'military' lines, the detail of each 'campaign' was planned and co-ordinated in the headquarters at Montreal, a logistics department ensured the regular flow of supplies, various echelons of responsibility controlled the operation in the field where each party and each man had a clearly defined job and the means of doing it. The efficiency of the organization is evidenced by the fact that 2700 tons of ore were indicated by each foot of drilling.

That there was some 'soul-searching' during this period can be seen from the fact that, in 1948, the original objective of 300 million tons was increased to 400 million tons. By 1950, however, the whole operation had been brought to a point where not only could further capital be brought in but a large loan obtained from insurance companies.

As soon as this final step in financing had been reached it was realized that the greatest overall economy which could be made on the project was one of time, even if it meant some overspending and a few imperfections on matters of detail.

A nucleus of men from the parent organization took charge of the operation as a separate company and an air transport company was formed to ensure the steady flow of equipment. Each man and each job was dovetailed with the rest and in some cases even the difficulties of climate were turned to advantage. Work was cut down to essentials and the remainder was left over for later financing from sale of ore; over 30 million tons have already been shipped from a mine which is still in the course of equipment.

The Iron Ore Company of Canada have successfully launched a difficult project in unusual circumstances and in record time, and in so doing have opened up an ore field with large reserves of direct shipping ores and great reserves of concentrating ores, and have paved the way for other companies.

There is much to learn from the way this project was handled, and an authoritative and detailed publication on the subject by the Company itself should form an interesting addition to any library concerned with developments in distant and difficult countries.

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Performance Analyses of Screens, Hydrocyclones, Jigs and Tables used in Recovering Heavy Accessory Minerals from an Intensely Decomposed Granite on the Jos Plateau, Nigeria*

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622.7.011:622.7–346.5
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SYNOPSIS

Methods of quantitative fragmental petrography originally devised for the physical analysis and valuation of various intensely decomposed granites were later used for investigating the plant-scale recovery of the resistant heavy accessory minerals cassiterite, columbite, zircon and xenotime from one of these granites. Performance analyses given show the recovery efficiencies of hydrocyclones, jigs and tables for recovering minerals in the specific gravity range 7·0 to 4·5 from 16 mesh down to sub-sieve sizes. The four flowsheets tested are described and future development discussed.

A simplified version of this plant was subsequently used for recovering heavy minerals in alluvial mining operations.

INTRODUCTION

THIS PAPER IS A SEQUEL to one published earlier[1] in which methods devised for valuing intensely decomposed granites containing columbite and other primary accessory minerals were described. Several areas rich enough to constitute orebodies were located and a plant was erected to treat one of them. In order to take advantage of the current demand and high price for columbite time could not be spared for experimentation with a pilot plant to investigate the problems of treating this unusual ore before proceeding to full-scale production, so that the plant erected had to serve the dual purpose of a commercial production installation and a pilot plant, with emphasis on the first function. Moreover, it had to be improvised from equipment and material available on the property.

Mr. R. H. W. Punnett‡ was responsible for the design engineering, the writer organizing the sampling programme and devising the system of sample analysis to provide data on which to base improvements to the flowsheet. Accordingly this paper records the sampling results but is not concerned with engineering detail.

Improvements had to be built into the plant with a minimum of stoppages and under the conditions prevailing it was not possible to make

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[1]WILLIAMS, F. A. The identification and valuation of decomposed columbite-bearing granites of the Jos–Bukuru Younger Granite Complex, Nigeria. *Trans. Instn Min. Metall., Lond.*, **65**, 1955–56, 169.

‡Then Technical Superintendent, A.O. Nigeria, Ltd.

the investigation as comprehensive as would have been appropriate in a true pilot plant; but as far as possible the performance analyses of unit processes were fundamental in conception. The results may therefore be of use outside the context of treating this particular ore. They are, in fact, already being used for guidance in the design engineering of jig plants to improve the recovery of columbite from alluvial leads containing cassiterite and subordinate columbite.

The investigation described presents an example of the extension of quantitative fragmental petrography from geological mapping and the valuation of reserves to a study of ore-dressing problems.

ANALYSIS OF THE ORE

From the ore-dresser's point of view the important characteristics of the feed to the recovery plant are summarized by the data presented in Table I. The valuations and screen analyses of the minerals include only 'free' mineral, the term 'free' being used here to denote that part of the total mineral content released by simple disintegration in water without crushing or grinding. Well-known examples of ore for which only the free mineral is brought into account in valuation and recovered in treatment are alluvial gold and tin deposits and beach deposits containing ilmenite, zircon, monazite and other heavy minerals. This intensely decomposed granite can be valued and treated by alluvial type methods for free mineral thus defined.

After disintegration in water about 50 per cent of the ore becomes slime. Most of the residue is coarser than 52 mesh, but most of the columbite is finer than this and fairly evenly distributed down to the sub-sieve sizes. The size distributions of the other accessory heavy minerals—cassiterite, zircon, thorite and xenotime—are similar to that of the columbite.

The laboratory techniques employed in making such analyses of intensely decomposed granites have been described in the earlier paper. With appropriate modification they could be applied to any incoherent ore in which the valuable mineral is heavy and a substantial part of it 'free'.

METHOD OF MINING

The decomposed granite was mechanically excavated with D.W.21 scraper loaders and dumped at the plant. From this dump it was bulldozed into a sump excavated in fresh granite and disintegrated by a monitor before elevation to the grizzly by a gravel pump. Consideration is now being given to breaking down the ore *in situ* with a high-pressure monitor and pumping it to the plant, thus dispensing with both mechanical excavation and bulldozing into a sump.

SAMPLING PROCEDURE

The columbite, being a primary accessory mineral of the granite, is very evenly distributed from the aspect of sampling. When mining started head values were calculated from the original sectionally sampled development bores on a 200-ft by 200-ft grid. As recovery in the plant improved a more accurate valuation of the feed became necessary in order to give significant

TABLE I

Average screen and mineral analysis of decomposed granite mined

Division	%	Free columbite content	
		lb/ton	Distribution %
Coarse sand + 52 B.S. mesh	35	1·2	7
Fine sand − 52 B.S. mesh	15	35·5	93
Slime	50	—	—

Deslimed sands

B.S. sieve	%	Free columbite content	
		lb/ton	Distribution %
+ 16	23·2	—	—
+ 25	22·0	0·2	0·5
+ 52	22·3	3·9	7·2
+ 72	6·7	21·9	10·7
+ 100	4·9	32·7	12·7
+ 150	6·0	40·5	19·6
+ 170	2·5	45·7	8·9
+ 240	3·9	44·7	13·1
+ 300	3·3	43·1	11·2
− 300	5·2	36·7	16·1

Screen analyses of associated accessory minerals

B.S. sieve	Cassiterite	Xenotime	Thorite	Zircon
+ 25	2·2	4·6	9·2	12·6
+ 52	3·5	26·4	23·3	20·8
+ 100	26·8	25·3	37·9	37·0
+ 200	38·0	28·7	20·6	26·7
− 200	29·5	15·0	9·0	2·9

performance analyses. To this end the area marked out for each day's cut in the open pit was first bored on a 20-ft by 20-ft grid to about the depth to be excavated and a composite sample valued daily.

From determinations made near the surface, when the development of reserves by boring was started, the dry weight of a cubic yard of this intensely decomposed granite was found to average about 2400 lb and this was the standard adopted for bore valuation. Later investigations in the open pit proved that the figure varied appreciably, being most commonly between 2450 and 2700 lb. A relationship was established between the percentage residue after desliming and the density of the ore from which the volume : dry weight ratio to be used in valuing any sample could be easily determined. This procedure gave a more reliable valuation of the feed to the recovery plant.

In the absence of facilities for sampling the large volume of the combined tailing discharged from the whole plant, overall recovery was computed from surveys of yardage treated, sampling in the open pit as already described and output sampling. To sample heads for unit processes—i.e. hydrocyclone, jigs and tables—was not feasible so only tailing and product were sampled, although even this proved difficult. For sampling hydrocyclone discharges and jig tails a cut-down 44-gal oil drum fitted with wooden shafts was used, taking the whole flow for a few seconds against a stop watch. These results proved sufficiently accurate for guidance in the design engineering and operation of the recovery plant and from an initial recovery of only 30 per cent nearly 80 per cent was finally achieved. The stage had then been reached at which further operational analyses, to be sufficiently accurate, would need to be based on automatic sampling over representative periods and the average of repeated determinations of rates of throughput. Such facilities it is planned to incorporate in an improved version of the plant, which will thus still retain the dual characteristics of a full-scale production installation and a pilot plant.

SAMPLE VALUATION

All plant samples consisted of the same three constituents as the disintegrated granite but in varying proportions—i.e. slime, sand and heavy minerals. Slime was removed by careful elutriation to ensure that no appreciable amount of fine columbite or other heavy mineral was lost in the overflow. The slime content was determined by difference, or by filtering, drying and weighing. The deslimed sands were dried, screen-sized and each fraction concentrated in a superpanner. Only when it was required to make a physical assay for all the heavy minerals present was grain counting carried out on untreated concentrate. Normally the superpanner product was treated to separate a higher grade concentrate containing columbite mainly, with subordinate xenotime and magnetic cassiterite. Each screen-sized fraction, or a magnetic separate from it, was leached for 48 hours in boiling concentrated hydrochloric acid. This dissolved ilmenite, as well as magnetite and other iron oxides, demagnetized part of the magnetic cassiterite and all the magnetic zircon, and reduced thorite to a silica skeleton, but it had no appreciable effect on the columbite and xenotime. After treatment with dilute hydrochloric acid on the zinc block to turn the cassiterite grey, each fraction was dried and the columbite, xenotime and remaining magnetic cassiterite were extracted together in a Franz isodynamic separator. As columbite usually comprised 90 per cent or more of the final magnetics, estimation under the microscope was rapid and accurate. Physical assaying for magnetic cassiterite and xenotime entailed counting a larger number of grains and was only carried out for special purposes. If the non-magnetics contained too much sand they were either

cleaned up in the superpanner or by centrifuging in bromoform before grain counting the non-magnetic cassiterite under the microscope.

With samples thus physically assayed for up to three minerals or more if necessary, covering a specific gravity range from 7·0 to 4·5 determined for sized fractions over a mesh range from 16 to 300, illuminating performance analyses were made of unit processes in the recovery plant.

PLANT DEVELOPMENT AND PERFORMANCE

Stage 1

The flowsheet of the initial recovery plant is shown in Fig. 1. As no moving screen was available a fixed grizzly with a gap of ½ in. was installed. It would have been preferable, nevertheless, to have screened at ¼-in or finer, as stones between ¼ in. and ½ in. in diameter and larger tended when flat to replace the hematite ragging in the first hutches of the primary jigs.

Hydrocyclones were installed ahead of the secondary and clean-up jigs only, mainly to remove excess water, but it was found that they also improved recovery by lowering the slime content of the feed. The hydrocyclone overflows and clean-up jig tails were provisionally returned in closed circuit to the sunken main feed bin to minimize loss, although this practice tended to reduce the new intake of the 10-in feed pump. The rate of throughput of decomposed granite was controlled by varying the rate of bulldozing into the sunken feed bin and by manipulating the monitor.

The first test run made was intended to ascertain the maximum throughput capacity of the recovery plant and during a period of two weeks an average throughput of 124 cu. yd an hour was maintained. This achievement was particularly noteworthy, as it was more than twice the average solids throughput of 10-in gravel pumps as used in alluvial mining on the Jos Plateau. However, that rate of feed to the jigs was about the same as would be normal in a dredge working an alluvial tin deposit.

TABLE II.—*Recovery of Columbite in Relation to Grain Size*

B.S. sieve	Flowsheets and cubic yards per hour				
	Stage 1		Stage 2	Stage 3	Stage 4
	124	63	77	69	70
	Col. 1	Col. 2	Col. 3	Col. 4	Col. 5
	%	%	%	%	%
+ 25 } + 52 }	99	99	99	99	99
+ 72	76				
+ 100	57	90	97	97	97
+ 150	20	80	83	94	96
+ 170	20	35	50	87	92
+ 240	15	30	34	75	86
+ 300	6	25	18	40	50
− 300	2	3	4	20	29
Total	30	60	69	74	78

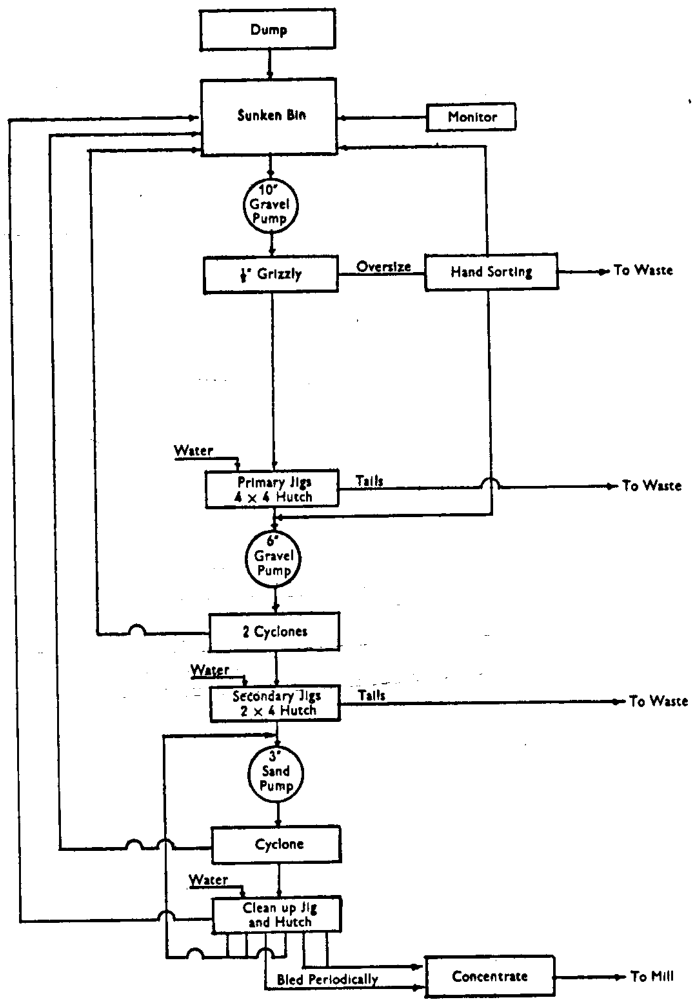

Fig. 1.—Flowsheet of experimental jig plant. Stage 1.

At this high rate of throughput recovery of columbite was only about 30 per cent of the bore values; recoveries in relation to grain size are shown in column 1 of Table II. These results presented two major problems in plant development: (1) to extend significant recovery of columbite to finer grain sizes and (2) to combine such higher percentage recovery with the proved capacity of the gravel pump to handle a feed of high pulp density.

The primary jig tails were then sampled. The slime was poured off and combined screen analyses and columbite valuations made on the two

portions. The results showed that about 70 per cent of the loss of 'free' columbite from the primary jigs was held in suspension in the rather thick slurry. Grains of columbite up to as coarse as 72 mesh were well represented in suspension. For comparison a sample of the tails from the secondary jigs was similarly analysed, as the slime content of the feed to those jigs had been considerably reduced by the hydrocyclones. The analysis showed that here only about 20 per cent of the columbite lost in the tails was held in suspension in this more dilute slurry and, furthermore, that very little of this columbite held in suspension was coarser than 150 mesh. It was therefore concluded that it would be worth while installing hydrocyclones and the necessary pump ahead of the primary jigs.

As a temporary measure and to check the overall recovery with less slime in the water the throughput of decomposed granite was reduced to approximately half while still using about the same quantity of water. The overall recovery of columbite then rose to 60 per cent of the bore value. The improvement in recovery in relation to grain size is shown in column 2 of Table II. Hydrocyclones were then installed ahead of the primary jigs to try and maintain or improve this percentage recovery and leave scope for stepping up the throughput again.

Stage 2

The flowsheet of the modified recovery plant is shown in Fig. 2. The improvement in the recovery of columbite of fine grain size resulting from the partial desliming thus effected is indicated in column 3 of Table II.

TABLE III.—*Performance Analysis of Secondary Hydrocyclones*

B.S. sieve	Reporting in underflow		
	Sand	Columbite	Cassiterite
	%	%	%
+ 16 B.S. mesh	98·1	—	—
+ 25 „ „	98·3	—	—
+ 52 „ „	98·0	99·95	—
+ 72 „ „	97·1	98·8	—
+ 100 „ „	95·4	98·7	—
+ 150 „ „	92·5	98·5	99·65
+ 170 „ „	88·8	98·4	99·5
+ 240 „ „	86·7	97·9	99·0
+ 300 „ „	97·7	96·4	98·4
+ 325 Com. mesh	92·2	95·6	97·5
− 325 „ „	94·0	91·0	89·5
Total	93·7	97·0	97·8
Slime	18·2		
Water	10·8		

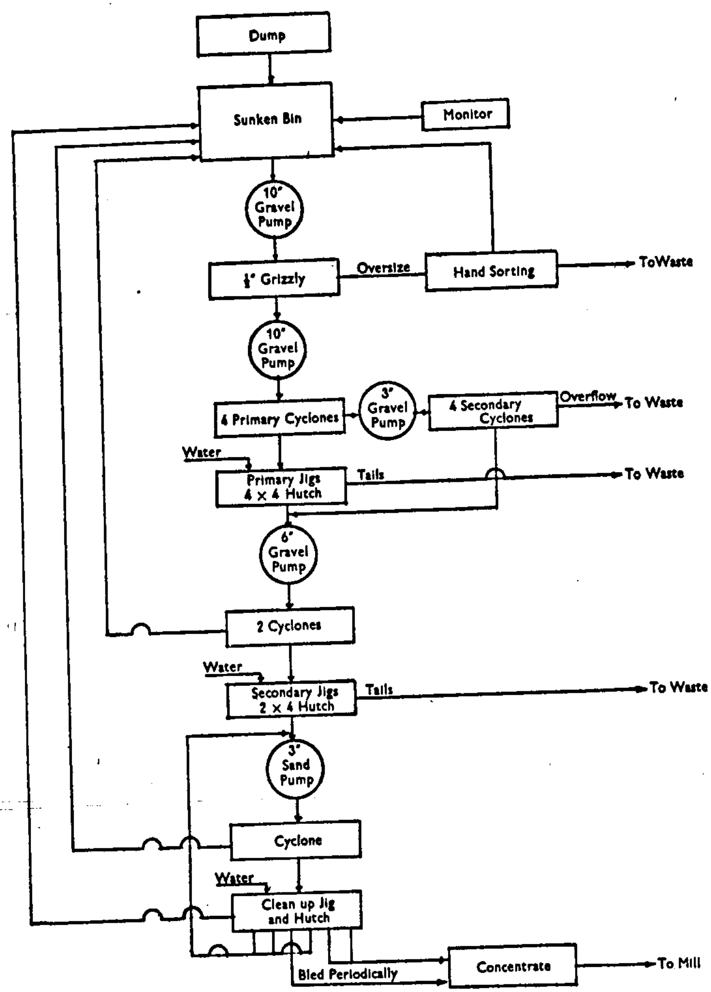

Fig. 2.—Flowsheet of experimental jig plant. Stage 2.

As shown by the flowsheet the amount of slime remaining in the underflow of the secondary cyclones was further reduced by dilution with the hutch products of the primary jigs and pumping through the hydrocyclones already installed ahead of the secondary jigs.

For guidance in future design engineering it was then decided to make a performance analysis of the secondary hydrocyclones and timed samples of the overflow and underflow were taken and analysed in the laboratory. From the results the performance analysis recorded in Table III was

calculated, showing the indicated efficiency of extraction of both cassiterite and columbite to be very good. Particularly noteworthy was the fact that recovery was well maintained even in the sub-sieve sizes. It was thought that if the grizzly was replaced by screens down to about ¼ in. mesh then the primary hydrocyclones might be expected to operate with similar efficiency and thus improve recovery in the primary jigs.

At this stage it was decided to make a performance analysis of the clean-up jig, the results being recorded in Table IV. The last four hutches

TABLE IV.—*Performance Analysis of Clean-up Jig*

B.S. sieve	Columbite recovery: lb/hour			Percentage of new feed represented by tails
	Hutches 1 and 2 withdrawn	Hutches 3–6 closed circuit	Tails	%
+ 16 B.S. mesh	0·3	—	—	—
+ 25 . .	4·8	1·0	0·4	8
+ 52 . .	44·2	5·8	1·6	4
+ 72 . .	53·0	71·1	0·8	1
+ 100 . .	76·0	248·0	1·9	2
+ 150 . .	57·2	163·0	7·6	12
+ 170 . .	21·7	132·7	8·6	28
+ 240 . .	19·7	106·2	24·6	55
+ 300 . .	3·4	17·7	8·4	70
+ 325 Com. mesh	1·1	5·7	5·8	85
− 325 . .	0·6	4·8	3·3	85
Total .	282·0	756·0	63·0	

TABLE V.—*Screen Analyses of Clean-up Jig Tails*

B.S. sieve	Sand	Columbite
	%	%
+ 16 B.S. mesh .	4·8	—
+ 25 . . .	24·6	0·7
+ 52 . . .	32·4	2·5
+ 72 . . .	10·3	1·9
+ 100 . . .	15·7	3·1
+ 120 . . .	2·1	1·8
+ 150 . . .	4·2	10·4
+ 170 . . .	2·3	13·7
+ 240 . . .	2·8	38·2
+ 300 . . .	0·6	13·3
+ 325 Com. mesh .	0·1	9·2
− 325 . . .	0·1	5·2
Total . .	100·0	100·0

were returned in closed circuit in order to maintain a high grade of concentrate from the first two, this alone being withdrawn for despatch to the mill. The loss of fines to jig tailing was high. Although such tailing was returned in closed circuit through the whole recovery plant it was

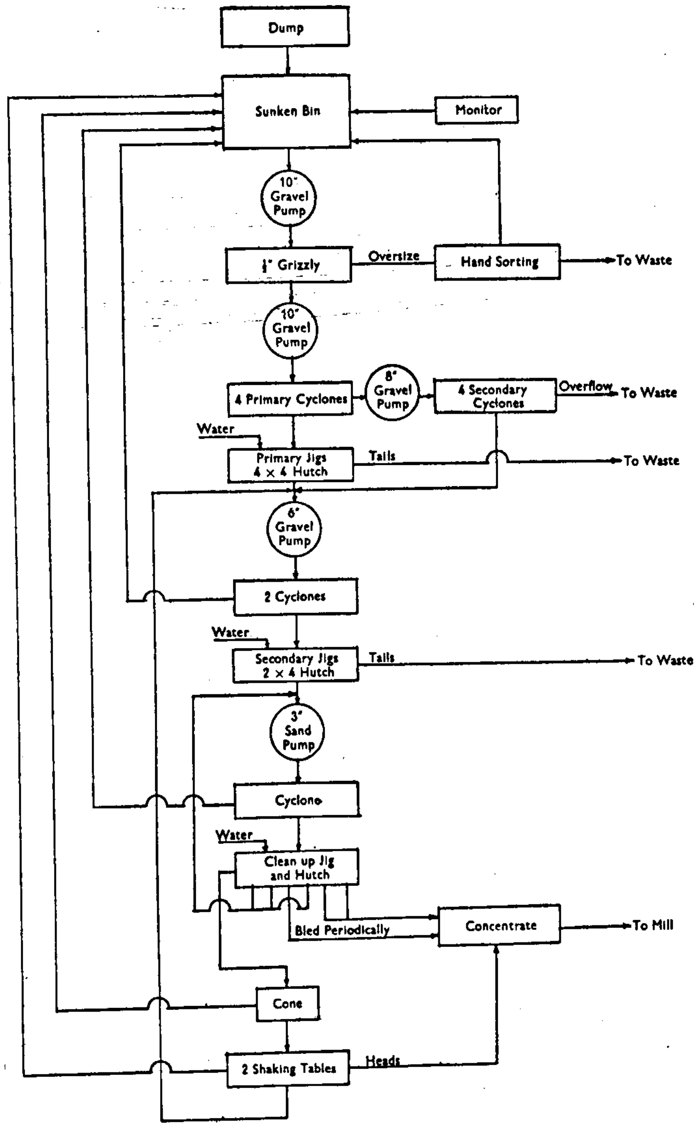

Fig. 3.—Flowsheet of experimental jig plant. Stage 3.

apparent that this practice was building up an excessive circulating load of fine heavy minerals, much of which would ultimately be lost in the primary and secondary jig tails. A combined screen and mineral analysis of the clean-up jig tailings, recorded in Table V, showed that this was a well classified product and it was decided, therefore, to install two Holman sand tables to scalp fine heavy mineral from this circulating load.

Stage 3

The flowsheet incorporating the two sand tables is shown in Fig. 3. Only a rich head cut, consisting mainly of fine columbite and cassiterite, was taken from these tables, lower-grade middling being returned to the secondary jigs. The table tailings thus scalped of fine mineral were returned in closed circuit through the whole recovery plant as a precaution against loss of coarse mineral. The efficiency of these tables was thus not critical in overall plant recovery. The improvement in the overall recovery in relation to grain size is shown in column 4 of Table II.

At this stage in the development of the recovery plant it was decided to make a detailed performance analysis of the secondary jigs. The percentage of slime in the feed to these jigs had already been reduced to about 6 per cent of the total solids and now they had been relieved of an excessive circulating load of fine heavy mineral. It was considered, therefore, that their performance must have reached about the practical optimum recovery for jigs of that length—i.e. four hutches totalling 13 ft 4 in. Furthermore, the results indicated what might be expected from primary jigs with adequate desliming of the feed. Timed samples were then taken from all the hutch discharges and the tailings discharge and analysed in the laboratory. The opportunity was taken to make this analysis in terms of three minerals—cassiterite, columbite and xenotime, covering a range of specific gravity 7·0 to 4·5. The jig performance analysis is recorded in Table VI overleaf. From these results it was apparent that although recovery should be improved by lengthening the jig to, say, six hutches

TABLE VII.—*Screen Analyses of Secondary Jig Tails*

B.S. sieve	Sand	Columbite	Cassiterite
	%	%	%
+ 16 B.S. mesh .	15·0	—	—
+ 25 . . .	32·2	—	—
+ 52 . . .	30·8	0·3	—
+ 72 . . .	6·8	0·3	—
+ 100 . . .	5·2	1·4	Tr
+ 150 . . .	4·4	4·5	1·2
+ 170 . . .	1·6	4·5	4·0
+ 240 . . .	2·3	21·1	20·0
+ 300 . . .	0·6	15·4	31·0
+ 325 Com. mesh .	0·5	20·0	29·0
− 325 . . .	0·6	32·5	14·8
Total . .	100·0	100·0	100·0

TABLE VI.—*Performance Analysis of Secondary Jigs (Pan American)*

B.S. sieve	Mineral	Distribution									
		Percentage recovered							Percentage lost		
		Hutch no.				Total					
		1	2	3	4	Cass-iterite	Col-umbite	Xeno-time	Cass-iterite	Col-umbite	Xeno-time
		%	%	%	%	%	%	%	%	%	%
+ 16	Cassiterite	100·0	—	—	—	100·0			—		
	Columbite	78·7	14·7	6·6	—		100·0			—	
	Xenotime	93·8	6·3	—	—			100·0			—
16/25	Cassiterite	93·6	6·4	—	—	100·0			—		
	Columbite	84·1	13·2	1·7	1·0		100·0			—	
	Xenotime	81·3	18·7	—	—			100·0			—
25/52	Cassiterite	72·8	24·8	2·4	—	100·0			—		
	Columbite	77·2	16·7	3·4	1·8		99·1			0·9	
	Xenotime	78·1	18·4	1·9	1·2			99·6			0·4
52/72	Cassiterite	68·5	28·2	1·2	2·1	100·0			—		
	Columbite	72·2	20·3	3·1	3·0		98·6			1·4	
	Xenotime	67·9	21·2	5·5	5·1			99·7			0·3
72/100	Cassiterite	67·3	25·7	3·1	3·8	99·9			0·1		
	Columbite	63·6	26·0	4·2	4·6		98·4			1·6	
	Xenotime	60·5	28·8	4·6	5·3			99·2			0·8
100/120	Cassiterite	62·3	28·1	4·7	4·5	99·6			0·4		
	Columbite	58·5	26·8	5·4	5·8		96·5			3·5	
	Xenotime	57·1	29·3	5·5	5·5			97·4			2·6
120/150	Cassiterite	56·9	29·3	8·3	5·1	99·6			0·4		
	Columbite	50·0	32·1	5·9	6·9		94·9			5·1	
	Xenotime	51·1	27·4	8·0	9·4			95·9			4·1
150/170	Cassiterite	48·2	34·7	10·0	6·7	99·6			0·4		
	Columbite	41·8	31·2	8·7	9·3		91·0			9·0	
	Xenotime	47·7	21·1	10·9	11·5			91·2			8·8
170/240	Cassiterite	46·3	27·6	14·4	10·9	99·2			0·8		
	Columbite	33·3	26·4	12·2	11·9		83·8			16·2	
	Xenotime	29·1	20·7	13·9	12·4			76·1			23·9
240/300	Cassiterite	26·3	25·5	27·1	15·4	94·3			5·7		
	Columbite	15·4	12·4	9·8	9·4		47·0			53·0	
	Xenotime	15·5	5·6	15·5	7·2			43·8			56·2
300/325 c	Cassiterite	20·6	24·8	24·0	18·2	87·6			12·4		
	Columbite	14·3	13·3	13·5	14·1		55·2			44·8	
	Xenotime	6·9	5·5	12·3	8·2			32·9			67·1
−325 c	Cassiterite	31·5	17·3	17·3	15·4	81·5			18·5		
	Columbite	9·2	8·0	8·3	10·6		36·1			63·9	
	Xenotime	2·0	1·8	1·8	4·4			10·0			90·0
Total	Cassiterite	53·5	28·0	9·7	7·2	98·4			1·6		
	Columbite	50·8	24·0	6·7	6·9		88·4			11·6	
	Xenotime	55·2	22·6	6·5	6·2			90·5			9·5
*Recovery lb/hr	Cassiterite	26·5	13·8	4·8	3·6	48·7			0·8		
	Columbite	108·9	51·4	14·2	14·7		189·2			24·7	
	Xenotime	28·6	11·7	3·3	3·2			46·8			4·9
Grade, lb/ton	Cassiterite	16·9	18·5	6·8	4·5	12·7			0·1		
	Columbite	69·4	69·0	20·1	18·3		49·4			2·2	
	Xenotime	18·2	15·7	4·7	4·0			12·2			0·4
*Tons/hr	Sand	1·570	0·745	0·705	0·805		3·825			11·050	
*Gal/min	Discharges	25·3	23·4	33·4	27·0		109·1			406·3	

*A pair of four hutch jigs.

and returning the products of the last two hutches in closed circuit to avoid overloading the clean-up jigs, there would still be a considerable loss of columbite finer than about 170 mesh. The combined screen and mineral analysis of these jig tails recorded in Table VII showed that, if the coarse fraction could be screened out for discharge to waste, the enriched screen undersize should be suitable for tabling. It was, therefore, decided to install a Symons V-screen and four Holman tables for treating the tailing from these jigs.

Stage 4

The flowsheet of the recovery plant incorporating the V-screen and sand tables for treating the screen undersize is shown in Fig. 4, the improvement in the overall recovery in relation to grain size being shown in column 5 of Table II.

The percentage of undersize over a range of grain sizes reporting with the oversize of the V-screen fitted with a screen having apertures equivalent to 25 B.S. mesh is recorded in Table VIII. Comparison with Table VII indicates that the loss of values to waste was not serious.

A screen and mineral analysis of the dewatering cone underflow passed to the sand tables is recorded in Table IX. Recovery from this battery of tables was less than half the amount anticipated from the sampling of the secondary jig tails. It was suspected at first that an appreciable loss of fine

TABLE VIII.—*Performance Analysis of Symons V-screen at 25 B.S. mesh*

B.S. sieve	Percentage of undersize reporting with oversize	
	Input 3·6 tons/hr	Input 9·0 tons/hr
	%	%
+ 30 B.S. mesh	50·7	72·0
+ 36	16·7	42·6
+ 44	7·9	31·8
+ 52	2·5	16·0
+ 60	1·5	10·2
+ 72	0·5	5·6
+ 85	0·3	3·0
+ 100	0·2	2·2
+ 120	0·1	2·2
+ 150	0·1	0·7
+ 170	0·1	1·0
+ 220	0·1	0·3
+ 240	0·1	0·1
+ 300	0·4	0·4
+ 325 Com. mesh	0·5	2·8
− 325	8·0*	3·6
Distribution—		
Oversize	0·7 tons/hr	4·2 tons/hr
Undersize	2·9 tons/hr	4·8 tons/hr

* Including slime.

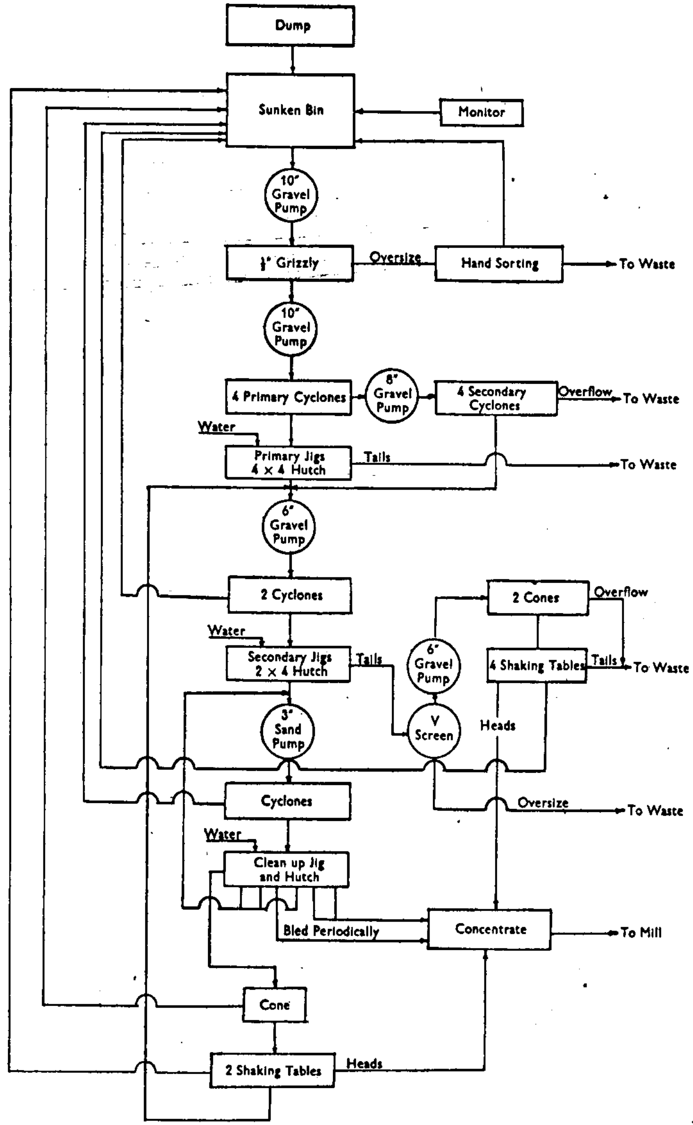

Fig. 4.—Flowsheet of experimental jig plant. Stage 4.

mineral might be occurring in the overflow of the dewatering cones used to thicken the feed to the tables, but as these cones were only temporary, pending the installation of hydrocyclones, priority was given to a study of table performance.

TABLE IX.—*Analysis of Feed to Holman Tables*

B.S. sieve	Total material	Cassiterite	Columbite	Zircon
	%	%	%	%
+ 52 B.S. mesh	27·20	0·21	0·01	1·09
+ 72	25·64	0·24	2·23	7·61
+ 100	20·21	0·85	6·61	9·52
+ 120	7·35	1·24	7·50	6·48
+ 150	5·74	0·46	6·39	7·79
+ 170	5·20	0·48	10·51	13·10
+ 240	4·71	17·64	19·97	24·34
+ 300	1·24	18·87	9·31	8·91
+ 325 Com. mesh	0·95	21·03	8·18	7·40
− 325	1·76	38·98	29·29	13·76
Total	100·00	100·00	100·00	100·00
Grade, lb/ton		0·04	3·30	4·77

Notes.

(1) Feed was V-Screen undersize from secondary jig tails.
(2) Analysis calculated from timed sample cuts nos. 1–6 of table discharge.

The tables appeared to be working satisfactorily, but it was proved in the event that they were only making about 40 per cent recovery of columbite in the head cut. Six timed samples were taken from one of the tables, covering the total discharge, as shown in the diagram Fig. 5. Screen and mineral analyses to include all the heavy minerals in case of need were made in the laboratory. Recovery of cassiterite in the head cut proved to

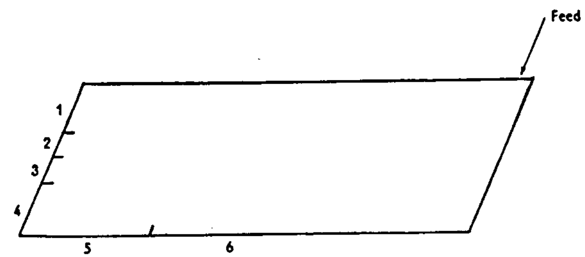

Fig. 5.—Harwell plant. Sample diagram of Holman table treating V-screen undersize from secondary jig tails.

Cut no. 1 *Black band, cassiterite, columbite, zircon.*
2 *Brown band, zircon predominates.*
3 *Clear band, quartz predominates.*
4, 5, 6 } *Opaque band, quartz and decomposed Felspar.*

TABLE X.—*Performance Analysis of Holman Table in Terms of Columbite*

B.S. screen	Feed	Cut No. 1	Cut No. 2	Cut No. 3	Cut No. 4	Cut No. 5	Cut No. 6
	Recovery of columbite from V-screen undersize from secondary jig tails						
	Screen analysis of columbite content						
	%	%	%	%	%	%	%
+ 52 B.S. mesh	—	—	—	—	—	—	—
+ 72	2·24	0·14	1·93	—	2·95	10·51	—
+ 100	6·61	1·39	6·02	0·30	7·29	30·47	—
+ 120	7·50	3·82	13·03	0·71	8·35	19·57	21·02
+ 150	6·39	4·63	7·04	1·46	9·89	5·99	31·84
+ 170	10·51	9·38	8·50	3·92	15·81	8·33	16·52
+ 240	19·97	27·50	12·76	8·93	20·09	7·85	14·41
+ 300	9·31	13·02	11·07	7·52	7·18	1·81	3·90
+ 325 Com. mesh	8·18	12·69	5·80	10·92	4·42	0·76	4·20
− 325	29·29	27·43	33·85	66·24	24·02	14·71	8·11
Total	100·00	100·00	100·00	100·00	100·00	100·00	100·00
	Distribution of columbite						
+ 52 B.S. mesh	—	—	—	—	—	—	—
+ 72	100·0	2·6	7·3	—	37·0	53·1	—
+ 100	100·0	8·7	7·7	0·5	31·0	52·1	—
+ 120	100·0	21·0	14·8	0·9	31·3	29·5	2·5
+ 150	100·0	29·8	9·4	2·3	43·5	10·6	4·4
+ 170	100·0	36·8	6·9	3·7	42·3	8·9	1·4
+ 240	100·0	56·7	5·4	4·5	28·3	4·4	0·7
+ 300	100·0	57·6	10·1	8·1	21·6	2·2	0·4
+ 325 Com. mesh	100·0	63·9	6·0	13·3	15·2	1·1	0·5
− 325	100·0	38·6	9·8	22·6	23·0	5·7	0·3
Columbite	100·0	41·2	8·5	10·0	28·1	11·3	0·9
Total feed	100·0	0·20	0·15	2·89	18·57	41·03	37·16
Grade	0·147	31·10	8·17	0·50	0·22	0·04	0·0033

Rate of feed 2116 lb/hr.

be nearly 100 per cent. A performance analysis in terms of columbite is presented in Table X and the poor recovery in the head cut, or even cuts 1 and 2 combined, will be noted. The distribution peak in cut 4 is of special interest, particularly as it is recognizable right through the range of grain sizes. From this analysis it was apparent that to achieve a good recovery of columbite cuts 1 and 2 should be accepted, cuts 3, 4 and 5 retabled, that only the comparatively small bulk represented by cut 6 was poor enough to discharge to waste and that closer classification should also improve recovery.

TABLE XI.—*Performance Analysis of Holman Table in Terms of Zircon*

B.S. screen	Feed	Cut No. 1	Cut No. 2	Cut No. 3	Cut No. 4	Cut No. 5	Cut No. 6
	Recovery of zircon from V-screen undersize from secondary jig tails						
	Screen analysis of zircon content						
	%	%	%	%	%	%	%
+ 52 B.S. mesh	1·09	0·04	0·49	5·05	0·47	4·16	—
+ 72	7·61	0·38	8·27	9·08	11·30	23·95	—
+ 100	9·52	2·83	13·65	2·07	11·64	23·16	59·93
+ 120	6·48	3·80	8·45	0·86	8·00	14·87	11·78
+ 150	7·79	6·95	7·64	1·99	11·84	4·55	5·89
+ 170	13·10	13·29	12·02	4·03	17·44	9·49	9·26
+ 240	24·34	36·87	22·45	13·71	19·55	5·97	4·04
+ 300	8·91	12·85	7·82	8·64	7·41	0·69	2·19
+ 325 Com. mesh	7·40	9·60	3·26	9·54	7·06	2·90	2·36
− 325	13·76	13·39	15·95	45·03	5·29	10·26	4·55
Total	100·00	100·00	100·00	100·00	100·00	100·00	100·00
	Distribution of zircon						
+ 72 B.S. mesh	100·0	1·8	10·3	15·6	41·3	31·0	—
+ 100	100·0	11·4	14·6	2·1	37·3	23·3	11·3
+ 120	100·0	22·5	13·3	1·3	37·7	22·0	3·2
+ 150	100·0	34·2	10·0	2·5	46·4	5·6	1·3
+ 170	100·0	38·8	9·4	3·0	40·6	7·0	1·2
+ 240	100·0	58·0	9·4	5·4	24·5	2·4	0·3
+ 300	100·0	55·2	8·9	9·3	25·4	0·8	0·4
+ 325 Com. mesh	100·0	49·7	4·4	12·4	29·1	3·8	0·6
− 325	100·0	37·3	11·8	31·4	11·7	7·2	0·6
Zircon	100·0	38·3	10·2	9·6	30·5	9·6	1·8
Total feed	100·0	0·20	0·15	2·89	18·57	41·03	37·16
Grade	0·213	40·88	14·49	0·71	0·35	0·05	0·01

Rate of feed 2116 lb/hour.

It was at first suspected that the poor recovery of columbite could be due in part to its shape, a high proportion of the grains being acicular or tabular in habit. However, examination of the concentrates under the microscope had not indicated that this was the cause. As a check the performance analyses in terms of zircon, presented in Table XI, was calculated from the screen and mineral analyses of the samples, as the zircon in the granite worked is characteristically equidimensional. Zircon showed the same characteristic distribution.

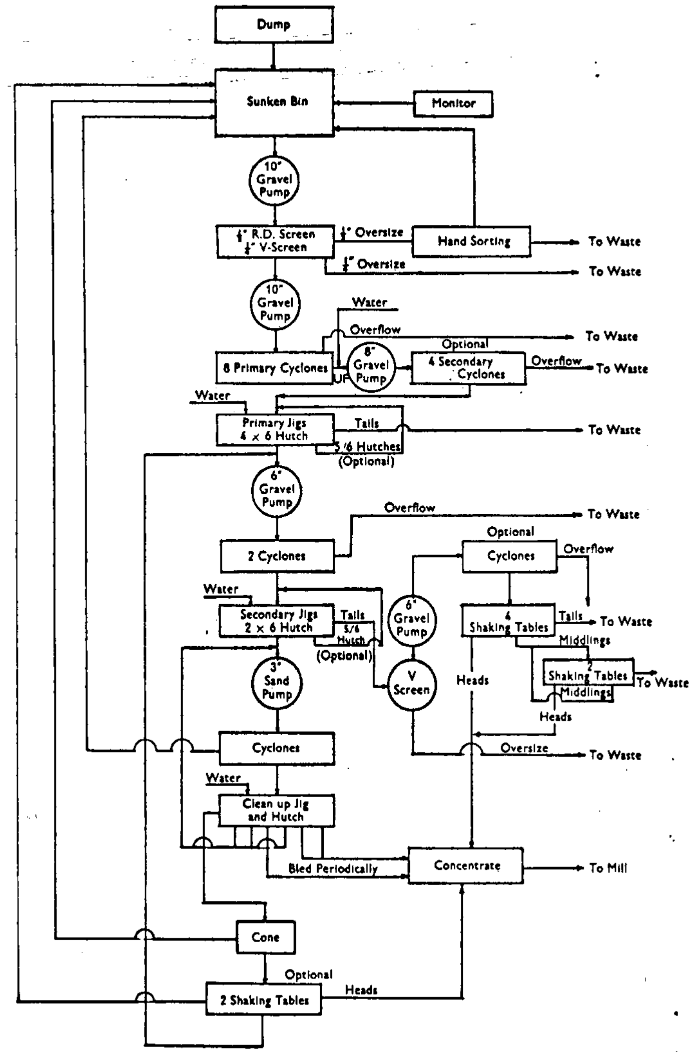

Fig. 6.—Flowsheet of experimental jig plant. Stage 5 (proposed.)

By the time the development of the recovery plant had reached Stage 4, the bottom had temporarily fallen out of the columbite market. The plant was therefore simplified and turned over to treating alluvial wash from an adjacent lead-containing cassiterite and subordinate columbite.

FUTURE DEVELOPMENT

As a basis for discussion of the design engineering of the next version of this plant to be used for treating the decomposed granite, the flowsheet shown in Fig. 6 was drawn. This presents several choices of optional features for improving recovery to whatever might be the economic limit of extraction as determined by operating costs, the grade of ore and the realization value of the products. The grizzly has already been replaced by a ¾-in rod-deck screen and Symons V-screens fitted with 3-mesh wire screens. For the decomposed granite better results in the primary cyclones appear desirable, owing to the detrimental effect of slime on the recovery of fine columbite. Part of the recovery which should be achieved with the full battery of tables depicted could be more cheaply obtained by using longer jigs, the reserve capacity of the jigs to take a higher rate of throughput being considerable, whereas the tables are already fully loaded. The several optional features shown in this flowsheet are by no means exhaustive. Although not strictly a pilot plant it is reasonably flexible and presents scope for experimentation.

APPLICATION TO ALLUVIAL DEPOSITS

Preliminary analyses of sectional bore samples from the alluvial lead indicated that the wash contained only about 30 per cent slime and that very little cassiterite or columbite was finer than 150 mesh, so that, compared with the decomposed granite, it would be relatively easy to attain the economic limit of extraction.

Except for replacing the grizzly by a ½-in rod-deck followed by V-screens fitted with 3-mesh wire screens, the recovery plant was therefore simplified to reduce the operating cost. The secondary hydrocyclones were directly connected to the overflow of the primary cyclones, thus dispensing with one pump. The battery of four tables treating the V-screen undersize from the secondary jig tails was then taken out of service after a week's test run had shown the yield from it to be negligible; there was very little heavy mineral fine enough to get past the secondary jigs. To reduce operating costs the two tables following the clean-up jig were also taken out of service and the tails from this jig returned to the feed of the secondary jigs. With the percentage of sand in the feed much higher than with the decomposed granite, the capacity of the primary hydrocyclones limited the plant throughput to about 80 cu. yd per hour. The next alteration, therefore, will be to install additional primary hydrocyclones up to the full capacity of the primary jigs to take the underflow. The plant should then be operating at near optimum throughput and near the economic limit of extraction for the wash from this particular alluvial lead.

The plant described is now serving as the prototype for the design engineering of additional jig plants to replace sluice boxes in alluvial mining. At the same time the methods of quantitative fragmental petrography originally devised for analysing the intensely decomposed granite are in process of being modified for the study of alluvial deposits in order that the most suitable recovery plant can be designed for each.

APPLICATION TO OTHER ORES

Practically any incoherent ore containing sufficient free heavy economic mineral could be treated with an appropriate sequence of coarse screens, hydrocyclones, jigs, fine screens and tables as in the example described. The range of ores would include intensely-decomposed igneous and metamorphic rocks, as well as eluvial, alluvial and beach deposits. Coarse screens will eliminate any quantity of stone and hydrocyclones will reduce the slime content to the required minimum without appreciable loss of heavy mineral. The lower limits of specific gravity and grain size at which jigs will still make a good recovery, if the feed is well deslimed, are approximately 7/300, 5/150 and probably about 4/100. Within practical limits, the longer the jig the better the recovery. Except on a dredge, where space is limited, jig tails can be effectively screened at about 25-mesh and only the undersize, a well-classified product, treated on tables. The plant can be made simple or elaborate according to the economic limit of extraction, as determined by the operating cost of unit processes, the grade of the ore and the realization value of the products.

The methods of quantitative fragmental petrography described elsewhere,[1] originally devised for the geological mapping and study of intensely decomposed granites, could be modified for analysing any incoherent ores and potential ores containing free heavy mineral. Furthermore, as described in the present paper, such methods can provide illuminating analyses of plant performance.

This paper has been written in the hope that it will stimulate both the search for new incoherent ores containing free heavy economic minerals and the reassessment of deposits already known or being worked, and, at the same time, foster use of the method of combined screen and mineral analyses of timed samples for fundamental research on the recovery efficiency of unit processes.

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[1] See reference on p. 89.

The Dressing of Flake Graphite in Kenya*

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622.7-336

SYNOPSIS

The paper describes the treatment of crucible flake graphite in Kenya. Separation may be effected by consecutive crushing and screening, and sluicing or winnowing; and by flotation. Two typical flowsheets are given: the first a small-worker's plant that employs two-stage crushing followed by sluicing and 'batch' flotation; the second is a larger plant with three-stage crushing, rougher flotation and cleaning of the concentrate in winnows.

THE EXISTENCE OF GRAPHITE in Kenya was known before 1914, but it was not worked until 1942 when required for strategic purposes. At that time it was very difficult to import machinery, and plants were improvised locally often using redundant machines from gold mines. Since the war the industry has been hampered by lack of capital, and there is still a marked tendency to improvise.

Specification

The market specifications for crucible-grade graphite has already been described in a previous paper.[1] These specifications have an important influence on the dressing of the ore.

Apart from the crucible trade, there is also a market for purer forms of graphite, but with much less stringent requirements as to size. The specifications are + 90 per cent, + 95 per cent or + 98 per cent carbon and such size requirements as + 50 or + 80 mesh.‡ A Kenya producing firm, Messrs. Shah Vershi Devshi, Ltd., is now supplying all its graphite to this type of specification.

Occurrence

All the important occurrences of flake graphite in Kenya are in graphite schists with quartz and felspar. Mica is found in some of these schists. It is more common in the adjoining country rock than in the ore itself, but with careful mining it may be excluded from the ore fed to the mill. Apatite and pyrite have also been noted, but only as very minor accessories, insufficient to affect the dressing.

Treatment

Comminution and Sizing

When severing graphite from the gangue it is important to do so with as little damage to the flake as possible so that the buyers' demand for

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†Mines and Geological Department, Kenya.
‡Screens are approximately B.S.S.
[1]etc. See list of references at the end of the paper.