cores. Naturally the expansion of these cores from their volumes *in situ* was then reflected in undervaluation of the bores. Although an adjustment for this expansion of the core is now usually made there must still be appreciable losses of fine cassiterite during desliming and panning which also result in such undervaluation. For far too long the recovery of bore values was accepted as indicating satisfactory performance of the recovery plants.

In the new Nigerian method that portion of the core which is recovered solid is measured immediately by water displacement. The remaining disintegrated portion of the core is then thoroughly deslimed and the solids content of the decanted slimes determined with a hydrometer. The clean sands from this portion of the sample are then placed in a measuring box with an excess of water and the box bumped until the volume occupied by

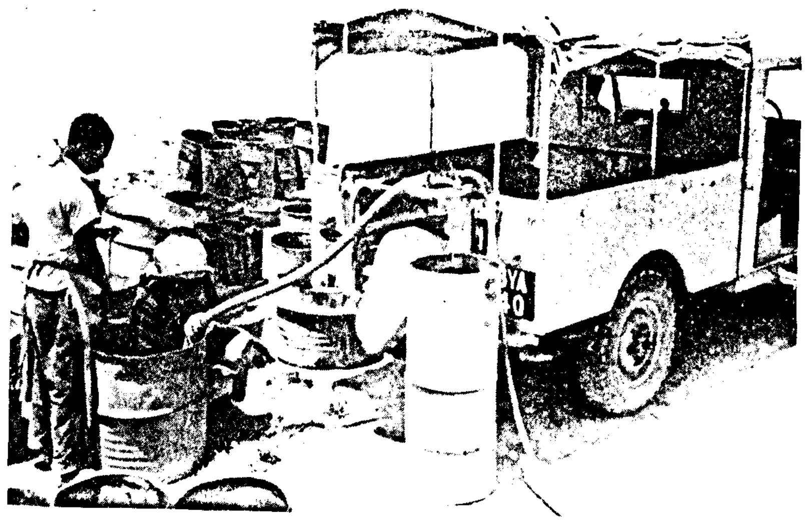

Fig. 1. Initial cyclone unit mounted on a Land-Rover.

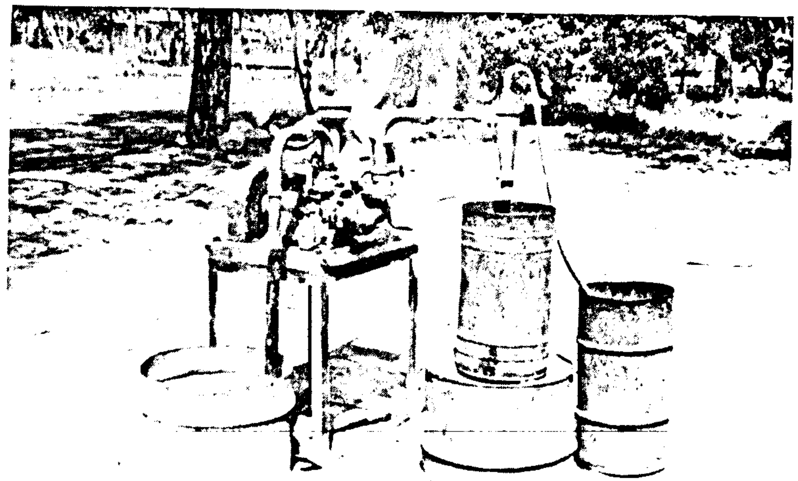

Fig. 2. Portable cyclone unit.

Plate II F. A. WILLIAMS: *Recovery of Fine Alluvial Cassiterite: Correlation of Bore Valuations with Plant-scale Recovery.*

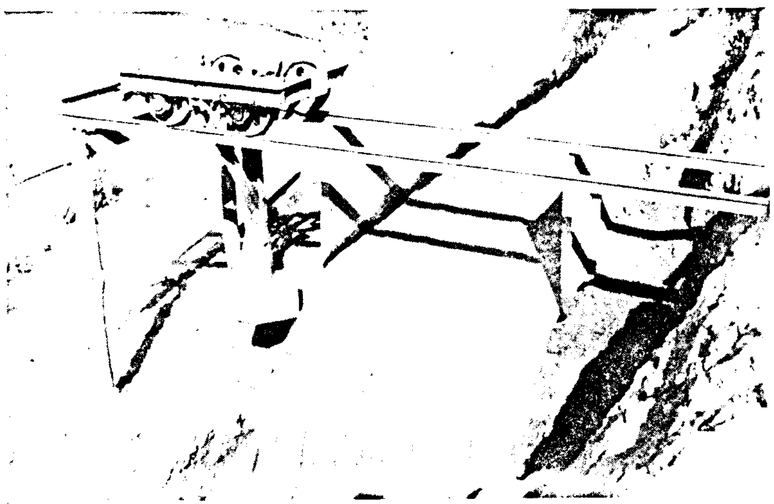

Fig. 3.—Sample cutter fitted across tailings launder from sluice boxes.

the sand ceases to diminish. To determine the average porosity a representative number of these samples of clean sand were dried and the net volume measured by water displacement. If the solids content of the decanted slimes exceeds the average porosity of the well packed sands the difference is added to this packed volume. The corrected volume of the sands from the disintegrated portion of the core plus the directly-determined volume of the undisintegrated portion is assumed to represent the original volume of the core *in situ*. This volume is used to calculate the value of the bore or sections of it.

The shoe factor is still widely used in Nigeria because it is a cheap and quick method of valuation. Its use can be justified for true prospecting and perhaps scout boring for preliminary valuation, but it would be incongruous to use a shoe factor in conjunction with the improved method of sample valuation described in this paper.

Laboratory Equipment and Procedure

The sands received from the field are concentrated in two stages, first by tabling and then by superpanning. An important aspect of the procedure is that one table can keep pace with the output of one boring gang except when they are drilling fast in shallow ground. The third stage is the physical assaying of the concentrates in terms of free cassiterite to yield both valuations and screen analyses.

Stage 1

Preliminary tests showed that the usual type of small laboratory table about 2 ft 3 in. long was too small for reliable results. A half-size Holman sand table was therefore fitted with an improvised rubber deck carrying rubber riffles fixed on with waterproof adhesive. This deck covering did not crack if it was allowed to dry and no grains of mineral could be caught under the riffles.

For concentrating the sand samples a standard routine procedure was adopted; classification was not used. The tailing from the first pass over the table is re-treated twice, and each time a head and a middling are collected. The final tailing is discarded. The combined middling is then re-treated and a tailing discarded. The final middling is then added to the combined heads. The sands from a 5-ft bore section can be put through in about half an hour. The final table tails were checked by superpanner on several occasions and the losses found to be negligible down to 300 mesh. A start was then made with installing more tables to keep pace with more than one boring gang and also with the simultaneous head and tail sampling of recovery plants.

Checking bulked table tailings by superpanning is now being considered. The water would be pumped through a 3-in cyclone and the underflow added to the sands. At convenient times the accumulated tailing would be run through a wet sample splitter. At the end of a week or month the accumulated splits would again be run through the wet sample splitter to give a small final sample for drying, screen sizing and superpanning. The additional recovery of each grain size, if appreciable, could be backproportioned over the original samples tabled. This would provide a check on careless operation of the table or the loss of some coarse cassiterite if present in the unclassified feed. It would also extend the group valuation to sub-sieve sizes of cassiterite. This is an example of the practical necessity mentioned for compromise between speed and detail.

The wet sample splitter has already been installed. It consists of a water supply to steel a launder to the discharge end of which a Geary-Jennings sampler has been fitted. The cutter of this sampler moves across the flow at a constant speed, the split and the remainder discharging into separate rectangular steel vats for volume measurements to determine the percentage represented by the portion split out.

Stage 2

The weights of the final sample from tabling and of the amount of sand left in it vary appreciably. One superpanner in operation full time and one part time will keep pace with one table. The concentrates are superpanned without sizing merely to remove the remaining small quantity of sand and to yield a clean concentrate of heavy and semi-heavy minerals.

The concentrates recovered by panning in the field and by tabling in the laboratory are kept separate in order to obtain figures for the percentage recovery in the field as some indication of correction factors which should be applied to old boring results. Preliminary tests had shown that losses of cassiterite and columbite during the desliming of the samples in the field were considerable and varied with the amount of water used, that amount tending to diminish with the distance it had to be carried. Cycloning the slimes eliminates the personal factor in desliming samples in the field and tabling and superpanning the −12-mesh fraction of the sands retrieves and concentrates the cassiterite (and columbite) lost in panning.

Stage 3

Physical assaying of the concentrates is preferred to chemical assaying for practical reasons. Separate grains of free cassiterite which have been seen under the microscope must all be considered as potentially recoverable at plant scale. In contrast chemical assays are less convincing and may often, moreover, actually be misleading owing to the presence of cassiterite included in grains of sand.

Although the separate complete valuation and analysis of each 5- or 10-ft section would have yielded interesting and useful information, a compromise had to be made between speed and detail. Only the field concentrates are valued section by section. For this purpose the magnetics are first removed by fixed electromagnets equipped with foot switches, the small amount of sand present is then separated from the non-magnetic fraction by tossing and the resulting fairly clean cassiterite is weighed. This gives a preliminary indication of the value of each section of the bore. After these results have been examined the samples are arbitrarily divided into overburden and wash and bulked to make only two samples of each type. Field and laboratory concentrates are still kept separate. Sometimes this procedure is carried a step further by separately bulking the overburden and wash samples from a line or group of bores. Screen and mineral analyses are then made on these bulked concentrates. This reduction in the

amount of grain counting is useful for the rapid preliminary re-valuation of blocks of reserves.

In Nigeria the valuation of the concentrates is complicated by the presence of considerable quantities of columbite, sometimes exceeding the amount of cassiterite, and by the fact that the proportion of magnetic cassiterite varies up to 50 per cent or more. In addition alluvial leads draining from one particular type of granite also contain magnetic zircon. For use in Malaya, where there is usually very little columbite or magnetic cassiterite, the Nigerian procedure could be greatly simplified, and valuation and screen analysis in terms of cassiterite only would usually suffice. The valuation of concentrates in Malaya should therefore be much quicker than in Nigeria. If the amount of magnetic cassiterite and columbite present were economically significant, bulked magnetics from groups of bores could be dealt with separately so as not to reduce the speed of sample valuation.

The next step at Stage 3 in the Nigerian procedure is to screen size the dried concentrates. The selection of screens used is always more closely spaced in the finer sizes because these sizes are the most difficult to recover at plant scale. Each screen-sized fraction is valued separately. For magnetic separation there is a choice between three types of equipment available in the laboratory—fixed electromagnets with foot switches, variable-strength Eclipse hand magnets and a Frantz Isodynamic Magnetic Separator. They will all lift ilmenite. Following the method described by Jones and MacLeod[3] the magnetics are leached for 48 hours in boiling concentrated hydrochloric acid to decompose the ilmenite. The residue is then treated with dilute hydrochloric acid on a hollowed-out zinc block (to turn the cassiterite grey for ease of recognition), dried and again magnetically separated. The building, equipped with fumes cupboards and exhaust fans, has a capacity of over 700 samples.

The cassiterite and columbite content of the screen-sized fractions is determined by grain counting under the microscope. The cleaner the concentrate the fewer the number of grains which must be counted to maintain acceptable standards of accuracy, hence the advisability of first removing and leaching the magnetics. The proportions in which other minerals are present in the final concentrates examined under the microscope are determined in order to arrive at a figure for the bulk specific gravity for use in converting the grain count percentage to a weight percentage. This procedure has been described by Bain.[4] The formula used is

$$a = \frac{100\ (\mathrm{A} \times \text{sp. gr.})}{(\mathrm{A} \times \text{sp. gr.}) + (\mathrm{B} \times \text{sp. gr.}) + (\mathrm{C} \times \text{sp. gr.}) + \ldots\ldots}$$

where a is the weight percentage of the mineral; A, B, C, etc., are the count percentages of the constituent minerals and sp. gr. is their specific gravity.

In Malaya it should be possible to make such physical assays of large numbers of samples by adapting a system seen in practice in South Africa, where the mineralogist in charge of a laboratory supporting the work of field geologists was assisted by a team of women sorting minerals under binocular microscopes. With such a team engaged in grain counting, large numbers of samples could be dealt with, and it might even prove practicable to value separately each section of the bore, usually of 5 ft or less.

The mineral analysis of samples of cassiterite concentrate which have been cleaned by magnetic separation and leaching with hydrochloric acid is not usually difficult. Using an artist's small water-colour brush a representative portion of each screen-sized fraction is spread out in a thin line on a glazed white tile, and moved across the field of view of a binocular microscope. Without looking up from the microscope the operator can easily count up to five different minerals and record the numbers of each on a mechanical keyboard counter. The type of counter used in the mine laboratory in Nigeria totalled the counts of the individual minerals and rang a bell at every hundred, which facilitated recording the results as frequency percentages. Usually a count of 300 gave a result of acceptable accuracy. If all the minerals are familiar to the operator and easy to recognize, a count of 300 grains can be completed in about 2 min or even less. The mental concentration, keyboard memory and finger dexterity required are comparable with touch typing. In Nigeria the geologists and mineralogists have done the grain counting themselves.

In Malaya it should be practicable to train a team of women, particularly typists and comptometrists, to make such routine mineral counts. The mineralogist in charge would then need to deal only with the relatively few samples presenting some special difficulty.

If an adequately fast technique can be devised for removing the remaining free impurities then grain counting could be eliminated. Gravity separation in molten lead chloride, with the aid of a centrifuge for the fine grain sizes, may perhaps solve this problem.

Mr. B. H. Flinter, senior mineralogist in the Federation of Malaya Geological Survey Department, Ipoh, arranged a demonstration of the routine procedure used in their laboratory for separating some heavy minerals from each other, the demonstration being carried out with a mixture of fairly coarse ilmenite and columbite. Notes on this laboratory procedure, described in the appendix to this paper, were passed to the author's former colleagues in Nigeria, who tried it out. The melting point of lead chloride, used in this method, is 501° C, and although there appears to be no suitable heated centrifuge on the market, such a comparatively simple problem of design engineering should not be allowed to hold up research in the alluvial tin mining industry.

The free impurities remaining in the non-magnetic leached cassiterite concentrates in Nigeria are usually only zircon (sp. gr. 4·5–4·7), anatase (sp. gr. 3·9) and topaz (sp. gr. 3·5). These minerals have recently been removed by hand panning in heavy liquids, which may prove to be faster and less troublesome than the lead chloride method. It is also planned to try to remove these minerals by concentration in heavy liquids on a Muller micropanner,[5] which has only recently become available commercially. The figures for concentration criteria in tetrabromoethane (TBE) and methylene iodide given in Table II indicate that this is a promising approach to the problem of removing the non-magnetic free impurities topaz, anatase and zircon from cassiterite. The micropanner being very much smaller than the Haultain Superpanner, a dozen or more could be installed in a line

TABLE II.—*Concentration Criteria**

Cassiterite (sp. gr. 7·0) from	In water	In heavy liquids	
		Tetrabromoethane (sp. gr. 2·96)	Methylene iodide (sp. gr. 3·32)
		Sink-float	Sink-float
Quartz, sp. gr. 2·65	3·7		
Topaz, sp. gr. 3·5	2·4	7·8	20·4
Anatase, sp. gr. 3·9	2·1	4·3	6·3
Zircon, sp. gr. 4·7	1·6	2·3	2·7

*The higher specific gravity over the lower specific gravity each diminished by the specific gravity of the liquid.

on a bench and operated by a locally recruited staff. A suitable type of bench through which heavy fumes are appropriately removed by down-draught suction ventilation is used in the laboratory of Minerais et Métaux in Paris. A large number of screen-sized samples might thus be finally dressed up at an adequate rate of throughput.

Field Recovery

Table III shows the recovery of cassiterite from bore samples in relation to grain size achieved by desliming the sample in the field and panning the washed sands. There are three main points of interest in this table. First, there is no sign of the rise in losses in the middle size range reported by Harris;[2] second, recovery diminishes progressively with decreasing grain size and is very poor for sizes finer than 150 mesh; last, the very poor recovery from the overburden, even in relation to grain size, suggests that very considerable losses probably occur in the desliming.

Each of the three sets of figures in Table III (p. 58) is the average for a line of bores across the same alluvial lead. The results show considerable variation of recovery in relation to grain size. This suggests that bore valuation without a cyclone and shaking table must be very unreliable.

It should be interesting and informative to have for comparison tables of results obtained from actual alluvial bore samples under Malayan conditions, using a hydrocyclone but a dulang instead of the Nigerian calabash. These would be more convincing than the results obtained by Harris[2] with synthetic samples and chemical assaying.

Grain Size Distribution of Cassiterite

Average screen analyses of the total free cassiterite recovered from the bores in several different areas on the Jos Plateau, Nigeria, are set out in Table IV. The possession of such data for all blocks of reserves is, of course, essential for a proper study of the design engineering and economics of recovery plants. A fairly simple jig plant would suffice for Area A, but more thorough desliming in hydrocyclones and more jig cells in series and/or in parallel could be considered for Area D. Actually the position is not quite as simple as that because of the relative amounts of columbite present.

TABLE III.—*Field Recovery of Cassiterite from Alluvial Bore Samples in Nigeria in Relation to Grain Size*

B.S. sieve no. (C = commercial)	From overburden	From wash	Average from surface to bedrock
		Line I	
	%	%	%
+ 25	48·1	92·8	90·1
25– 52	39·5	87·9	85·1
52– 72	44·7	82·5	79·8
72–150	15·9	35·6	31·5
150–240	0·8	11·2	4·1
240–325 C	0·2	0·9	0·5
–325 C	0·2	3·0	1·2
Total recovery	9·9	67·2	54·1
		Line 2	
	%	%	%
+ 25	50·1	83·4	79·4
25– 52	49·3	95·5	91·9
52– 72	71·2	90·9	89·1
72–150	35·6	70·9	66·3
150–240	7·0	19·9	14·6
240–325 C	0·2	0·7	0·4
–325 C	0·7	2·1	1·4
Total recovery	17·8	67·7	56·6
		Line 3	
	%	%	%
+ 25	42·6	93·2	91·4
25– 52	63·4	95·4	94·2
52– 72	73·0	91·4	93·0
72–150	32·9	72·9	67·0
150–240	7·1	25·3	19·3
240–325 C	2·8	6·1	4·5
–325 C	3·1	1·0	1·9
Total recovery	32·2	86·5	82·2

Such screen analyses of the total free cassiterite in the ground extending reliably to fine grain sizes, not previously available in Malaya, could now be obtained by the method just described. Areas previously thought to be of marginal value may prove to be payable if the full amount of recoverable fine cassiterite is taken into account. The total amount of fine cassiterite present in such areas and its grain size analysis can now be determined by check boring.

Some screen analyses of cassiterite concentrates from field recovery only in alluvial boring in Malaya are recorded in Table IVA. These concentrates still contain some other minerals—actually as much as 50 per cent in the

TABLE IV.—*Alluvial Bore Valuations, Nigeria: Total Recovery in Field and Laboratory : Screen Analyses of Cassiterite from Surface to Bedrock*

B.S. sieve no.	Area A	Area B	Area C	Area D
	%	%	%	%
+ 25	56·0	22·9	16·1	15·3
25– 52	24·4	29·7	29·4	36·6
52– 72	10·1	26·0	23·1	15·3
72–150	7·7	19·1	24·3	13·9
150–240	1·5	2·1	5·2	13·5
240–300	0·2	0·1	1·1	3·4
–300	0·1	0·1	0·8	2·3
	100·0	100·0	100·0	100·0

TABLE IVA.—*Alluvial Bore Valuations, Malaya: Field Recovery Only, Surface to Bedrock Screen Analyses of Cassiterite Concentrate**

B.S. sieve no.	From areas progressively further downstream			
	Area 1	Area 2	Area 3	Area 4
	%	%	%	%
+ 25	2·1	2·2	0·2	0·1
25– 52	17·7	16·0	7·3	0·1
52– 72	36·0	28·0	31·2	3·2
72–150	41·9	49·7	59·5	80·2
–150	2·3	4·1	1·8	16·4
Total	100·0	100·0	100·0	100·0
B.S. sieve no.	From scattered areas arranged according to diminishing average grain size			
	Area 5	Area 6	Area 7	Area 8
	%	%	%	%
+ 25	30·6	0·5	4·7	0·5
25– 72	52·2	63·4	26·2	32·1
72–100	9·1	20·4	39·6	28·8
100–120	2·6	4·9	13·6	12·6
–120	5·5	10·8	15·9	26·0
Total	100·0	100·0	100·0	100·0

*Still contains some free impurities particularly in the finer sizes.

finer fractions. Unless the field recovery in relation to grain size from bore samples in Malaya is very much better than that recorded for Nigeria (see Table III) it would appear that in at least some Malayan areas there must be very considerable amounts of unrecorded fine cassiterite.

Reading the figures in the last column of Table III with those for Area 4 in Table IVA, it would appear that the bores in the latter area may have revealed less than half the actual amount of free cassiterite in the ground. Although this inference may not be entirely justifiable it certainly indicates the need for direct investigation.

The valuation of tailings can now also be undertaken with confidence and a study be made of the problems of recovering fine cassiterite previously lost from palongs fed by hydraulic elevators or gravel pumps and from jigs without hydrocyclones on dredges. The practical method of sample valuation described opens up a new approach to projects for the removal and retreatment of old dredge tailings as the first step to gaining access to the rich wash left behind between limestone pinnacles. Chinese miners are already reworking tailings profitably with gravel pumps and small jig plants, but the results of the accurate valuation of large areas of old tailing could conceivably justify the necessary capital outlay for larger-scale operations in some localities.

II.—INVESTIGATION OF PLANT-SCALE RECOVERY

The basic unit for plant-scale recovery of cassiterite is the jig. It has been standard equipment on bucket dredges for over 30 years and few dredges remain still equipped with sluice boxes, while jigs are now rapidly replacing sluice boxes in gravel-pump mining. Recent research work in Nigeria[6,7] and by the Mines Department, Research Division, in Malaya[8] has demonstrated that effective recovery of cassiterite in jigs can be extended to much finer grain sizes if the slime content of the feed is considerably reduced by first passing it through hydrocyclones.

Table V-A shows that, after the feed has been cycloned once, recovery in a 4-cell jig is still over 95 per cent at the fairly fine grain size of 150–170 mesh, but thereafter recovery diminishes rapidly with decreasing grain size. Table V-B shows that after the feed has been diluted and cycloned a second time, thus reducing the amount of slime to negligible proportions, recovery is still about 95 per cent at the extremely fine grain size of 240–300 mesh. In both cases the tailing values were obtained by direct sampling. There is no sign of the rise in tailing losses in the middle size range reported by Harris.[2] Instead recovery diminishes progressively with decreasing grain size. These two tables also show the usefulness of the third and fourth cells for obtaining a high recovery in the fine grain sizes from hydrocyclone underflows.

It is thus apparent that virtually all the cassiterite down to 300 mesh which can be physically recovered from bore samples by the method described can also be recovered at plant scale by the use of hydrocyclones followed by jigs of four cells in series. What remains to be determined for individual properties is whether or not the value of the additional recovery would show a profit after allowing for the cost of desliming the feed to the jigs. If necessary the investigation can be extended to sub-sieve sizes of cassiterite as described in the previous section of the paper.

There is need for more detailed research work on mineral recovery through the screen in jigs. For instance, the recovery of cassiterite over a range of grain sizes has not yet, to the author's knowledge, been systematically studied hutch by hutch in relation to all the major variables—grade of feed, rate of feed, dilution of feed, amount of slime suspended in the water, length of stroke, speed of stroke, depth and nature of ragging and hutch water control. It is not even known which of the two arrangements of cells, one line of four in series or two lines of two in series, gives the better recovery from the same quantity of feed. It is particularly important that the conditions under which high losses occur in the middle size range, as reported by Harris,[2] should be defined. Sufficient work has been done in Nigeria to show that this phenomenon does not normally occur there.

The Nigerian results[6] were obtained under difficulty, without the benefit of installed samplers, either fixed or automatic, or by-pass launders for measuring rates of feed and tailing discharge. In Malaya adequate sampling facilities may be provided for investigating jig performance in detail on one or more dredges working different types of ground; but in both

TABLE V.—*Recovery of Cassiterite in Jigs: Distribution*

B.S. sieve no.	Recovery, % Hutch no. 1	2	3	4	Total	Lost, %
A: Feed: 10 tons/hr; screened through ½ in. Cycloned once. Speed: 129 rev/min. Stroke: 1½ in.						
16	98·0	1·8	0·1	– –	99·9	0·1
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
B: Feed: 7·4 tons/hr; screened through ¼ in. Cycloned twice. Speed: 155 rev/min. Stroke: ½ in.						
16	100·0	– –	– –	– –	100·0	– –
16– 25	93·6	6·4	– –	– –	100·0	– –
25– 52	72·8	24·8	2·4	– –	100·0	– –
52– 72	68·5	28·2	1·2	2·1	100·0	– –
72–100	67·3	25·7	3·1	3·8	99·9	0·1
100–120	62·3	28·1	4·7	4·5	99·6	0·4
120–150	56·9	29·3	8·3	5·1	99·6	0·4
150–170	48·2	34·7	10·0	6·7	99·6	0·4
170–240	46·3	27·6	14·4	10·9	99·2	0·8
240–300	26·3	25·5	27·1	15·4	94·3	5·7

countries variation in the composition of the feed presents a major sampling difficulty.

A selection of four-cell jig assemblies for this much-needed plant-scale research work is shown in Fig. 4. The most versatile, although necessarily the most expensive, is type A in which both the speed and the stroke of each cell can be varied individually. The removable launder between the two pairs of cells is necessary for checking the validity of the type of flow-sheet recommended by Harris.[8] A by-pass launder can be inserted at the tail end of the second cell and a short feed launder ahead of the third cell.

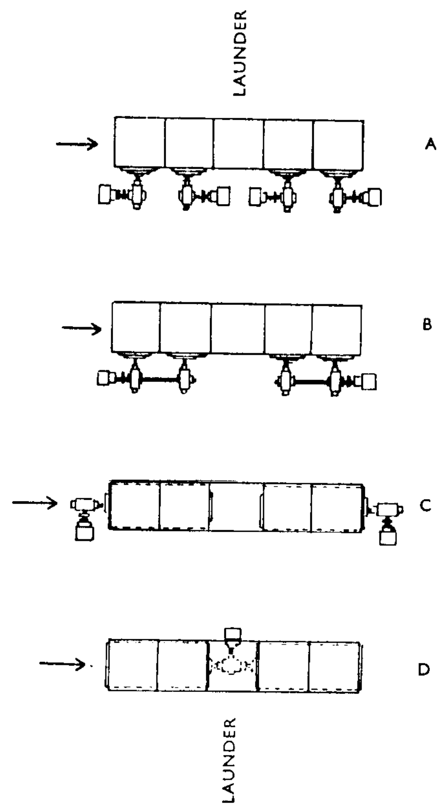

Fig. 4.—Four-cell jigs for test work; variable speed motors.

The tailing from the second cell can then be screened and the undersize dewatered and fed back to the third cell to ascertain whether or not with a given feed this refinement results in a worthwhile additional recovery.

Simultaneously the recovery of cassiterite in relation to grain size in the hydrocyclones themselves could be investigated using samplers, by-pass launders and the same techniques of sample valuation and analysis as

applied to the bore and jig samples. Both low- and high-pressure hydrocyclones should be tested. The results should be reported in terms of cassiterite as well as sand and slime (as in Table III of the author's earlier paper[6]).

By this means the industry would achieve a correlation between the recovery of fine cassiterite from bores and at plant scale. The recovery plants on dredges could then, if necessary, be re-designed to extend the recovery of fine cassiterite to the economic limit according to the amount found to be present in the ground, its range of grain size and the amount of slime present tending to diminish recovery. Another limiting factor is the space available on the dredge, of which better use might be made by installing modern jigs. The Dredge Data Chart for 1959 produced by the Mines Department of the Federation of Malaya shows that over 30 per cent of the jigs on dredges were then still of the obsolete Harz type.

Performance Analyses of Jigs

Sampling Procedure

The minimum objectives should be to determine separately the recovery of cassiterite in short-range grain size fractions from each hutch and also the tailing loss, for which purpose sampling alone is not sufficient. It is also desirable to determine the rates of discharge of water and solids at the points sampled. Standard time/weight sampling is usually easy to carry out for the spigot discharges of the hutches. Vessels with capacities of a few gallons can be thrust under each spigot in succession for a short period which can be measured with a stop-watch. The samples obtained can be put into larger vessels, one for each hutch. This procedure can be repeated at regular intervals and may extend over several shifts.

For the jig tailing discharge the combination of sampling and determination of rates of discharge often entails some engineering work. Pending better facilities, half an oil drum can be used to take the whole flow for a short measured period when the discharge is through a pipe. Usually recovery plants, on dredges and ashore in both Nigeria and Malaya, are designed and erected with a total disregard of the need for sampling, while, in marked contrast, it is the accepted practice in comminution mills to provide adequate facilities for weighing and sampling the flow of ore. An alluvial tin mining company in the Belgian Congo made adequate provision for both the sampling and the direct determination of the rate of discharge of the total tailing from shore-based jig plants. The sampling was done by an automatic splitter moving across the discharge lip at regular intervals, and the rate of discharge was determined by by-passing the whole flow for measured periods of time into a large steel vat where the volume could be measured. Comparison with the volume cut out by the automatic sampler calibrated the latter as yielding a certain percentage of the total flow. This procedure could be used on a dredge or in a shore-based plant to sample and determine the rate of flow of the tailing discharge from a single four-cell or two-cell jig selected for making a detailed analysis of its performance.

As the method of sample valuation described here can deal with fairly large quantities, the accumulated total tailing flow by-passed for repeated short measured periods of time could itself sometimes be accepted as the sample. This would apply particularly to cyclone overflows.

Unless it is desired to make a complete check in/check out metallurgical balance, as described by Fitzgerald,[9] it is not necessary to sample the heads. In any case the heads are usually the most difficult to sample and are only indirectly of analytical importance. The most significant values are those of hydrocyclone overflows and jig tailing which should, therefore, be determined by direct sampling.

Analysis of Samples

The total volume of the sample, including the water, should first be measured. The remaining procedure can be practically the same as with bore samples but in this case there is no point in keeping the field and laboratory concentrates separate. The total dry weight of sand tailings from each sample should be recorded along with the dry weight of the slimes previously taken out by cycloning. It may also be useful to have a screen analysis of the sands from each sample, which may prove to have some bearing on the recovery of cassiterite in the jigs in relation to grain size.

The concentrates should preferably be screen sized into at least ten sized fractions, particular importance attaching to sizes finer than 100 mesh. The cassiterite content of each screen-sized fraction can then be determined physically by grain counting under the microscope with or without the prior removal of other minerals by magnetic separation and/or leaching with hydrochloric acid.

Performance Analyses of Hydrocyclones

Proper facilities for the automatic sampling and determination of rates of discharge of both the overflow and the underflow are advisable for this ~~purpose also. It is not necessary to sample the heads~~ unless it is desired to make a fully checked metallurgical balance. The samples will consist of slime, sand and heavy minerals in varying proportions. As in the case of the bore and jig samples, they can be analysed and valued by the same procedure. The small hydrocyclone used in valuing the samples will recover the fine cassiterite lost from the large hydrocyclones.

As an alternative to tabling the whole sample, a wet sample splitter will enable a small representative portion to be obtained easily for superpanning, with or without screen sizing. This procedure might be preferable for samples of very fine grain size from the overflow of large hydrocyclones.

It is desirable that the separating efficiency of the large hydrocyclones should be reported in terms of cassiterite as well as quartz and over a range of grain sizes both coarser and finer than the nominal mesh of separation.

Performance Analyses of Spirals

If spirals are used their performance should be analysed to determine the percentage recovery of cassiterite in relation to grain size. The results could then be compared with performance analyses of jigs and shaking tables.

No adequate performance analyses of spirals in relation to grain size have yet been published within the writer's knowledge. In view of their widespread use in ore dressing the publication of such detailed data is overdue.

III.—FLOWSHEETS

As already noted, thanks to the excellent work carried out by the Research Division of the Mines Department in Malaya with the support of the Geological Survey, the Chinese gravel-pump mining industry is now in the process of changing over from sluice boxes to recovery plants incorporating screens, hydrocyclones and jigs. It has proved profitable to do so, but flowsheets have not yet been standardized or designed to the best advantage, the amount of fine cassiterite present not being accurately known. The theory of serial gravity concentration put forward by Harris[2] is not everywhere accepted, while the valuation of samples by means of chemical assays has also not always carried conviction. That is a big argument in favour of the physical methods of sample valuation tested in Nigeria and advocated here. Cassiterite which has been separated from a sample by gravity concentrate and has been seen and counted as individual grains under the microscope is more readily accepted as potentially recoverable by improvements in flowsheets.

In Nigeria the European gravel-pump mining industry is also in the process of changing over from sluice boxes to jig plants, but there are still big differences in the flowsheets used by the various companies. This is possibly due to insufficient fundamental information on the performance of hydrocyclones and jigs as much as to differences in the amounts and size analyses of the cassiterite and columbite in the ground and the amount of clay being slimed. The introduction of the author's methods of sample valuation is beginning to achieve realistic correlations between bore valuations and plant-scale recovery and flowsheets of jig plants are now being designed to suit individual deposits.

While the Malayan dredging industry continues to lack reliable information about the amount of fine cassiterite in the ground, its range of grain size and details of cassiterite recovery, it cannot reliably decide whether dredge flowsheets can be improved without overstepping the economic limit of recovery, or whether the capital expenditure involved would be amortized with known reserves. Check boring and the analysis of samples by the methods described here would determine the amount of fine cassiterite in the ground and its size range as well as the percentage of clay present. Experiments with hydrocyclones and jigs on dredges and the analysis of samples by the same methods could be carried out to define the conditions of feed and operation under which more fine cassiterite of any particular grain size down to 300 mesh could best be recovered, in spite of the amount of slime yielded by the disintegration of the clay. Consideration might then be given to the economics of incorporating some of the following additions into dredge flowsheets, according to the nature of the ground to be worked, the size range of its cassiterite content and the accompanying problems of design engineering:

(1) More high-pressure water in the trommel for the dual purpose of disintegrating more clay and diluting the pulp to give more effective separation in subsequent cycloning.

(2) Larger holes in the screen to avoid the discharge of appreciable

amounts of slime from the tail end of the trommel. This could be combined with research on the results achieved by jigging with long strokes at slow speeds without ragging, as is the practice on one dredge in Malaya.

(3) Dilution of the trommel undersize with low-pressure water and/or by using hydraulic elevators to feed the pulp to the cyclones. (The second method has been tried by one company in Malaya.)

(4) Primary cycloning of the dilute pulp in large low-pressure cyclones or higher pressure cyclones if separation in the latter is found to be more effective and not too expensive in terms of power and maintenance.

(5) Re-cycloning primary cyclone overflow if it still contains an appreciable amount of fine cassiterite particularly during surges of sand and possibly the use of smaller cyclones and higher pressures for this purpose.

(6) Dilution and re-cycloning of the primary cyclone underflow to remove more slime. (One company in Malaya has experimented with the injection of water under pressure to reduce the amount of slime in the primary cyclone underflow.)

(7) Dilution of the pulp to the primary jigs if it is dense or as an alternative to (6).

(8) Replacement of all Harz jigs by modern jigs to give greater concentrating area.

(9) Introduction of the method of primary jigging recommended by Harris[2] in which the tailing from only two cells in series is screened and the screen undersize dewatered by cycloning before passing to two more cells in series. (Screening may not be necessary. Furthermore, as the pulp will have been diluted with hutch water, cycloning at this point could be an alternative to (6).)

(10) Bringing clean water from ashore on to the dredge for use as hutch water and possibly also for (7) and (6).

(11) Pumping all slimes ashore and using large settling areas and chemical treatment to provide clean water for (10). (One company in Malaya has achieved low-cost clarification with alum and lime and another company is operating a dredge from which all the slime and sand tailing is pumped ashore.)

(12) Three-stage concentration in jigs combined with one or more of the following closed circuits: (*a*) the last two hutches of the four-cell cleaner jigs to the head of the same jigs; (*b*) the cleaner jig tailing to the head of the secondary jigs; and (*c*) the secondary jig tailing to the head of the primary jigs.

(13) Use of a shaking table in each of the closed circuits (12) (*a*), in order to remove fine cassiterite as a head and also semi-heavy minerals as a middling, both of which might otherwise build up in closed circuit with continuous loss of fine cassiterite. (In such closed circuits temporary malfunctioning of the tables could be tolerated.)

(14) Screening the primary jig tails, cycloning the screen undersize and treating the cyclone underflow on spirals or shaking tables or jigs operated at high speeds and correspondingly short strokes. (This system has the advantage that it comes *after* the main recovery in jigs and can be brought into use only when the size range of the cassiterite in the ground being worked makes it profitable to do so.)

(15) The use of two lines of two-cell primary jigs, as on one dredge in Malaya, instead of the more usual one line of four-cell jigs on each side of the dredge. (If the undersize from the trommel is to be delivered by hydraulic elevators or pumps to a hydrocyclone at the head of each primary jig, the distribution for this system might be easier to design.)

(16) Arrangement of the primary jigs in two tiers, one above the other. (The use of pumps with or without cyclones instead of the usual gravity feed flow to the primary jigs would allow the headroom available for this purpose to be increased. The feed to the primary jigs would then be spread over approximately twice the jigging area which should increase the recovery of very fine cassiterite.)

It is apparent that the magnitude of the check boring and research programme required calls for systematic planning and execution with adequate staff, sampling facilities and laboratory equipment. The ground would have to be exceptionally rich in very fine cassiterite to pay for the fullest possible elaboration of the flowsheet along the lines indicated, but at least some improvements are likely to be profitable in many areas. A major difficulty, which is not so pronounced in the case of gravel pumping, arises from the extreme variation and high maximum of the percentage of sand in the trommel undersize to be cycloned before jigging. Several large groups of companies in Malaya have already carried out experiments along some of the lines indicated but as the work is still in progress no results have yet been published. Furthermore it is doubtful whether the methods of sample analysis being employed are adequate in all cases. Certainly the occurrence of fine cassiterite in the ground has not yet been sufficiently investigated because of the previous lack of a suitable fast method of sample valuation and analysis.

TABLE VI.—*Dredge Outputs, Malaya: Screen Analyses of Cassiterite Contents*

B.S. sieve no.	1	2	3	4	5	6	7
	%	%	%	%	%	%	%
+ 22 . .	17·9	0·1	0·6	0·1	1·8	1·0	1·5
22– 52 . .	34·9	6·9	6·0	4·8	10·1	7·3	2·8
52–100 . .	24·1	58·6	49·1	67·4	69·1	69·2	33·8
100–120 . .	16·5	25·2	35·7	22·5	11·8	19·1	49·8
120–150 . .	5·3	7·4	6·2	3·9	6·3	3·1	10·5
150–200 . .	1·1	1·7	2·0	1·0	0·7	0·3	1·5
–200 . .	0·2	0·1	0·4	0·3	0·2	Tr	0·1
	100·0	100·0	100·0	100·0	100·0	100·0	100·0

Table VI records a selection of typical screen analyses of the cassiterite recovered by the normal types of jig plants without hydrocyclones on bucket dredges in Malaya. In each case the amounts fall off very noticeably below 120 mesh, but the relative proportions in which this must be attributed to the grain size distribution of cassiterite in the ground and to poor percentage recoveries achieved in the fine grain size range remains unknown. By using the methods of sample valuation described the dredging industry could now find out how much fine cassiterite there actually is in the ground, how much is being lost with the present flowsheets and what additional recovery could be made by various arrangements for cycloning the feeds to the jigs and for treating jig tailing in other types of concentrators.

Organization of Research

It is of interest to note that although the tin mining industry supports a very active international research organization, mainly concerned with developing new uses for tin metal, there is no counterpart organization carrying out research on the recovery of the original cassiterite and associated saleable and potentially saleable minerals. The latter aspect is probably as important to the future prosperity of the industry as the former, but the problems of recovering cassiterite and associated minerals are not so readily amenable to centralized research. A central liaison office could, however, help to co-ordinate research programmes of individual companies so minimizing unnecessary costly duplication, and could also receive, catalogue and distribute the reported results of research.

In an adequate programme of research the economic geology of the alluvial deposits needs to be studied in great detail, but, owing to the element of competition which exists in the acquisition of land, geological investigation by the tin mining industry is much less amenable to centralized advice than the investigation of plant-scale recovery. Not until reserves have been adequately defined in all their economic aspects can decisions regarding the conversion of recovery plants on dredges to new flowsheets be safely made.

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APPENDIX

SEPARATION OF HEAVY MINERALS WITH MOLTEN LEAD CHLORIDE (sp. gr. 5·0, m.p. 501° C)

Laboratory Procedure

(1) Take not more than 1 g of the mineral mixture.

(2) Weigh out sufficient lead chloride to two-thirds fill a ½-in Pyrex test-tube after compacting the charge by gently bumping the bottom of the test-tube on the bench.

(3) Place about ½ in. of this lead chloride in the bottom of the test-tube.

(4) Mix the mineral sample thoroughly with the remainder of the lead chloride and pour the mixture into the test-tube bumping it to compact the charge.

(5) Fuse over a Bunsen burner, heating the charge over its full length and on two sides to ensure uniform melting (the test-tube holder should preferably be made from hard asbestos board).

(6) Shake the test-tube gently to release any heavy mineral in the float portion and keep fused for about 5 min.

(7) With a sharp grease pencil mark the middle of the clear zone between float and sink (the total length of the melt will be about 1½ in.).

(8) Allow to cool.

(9) Use a diamond pencil to score right round the test-tube at the marked level.

(10) Scrape the inside of the test-tube with a broken hack-saw blade to release any particles of float mineral fused to the sides of the test-tube above the melt.

(11) Wrap the test-tube in a cloth and break in two gently with a pestle.

(12) Separate the sink and the float portions of the melt.

(13) Break and grind each portion gently in a porcelain mortar to reduce the fused lead chloride to a fine powder without appreciable breakage of mineral grains. (This differential grinding is the most important feature of the technique, otherwise it takes too long to dissolve the lead chloride.)

(14) Place the ground sink and float portions of the melt in separate beakers, add water and bring to the boil; decant the water and repeat until all the lead chloride has been removed partly in suspension and partly in solution (time required about 1 hour).

(15) Dry the sink and the float mineral fractions.

Note.—Tests in Nigeria have given satisfactory separations down to 170 B.S. mesh without centrifuging.

European man-day refer to prospecting surveys with which we have been associated. They refer to the operational cost of drainage reconnaissance in areas where cross-country transport by Land-Rover is practicable, where a sample density of about three samples per sq. mile is permissible and where the company is based in the territory under investigation. The costs do not include any extensive follow-up work on the anomalies detected by the reconnaissance survey. The samples are collected by Europeans each operating from a Land-Rover; the ratio of Africans to Europeans was 2 or 3 to 1 both in the field and laboratory.

In reply to Dr. Shaw's analytical query, carbon tetrachloride is not suitable for the cold-extraction copper test because, being heavier than the aqueous phase, it would be extremely difficult to estimate the colour of the organic phase in the presence of the sediment sample. Final modification of the cobalt test has taken longer than expected, but it is hoped that the details requested by Dr. Hosking may be available in the near future.

It is impossible in a short space to reply in any detail to the questions concerning more general applications of geochemical methods. In outline, therefore, prospecting desert terrain, mentioned by Dr. Mackay, poses a whole series of specific problems to which, in most cases, the methods outlined in our paper would not be directly applicable. We expect, however, to be engaged on a study of desert problems in the coming year. Professor Williams commented on the possible role of geochemical surveys on a regional or provincial basis. The Research Centre at Imperial College has been engaged on a programme of research in this field since 1955 and the first geochemical maps covering an area of more than 2000 sq. miles in Northern Rhodesia are currently being prepared. There is no doubt that maps of this nature provide information of fundamental as well as practical value. Research is also being undertaken on the primary metal content of rocks associated with ore, complementary to similar problems which were attracting increasing attention in North America. Although these studies are still strictly experimental they hold great promise of practical application in the future.

In reply to Mr. Wilson, the conventional way in which to carry out a comprehensive survey is to include analytical and sampling coverage for the full suite of metals or their associated pathfinder elements. This principle is being used in the regional geochemical approach mentioned in the preceding paragraph.

With regard to Dr. Shaw's remarks concerning the relative roles of geophysical and geochemical methods, it will be appreciated that geochemical drainage anomalies serve only as a rapid primary method of delineating promising catchment areas wherein to concentrate further work. Before drilling can be contemplated it is necessary always to follow up the anomalies by more detailed surveys. In the Copperbelt this can most conveniently be done by soil sampling, based on such geological information as may be avilable, followed by pitting as a prelude to drilling. In certain other areas, geophysical methods may well be the best means of further reducing the size of the target. The optimum integration of all available methods is, of course, one of the prime problems in mineral exploration.

In conclusion, we should like to stress again how much we appreciate the generous assistance we have received from many quarters, to which reference was made in the kind remarks of Dr. Dixey, Mr. Thomson and Professor Williams.

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Recovery of Fine Alluvial Cassiterite: Correlation of Bore Valuations with Plant-Scale Recovery

F. A. WILLIAMS, B.Sc., MEMBER

Author's reply to discussion on paper published in November, 1960* (Transactions *vol. 70, 1960–61*), *pp. 49–69*

Mr. F. A. Williams: I wish to thank Mr. J. L. Farrington for introducing the paper to the Nigerian Section, also all those who contributed to the discussion at the London and Nigerian meetings, or in writing.

In replying to the discussion I have used the same headings as in the original paper.

I. *Valuation and analysis of alluvial bore samples*

I am in agreement with Mr. Michell's comment that it would have added considerably to the value of the paper to have had some indication of the relative amount recovered from the underflow of the cyclone, and the amount of additional recovery made by retreatment of the tailing from field panning. There had not been sufficient time before my retirement

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from Nigeria to test any possible modications of the method to meet varying degrees of emphasis on speed, cost, reliability and detail. Early in the paper, attention was drawn to the fact that all practical methods of valuing alluvial bore and plant samples have to be compromises between these four conflicting aspects. In this connexion the contribution from Mr. J. K. Broadhurst supplies some interesting information relating to Malayan conditions. The criticisms in parts of the contributions from Dr. E. Cohen, Mr. M. P. Jones and Mr. P. M. Sheahan do not make allowance for the practical need for compromise which often entails acceptance of standards of accuracy appreciably lower than the optimum otherwise obtainable.

Previously-accepted methods of sample valuation lost appreciable quantities of cassiterite of even fairly coarse particle sizes and, as shown by Table III (p. 58), such losses increased rapidly with decreasing particle size until, approaching 300 mesh, recovery became negligible. In marked contrast, the method described gives practically complete recovery of all sizes down to 300 mesh. Thereafter, recovery presumably diminishes with size although, as mentioned in the paper, accurate valuation can be extended to finer sizes than 300 mesh by splitting out a small portion of the total table tailing for concentration on a Haultain Superpanner. This method thus represents a very substantial improvement.

As Mr. Arthur D. Hughes pointed out, there is an economic limit to plant-scale recovery. It is doubtful whether any plant would recover an appreciable amount of cassiterite finer than the finest recoverable on a Haultain Superpanner and indeed very little finer than that recovered by putting the deslimed unsized sample three times over a half-size Holman sand table as in my usual procedure. Furthermore the results in Table IV (p. 59) are in agreement with the basic geological expectations that alluvial deposits do not in fact usually contain much cassiterite finer than about 300 mesh. The appreciable loss of coarse cassiterite shown by Table III, which Mr. Sheahan had also noted in Malaya, might conveniently be checked by coarse sizing of the sands and using Pleitz jigs, as in the valuation of diamond-bearing gravels.

Mr. Sheahan found that heavy minerals were concentrating in the end housing of their Monopump. The suppliers advise that this type of pump can be quickly dismantled for cleaning by fitting four 'Collet' nuts in the centre flange of the casing thus allowing the whole casing to be removed easily from the rotor. The nuts referred to are split vertically and have bevelled bases which fit into countersunk bolt holes. All that is required to remove the nuts is a turn with a spanner and the nuts fall in half. To replace them, each nut is placed over the thread of the stud and tightened. Mr. Sheahan's note on the use of a conical-bottomed container and a plastic pipe instead of a Monopump appears to be an interesting practical idea provided the sample of slime can be kept adequately stirred to ensure efficient operation of the cyclone. Mr. Sheahan's suggestion that my tabling procedure could have been greatly assisted by screening the feed into three sizes ignores a practical consideration. Fine wet screening is a very slow process, so it was much quicker to table the deslimed but unsized sample three times as a safeguard against tails losses.

Dr. Cohen's insistence on chemical assaying which, to use his own words 'would also account for all the locked tin' is out of place. The alluvial tin mining industry does *not* want that account since it is geared economically to the low-cost recovery of free cassiterite. The grinding of alluvial sand tailing to release locked cassiterite is not at present a practical proposition. Even if, at some future date, the price of tin were to make such grinding profitable it would still be necessary to use a physical method to value the samples in terms of free cassiterite recoverable at relatively low cost, before using chemical assaying to indicate the amount of locked cassiterite. In his contribution Mr. F. A. Garner also ignores this practical economic aspect. The alluvial mining industry needs mineral processing engineers who can and will analyse samples in terms of free minerals. In this connexion I am pleased to note Mr. F. B. Michell's support for more mineralogical analyses. Mr. Broadhurst concluded his contribution with the statement 'it would appear that the scale of supposed under-valuation and plant losses has tended to be exaggerated in some recent papers'. Such exaggeration in Malaya may have arisen from the use of chemical assaying where physical assaying would have been more appropriate.

Mr. Farrington drew attention to the fact that the grade of ore in the alluvial tin fields of the world is only 0·015 to 0·05 per cent. The figures submitted by Mr. Michell on page 304, relating to the limits of accuracy of the direct chemical assaying of alluvial samples, are timely. They highlight the inadequacy of the chemical assaying of such samples favoured by Dr. Cohen, Mr. Sheahan and Mr. Jones. I am in agreement, however, with Mr. Michell that there is much less objection, if any, to chemical assaying of the concentrates separated from the alluvial samples by gravity means. Any locked cassiterite or tin in other minerals recorded by such assays would at least relate to particles heavy enough to report in the plant-scale jig concentrates sent to the mill for further processing. Particular interest attaches to the instrumentation for high-speed chemical assaying referred to by Dr. Cohen and Mr. Sheahan. Each sample of alluvial concentrate should of course be screen-sized into about ten fractions and each of these should be assayed for tin. A size analysis of the cassiterite could then be calculated. Adequate information on the size range of the cassiterite is essential both for research work and for routine mineral accounting and control.

Mr. Chaston's statistical analysis of the physical assaying problem is interesting and actually lends support to my use of leaching and magnetic separation to produce a high-grade concentrate before grain counting. The table would have been of much more practical use and significance if it had been extended upward to include the range 60 to 95 per cent true assay in which range the number of grains to be counted in order to achieve the required accuracy becomes quite small.

Mr. Arthur D. Hughes mentioned the need to apply recovery factors to bore valuations in order to arrive at estimates for plant-scale recovery. For this purpose the method of sample valuation described in my paper, which in Nigeria is being applied to both bore and plant samples, is particularly suitable. When valuing reserves it provides a size analysis of the cassiterite down to 300 mesh and to even finer sizes if required. Applied to the valuation of head and tailing samples from recovery plants it provides figures showing the percentage recovery of each particle size. From the original bore valuations recoverable values can be calculated for different types of recovery plant.

Mr. Hughes also mentions applying low recovery factors to bores in some properties in Malaya containing large amounts of clay or where the bedrock is irregular limestone. The economic geology of alluvial clays has not yet been adequately studied in Malaya. To allow for limestone bedrock there the accepted but rather arbitrary practice is to delete the last one or more samples from the bores when calculating recoverable reserves. An outstanding problem of great economic importance in Malaya is how best to excavate the ground below dredging depth and between limestone pinnacles instead of burying it beneath low-grade tailings as at present. Co-ordination of bore values and plant-scale recovery in Malaya involves more than consideration of recovery in relation to particle size. The difficulties involved, although mainly in the realm of engineering, cannot be divorced from study of the geological features of the alluvial deposits.

The company with which I was formerly connected in Nigeria employed a team of up to five geologists. For a number of years it has been actively engaged not only in geological investigations and the introduction of new methods for the valuation of reserves but also in research on plant recovery. On visiting Malaya in 1960 I was surprised to find that none of the European alluvial tin mining companies or groups of companies maintained similar research departments staffed with either geologists or mineral processing engineers and supported by suitably equipped laboratories, although the Malayan industry dwarfs that of Nigeria. Before the war I spent many years in Malaya check-valuing alluvial tin deposits and sampling recovery plants and I know from examples investigated how very inefficient have been some alluvial prospecting and mining operations undertaken without adequate study of the geology of the deposits.

The contributions to the discussion from Mr. C. M. Richardson and Mr. P. J. H. Rich serve to draw attention to one of the principal problems in the economics of valuing alluvial deposits. As such deposits can often be worked at very low cost, average values tend to be low and it is seldom economically practicable to bore a small block close enough to give a reliable average valuation. Only in the case of large blocks, as for instance the total reserves for the life of a bucket dredge, is the calculated average value reasonably reliable with the range of bore spacings normally used. This problem is particularly acute in Nigeria, where narrow leads are worked with gravel pumping plants which treat in a whole year less than a large Malayan tin dredge treats in a single month. Mr. Rich recorded how inaccurate monthly or even yearly estimates of ground values can be in the case of Malayan bucket-dredging operations. Mr. Richardson's action in submitting his sampling results for analysis by an expert statistician not only exemplifies the correct approach, but may also be a pointer to the future. It would certainly produce some thought-provoking results if average values based on bore or pit samples were always accompanied by a concise expression of their probable error arising from the number of samples in relation to the range of variation. We can perhaps look forward to the day when the large European alluvial tin mining companies and groups employ statisticians as well as geologists and mineral processing engineers for a concerted attack on the problems of mineral accounting and control. If necessary, the money could be found by reducing the expenditure of conventional monetary accounting in order to bring the two into a more rational balance of reliability. It seems rather pointless to spend a lot of time and money on budgetary accounting to predict the monthly expenditure on an operation to within say, $\pm$ 5 per cent or less when, as the figures given by Mr. Rich indicate, the prediction of the monthly output may be wrong by as much as $\pm$ 40 per cent. Regarding the intensity of boring required to justify the use of my method of sample valuation, the probable error of the averaged value of a number of bore values varies inversely as the square root of the number, i.e. to halve the probable error it is necessary to have four times as many bores, and so on. Obviously the economic limit to improving the reliability of valuations is soon reached in this direction. There are two practical ways of tackling this problem and preferably both of them should be used:

(1) cycloning the slimes and retaining both the underflow and the pan tailings from all bore samples but *bulking* these, and if necessary splitting the bulk down, before applying the suggested method for retrieving fine cassiterite lost. Such bulking of samples was recommended by Professor B. W. Holman. Separate bulking of overburden and wash samples is often practicable. It is an accepted practice in Malaya to bulk separately bore concentrates and amang representing large blocks and to submit these bulked samples for screen analysis and chemical assay. The chemical assays are then used to adjust the average value to 72 per cent Sn. Concentrates from bulked samples of slime and sand tailing from the same group of bores obtained by my method could be submitted for screen analysis and chemical assay. An additional adjustment could then be applied to the average value, and the combined screen analysis would be truly representative;

(2) making use of my method for the valuation of large adequately representative samples taken on dredges to determine the percentage recovery of each grain size. In the latter connexion it will be informative to compare the hard ore and alluvial mining industries. The cut-off grade in hard ore mining is usually more than 20s./ton while in dredging it is usually less than 2s./cu. yd. In other words their cost structures differ by a factor of more than 10. The hard ore mining industries can afford to (and usually do) sample the orebodies closely enough for reliable guidance in mining operations. Nevertheless the average indicated value of the ore mined *in situ* is not usually accepted as the head value of the mill feed, if for no other reason than the undeterminable amount of dilution. Instead, metallurgical accounting and control of mill operation is based on direct sampling in the mill. Similarly, and particularly under the greater compulsion of an allowable expenditure of less than $\frac{1}{10}$ per unit weight or volume in valuing the ore *in situ*, the alluvial mining industry should implement mineral accounting and control of recovery plants by direct sampling within these plants. It is therefore

particularly significant that in Nigeria my method is being used for the valuation of both bore and plant samples.

II. *Investigation of Plant Scale Recovery*

Since reading Mr. Richardson's contribution, I have visited Ghana and Sierra Leone to study, *inter alia*, the sampling problem described against its geological background. Mr. Richardson concluded that, as in the alluvial tin mining industry, the valuation of the ground *in situ* could not be used to determine the head value to the plant with sufficient accuracy. There is need for systematic sampling of recovery plants to provide reliable information on recovery. This applies both to overall recovery and to recoveries in the various unit processes employed. Mr. L. O'N. Thomson's reference to the scope for using differential-conductivity meters is interesting but capital cost could rule them out. It is not essential to sample the head, output, and tails, as samples of any two will suffice. In both the alluvial tin and alluvial diamond mining industries the output is accurately weighed and a size analysis is normally made. The choice therefore lies between head and tails sampling. Unfortunately it is rare to find that recovery plants either on dredges or ashore, are provided with built-in facilities for such sampling. Provision for either head or tails sampling should always be incorporated in the design engineering of recovery plants. Without such provision technical progress will be greatly retarded.

An established practice in some alluvial diamond recovery plants is to take a continuous automatic cut of the pan tailing and pass it to the more effective sample jigs to check losses. Similarly on a tin dredge the total jig tailing discharge could be continuously sampled and the split continuously concentrated on shaking tables to determine the loss of fine cassiterite as in my method used for separate samples. It should be possible to install such sample valuation tables on adjustable platforms to allow for changes in the level of the deck. On the score of cost it may be necessary to compromise at least initially and carry out such an investigation only periodically on most dredges although one might be selected for fundamental research over a longer period.

It is thus apparent that the answer to the problem presented in the contributions to the discussion by Mr. Richardson and Mr. Rich will be found in an adequate study of the geology of the alluvial deposits in order to make the best possible use of the necessarily rather limited amount of sampling *in situ* coupled with better provision for plant sampling and the analysis of samples.

Mr. Michell twice refers to the use of spirals for recovering fine cassiterite. In view of the current experimental use of spirals in association with jigs there is urgent need for the publication of quantitative data on the recovery of cassiterite in spirals in relation to particle size for comparison with what can be achieved in jigs. The data on spiral performance in the paper* by H. Dalton-Brown were not encouraging.

*Recovery of cassiterite at the Sungei Besi mines, Selangor, Malaya. *Trans. Instn Min. Metall., Lond.*, **70**, 1960–61 (*Bull. Instn Min. Metall., Lond.*, no. 648, Nov. 1960), 33–48.

It was of particular interest to note Mr. Sheahan's statement that 'new equipment currently being tested by the Research Division is expected to provide simple and economic methods of extending the range of recovery to sizes finer than 300 mesh BS'. It is to be hoped that details will eventually be published by the Institution so that they can be discussed.

III. *Flowsheets*

Mr. Hughes and Mr. Cleaveland relate how jigs were first introduced into the Malayan alluvial tin mining industry by an economic geologist, Mr. J. F. Newson, who had studied the operation of Harz jigs in the U.S.A. In Malaya jigs soon became almost universal on dredges and now the Chinese gravel pump miners are changing over from sluice boxes to jigs. In Nigeria, although there are now a fair number of jig plants in the alluvial gravel pumping industry, far too many sluice boxes survive and continue to lose appreciable amounts of fine cassiterite and, particularly, columbite. The fact that considerable tails losses were occurring from the sluice boxes was always apparent from the recoveries made by tributers reworking the tails. Such losses can now be accurately assessed by the new methods of sample valuation. The results should hasten the replacement of the remaining sluice boxes by jig plants.

Mr. Cleaveland mentions three innovations by Pacific Tin in actual use on their dredges: (1) jigging without ragging; (2) a flowsheet designed to make dewatering of the feed unnecessary; and (3) pumping all the sand and slime tailing ashore. Dispensing with ragging represents an appreciable saving. Mr. Cleaveland's claim to achieve a better recovery on the dredge by this means could be checked by my sample valuation method.

The company's approach to designing a flowsheet to make dewatering of the feed unnecessary was to equip one of their dredges with a double line of jigs of 2 cells in series on each side of the dredge instead of the usual one line of 4 cells in series on each side. I suggest for engineering consideration that to provide additional jigging area two tiers of jigs, either 2-cell or 4-cell, could be installed on dredges. The feed could gravitate to the lower tier but would have to be pumped to the upper tier. The 'splitter distributor' mentioned by Mr. H. Hocking might be incorporated to advantage in such a flowsheet. The writer duly noted that Mr. Hocking was also an advocate of increasing the jigging area. The use of cyclones would be optional. Pumping all sand and slime ashore would allow for installing the bottom tier of jigs at a lower level thus increasing headroom and improving stability. As pointed out by Mr. Cleaveland, increasing the limit of feed size to well over an inch would help to increase the permissible throughput of the trommel as would the advantageous use of more water. Mr. Cleaveland's idea of designing a flowsheet to make dewatering unnecessary might also be considered for use in connexion with gravel pumping or, to carry the idea to its logical conclusion, even with hydraulic elevating.

It is the third innovation mentioned by Mr. Cleaveland—pumping jig tailing ashore—which interests me particularly. The immediate justification of the cost is the better recovery of fine cassiterite in the jigs when cleaner hutch water can be obtained from the paddock. It might, however, prove to be the starting point from which a solution could be found to the major alluvial mining problem in Malaya already mentioned—how to mine profitably ground too deep for dredging and between limestone pinnacles concurrently with dredging.

Having pumped all the jig tailing ashore through a flexible pipe-line from the moving dredge it should be relatively easy to use one or more auxiliary pumps to move it any required distance as in reclamation work. The paddock could then be kept open for any desired distance behind the dredge. The discharge of the stone chute could then be transported by barges to the far end of this elongated paddock as in harbour work. Deep ground below the digging depth of the bucket dredge could then be worked with a grab dredge equipped with its own jig plant. For working deep ground such an auxiliary should represent less capital outlay than would be required for a bucket dredge of a size necessary to reach the required depth, and in some places in Malaya the ground is out of reach of the deepest-digging dredge at present existing. In such open water a grab dredge might prove to be suitable for excavating at least some of the ground between limestone pinnacles, particularly if special small grabs were used. Alternatively, or in addition, a still smaller jet dredge also equipped with jigs might be tried for cleaning irregular limestone bottom.

Improvements in the flowsheets of dredge recovery plants designed to increase the recovery of fine cassiterite will also increase the recovery of the associated heavy and semi-heavy minerals. This will place an added burden on the plants used for separating the cassiterite from the jig concentrates. When visiting Malaya I frequently noticed that the tailing from such dressing plants was finally scavenged by hand sluicing. This indicates room for improvement in the mechanical processing. Mr. Hocking mentioned that four modernized gravity and electrostatic-magnetic plants have been installed during the last seven years to treat dredge concentrates. In Nigeria the tendency during that period has been to reduce the number of wet shaking tables and to make more use of all-dry sequences of operations including dry screening in cascade screens fitted with nylon mesh before any minerals separation, followed by various sequences of high-tension separators, induced-roll and belt-type magnetic separators and pneumatic tables. These types of flowsheet might be worth considering for use in Malayan conditions.

Finally I would point out that technical progress cannot be assessed in an economic vacuum. In their own interests and in the interest of the alluvial tin mining industry as a whole, those companies who already publish their cost per cubic yard could well consider releasing break-downs of these costs for incorporation in technical papers. This applies particularly to dredging operations in Malaya. As Mr. Michell said: '. . . in many dredges the design did not permit major alteration to flow-sheets without virtually redesigning the whole structure.' Such redesigning could well be economically justified, but the potentialities cannot be studied only in terms of greater throughput and increased recovery ignoring detailed consideration of capital and operating costs.

Experimental Study of Adsorption and Desorption of Xanthate by Sphalerite

TOSHIAKI YONEZAWA

Author's reply to discussion on paper published in March, 1961* (Transactions *vol. 70*) *pp. 329–353*

Mr. T. Yonezawa : Concerning Dr. A. P. Prosser's first comment, it is shown that a copper-activated sphalerite surface adsorbs more xanthate than an unactivated surface (p. 332). Hence, I believe that saturation adsorption of xanthate on the surface of sphalerite should define the degree of copper ion activation, or surface coverage, of the sphalerite. This experiment was, therefore, designed to ascertain the saturation adsorption of xanthate on sphalerite surfaces which had been pre-coated with copper to different extents by using solutions of different initial copper concentration (column 1 of Table VII, p. 344). In each case xanthate adsorption was determined from solutions of fairly high xanthate concentration in order to ensure saturation adsorption, i.e. 100 mg/l. The pH value used was not regulated, but was about 6·5, the case of xanthate adsorbed to the maximum possible extent on surfaces pre-coated with copper to different extents being discussed. It is true that the fact that the ratio of copper adsorbed to xanthate adsorbed is 2·3 is significant only in the case of approximately 100 per cent surface coverage, as stated by Dr. Prosser. I also agree with his statements published in the latter half of the last paragraph on page 550.

I am convinced that on a copper-coated sphalerite surface one molecule of xanthate always reacts with one copper atom. In other words, the product of the reaction is principally cuprous xanthate. The reasons are as follows:

First, based on the fact that xanthate is a strong reducing agent, as was also pointed out by Dr. Burkin, I inferred that it must reduce cupric to cuprous ion, not only in the solution, but on the surface of sphalerite as well. In fact, R. Sato investigated the action of both xanthate and dixanthogen on copper-activated sphalerite cleavage faces by means of electron diffraction†. In each case, he deduced the reaction product to be cuprous xanthate; nevertheless the mechanism of the chemical reaction could not be clarified in the case of dixanthogen.

Secondly, judging from the fact that when there is low surface coverage of Cu ion, as shown in Table VII, the molar ratio of uptake Cu/EtX can be one or less, together with the fact that unactivated sphalerite adsorbs a small amount of xanthate (i.e. $0 \cdot 54 \times 10^{-7} - 0 \cdot 65 \times 10^{-7}$ mol/g) as shown in Table VII, I consider it is rational to infer that one molecule of xanthate may be adsorbed on the mineral surface stoichiometrically with one copper atom in the form of cuprous ethyl xanthate (in this case

**Trans.*, vol. 70, pp. 545–553.

†Reference 5 on p. 352 of my paper.