GOLD CONCENTRATION AT THE AMALGAMATED BANKET AREAS REDUCTION PLANT

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ABSTRACT

The principal subject of the present paper is a description of mechanical methods which have been introduced in place of straking for the recovery of free gold at the reduction plant of Amalgamated Banket Areas, Ltd. Relevant to gold concentration, reference is also made to the mineralogical characteristics of the banket ore; the general arrangement of plant; security aspects; overall performance of the plant when straking was practised and as now operating; the association of gold with ore minerals in relation to the necessity for selective regrinding of such minerals, and alternatives to concentration.

INTRODUCTION

The property of Amalgamated Banket Areas, Ltd., is at Tarkwa, 39 miles by rail from the modern port of Takoradi in the Gold Coast, and includes two of the original (and in their day prominent) mines located and worked on the banket lode—Tarkwa and Abbontiakoon. Whereas the descriptive term 'banket' originated in the Transvaal and was later applied to the auriferous conglomerates of the Gold Coast, it is of interest that the latter were, in fact, the first discovered (in 1877) and developed, some seven to eight years before the discovery and exploitation of the more famous and extensive Witwatersrand field.

Two main classes of ore are being received at the Amalgamated Banket Areas plant for treatment—deep-mine unaltered banket, and lower-grade weathered ore from a flat anticlinal banket capping at Pepe, which is mined by opencast methods. The two are treated in parallel, but practically distinct, circuits. It is with the treatment of the underground ore that the present paper is mainly concerned.

THE BANKET ORE

The Gold Coast 'banket' has been described in memoirs of the Gold Coast Geological Survey. Some of the principal characteristics are briefly mentioned here in their relevance to the metallurgical problem.

The conglomerate is comprised substantially of ellipsoidal water-worn pebbles of quartz, some clear and glassy, others white and opaque. Occasional pebbles of amethystine quartz and of pink felsite are also present. The pebbles vary in size over an average range of one to three inches in length, although smaller and larger pebbles are encountered. Distributed through the pebble interstices are sand grains and small crystals of specular haematite, with minor amounts of ilmenite and magnetite. The whole has been cemented by silicification into an extremely hard compact conglomerate superficially resembling the Witwatersrand banket.

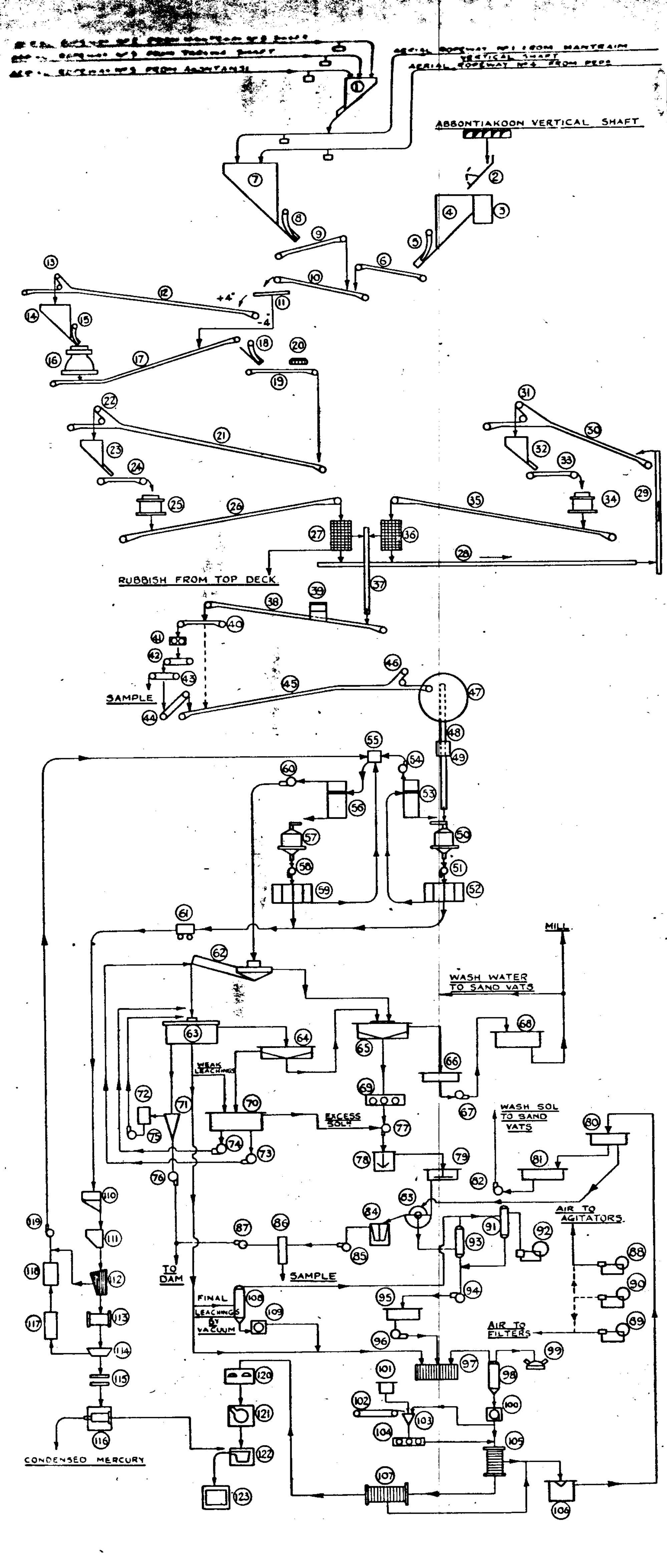
The haematite, ranging up to 7 per cent by weight of the ore, is of primary origin and persists to the deepest levels (2,500 ft at Taquah and Abosso Mines, Ltd.) yet worked. The gold, which is unevenly distributed through the secondary quartz matrix, is crystalline and in extremely fine grains, being rarely visible to the unaided eye. Sulphides are so seldom encountered as, for metallurgical purposes, to be considered absent. The quartz pebbles are substantially barren. There is microscopic evidence of occasional occurrence of gold in haematite itself, but our investigations have shown that this is of minor significance compared with the proportions that obtain for gold in the pyrite of the Witwatersrand banket.

The underground ore treated at the plant is comprised of banket, banded auriferous haematite-quartzite, and varying dilutions of quartzite wall rocks. As received the ore ranges from 4 to 6 dwt/ton.*

FORMER PRACTICE—STRAKING

The flow-sheet to which the plant originally operated is set out in Fig. 1 which includes descriptive data covering

* All assays refer to the ton of 2,000 lb.



1	[illegible] BIN [illegible] TONS CAPACITY
2	BY PASS GATE FOR WASTE
3	WASTE BIN 150 TONS CAPACITY
4	SHAFT ORE BIN 500 TONS CAPACITY
5	2-Nº5 ROSS CHAIN FEEDERS
6	36" BELT CONVEYOR Nº1A
7	ROPEWAY UNLOADING BIN 1000 TONS
8	4-Nº5 ROSS CHAIN FEEDERS
9	36" BELT CONVEYOR Nº1B
10	36" BELT CONVEYOR Nº2
11	SHAKING GRIZZLEY 3' x 6'
12	36" BELT CONVEYOR Nº5
13	HAND TRAVERSED TRIPPER
14	PRIMARY CRUSHER BIN 300 TONS CAPY.
15	2-Nº5 ROSS CHAIN FEEDERS
16	2-13" Mc CULLY CRUSHERS
17	24" BELT CONVEYOR Nº6
18	Nº3 ROSS CHAIN FEEDER.
19	48" BELT CONVEYOR Nº7.
20	56" DIA. ELECTRO MAGNET
21	24" BELT CONVEYOR Nº8
22	HAND TRAVERSED TRIPPER
23	SURGE BIN 150 TONS CAPACITY
24	2-30" APRON FEEDERS
25	2-4' STD. SYMON CRUSHERS
26	2-16" BELT CONVEYORS Nº 12A & 12B
27	2-3'-10' GYREX DOUBLE DECK SCREENS
28	24" BELT CONVEYOR Nº9
29	24" BELT CONVEYOR Nº10
30	24" BELT CONVEYOR Nº11.
31	HAND TRAVERSED TRIPPER
32	SURGE BIN 150 TONS CAPACITY
33	2-30" BELT FEEDERS
34	2-4' SHORTHEAD SYMONS CRUSHERS
35	2-16" BELT CONVEYORS Nº 12C & 12D
36	2-5' x 10' GYREX SINGLE DECK SCREENS
37	24" BELT CONVEYOR Nº13A
38	24" BELT CONVEYOR Nº13B
39	MERRICK WEIGHTOMETER
40	BUCKET SAMPLER 1% CUT.
41	18" x 12" SAMPLE CRUSHING ROLLS
42	12" BELT FEEDER
43	BUCKET SAMPLER 0·5% CUT
44	BUCKET ELEVATOR
45	24" BELT CONVEYOR Nº14
46	AUTOMATIC TRAVERSING TRIPPER
47	4-FINE ORE STORAGE BINS. EACH 1000 TONS
48	4-16" BELT CONVEYORS Nºs 15A B C & D
49	4-MERRICK WEIGHTOMETERS
50	4-PRIMARY BALL MILLS 8' DIA x 6' LONG
51	2-4" WILFLEY PUMPS AT EACH MILL (1-STANDBY)
52	PRIMARY CORDUROY STRAKES (4 BANKS)
53	4-4' WIDE x 20' LONG DORR CLASSIFIERS
54	4-4" WILFLEY PUMPS
55	2-DISTRIBUTION BOXES
56	4-8' WIDE x 20' LONG DORR CLASSIFIERS
57	4-SECONDARY BALL MILLS 8' DIA x 6' LONG
58	2-4" WILFLEY PUMPS AT EACH MILL (1-STANDBY)
59	SECONDARY CORDUROY STRAKES (4-BANKS)
60	4-6" WILFLEY PUMPS (2-STANDBY)
61	TROLLEY FOR HANDLING CORDUROY CLOTHS
62	2-DORR TURRET TYPE CLASSIFIERS 8' x 20' x 14' BOWL
63	8-SAND VATS 50' DIA x 15' DEEP
64	DORR THICKENER 30' DIA. x 8' DEEP
65	3-DORR THICKENERS 50' DIA x 10' DEEP
66	THICKENER OVERFLOW TANK 20' DIA x 6'
67	2-6"/7" HARLAND PUMPS (1-STANDBY)
68	2-MILL WATER TANKS 40' DIA x 10'
69	3-DORRCO DIAPHRAGM PUMPS
70	CIRCULATING SOLUTION TANK 50' DIA x 1[illegible]
71	4-DEWATERING CONES 8' DIA x 10' DEEP
72	CONCRETE WATER TANK
73	1-HARLAND 4"/5" PUMP
74	1-HARLAND 4"/5" PUMP
75	1-5"/7" TWO STAGE SLUICING PUMP
76	2-6" WILFLEY PUMPS (1-STANDBY)
77	2-4" WILFLEY PUMPS (1-STANDBY)
78	8-DORR AGITATORS 20' DIA x 20' DEEP
79	SURGE AGITATOR 25' DIA x 10' DEEP
80	SPENT SOLUTION TANK 25' DIA x 10' DEEP
81	2-WASH SOLUTION TANKS 32' DIA x 10' DEEP
82	2-4"/5" HARLAND PUMPS (1-STANDBY)
83	4-OLIVER FILTERS 14' DIA x 16' FACE
84	4-VORTEX MIXERS 4' DIA x 4' DEEP
85	4-3" WILFLEY PUMPS (2-STANDBY)
86	GEARY JENNINGS SAMPLER
87	2-4" WILFLEY PUMPS (1-STANDBY)
88	2-COMPRESSORS FOR AGITATOR SERVICE
89	2-COMPRESSORS FOR FILTER SERVICE
90	1-COMPRESSOR STANDBY FOR 88 OR 89
91	2-MOISTURE TRAPS 34" DIA x 72"
92	3-DRY VACUUM PUMPS. (1-STANDBY.)
93	2-VACUUM RECEIVERS 34" DIA x 72"
94	2-4" SUBMERGED EXTRACTION PUMPS
95	UNCLARIFIED SOLUTION TANK 25' DIA x 10'.
96	2-5" HARLAND PUMPS. (1-STANDBY)
97	GOLD SOLUTION & CLARIFIER TANK 25' x 8'
98	MERRILL CROWE VAC RECEIVER 72" x 12'-0"
99	2-VACCUUM PUMPS (1-STANDBY)
100	1-8" SUBMERGED EXTRACTION PUMP
101	3-REAGENT FEEDERS.
102	3-ZINC DUST BELT FEEDER.
103	3-MIXING CONES
104	3-THREE THROW ZINC EMULSION, PUMPS.
105	3-36 CHAMBER 36" SQ PRECIP PRESSES
106	LEA-RECORDER
107	1-12 CHAMBER 25" SQ CLEAN-UP PRESS
108	VACUUM RECEIVER 34" DIA x 72".
109	1-4" SUBMERGED EXTRACTION PUMP
110	3-WASHING BOXES FOR CORDUROY
111	CONCENTRATES BIN
112	WILFLEY TABLE
113	2-AMALGAM BARRELS
114	BATEA
115	AMALGAM PRESS
116	AMALGAM RETORT FURNACE.
117	COPPER PLATE
118	GUARD STRAKE
119	1-2" WILFLEY PUMP
120	CALCINING FURNACE
121	2-TILTING RETORT FURNACES
122	BULLION FURNACE
123	VAULT

Fig. 1—ORIGINAL FLOW-SHEET FOR TREATMENT OF 60,000 TONS OF ORE PER MONTH

he principal items of equipment. The original layout, it will be noted, included straking for recovery of free gold, the nstallation being set in a strake house 315 ft long by 22 ft wide (the central bay of which is 70 ft long by 30 ft wide) and in this were accommodated the strake washing boxes.

The strakes were each 10 ft by 1 ft 6 in wide, arranged in eight banks of sixteen strakes each. With four primary and four secondary 8 ft diameter by 6 ft ball-mills installed, one bank of strakes was available for the treatment of the discharge from each ball-mill.

As originally operated Pepe weathered ore was also straked, but there have been changes in Pepe ore treatment, one of which has been the discontinuation of straking. Recovery of gold from the main bulk of this weathered low-grade ore is now effected by cyanidation only.

One European and twelve Africans per shift were required for supervision and changing and washing of strakes. The installation operated efficiently to recover 52 per cent of the total gold from underground ore treated in the plant.

Originally the stored strake concentrates were reconcentrated on the day shift on the Wilfley table, from which a deep-cut concentrate was removed for working up in the amalgam barrels. Later amalgamation was discontinued and a novel procedure was adopted, accumulated Wilfley table concentrate being fed to two small flotation machines with additions of xanthate and pine oil. By this means a clean gold concentrate suitable for drying and direct smelting was floated in a froth of unique beauty, recoveries closely approximating those by amalgamation. Such a procedure was practicable due to the average fineness of the free gold and to the absence of sulphide minerals from the original ore.

Set out in Table I is a metallurgical balance compiled from recorded data for the nine months of operation immediately prior to the suspension of milling at this plant in March, 1943, under the Concentration Scheme, a war-time arrangement whereby, with the supplies of diesel oil and other stores restricted by the emergency and Government action, certain mines suspended operation in order that the others might continue to operate at their normal rates of production.

This period of operation has been chosen for present purposes as, during these nine months, the plant was milling underground ore only, treatment of Pepe ore having been suspended. It therefore affords data on straking and amalgamation comparable with current concentrating practice, which is being applied to the whole of the deep mine ore *plus* a proportion (15 to 18 per cent of the total) of low-grade coarse ore from the Pepe workings.

THE SECURITY PROBLEM

Metallurgically the concentrating layout and operations briefly described yielded very good results. The snag was the temptation offered to gold thieves and the

TABLE I

METALLURGICAL BALANCE PRACTISING STRAKING AND AMALGAMATION

Product	Weight per cent	Dwt per ton	Per cent of total gold
Ore feed	100·0	4·417	100·0
Recovery by straking and amalgamation	100·0	2·297	52·0
Recovery by cyanidation	100·0	1·853	42·0
Strake tailing ...	100·0	2·120	48·0
Sand to cyanidation	48·6	2·730	30·0
Slime to cyanidation	51·4	1·543	18·0
Sand residue—total...	48·6	0·329	3·6
Slime residue—total	51·4	0·209	2·4
Total residue ...	100·0	0·267	6·0
Total recovery ...	100·0	4·150	94·0

GRADING ANALYSES (I.M.M.)

Product		Per cent
Sand residue:	+100	67·3
	−100+200	14·3
	−200	18·4
Slime residue:	+100	5·5
	−100+200	11·1
	−200	83·4

depressing prison-like conditions for the operating shift in the locked strake house.

As its name might imply the Gold Coast has an historical association with the winning and working of gold which dates back some centuries before the arrival on the scene of the European miner. There is in the country an ancient, widely developed and skilled guild of African goldsmiths working under Government licence. Gold ornament figures prominently among the insignia of chiefs and in the adornment of African women and children. International borders are not so very far distant from most centres of gold production and the dense bush and forest preclude close control. Minor gold occurrences are fairly frequent apart from the major occurrences being worked by European enterprise and Africans still win gold with the traditional calabash and more recently acquired prospector's pan, the adept panners being the women. Legislation regarding the possession of unwrought gold is nothing like so restrictive as in other comparable fields.

Thus it will be appreciated that there is probably no other field in Africa in which the local population in general is so gold conscious and in which the identity of illicit gold can be so simply and completely lost. Analyses of detected thefts of gold at a majority of the producing mines showed that over 90 per cent of the surface thefts and a considerably higher proportion of the value, were of gold in concentrate. Examination of the records summarized in Table I disclosed that under-recoveries of gold by straking and amalgamation were reported in seven of the nine months, two months only reflecting small over-recoveries. Based on the head sample assay of the ore entering the plant the indicated shortage on gold call over the whole plant for the period was 2·8 per cent of gold received in ore.

Since the war, losses from plants due to theft have been reduced to minor proportions by the more thorough organization of security measures and the exercise of unremitting vigilance. Apart from these preventive measures the author has felt that the surer safeguard and the greater peace of mind would result from modifications of metallurgical practice by which high-grade intermediary products in process were kept out of sight and reach of the potential thief. The problem is not peculiar to this field, although the complexities which attend it possibly are.

Such then is the background against which a variation from the orthodox and metallurgically efficient but, in other respects, vulnerable and labour-consuming practice of straking and amalgamation is to be viewed.

PRESENT PRACTICE—MECHANICAL CONCENTRATION

The alternative adopted was to use continuous discharge Pan American jigs in the primary mill-classifier circuit and the comparatively uncommon and, it is to be feared, neglected Johnson concentrator, in the secondary mill-classifier circuit. Both are machines of low capital and operating cost. The savings on labour alone in one year of operation were sufficient to meet the cost of installation of jigs and concentrators. Further, the installation required has been comfortably accommodated with adequate room for extension in a portion of the central 70 ft by 30 ft bay of the strake house. For a similar layout installed *ab initio* a capital economy in building accommodation over that required for straking is thus implied.

Fig. 2 shows the original strake installation in plan and section, and in Fig. 3 the new jig-Johnson concentrator installation is set out in plan and section. The unit illustrated in Fig. 3—i.e. two primary ball-mills, two jigs, one secondary ball-mill, and two Johnson concentrators, is treating a feed of *minus* ½ inch banket ore at the rate of 900 tons per 24 hours.

The flow-sheet to which the concentrating section is now operating is set out in Fig. 4.

The jig and Johnson concentrates discharge via pipes through the concrete floor to the concentrate storage set in the gold room below. The concentrates are out of sight and reach and are never handled. The jigs and Johnson concentrators operate behind a locked door, the machines being visited periodically during shifts by the European mill shift bosses. The stored concentrates from both machines are re-concentrated on the morning shift by passage over a Wilfley table, from which a clean gold concentrate suitable for direct smelting is recovered. Wilfley tailings pass over two strakes each 5 ft by 1 ft 9 in wide

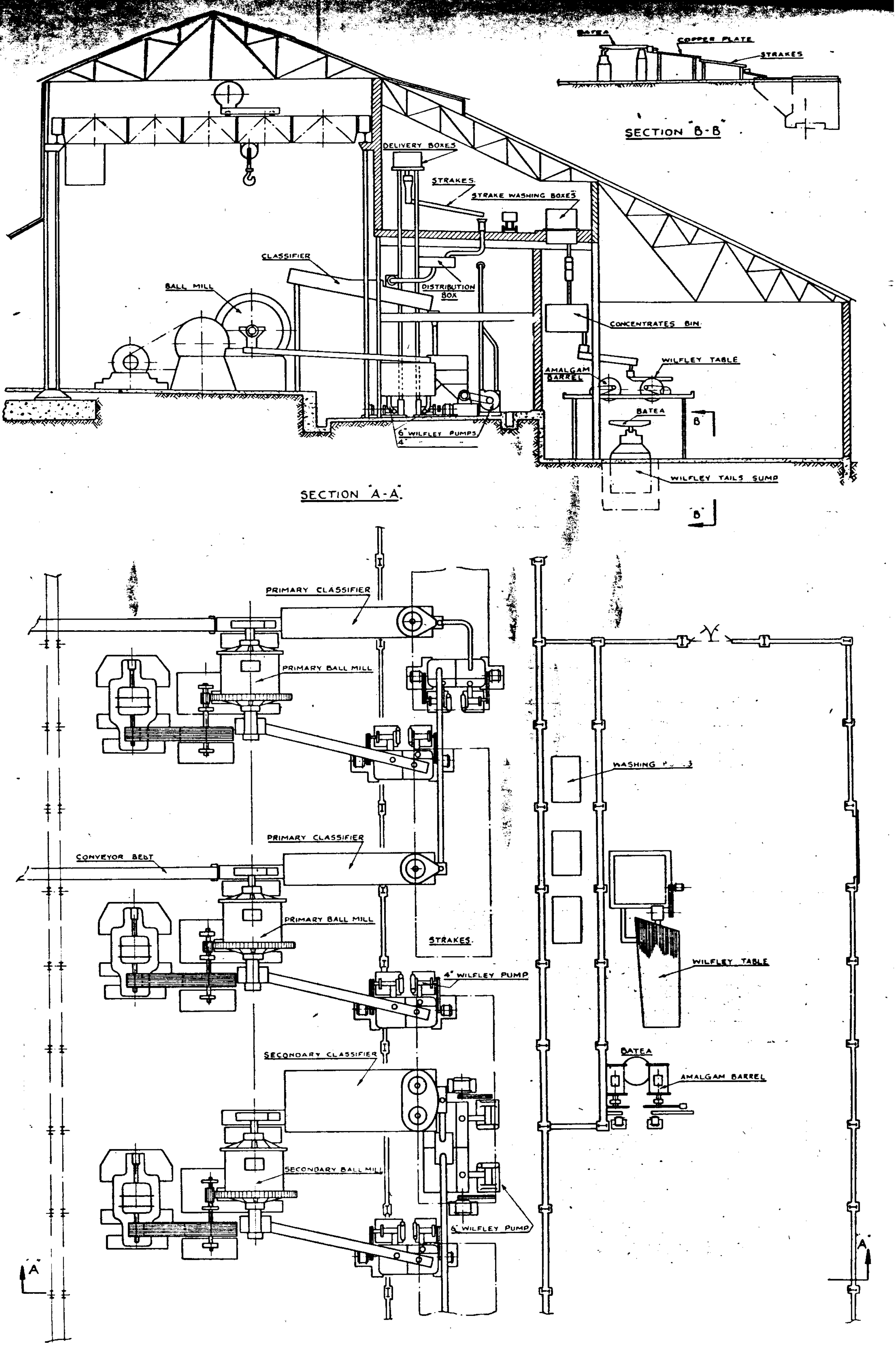
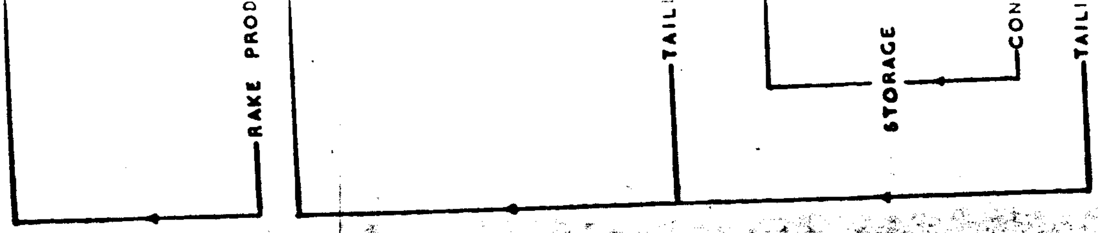


Fig. 2—ORIGINAL MILLING AND STRAKE INSTALLATION—SECTION AND PART PLAN



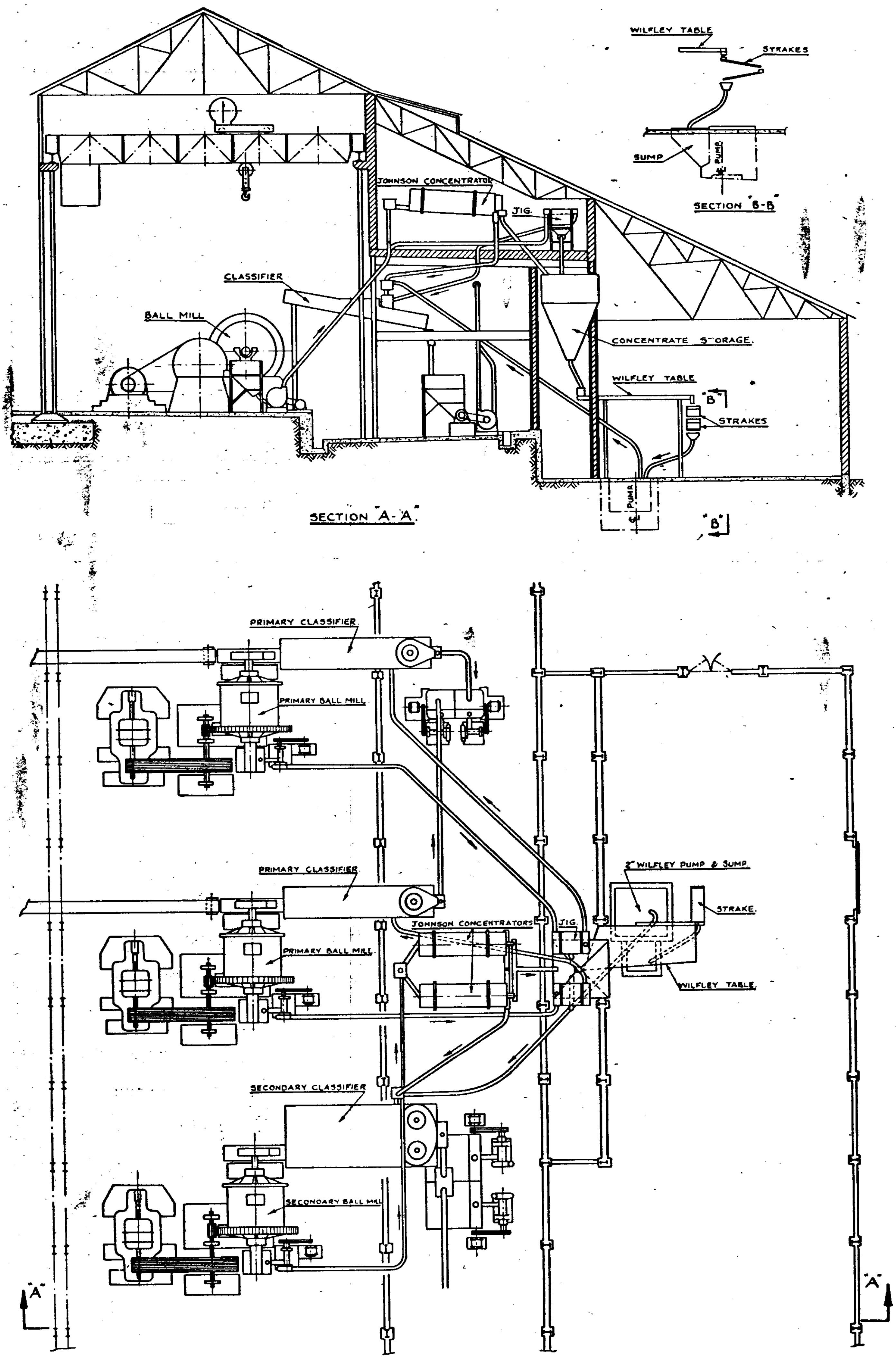


Fig. 3—NEW MILLING AND CONCENTRATOR INSTALLATION—PLAN AND SECTION

strake concentrate, very small in bulk, is stored in the strong room and twice a month is separately reconcentrated on the Wilfley table.

The installation operates smoothly with the minimum of attention. With the jigs operating alone and no concentration step in the secondary mill circuit the recovery of free gold into bullion averages 25 per cent of the gold in ore, the finer gold escaping to the cyanide plant. With the Johnson concentrators also operating in the secondary mill circuit, gold recovery is averaging 47·8 per cent made up as follows :

	Per cent
Recovery from jigs	25·0
Recovery from Johnson concentrators (30·4 per cent of 75 per cent)	22·8
	47·8

TABLE II

Metallurgical Balance Practising Concentration

Product	Weight per cent	Dwt per ton	Per cent of total gold
Ore feed	100·0	4·117	100·0
Recovery by concentration	100·0	1·967	47·8
Recovery by cyanidation	100·0	2·041	49·6
Concentration tailing	100·0	2·150	52·2
Sand to cyanidation	44·4	2·849	30·7
Slime to cyanidation	55·6	1·592	21·5
Sand residue—total...	44·4	0·149	1·6
Slime residue—total	55·6	0·077	1·0
Total residue ...	100·0	0·109	2·6
Total recovery ...	100·0	4·008	97·4

Grading Analyses (I.M.M.)

Product	Per cent
Sand residue : +100	61·9
−100+200	22·6
−200	15·5
Slime residue : +100	3·9
−100+200	8·3
−200	87·8

the new concentrating installation are set out in Table II, which is compiled from current operating records.* The results in this Table are more or less directly comparable with those set out in Table I. There are qualifications : First, the ore feed to the section was comprised of 83·5 per cent of underground ore averaging 4·554 dwt/ton and 16·5 per cent of weathered coarse ore from Pepe averaging 1·906 dwt/ton. The concentratable gold content of the weathered ore is low, being of the order 10 to 15 per cent. Further, only 83 per cent of the ore milled passed the jigs, the remaining 17 per cent having been milled in a third primary ball-mill which is not at the moment in circuit with the jigs. The product from the secondary stage of grinding of the 17 per cent did pass to the Johnson concentrators.

There have been improvements in cyanidation practice which, while falling outside the scope of this paper, are to be borne in mind when comparing total recoveries in Tables I and II.

By taking a larger concentrate from the Wilfley table and working this up by amalgamation, gold recovery by concentration no doubt could be improved. Also by increasing the amount of concentrate drawn from the Johnson concentrators (see Table III), and running the Wilfley table for a longer period, or by installing a second Wilfley table, recoveries by concentration could be raised. In view of the performance of the cyanide plant on the current concentration tailing neither of these courses has been considered necessary. It is of interest to add that the gold call on the plant is now being regularly and satisfactorily met.

Johnson Concentrator Recoveries

The Pan American jigs (36 in by 36 in) are standard equipment, familiar in dredge operation. The Johnson concentrator is probably less familiar to many members. This machine was described† by that doyen of Rand metallurgists, Mr E. H. Johnson, by whom the concentrator was developed. For convenient reference a drawing of the machine is included in Fig. 5.

* February and March, 1949

† Concentration and selective regrinding. *J. Chem. Soc. S. Afr.*, 27th April, 1927, 215.

Johnson Concentration Recoveries Under Varying Conditions

Test No.	1	2	3	4	5	6
Drum, r.p.m.	7	5	7	5	5	7
Per cent moisture in feed	35	33	35	30·5	40	34
Feed to concentrators—						
* Tons per concentrator/24 hr ...	283	324	250	448	303	332
Dwt/ton	5·60	4·67	3·39	3·81	4·61	3·25
Concentrate—						
Tons per 24 hr	9·04	8·79	4·88	4·70	2·27	2·21
Weight—per cent of feed	3·20	2·72	1·95	1·05	0·75	0·67
Dwt/ton	94·8	79·5	74·6	129·4	204·2	136·8
Recovery—dwt/ton of feed	3·03	2·16	1·45	1·36	1·53	0·92
Amalgamation of concentrate—						
Tailing—dwt/ton	10·2	10·9	6·6	4·8	8·8	8·2
Tailing—dwt/ton of feed	0·33	0·30	0·13	0·05	0·07	0·05
Recovery—dwt/ton of feed	2·70	1·86	1·32	1·31	1·46	0·87
Concentrator tailing—						
Weight—per cent feed	96·8	97·28	98·05	98·95	99·25	99·33
Dwt/ton	2·65	2·58	1·98	2·48	3·10	2·35
Dwt/ton of feed	2·57	2·51	1·94	2·45	3·08	2·33
Recoveries—						
(1) In Johnson concentrate : per cent gold in feed	54·1	46·2	42·9	35·6	33·2	28·0
(2) By amalgamation of concentrate : per cent gold in concentrate ...	89·2	86·3	91·1	96·3	95·7	94·0
(3) By amalgamation of concentrate : per cent gold in feed	48·2	39·8	39·1	34·3	31·7	26·3

* Includes circulating load.

TABLE IV

Grading Analyses (Tyler)* of Products from Johnson Concentrator Tests (Table III)

Test No.	1	2	3	4	5	6
	Per cent	Per cent	Per cent	Per cent	Per cent	Per cent
Feed—						
+ 48 mesh	10·6	13·0	11·1	10·2	11·3	10·2
− 48+100 mesh	30·2	34·7	29·3	31·0	31·5	31·0
−100+200 mesh	23·3	25·0	23·2	25·7	23·6	25·7
−200 mesh	35·9	27·3	36·4	33·1	33·6	33·1
Concentrate—						
+ 48 mesh	2·1	2·3	3·3	2·8	2·7	2·9
− 48+100 mesh	17·5	19·0	16·6	21·5	15·7	12·6
−100+200 mesh	45·6	45·7	42·9	43·6	46·5	41·8
−200 mesh	34·8	33·0	37·2	32·1	35·1	42·7
Tailing—						
+ 48 mesh	10·4	12·7	10·9	10·0	11·8	10·0
− 48+100 mesh	30·8	34·2	30·5	30·7	29·7	30·7
−100+200 mesh	23·2	25·6	23·0	26·1	23·4	26·1
−200 mesh	35·6	27·5	35·6	33·2	35·1	33·2

* An explanation of the appearance of a second grading series is necessary. Plant gradings have hitherto been conducted with I.M.M. sieves, but are shortly to be changed to the British Standard which closely approximates the Tyler series. At the West African Gold Corporation's laboratory the Tyler series is used.

In Table III the results from a series of controlled tests, in which conditions of operation of the Johnson concentrators were studied, are set out. Rotation was varied at speeds of 5 and 7 r.p.m. No appreciable difference in performance was noted. For normal operation the slower speed has been adopted. The percentage weight of concentrate was controlled by varying the amount of water used in the side sprays. Gold recovery was found to vary directly with the amount of concentrate made. A further variable possible with the machine operating on finer feeds is the inclination of the rotary drum. This has not yet been studied.

During the period of each of the tests recorded in Table III it is to be noted that Wilfley table tailings were not being returned to circuit. Each of the tests was of about three hours' duration. Concentrate recovered from each test was carefully riffled down to a sample of approximately 5,000 g, which was amalgamated without grinding. The concentrate values reported are built up from the gold recovered in amalgam *plus* gold found in the amalgamation tailing. It will be noted that an average of 90 per cent of the gold recovered in concentrate was amalgamable. Moreover, the assays of the tailings from amalgamation reflect but a minor lock-up of gold in haematite.

The grading analyses of products to and from the Johnson concentrator are set out in Table IV.

There is little doubt that the jigs could be excluded from the concentrating circuit and that with the Johnson concentrators operating alone on the secondary ball-mill discharge an equally satisfactory overall recovery of gold from the ore would be possible. There would almost certainly result a build-up in gold content of the rake product from the primary simplex classifiers which would almost as surely attract attention from the receivers' metallurgical scouts. It has been considered advisable to retain the jigs on their duty of recovering the coarser fraction of what is, on the average, very fine free gold.

The operation of the Johnson concentrator differs in several respects from South African practice. As described by E. H. Johnson, this machine operated in a single-stage tube-milling circuit, in which the size range of particles in the feed to the concentrator would be considerable. It was thus required to handle the total tonnage of ore under treatment including the circulating load. The operation of this machine on the reduced tonnage and finer sizings of a secondary milling circuit may be new. Its efficiency on these sizings has been impressive.

ENCASEMENT OF GOLD IN ORE MINERALS

On the Witwatersrand there is also an important association of gold enclosed in pyrite, and concentration of pyrite for separate selective regrinding and exposure of encased gold in preparation for cyanidation was an important function of the machine as described by E. H. Johnson. The author's investigations have not disclosed any such intimate association of gold with the haematite of the Gold Coast banket. At Amalgamated Banket Areas the first intention was to return Wilfley table tailings to the primary ball-mill feed, but in the light of experience it has been found sufficient for treatment requirements to return the Wilfley tailings to the secondary classifier so that haematite trapped in the concentrating section retraverses the secondary stage of classification and grinding until size reduction has progressed to the point at which it escapes in the secondary classifier overflow.

The following examination of a representative sample of current Johnson concentrate illustrates the negligible encasement of gold in haematite :—

Johnson concentrate—22·1 oz Au per ton.
Grading Analysis (Tyler) :

+ 48 :	5·1 per cent
− 48 + 100 :	16·4 per cent
− 100 + 200 :	28·2 per cent
− 200 :	50·3 per cent

The concentrate was amalgamated without grinding and yielded :—

By amalgamation	: 21·9 oz Au per ton con.
Amalgamation tailing	: 4·0 dwt per ton.

The amalgamation tailing was ground in the laboratory ball-mill and re-amalgamated with the following results :—

By amalgamation	3·0 dwt Au per ton concentrate
Final amalgamation tailing	1·0 dwt per ton.

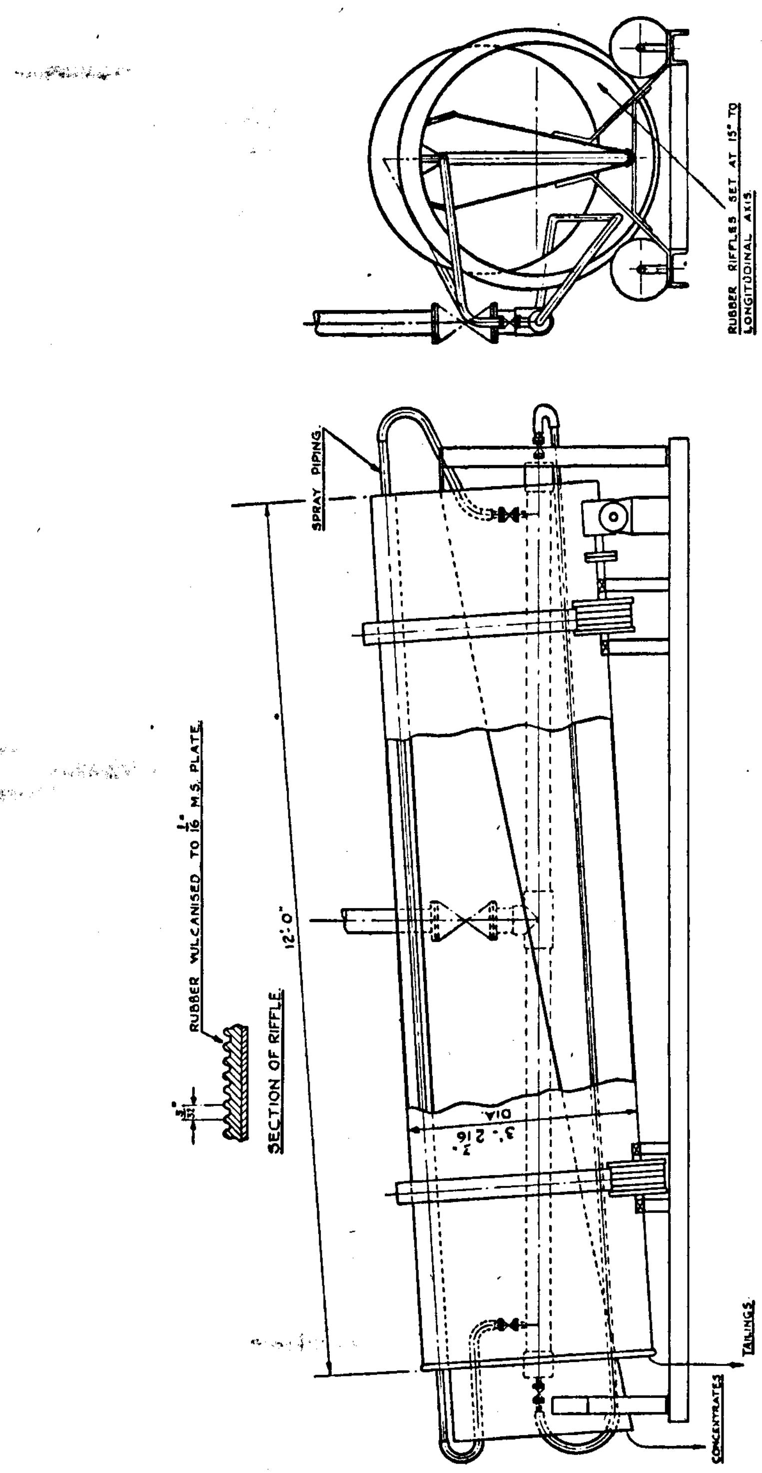


Fig. 5 JOHNSON CONCENTRATOR

TABLE V

Product	Slime	Flotation: Feed	Flotation: Concentrate	Flotation: Middling	Flotation: Tailing
Weight—per cent of original	24·7	75·3	30·5	33·7	11·1
Weight—per cent of flotation feed	—	100·0	40·5	44·8	14·7
Gradings (Tyler)—					
+ 48 per cent	—	0·1	—	—	0·7
− 48+100 per cent	0·1	2·7	0·9	1·6	11·0
−100+200 per cent	1·9	29·7	16·6	28·7	68·9
−200 per cent	98·0	67·5	82·5	69·7	19·4
Partial Analyses—					
Au, dwt/ton	0·77	1·10	1·15	1·15	0·77
Fe_2O_3 per cent	10·3	30·1	43·5	25·2	5·7
Magnetics, per cent	Nil	4·0	5·4	5·2	4·0
Insoluble, per cent	85·0	63·3	47·8	67·1	89·1
Undetermined, per cent	4·7	2·6	3·3	2·5	1·2

TABLE VI

Cyanidation of Mantraim Ore by Agitation

Ground to 65·6 per cent *minus* 200-mesh (Tyler)

Cyanided To cyanidation—dwt/ton	Directly 4·87			After amalgamation 1·08		
Agitation—hours	4	8	16	4	8	16
(A) Cyanided at NaCN per cent * ...	·026	·026	·027	·030	·030	·030
Washed residue, dwt/ton	0·86	0·25	0·10	0·67	0·23	0·12
Total extraction, per cent ...	82·3	94·9	97·9	86·2	95·3	97·5
(B) Cyanided at NaCN, per cent ...	·047	·048	·047	·051	·051	·045
Washed residue, dwt/ton	0·52	0·22	0·10	0·30	0·17	0·10
Total extraction, per cent ...	89·3	95·5	97·9	93·8	96·5	97·9

* At end of agitation period.

TABLE VII

Cyanidation of Mantraim Ore by Agitation

Ground to 75·1 per cent *minus* 200-mesh (Tyler)

Cyanided To cyanidation—dwt/ton	Directly 4·40			After amalgamation 1·53		
Agitation—hours	4	8	16	4	8	16
(A) Cyanided at NaCN, per cent ...	·030	·031	·030	·028	·029	·025
Washed residue, dwt/ton	0·58	0·28	0·12	·042	0·10	0·12
Total extraction, per cent ...	86·8	93·6	97·3	90·5	97·7	97·3
(B) Cyanided at NaCN, per cent ...	·053	·053	·052	·050	·049	·048
Washed residue, dwt/ton	0·38	0·20	0·10	0·12	0·07	0·05
Total extraction, per cent ...	91·4	95·5	97·7	97·3	98·4	98·9

TABLE VIII

Cyanidation of Mantraim Ore by Agitation

Ground to 90·8 per cent *minus* 200-mesh (Tyler)

Cyanided To cyanidation—dwt/ton	Directly 4·63			After amalgamation 1·30		
Agitation—hours	4	8	16	4	8	16
Cyanided at NaCN, per cent	·031	·029	·029	·029	·027	·029
Washed residue, dwt/ton	0·85	0·45	0·12	0·11	0·10	0·06
Total extraction, per cent	81·6	90·3	97·4	97·6	97·8	98·7

TABLE IX

Cyanidation of Abbontiakoon Ore by Agitation

Ground to 66·4 per cent *minus* 200-mesh (Tyler)

Cyanided To cyanidation—dwt/ton	Directly 5·55			After amalgamation 1·67		
Agitation—hours	4	8	16	4	8	16
(A) Cyanided at NaCN, per cent ...	·026	·027	·026	·027	·027	·026
Washed residue, dwt/ton	1·83	0·57	0·13	0·27	0·20	0·15
Total extraction, per cent ...	67·0	89·7	97·7	95·1	96·4	97·3
(B) Cyanided at NaCN, per cent ...	·046	·045	·044	·044	·043	·044
Washed residue, dwt/ton	0·80	0·37	0·12	0·20	0·18	0·15
Total extraction, per cent ...	85·6	93·3	97·8	96·4	96·8	97·3

TABLE X

Cyanidation of Abbontiakoon Ore by Agitation

Ground to 74·6 per cent *minus* 200-mesh (Tyler)

Cyanided To cyanidation—dwt/ton	Directly 5·23			After amalgamation 1·88		
Agitation—hours	4	8	16	4	8	16
(A) Cyanided at NaCN, per cent ...	·027	·027	·028	·027	·027	·026
Washed residue, dwt/ton	1·78	0·48	0·23	0·35	0·13	0·10
Total extraction, per cent ...	66·0	90·8	95·6	93·3	97·5	98·1
(B) Cyanided at NaCN, per cent ...	0·046	0·045	0·044	0·046	0·045	0·045
Washed residue, dwt/ton	1·22	0·30	0·20	0·22	0·13	0·07
Total extraction, per cent ...	76·7	94·3	96·2	95·8	97·5	98·7

The final tailing from amalgamation was deslimed and the sand fraction was floated with Reagent 801 to effect a concentration of haematite. The products from flotation were a concentrate, middling and tailing.

In Table V are set out the percentage weights of the several products—their gradings, gold content, and partial analyses.

The magnetics were composed mainly of metallic iron. The concentration of haematite effected was not of a high order but sufficient to provide a comparison of the relationships between the Fe_2O_3 and gold contents of the several products. The evidence is that the encasement of gold in haematite is insignificant.

'ALL-SLIMING' AS AN ALTERNATIVE

Having described the new concentrating layout the author is still aware that there are two schools of thought on the subject of concentration of gold from banket ores, the second of which poses the provocative query 'Why bother?'

The first argues that by concentration a high proportion of the total gold is earlier in the bank, the cyanide plant is relieved of the duty of dissolution of all but the finest gold, treatment time in the cyanide plant may be shortened and as a corollary less equipment capacity will be required, and residue washing will not be attended by the same risks of dissolved gold loss. These considerations assume greater importance in the plant cyaniding a portion of the mill product as sand by leaching.

The second school is confined to advocates of the one-pulp plant, in which grinding is carried far enough to enable the whole of the ore to be cyanided by agitation and filtration or counter-current decantation. This school argues that with the mill-classifier circuit suitably arranged the gold entering the cyanide plant will have been sufficiently reduced in size for effective recovery in reasonable time by cyanidation alone—although it is to be remarked that this time is likely to be longer than if a concentration step were included. Further, the result is a simplified flow-sheet from which the processes and labour of working up a gold concentrate are eliminated. Also, in consequence, this type of plant does not present the temptation of or opportunity for gold theft that are present when the handling of a high-grade concentrate, often containing visible gold, is a routine shift duty. (In parenthesis one must enter a caveat regarding the classifier sands in this type of plant.)

Two 'all-sliming' plants have operated on the Gold Coast banket with no concentration step and one continues to operate, metallurgical extractions being commendably high. In Tables VI, VII, VIII, IX, and X are set out the results from a series of tests conducted on Amalgamated Banket Areas deep mine banket ores. These results, with due allowance for advantages in favour of laboratory tests, are to be compared with those from current plant operation (Table II). The 'After Amalgamation' tests were conducted on pulp which had been subjected to bottle amalgamation. These are not strictly pertinent to the present discussion, but are included as being of interest and reflecting approximate maxima of recoverable free gold.

In the change from straking to mechanical concentration the installed cyanide plant was, of course, a governing consideration. Even so and in view of the fact that in current operation the grading of the combined sand *plus* slime is averaging 55 to 56 per cent *minus* 200-mesh I.M.M., Table II would seem to provide whatever further justification might be required of current practice.

ACKNOWLEDGMENTS

The author's thanks are due to the directors of Amalgamated Banket Areas, Ltd., for authorizing the publication of information contained in this paper; to the consulting engineers, the West African Gold Corporation, Ltd., on whose recommendation the change in concentrating practice was made and at whose laboratory research was conducted; to the mechanical engineering department of New Consolidated Goldfields, Ltd., for the preparation of the drawings, and to the general manager and colleagues on the plant and in the Corporation's laboratory for co-operation in establishing the new concentrating routine on a successful operating basis.

DISCUSSION

A. Clemes (Past President): I feel that this paper by Mr Chad Norris is opportune as far as metallurgical practice on the Witwatersrand is concerned, because with the present and anticipated shortage of native labour and increased risk of theft due to the higher price of gold, attention is being focussed on means of concentration which reduce both labour and risk.

Since this paper is a joint one with the Institution of Mining and Metallurgy, I may perhaps be excused if I open my contribution to discussion with a few notes on the practice of gold recovery here, since it may not generally be known away from South Africa.

Between 1918 and 1922 the use of amalgamated plates was displaced by corduroy to reduce lock-up of gold in the mill circuit, loss by theft and mercurial poisoning of workmen. Two Groups went further and dispensed with amalgamation altogether on two existing mines and those subsequently erected by them have also depended entirely on cyanide treatment for the total gold recovery.

The matter of whether a primary recovery of gold in the milling circuit by corduroy tables, or mechanical concentrators is worthwhile is still a matter of controversy. The plants which do not employ gravity concentration in the milling circuit avoid possible repercussions on residue values by the use of one, or possibly a combination of two, of the following practices.

(*a*) Fine grinding the whole of the ore to, say 80 per cent minus 200 mesh (Tyler) in two or three stages.

(*b*) Employing three-stage classification to ensure that coarse gold, or gold encased in pyrite, is retained in the milling circuit until it is in a condition to be readily attacked by cyanide.

(*c*) Milling in cyanide solution.

In general practice it appears that a compromise of a modest two stage grind to, say, better than 75 per cent minus 200 mesh, with ample classifier capacity, meets the case of preparing ore for direct cyaniding.

Under these conditions the average particle size of quartz leaving the milling circuit is under 5 microns (in the minus 200 mesh about 3 microns) and particles of free gold or "chats" are obviously of such small dimensions by reason of higher specific gravity. It is very problematical if any totally encased gold exists above one micron size, unless it is coated with some oxide or sulphide film produced during treatment, or inherent in the ore.

It appears to me that three-stage or "tandem" classification, claiming to give especial preferential grinding to the gold encased in pyrite, has no advantage over a "straight" classifier circuit of ample capacity, which should give the same results. In capital and maintenance costs there are advantages in having a few large classifiers in the final (secondary) circuit rather than a multiplicity of small and large machines.

Milling in cyanide is still retained on a few mines, but only to the extent of using any excess precipitated solution, or return from slimes dams, as make-up to the mill water circuit. Addition of cyanide to the mill circuit was early abandoned as it led to high cyanide consumption and under certain circumstances could be deleterious to extraction by reason of the formation of sulphocyanides, etc. especially in the presence of pyrrhotite, which is present in some ores of the Far East Rand.

As a case for concentration in the milling section of the ore treatment plant, which is still employed by mines milling over sixty percent of the tonnage crushed on the Rand, I would say that it performs equally with the practice of, so called, preferential classification and is slightly cheaper than fine grinding of all the ore. There are, of course, also the advantages that from 45 to 50 per cent of the "gold-call" is available daily in the form of amalgam and that there is some reduction in gold "locked-up" in tube mill liners and in classifier beds. The daily recovery of gold from concentration gives a more positive indication of grade of ore to mill than sampling alone, but it is improbable that lock-up of gold in plant is very materially lessened. If we assume a life of 200 days for a tube mill liner, it would appear that with, or without, removal of concentrate from the discharge it would arrive at the point of "saturation" of retaining gold concentrates within, say, the first 30 days.

How much gold a tube mill liner holds up is probably governed more by the type of liner used. To quote two extremes, a test on a 9 ft by 10 ft primary ball mill lined with manganese blocks and without corduroy in circuit resulted in a recovery of 20 ounces of gold. On another plant, 5 ft 6 in by 22 ft primary tube mills lined with Osborne bars and in circuit with corduroys, normally yield 300 ounces on relining and this can amount to 900 ounces if the liner is not a close fit.

I also feel that the final amount of gold locked up in a classifier depends more on the time it is in operation between changing or rakes than on the value of the feed.

To my mind the main justification of the use of concentration lies in two directions.

Firstly, on a mine treating high grade ore, say, seven dwt and over, or where the milling plant is loaded beyond its capacity and grinding is at times adversely affected. In both cases there can be periods when coarse or encased gold escapes to the cyanide plant with resultant deterioration in recovery. Concentration in the milling circuit largely circumvents this.

Secondly, the removal of concentrates in the milling circuit enables a reasonably high extraction of platinum group metals to be achieved. In the Group which I represent the net earnings from this source have totalled £43,000 over the past five years. At today's price for this material this is equivalent to about 0·30 pence per ton milled. Over our ten producing mines this does not cover the cost of corduroy extraction, but on about one-third of our mines it breaks about square—with the additional safeguard of protecting residue values.

Although the paper by Mr Chad Norris shows very clearly an appreciable gain in overall recovery by a change in the concentration circuit, I have found it rather difficult to assess how much the introduction of jigs and Johnson concentrators have assisted in reducing sand and slime residues by better than 50 per cent, thus improving overall recovery from 94·0 to 97·5 per cent.

From Tables I and II it appears that recovery by "straking" was 52·0 per cent, and fell to 47·8 per cent by jigging and concentrating in Johnson concentrators.

From the evidence submitted it appears that the improvement in metallurgical recovery can mainly be accounted for by :—

(a) an improvement in grinding from 51·8 to 55·7 per cent minus 200 mesh (calculated from Tables I and II)
(b) a reduction in ore value
(c) improvements in cyanide practice
(d) possibly to some extent a change in ore.

It will be noted from these two tabulations that under the new conditions the plus 100 mesh has decreased by about 6 per cent, which is not inconsiderable. In terms of Witwatersrand ore this tie-up between coarseness of grind and residue values would indicate encasement of gold, but the author has proved that this is not the case at Amalgamated Banket Areas. It appears, therefore, that the effect of finer grindings is to prevent coarse gold escaping to the cyanide circuit.

This is, however, not entirely borne out by the test work on Mantraim and Abbontiakoon ores treated as a single pulp by agitation, since improved grinding from 66 per cent minus 200 mesh to 75 per cent minus 200 mesh (and in the case of Mantriam to 90·8 per cent) did not increase recovery unless amalgamation was included in the tests. The author's view on this particular aspect would be appreciated.

In the direction of improving security, although conditions on the Gold Coast are for various reasons not so good as on the Witwatersrand, we are becoming more conscious of the increasing danger from theft. On those of our plants which employ blanket strakes the danger of organized theft of partially finished material (high grade concentrate, or amalgam) is obviously high and requires special precautionary measures as normally applied to the cyanide gold recovery section and smelt house. Against this plants employing direct cyanidation expose more low grade concentrates in the various portions of the milling circuit where these accumulate and more petty thieving may result.

A good many of us on the Rand who operate plants embodying blanket strakes, are today turning our attention to forms of concentration which will both save labour and reduce the possibility of loss by theft. There are numerous devices to be investigated, each of which has its advantages and disadvantages in the following respects :—

(a) capital cost
(b) maintenance cost

(c) security
(d) saving in labour
(e) applicability to existing plants
(f) control of operating conditions in the milling circuit, i.e. control of moisture
(g) efficiency

Whilst I may be criticized for placing efficiency last, I do not consider that a drop of recovery by amalgamation from, say 50 to 45 per cent will on a well equipped plant result in a higher residue. In my Group we abandoned corduroys in primary mill circuits of new plants in 1936 with a drop in recovery from about 60 to 50 per cent. More recently we have changed the type of "blanket" used with a further drop of about 3 per cent. Overall recovery has not suffered and I, personally, would not be perturbed if amalgamation recovery fell to, say 40 per cent, except on a high grade mine offering peculiar metallurgical difficulties.

Although the paper by Mr Chad Norris claims by its title to refer mainly to the practices of concentration employed by Amalgamated Banket Areas Ltd., it has a very much wider scope than this in giving us a very complete and most interesting description of the nature of the ore treated and the type of plant used to obtain an extraction which must be most gratifying to the management. In addition a careful study of the paper brings to light quite a few aspects of metallurgy which at first glance may appear of less direct interest. For example, I was particularly intrigued with the idea of the application of flotation concentration to the product of the Wilfley table.

I feel that Mr Chad Norris is to be warmly congratulated for his very worthwhile paper.

N. B. Locke (Member): Mr Chad Norris is to be congratulated on his interesting and informative paper on the subject of gold concentration at the Amalgamated Banket Areas, which was published by the Institution of Mining and Metallurgy, and read to this Society, in his absence, by Mr H. E. Cross.

The paper offers very little scope for criticism for it appears that the author was not faced with any metallurgical problem, but with the circumvention of the theft of gold concentrates, in a country in which legislation concerning illicit gold traffic is less restrictive than in other comparable fields, and in which the identity of illicit gold can be lost with comparative ease.

If, during the nine months when straking was being practised, the shortfall of gold was 2·8 per cent of the gold received in the ore, the discrepancy, based on a head value of 4·417 dwt per ton, and 60,000 tons milled per month, would amount to 371 ounces fine gold per month, worth £4,605 at the current price of gold.

Following upon the change in metallurgical practice and improvement in cyanidation, Mr Chad Norris states that the gold call on the plant is now being regularly and satisfactorily met. It might be inferred therefore, that as the result of installing mechanical concentrators, the rich products from which are now out of sight and reach of potential thieves, the former loss of 2·8 per cent of the gold in the ore delivered to the plant was due to theft.

In the absence of data as to the gold content of the strake concentrate formerly recovered at Amalgamated Banket Areas, it might be of interest to apply the figure of 143 ounces gold per ton, which is the gold content of strake concentrates obtained at the Angelo Plant, East Rand Proprietary Mines, Ltd., to arrive at the quantity of concentrates that was being stolen each month. It would require 2·6 tons of concentrate, at 143 ounces per ton, to satisfy a loss of 2·8 per cent of the gold delivered to the plant. This is a large amount of concentrate to lose without detection, month after month, especially from a locked strake house. However, richer products, such as Wilfley table concentrates, may have been the object of the thieves' attention. I mention concentrates specifically, because it is stated that 90 per cent of the detected surface thefts were of gold concentrates. I am of the opinion that discrepancies in tonnage determinations, assaying and sampling, were probably contributing factors to the shortfall.

Nevertheless, to obtain peace of mind, Mr Chad Norris was faced with the alternative of cyaniding all of the pulp without the concentration step, or of introducing some other means to safeguard the concentrates if he wished to retain the step in his flowsheet. He chose the latter course, and replaced the metallurgically efficient, but vulnerable strakes, by Pan American jigs and Johnson concentrators. Thus, the recovery of gold by concentration declined from 52·2 per cent to 47·8 per cent.

As the plant is operating to produce sand and slime for cyanidation, I am of the opinion that by retaining the concentration step in his flow-sheet, Mr Chad Norris chose wisely. Had he eliminated the concentration step entirely, it is probable that he would have been faced with costs far in excess of those entailed by its retention, in an altered form, by additions to the cyanide plant, for, it may have been necessary to introduce "all sliming," with the installation of additional thickening and filtering equipment, to achieve comparable results.

As it was, a considerable labour force was dispensed with, the savings from which, alone, were sufficient to redeem the capital cost of the installation in one year. Furthermore, the results of the laboratory cyanidation tests carried out on Mantraim and Abbontiakoon ores directly, and after amalgamation, disclose that extractions are slightly in favour of cyanidation after amalgamation and, on this basis, the retention of the concentration step appears justified.

Incidentally, some other interesting points emerge from the laboratory tests, the results of which are shown in Tables VI–X in the paper. The finer grinding of the ores above 66 per cent minus 200 mesh (Tyler) does not have the marked beneficial effect on the per cent extraction that one would expect. The agitation time required for the complete dissolution of the gold seems to be at least 16 hours; and higher cyanide strengths also appear to be beneficial.

The location of the Johnson concentrators in the secondary circuit at the Amalgamated Banket Areas is novel. At the three reduction plants of the Central Mining—Rand Mines group, in which Johnson concentrators are installed, relatively coarse pulp from the primary tube mills is fed to the machines.

At this point in the discussion it might be appropriate and of interest to members, to disclose the results of comparative tests carried out on four Johnson concentrators, and a 16 inch by 24 inch Denver mineral jig, towards the end of 1937, at the Cason Plant, East Rand Proprietary Mines, Ltd.

The tests indicated that, when the weight of concentrates produced by the jig and the concentrators was 10·45 per cent and 10·77 per cent respectively, of the feed, the extraction by the jig was 42 per cent of the gold and 35 per cent of the pyrite, compared with 47 per cent of the gold and 42 per cent of the pyrite by the concentrators.

The jig concentrate was definitely coarser than the Johnson concentrate (82 per cent plus 100 mesh compared with 49 per cent plus 100 mesh) and of the total gold in the jig concentrate, 35 per cent occurred in the plus 100 mesh portion; whereas only 28 per cent appeared in the plus 100 mesh portion of the Johnson concentrates. These facts indicate a favourable selectivity by the jig in recovering that portion of the gold and pyrite most in need of intensive regrinding.

Thus, in using the jigs in the primary circuit, and the Johnson concentrators in the secondary circuit, Mr Chad Norris has disposed his machines with discernment. Other tests carried out at the Cason Plant, to determine the suitability of the jig to replace the corduroy strakes over which the Johnson concentrates pass, before entering the regrinding circuit, indicated that the jig was not as effective as corduroy for the purpose of collecting free gold in small bulk for amalgamation.

Although the Johnson concentrator was originally designed to withdraw the pyritic portion of the ore for intensive regrinding to liberate the encased gold, the concentrates recovered by the jigs and Johnson concentrators at the Amalgamated Banket Areas are not reground prior to concentration on the Wilfley table, although the Wilfley tailings are reground in the secondary circuit. Had Mr Chad Norris been faced with the necessity of installing, and maintaining, a separate grinding and classification circuit to handle the products from the jigs and concentrators, the additional expenditure that would have been entailed may not have made the scheme so attractive.

Mr Chad Norris does not supply any mechanical details of his concentrator layout, so the following brief notes may be of interest.

As originally installed at the East Rand Proprietary Mines, Ltd., the Johnson concentrators were direct driven from a lineshaft by crossed driving belts over the concentrator drums. Subsequently, this method was modified, and the concentrators were driven from a lineshaft through bevel gears and pinions fitted to one of the supporting rollers on each concentrator.

Experience has shown that the pinions are prone to rapid wear, and destruction, when wear takes place on the rollers, and a considerable amount of replacement is required. Loss of running time also seriously affects gold recovery. Recently, at the Consolidated Main Reef Mines & Estate, Ltd., some concentrators have been equipped with independent direct motor drives, through speed reduction gears, onto one of the supporting rollers. They are functioning satisfactorily.

At the Cason Plant, the lineshaft has been retained, and a reduction gear is driven through fast and loose pulleys therefrom. The concentrator roller is driven from the reduction gear by Vee belts. This method has merit in that it is flexible and any wear on the bearings of the driving roller does not stress the couplings of the gear box assembly. Furthermore, it is very simple to alter the speed of the machine, should this become desirable, by altering the pulley diameters.

The de-sliming or side sprays, are important for controlling the amount and cleanliness of the concentrates recovered by a Johnson concentrator. At the Cason Plant, the de-sliming spray pipes have been divided into two sections by means of a plug inserted halfway along the length of the pipe, and the amount of spray can thus be closely controlled over the feed and discharge sections of the concentrators, respectively.

The cost of maintenance of a Johnson concentrator, excluding labour, is approximately £3 per month.

THE USE OF THE WET KATA THERMOMETER ON THE WITWATERSRAND

By P. H. Kitto (Visitor)

Published in *Journal*, July, 1949

Discussion

J. P. Rees (Member): I am very pleased that the Society has published this paper by Mr Kitto describing his tests on the kata thermometer.

The kata has been and is widely used on the Witwatersrand because it is the most suitable instrument available for obtaining directly the combined effect of heat and air movement on an instrument. Yet it is in a few respects not a very suitable instrument for use in mines. It is fragile and the operations involved in heating it are not convenient. It could be improved, I think, and ways of improving it have been tentatively discussed and planned by Mr Kitto and the Dust and Ventilation Department of the Transvaal Chamber of Mines. However, before attempting to go ahead with any improvements, the value of the instrument has to be carefully considered and this is especially necessary because the kata thermometer has been severely criticized by various authorities. The criticism is chiefly directed to the question of whether the cooling power of the air measured by the kata thermometer is a reliable guide to the cooling power of the air on human beings. On this question very little information has come forward on the Witwatersrand from the mining side, that is whether in stopes which have low kata readings the manpower required for a given output is higher than in stopes having a high kata reading all other conditions being as nearly as possible similar. The question can also be tackled from the experimental angle under the guidance and with the co-operation of the medical profession.

Mr Kitto has not tackled this question, he has however investigated the instrument from the physical side and building on the valuable work done by Mr Buist has shown