

FLOTATION TRIALS AT ROOIBERG TIN MINE
USING A PORTABLE COLUMN CELL

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ABSTRACT

This paper presents the results of column flotation plant trials that were conducted at the Rooiberg tin mine in the Northern Transvaal, using a portable test rig designed at the University of Cape Town. The work forms part of a preliminary investigation to determine the feasibility of beneficiating the -6 micron slimes fraction that is presently discarded. The column was operated in parallel with the plant rougher bank for the bulk of the testwork. The performance of the column was compared to that of the rougher, with particular emphasis on the behaviour of the misplaced slimes in the flotation feed. It was found that the column was able to produce higher concentrate grades than the plant rougher at similar tin recoveries, in significantly shorter residence times. However, the higher grades were as a result of a significant reduction in the amount of slimes collected into the concentrate by the column. The rougher achieved very high recoveries of tin in the slimes fraction, but at a very low grade, indicating that the recovery of this fraction was more by entrainment than flotation. A large portion of the slimes in the rougher concentrate was rejected in the cleaner bank, which resulted in a substantial build up of this fraction in the plant circulating load. Tests performed on undeslimed flotation feed using the column proved unsuccessful. A test performed in which the column was used as a recleaner produced encouraging results, and this aspect warrants further investigation.

INTRODUCTION

Column flotation is finding extensive application world wide for the beneficiation of many different minerals. Among the advantages that column flotation offers over conventional flotation technology are improved metallurgical performance, cheaper capital and operating costs, and greater ease of control.

In South Africa, with its considerable mineral wealth, column flotation is receiving more and more attention. As a result of this, a portable pilot scale column flotation cell has been designed and built in the Department of Chemical Engineering at the University of Cape Town with the aim of performing on-site testwork. To date, the equipment has been used to conduct on-site trials on four different mines, operating for approximately a month in each case.

This paper presents the results of one of these trials, on a cassiterite ore, conducted at the Rooiberg tin mine in the Northern Transvaal. The column was operated in parallel with the rougher section of the Rooiberg flotation circuit. The aim of the work was to compare the performance of the column with the plant rougher cells, with particular emphasis on recovery as a function of particle size. The work forms part of a preliminary investigation to evaluate the possibility of beneficiating the slime fraction (-6 micron) which is presently removed prior to flotation and discarded.

OPERATION AT ROOIBERG

CASSITERITE BENEFICIATION

The concentration of cassiterite is achieved mainly by gravity methods due to the high relative density of this mineral (SnO_2 , RD 6,8 - 7,1). After crushing, the run-of-mine ore is subjected to dense medium separation, and the sinks are subjected to successive stages of concentration using shaking tables and spirals. This yields a high grade concentrate that is typically about 60 %Sn. However these methods

are only effective down to 45 micron. Beneficiation of the -45 micron fraction can only be achieved by flotation.

Flotation of cassiterite ore is a relatively new process. Up until about twenty years ago the -45 micron fraction was discarded. So, in comparison to the well established technology of sulphide flotation, the flotation behaviour of cassiterite is still relatively poorly understood.

The flotation of cassiterite, as with the flotation of other oxide minerals, is characterised by selectivity problems. Iron oxides present in the ore tend to float as readily or more readily than the cassiterite. Thus the final concentrate grade that is obtained by flotation is typically quite low (15 - 25 %Sn) in comparison to that obtained from the gravity circuit. Reagents and operating conditions are selected to suit the ore and the local conditions. What works well at one mine can often be ineffective at another. As at other tin mines, desliming of the flotation feed at Rooiberg has been found to be essential to ensure good recovery; the -6 micron material is removed prior to flotation, and discarded.

FLOTATION AT ROOIBERG

The first flotation plant at Rooiberg was commissioned in 1972 (at "A" mine). That plant is presently out of operation, owing to the slump in the tin market in the last few years. A second flotation plant was commissioned at Rooiberg in 1981 (at "C" mine), and it was on this plant that the work described in this paper was performed.

A schematic representation of the flotation circuit at Rooiberg "C" mine is presented in Fig. 1, along with reagent additions and the typical grades of the various process streams. The flotation plant processes approximately 2800 tons of ore per month, consisting of the -45 micron material from the gravity circuit and reclaimed slimes from the dumps that were deposited prior to the introduction of flotation.

Reagents that are used consist of the following:

Xanthate as collector and TEB as frother in the sulphide float
Sodium silicate as a gangue depressant prior to the tin float
Sulphuric acid as a pH modifier prior to the tin float
Styrene phosphonic acid (SPA) as collector in the tin float

No frother is used in the tin float, as SPA has sufficient frothing properties. As SPA is only partly soluble, it is dissolved in a caustic soda solution prior to addition.

Improvement in the flotation performance at Rooiberg over the years has been considerable. Tin recovery at the "A" mine flotation plant improved from an average of around 50% in the early 70's to around 70% in 1980 (Bulled, 1982). Tin recovery at the flotation plant at "C" mine has improved from around 80% in 1981 (Bulled, 1982) to approximately 90% at present.

However, as an underground operation, the mining costs at Rooiberg are relatively high, and the tin price has been very low. Although the tin price is improving, to ensure future profitability, further improvement in efficiency is desirable. Two of the main areas in which efficiency can be improved are:

- 1) beneficiation of the -6 micron slimes which are presently discarded.
- 2) improvement of the flotation concentrate grade, without sacrificing recovery, to decrease smelting costs and increase smelter capacity.

With this in mind, an investigation of column flotation is being undertaken. Column cells have proven extremely effective in a wide range of metallurgical applications. In particular, columns have been found to favour the flotation of very fine particles, and achieve significant improvement in concentrate grades.

The flotation of ultrafine particles is generally enhanced by the use of very small bubbles. This is a result of the increase in the

probability of bubble-particle collision - low collision rates being the main cause of the slow flotation rate of these particles (Trahar, 1981). However, in conventional flotation, water recovery (and, therefore entrainment) increases when bubble size is reduced. By counter-current washing of the froth, the problem of entrainment is prevented in column flotation, allowing the use of smaller bubbles (Dobby and Finch, 1986). The recovery of ultrafine particles is further enhanced by the quiescent conditions in the column compared to the high turbulence in conventional cells. Thus the application of column cell technology to the flotation of ultrafine appears to be entirely logical.

A preliminary investigation has been completed in which a pilot column cell was operated in open circuit in parallel with the existing rougher bank at Rooiberg. The aim of the work was to compare the performance of the column cell with that of the existing cells, with particular emphasis on recovery as a function of particle size. One test was also made using the column as a recleaner, and two tests were performed on undeslimed feed. The ultimate aim will be to try to beneficiate the ultrafines.

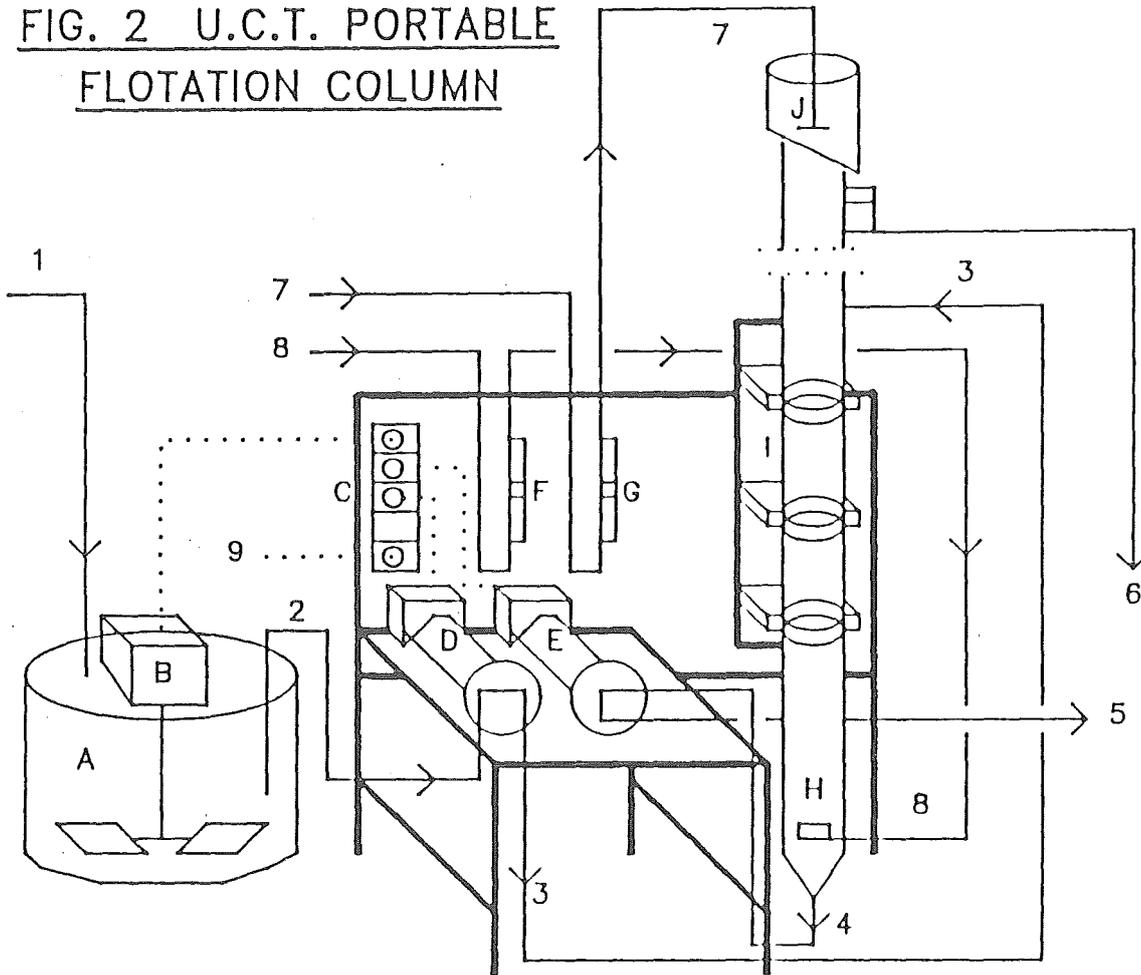
EQUIPMENT, PROCEDURE AND EXPERIMENTAL CONDITIONS

EQUIPMENT

A portable column flotation cell was designed and built at the University of Cape Town, in order to facilitate the performance of on-site trials at various locations. The system was designed to be self standing, and easy to dismantle, transport and assemble. A schematic representation of the column rig is shown in Fig. 2. All the equipment fits comfortably onto the back of a light commercial vehicle.

A 120 litre PVC tank with a mechanical agitator is used as a conditioner. The column itself is made up of half meter lengths of 9,4 cm I.D. flanged perspex tubing. The column height can be varied by the addition or removal of one or more segments; the maximum height is

FIG. 2 U.C.T. PORTABLE
FLOTATION COLUMN



Stream	Description	Equipment	Description
1	Feed from plant	A	Conditioner
2	Inlet line to feed pump	B	Agitator
3	Feed line to column	C	Plug Bank
4	Column tails to tails pump	D	Feed pump (Masterflex Peristaltic pump)
5	Column tails to tails sump	E	Tails pump (Masterflex Peristaltic pump)
6	Concentrate	F	Air rotameter
7	Wash water	G	Water rotameter
8	Compressed air	H	Air sparger (sintered glass frits)
9	220 V, 2 phase electrical supply	I	Column Support
		J	Wash water distributor

constrained by the need to have access to the top of the column. For the trials at Rooiberg a height of 4,5 m was used.

The feed to the column and the tailings are pumped using two variable speed peristaltic pumps. The tailings pump rate is used to control the level of the froth-pulp interface (monitored visually), thereby maintaining the froth height. At Rooiberg, plant compressed air was passed via a rotameter to the air sparger system, which consisted of four sintered glass frits. Water rate to the wash water distributor at the top of the column was measured using a rotameter. In these trials, both potable and clarified plant water were used.

PROCEDURE

The following procedure was used when the column was operated in parallel to the plant rougher. Variations to this procedure are discussed with the experiments in which they occur.

The column conditioning tank was filled by gravity from the plant conditioner. Thus the slurry had already been conditioned with sodium silicate, and the pH adjusted by the addition of sulphuric acid. The pH of the slurry sample was checked, and adjusted if necessary to a value of 5,5. The specific gravity of the pulp was measured, in order to calculate the volumetric feed rate required to achieve the desired solids feed rate, and to calculate the collector addition required. The appropriate volume of SPA was then added, and the slurry conditioned for 5 minutes.

While the slurry was being conditioned, the air was switched on and adjusted to the desired rate. The column was charged with water via the wash water distributor to a few centimetres below the feed point. Slurry was then introduced into the column at the required feed rate. After a reasonable build up of solids in the column, the wash water was gradually introduced, and adjusted to the desired rate.

After achieving a stable operation (usu. 10 to 15 minutes), thirty minutes were allowed to pass for the column to reach steady state. At this point timed samples were taken of both the concentrate and the tailings. Sample times varied from between 15 sec (tailings) to 2 minutes (concentrate), depending on the flowrates. The operating conditions of the column were then changed, and the column again allowed to reach steady state (about 30 minutes), before further samples were taken.

Sample preparation and analysis were carried out by the plant assay staff. Samples were filtered, dried and analysed for tin content. Provisional assay results were obtained using a portable isotope X-ray fluorescence (P.I.F.) analyser, but all samples were subsequently accurately analysed by the standard wet chemical method used daily by the staff of the Rooiberg laboratory.

EXPERIMENTAL CONDITIONS

The conditions that were used and the factors that were varied during the trials were as follows:

FEED:	Rougher feed (19 tests) Undeslimed plant feed (2 tests) Cleaner concentrate (1 test).
WASH WATER:	Potable water (Rand Water Board) Plant water.
FEED RATE:	13,0 - 51,5 kg/h (1,9 - 7,4 t/h/m ²)
AIR RATE:	1,5 - 2,0 cm/s
WASH WATER RATE:	1 - 3 l/min
FROTH HEIGHT:	0,5 - 1,5 m
COLLECTOR CONCENTRATION:	120 - 2500 g/t
COLUMN HEIGHT:	4,5 m
pH:	5,5

RESULTS AND DISCUSSION

TESTS USING POTABLE WASH WATER

In the first stage of the investigation it was necessary to identify operating conditions under which the column would produce a similar recovery to the rougher bank. This would enable a comparison of concentrate grades, and the recovery-particle size relationship, particularly in the -6 micron fraction. Typical performance of the plant rougher was as follows:

Rougher Feed: 2 %Sn	Rougher Concentrate: 9 %Sn
Recovery: 70 %	Rougher Tails: 0,6 %Sn

It was decided initially to use potable Rand Water Board (R.W.B.) water as wash water. Problems had been experienced using reclaimed clarified water during column flotation trials on other plants, mainly as a result of slimes contamination. However, the experiments using R.W.B. water all produced relatively poor recoveries. Great problems were experienced in obtaining and maintaining a stable froth during the operation of the column. A large number of tests were performed over a wide range of conditions in an attempt to identify the problem. The conditions and results of these tests are summarised in Table 1.

The tests at low reagent dosages were characterised by an unstable, brittle froth that was very difficult to maintain, but was very highly selective. Very high grades were obtained, up to 33,0 %Sn, compared to 9 %Sn for the plant rougher concentrate, and 20-23 %Sn for the plant cleaner concentrate. However the recoveries remained low, at between 40 and 50 %.

Attempts were made to increase the recovery by substantially increasing the reagent dosage. This resulted in much more stable froths, but recoveries did not improve, and in most cases worsened. The concentrate grades also decreased markedly.

It had been noted during these tests that the potable water at Rooiberg was extremely hard. The effects of water quality on the flotation of cassiterite have been widely reported in the literature (Bulled, 1982; Wright, 1982). The presence of high concentrations of ions (such as Ca and Mg) can seriously effect the performance of the collector. It was therefore decided to evaluate the affect of using clarified plant water on column performance.

TABLE 1

Summary of the tests using potable (R.W.B.) wash water.

Run No	Feed Rate kg/h	Collector Conc. g/t	Froth Height m	Air Rate cm/s	Wash Water l/min	Recovery %	Grade %Sn.
1	31,2	204	0,5	2,0	1,0	44,7	16,1
2	51,5	127	1,0	2,0	1,0	40,6	29,7
3	51,5	127	1,0	2,0	2,0	49,2	27,9
4	51,5	127	1,0	2,0	3,0	45,4	33,0
5	51,5	191	0,5	2,0	3,0	31,6	17,8
6	41,9	191	0,5	2,0	3,0	35,3	18,7
7	44,6	735	1,0	2,0	3,0	29,1	19,3
8	13,0	2500	1,0	2,0	3,0	23,6	14,7
9	36,3	900	1,0	2,0	3,0	38,6	11,2
10	13,0	2500	1,5	1,5	3,3	47,1	7,1
11	36,3	900	1,0	1,5	3,3	37,4	10,6
12	36,3	900	1,0	2,0	3,3	46,8	9,4
13	30,7	1070	1,0	2,0	3,0	53,6	12,3
14	30,7	174	1,5	2,0	3,0	44,9	21,4
15	33,5	159	2,5	2,0	3,0	44,7	20,1

TESTS USING CLARIFIED PLANT WATER

The effect of using the clarified plant water as wash water in place of the R.W.B. water was clearly apparent, by visual observation of the froth phase. Stable froths could be formed with ease, even in the absence of solids in the column. This can only be attributed to a build up of frother in the recirculated water. Although no frother is added to the tin float, frother is used in the pyrite float, and the concentration in the plant water has no doubt built up to a high level

(owing to water shortage in the area, all the plant water remains permanently in circulation; make up water is only added to replace that lost as a result of evaporation). Runs 16 to 18 (Table 2) were carried out using clarified plant water as wash water.

Another factor that was thought to be contributing to the low recoveries obtained by the column in comparison to the plant rougher was insufficient residence time. Cassiterite is a relatively weakly floating mineral that exhibits slow flotation kinetics. Laboratory flotation tests on Rooiberg ore have shown that cassiterite can require a residence time of 20 minutes or more to float to completion (Jones, 1983; Loe and Allen, 1983). Hence in two of the tests that were performed using plant water (Runs 17 and 18), the feed was passed through the column twice in order to double the residence time. The procedure that was used was the same as in the other tests, except that after steady state was achieved, the tails were collected in a separate vessel. After the original feed had been exhausted, the conditioner was cleaned, and the tails from stage one became the feed to stage two.

The conditions and results of the tests that were performed using plant water are summarised in Table 2.

TABLE 2

Summary of the tests using clarified plant wash water.

Run No	Feed Rate kg/h	Collector Conc. g/t	Froth Height m	Air Rate cm/s	Wash Water l/min	Recovery %	Grade %Sn.
16	24,9	255	1,0	1,7	3,0	57,2	11,5
17	31,2	229	1,0	2,0	3,0	59,6 70,2	15,0 ¹ 12,5 ²
18	27,0	264	1,0	2,0	3,0	63,0 72,9	12,6 ¹ 11,4 ²

1: 1st pass through column
2: 2nd pass through column

a) Effect of Plant Water

The change to plant water resulted in a substantial increase in the recovery of tin that was obtained. Recoveries increased by more than 10% when compared to equivalent experiments using R.W.B. wash water (cf. Table 1). It was apparent however that the froth was much less selective, and substantially lower grades were obtained.

b) Effect of Residence Time

The residence time of the column at the volumetric feedrates used in these tests was about 4.4 minutes. The residence time of the plant rougher cells is between 14 and 20 minutes, depending on the circulating load (15 - 60 %). Hence the residence time in the column was approximately a quarter of that in the rougher cells.

Doubling the residence time in the column by refloating the tails from the first column pass resulted in an increase in recovery of about 10 % (absolute) to around 70 %. Thus the column was able to achieve a recovery equivalent to that obtained in the rougher bank, in half the residence time. This is shown in Fig. 3, which plots the tails assay (which is inversely proportional to the recovery) down the rougher bank versus residence time in comparison to the results obtained on the column in runs 17 and 18.

c) Comparison of Rougher and Column Performance

Simultaneous to the performance of Runs 17 and 18 (which were performed a week apart), samples of the rougher feed, concentrate and tails were taken in order to make a direct comparison of the two operations. These results are summarised in Table 3.

It can be seen that the recoveries achieved by both the rougher and the column were very comparable. However, the concentrate grade achieved by the column was significantly higher in both experiments.

FIGURE 3 Comparison of tails assay as a function of residence time for the plant rougher and the column.

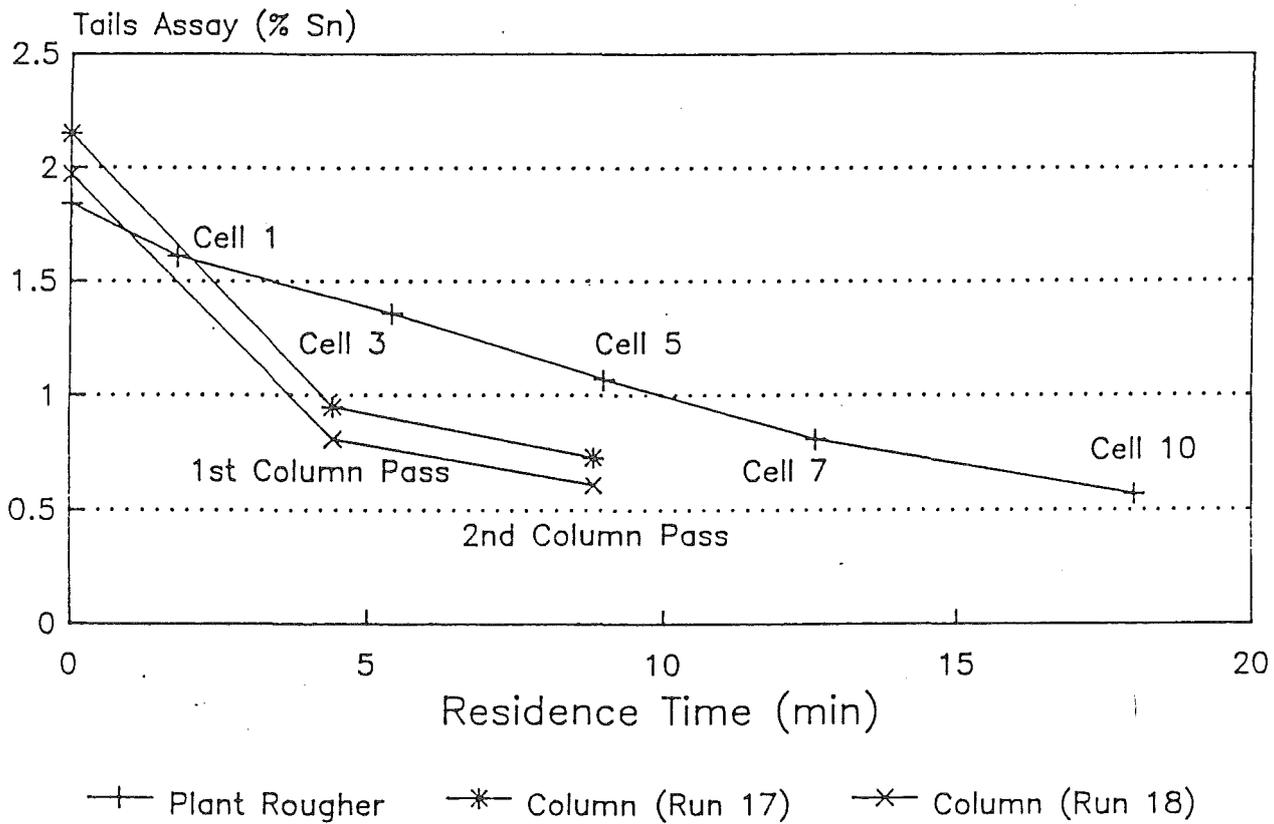


TABLE 3

Comparison of column and rougher performance

Run No	Rougher		Column	
	Recovery	Grade	Recovery	Grade
17	68,8	11,3	70,2	12,5
18	76,6	9,4	72,9	11,4

COMPARISON OF RECOVERY BY SIZE FRACTION

One of the main aims of the investigation was to compare the performance of the column and the rougher on the different size

fractions that constituted the feed. Of particular interest was the behaviour of the -6 micron fraction that remained in the flotation feed as a result of the imperfect operation of the desliming cyclone. To this end, samples of the feed, concentrate and tails of Runs 17 and 18 were taken and graded by size fraction. These results were then compared with those obtained from samples of the rougher feed, concentrate and tails, taken simultaneously to the performance of each of the column tests.

a) Tin Distribution in the Rougher Feed by Size

The distribution of the tin in the feed to the rougher during the trials is presented in Table 4. It can be seen that about 70 % of the tin was contained in the 23 to 9 micron size range. The amount of tin contained in the +45 micron size fraction was negligible. The -6 micron fraction contained 8 to 14 % of the tin.

TABLE 4

Distribution of tin by size in rougher feed

Size Fraction (micron)	Tin Distribution (%)
+75	,5 - 1
+45-75	1 - 3
+23-45	6 - 12
+17-23	20 - 30
+13-17	25 - 35
+9-13	13 - 17
+6-9	6 - 9
-6	8 - 14

b) Size Distribution of the Feed, and Column and Rougher Concentrates

The size distribution of the feed and the column and rougher concentrates from Runs 17 and 18 are presented in Figs. 4 and 5 respectively. It can be seen that significant quantities of -6 micron material were present in the rougher feed during Runs 17 and 18 (19 % and 15 % resp.). More dramatic, however, was the very high proportion of this fraction in the rougher concentrate (33 % and 48 % resp.). In

FIGURE 4 Comparison of the size distribution of the column and rougher concentrate streams for Run 17.

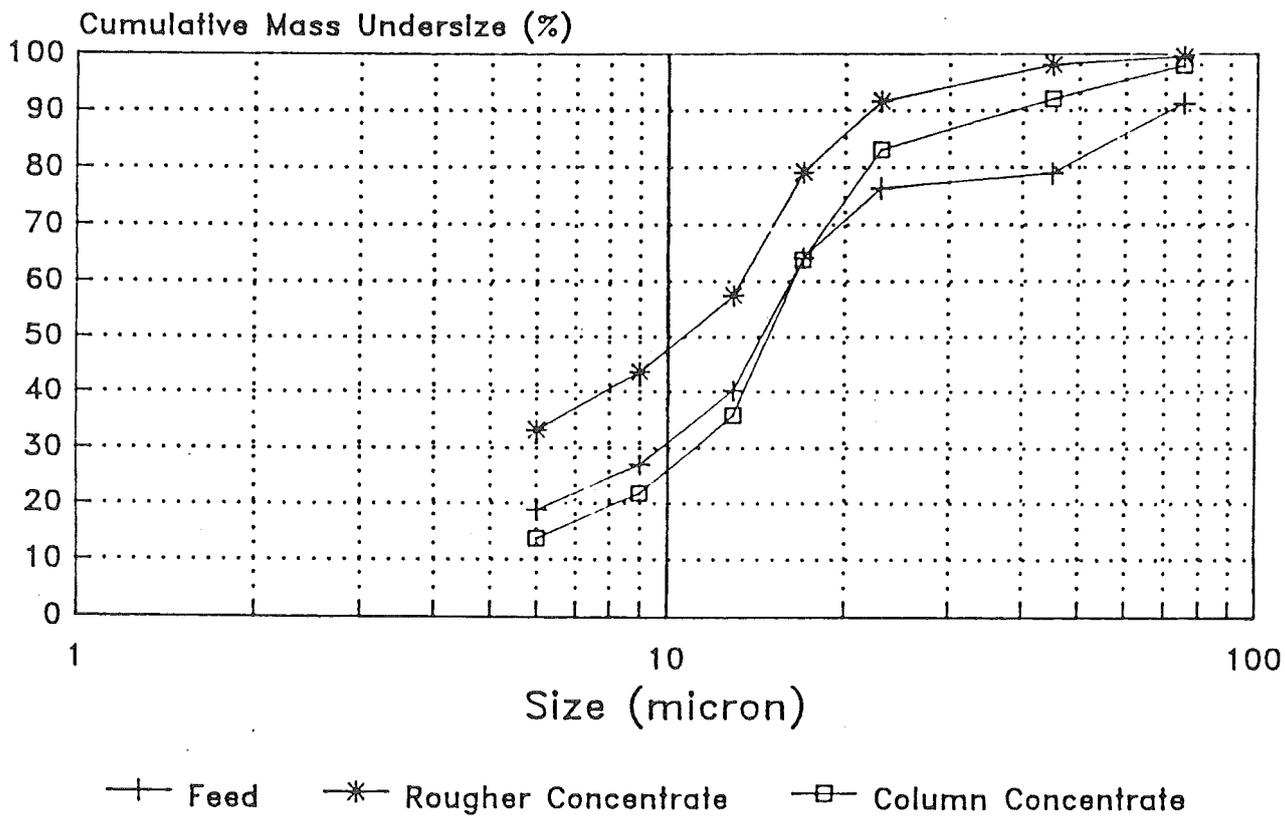
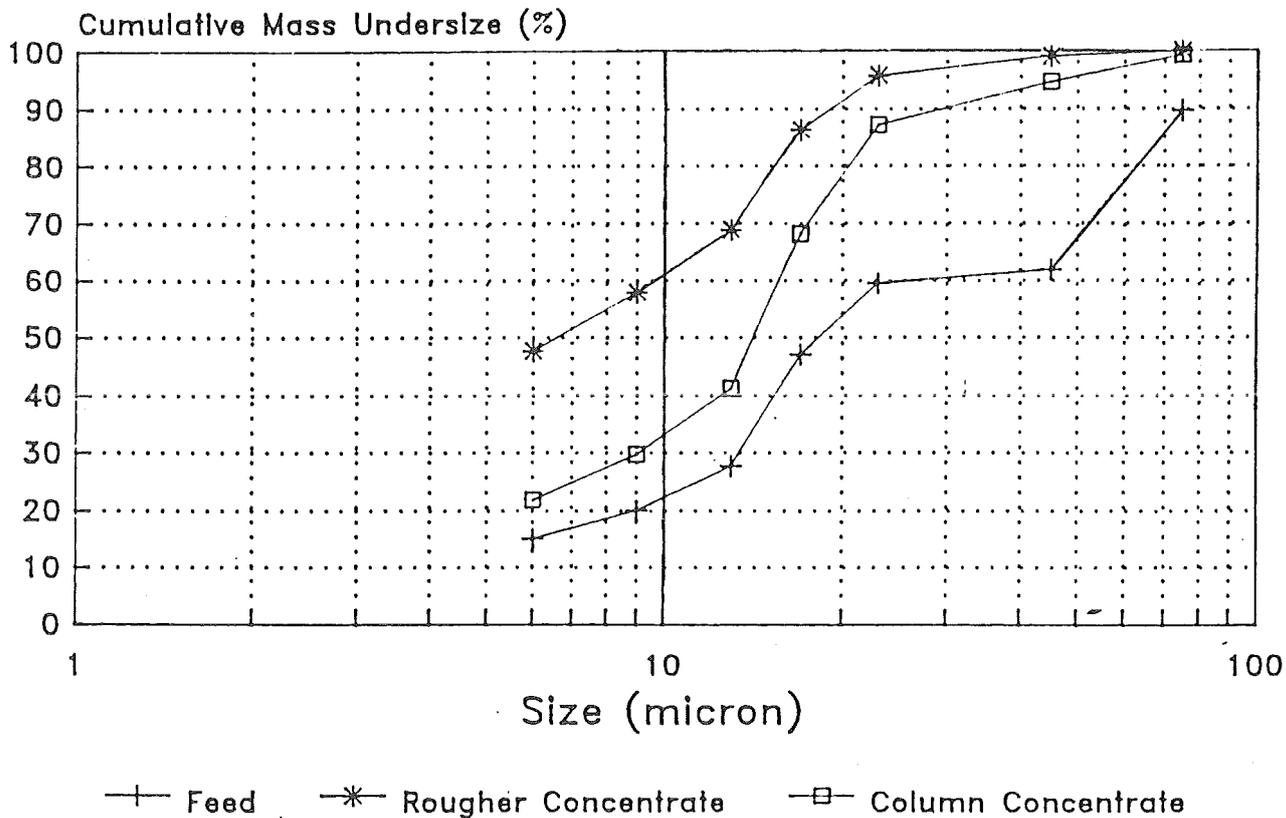


FIGURE 5 Comparison of the size distribution of the column and rougher concentrate streams for Run 18.



comparison, the amount of -6 micron material in the column concentrate was very much lower (13 % and 21 % resp.).

In the coarser size fractions (+45 micron), significantly higher proportions were collected in the column than in the rougher, although in both cases the yield was relatively low when compared to the amount of this material in the feed.

c) Comparison of Tin Recovery by Size Fraction

The tin recovery by size fraction obtained by the column and the rougher in Runs 17 and 18 is presented in Figs. 6 and 7 respectively. Each point is labelled with the grade of the individual size fraction in the concentrate. In both cases it can be seen that the column obtained considerably higher tin recoveries than the plant rougher in the +13 micron size ranges, with this trend reversed for the -13 micron material. In the +45 micron size fraction, both the rougher and the column obtained very low tin recoveries. Of particular note, however, is the very high tin recovery obtained in the -6 micron fraction by the rougher (83 % and 79 % resp.), in comparison to the column (42 % and 46 % resp.).

If the +45 micron fractions are ignored, it can be seen that the grade tended to decrease with decreasing particle size, in both the rougher and the column concentrates. Particularly low grades were obtained in the -6 micron size fraction (3,5 and 2,5 %Sn resp. in the rougher and 5,0 and 4,0 %Sn resp. in the column). In the other size fractions, the rougher grades tended to be higher than the column grades.

d) Discussion of the Size - Grading Results

An examination of the size - grading results presented above shows that although the overall tin recoveries obtained by the column and the rougher bank were very similar (cf. Table 3), they each exhibited a markedly different recovery - size relationship. This difference is most apparent when the -6 micron fraction is examined.

FIGURE 6 Comparison of the recovery by size relationship between the rougher and the column for Run 17. Labels indicate concentrate grades for each size fraction (Normal=column; *Italic*=Rougher).

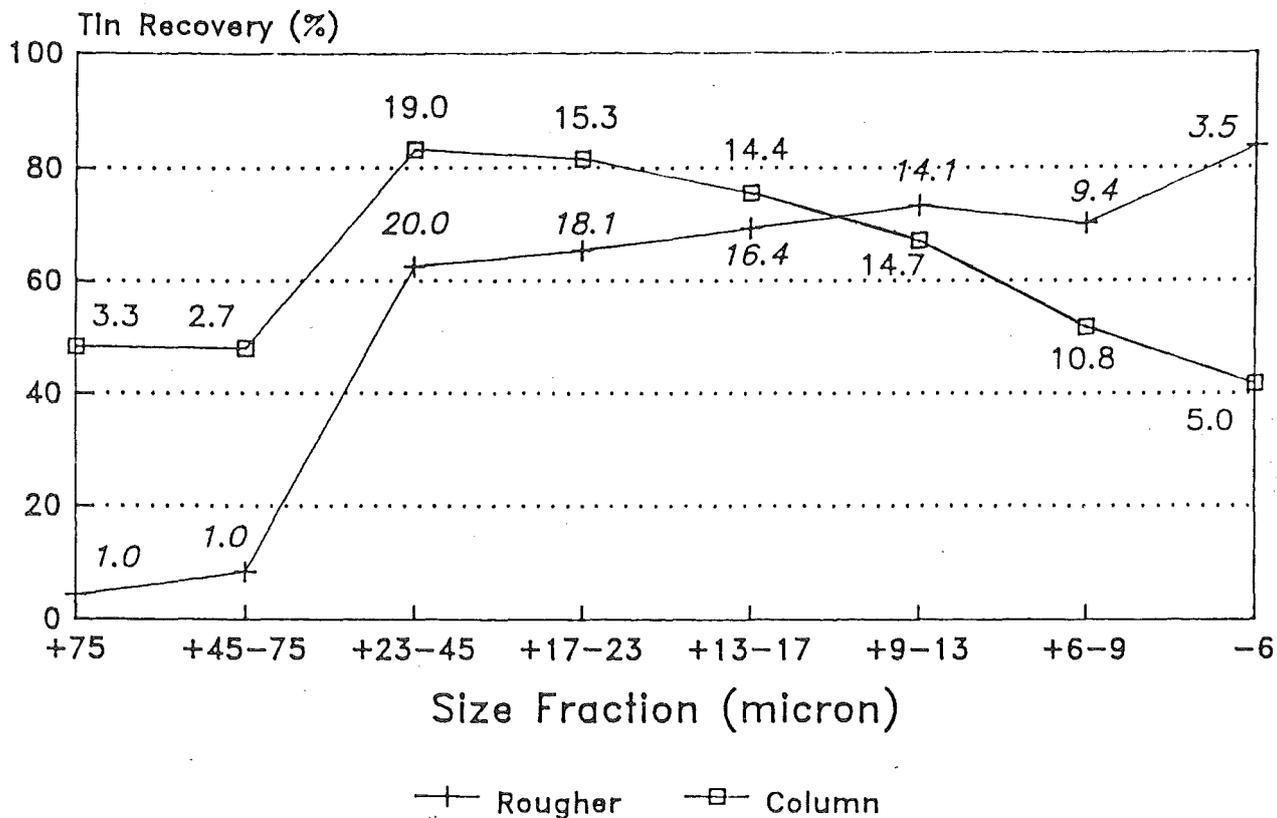
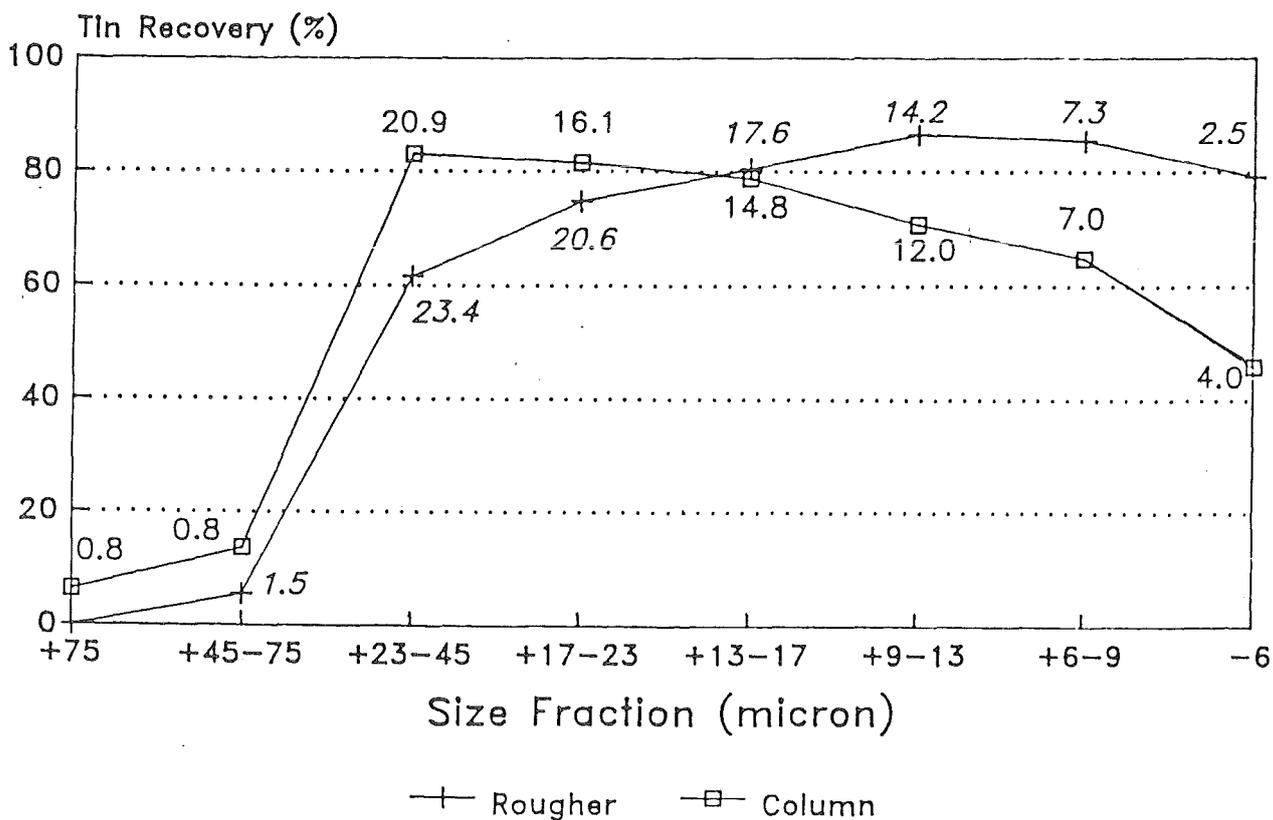


FIGURE 7 Comparison of the recovery by size relationship between the rougher and the column for Run 18. Labels indicate concentrate grades for each size fraction (Normal=column; *Italic*=Rougher).



An extremely high proportion of the rougher concentrate consisted of -6 micron slimes. As a result, the rougher achieved a very high tin recovery in this fraction. However, the grade of the recovered slimes was extremely low. Thus the effect was to significantly reduce the overall grade achieved by the rougher. In comparison, the column recovered a much smaller proportion of the slimes, exhibiting a marked preference for the recovery of the coarser tin particles, thereby achieving a higher overall grade.

A sharp decline in the grade of the concentrate size fractions as the particle size decreases can be observed [In this regard, the +45 micron fraction can be ignored, as this contain only about 4 % of the tin in the feed, and is in any case too coarse for effective cassiterite flotation (R.D. 6,8 to 7,1)]. This decline in grade was most extreme in the slimes fraction. The -6 micron fraction of the feed in Runs 17 and 18 assayed at 2,5 and 1,7 %Sn respectively. In the rougher concentrate, this fraction was upgraded to only 3,5 and 2,5 %Sn respectively. The grades obtained in the column were 5,0 and 4,0 %Sn respectively. The same phenomenon can be observed, to a lesser extent, in the -9+6 micron fraction.

Thus the results indicate that, under the present plant operating conditions, the ultrafine particles of this ore exhibit a very low degree of floatability, and their recovery occurs predominantly by entrainment. Therefore the marked difference in the recovery - size relationship of the plant rougher and the column can most probably be ascribed to a reduction in the entrainment of weakly floatable, or non floatable, ultrafine particles by the column cell.

The significant reduction in the entrainment of non floatable ultrafine particles that can be achieved in a column flotation cell in comparison to conventional mechanical cells has been previously reported (Espinosa-Gomez et. al., 1988). This is due to the deeper froth depth that can be achieved in a column cell, and the washing of the froth by counter-current spray.

Thus, from these results, it appears that different reagents and/or conditions to those presently employed at Rooiberg will be required if the ultrafine cassiterite particles are to be rendered sufficiently floatable to enable effective recovery of the tin from the slimes.

e) Slime Build Up in the Rooiberg Flotation Circuit

The underflow from the desliming cyclone prior to flotation typically contains less than 10 % -6 micron material. However it appears that a significant proportion of this material builds up and remains in circulation in the plant. This is clearly shown by the following values of the slimes content of a number of plant streams:

Rougher Feed:	15 - 20 % -6 micron
Rougher Concentrate:	30 - 50 % -6 micron
Cleaner Concentrate:	15 - 20 % -6 micron

From estimates of the stream mass flows, it can be calculated that 90 to 95 % of the slime particles in the rougher concentrate are recirculated back to the conditioner as cleaner tails. The cleaner tails can contain up to 55 % -6 micron material. This may have a significant effect on the operating efficiency of the flotation circuit. The implications of reducing the entrainment of the ultrafines by installing a column in place of the rougher or cleaner bank are too complicated to assess without more extensive testwork.

TESTS ON UNDESLIMED FEED

Two tests were performed in an attempt to float undeslimed plant feed. The samples were taken directly from the thickener prior to the desliming cyclones. The pyrite was floated off in the column using a Xanthate collector, and the tailings collected for the subsequent cassiterite float. The pyrite floated very readily, and no difficulty was experienced with this part of the experiment. The pyrite that was removed assayed at about 40 % sulphur.

For the cassiterite float the pH was adjusted to 5,5 and the pulp was conditioned for 15 minutes with 1000 g/t sodium silicate. An initial

dose of 300 g/t of SPA was added, and the pulp was conditioned for a further 10 minutes.

However, when an attempt was made to float the slurry, no material reported to the concentrate. All the column parameters were varied in an attempt to induce a response, including a gradual increase in the collector concentration to more than 3000 g/t, but the froth remained completely barren throughout.

The reason for the complete lack of flotation response of this material is unknown, but it is most probably as a result of slime coating of the cassiterite particles. An investigation of the flotation of this material is presently being conducted at U.C.T.

COLUMN AS A RECLENER

One test was performed on cleaner concentrate, to obtain an indication of the degree to which the concentrate assay could be improved in the column. Conditions and results were as follows:

FEED RATE: 22,8 kg/h	FROTH HEIGHT: 1 m
AIR RATE: 1,5 cm/s	WASH WATER RATE: 3 l/min
REAGENTS: None	

RESULTS: Feed = 24,8 %Sn	Concentrate = 34,0 %Sn
Tails = 18,3 %Sn	Recovery = 57,1 %

Since no attempt was made to optimise column performance using this feed, this result is extremely encouraging. It is likely that the feed rate chosen was far too high for this material and the carrying capacity of the column was being exceeded. In the rougher investigation, typical concentrate rates were between 290 and 430 kg/h/m², whereas in this case a concentrate rate of 1370 kg/h/m² was obtained.

A more thorough investigation of the column as a cleaner is certainly required, as the improvement in grade would substantially decrease smelting costs and increase smelting capacity.

CONCLUSIONS

Column flotation trials were conducted at the Rooiberg tin mine using a portable column flotation cell. The work formed part of a preliminary investigation of the beneficiation of the -6 micron slimes that are presently discarded.

For most of the work the column was operated in parallel with the plant rougher circuit. The aim of these tests was to compare column and rougher performance, with particular emphasis on the behaviour of the -6 micron size fraction present in the flotation feed.

Initially the tests were performed using potable (Rand Water Board) water as wash water. However, this water is extremely hard. At low collector dosages, these experiments were characterised by very unstable but highly selective froths. Grades of up to 33 %Sn were obtained, but tin recoveries remained below 50 %, which was substantially lower than the recovery of the rougher circuit, which was generally around 70 %. Increasing the collector concentration resulted in more stable froths, but resulted in a significant drop in both recovery and grade.

Tests were then performed using clarified plant water as wash water. This resulted in a marked improvement in the froth stability, most probably caused by a build up of frother in the recirculated water. As a consequence, tin recovery increased to around 60 %.

A further increase in the tin recovery, to about 70 %, was achieved by increasing the residence time in the column by reprocessing the tails. The two passes through the column together corresponded to about half the residence time of the plant rougher. Under these conditions, the column produced higher concentrate grades than the plant rougher at similar tin recoveries.

Comparison of size analyses of rougher and column concentrate showed that the higher grades were as a result of a significant reduction in the amount of slimes collected into the concentrate by the column. The rougher achieved very high recoveries of tin in the slimes fraction, but at a very low grade, indicating that the recovery of this fraction was more by entrainment than flotation. A large portion of the slimes in the rougher concentrate is rejected in the cleaner bank, resulting in a substantial build up of this fraction in the plant circulating load.

Tests performed on undeslimed flotation feed using the column proved unsuccessful. No recovery was obtained, probably as a result of slime coating of the cassiterite particles.

Finally, a test was performed in which the column was used as a recleaner. This produced encouraging results, and this aspect warrants further investigation.

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REFERENCES

- Bulled, D. A. "Practical Aspects of Cassiterite Flotation", Proceedings of the XIV International Mineral Processing Congress, Toronto, Oct. 1982.
- Dobby, G. S. and Finch, J. A. "Particle Collection in Columns - Gas Rate and Bubble Size Effects", Canadian Metallurgical Quarterly, vol. 1, pp. 913, 1986.
- Espinosa-Gomez, R., Finch, J. A. and Johnson, N.W. "Column Flotation of Very Fine Particles", Minerals Eng., vol. 1, pp. 3-18, 1988.
- Falcon, L. M. "Tin in South Africa", J. S. Afr. Inst. Min. Metall., vol. 85, pp. 333-345, Oct. 1985.
- Jones, J. A. "The Effects of Sodium Silicate, Fluoride and Silicofluoride Depressants on Cassiterite Flotation at Rooiberg Tin", Report RBG/72, Goldfields Laboratories, 1983.
- Loe, M. W. and Allen, G. R. "Alternative Collectors to Tenneco S.P.A. - A Further Comparative Laboratory Evaluation of Selected Flotation Reagents using C mine Cassiterite Flotation Feed", Report RBG/74, Goldfields Laboratories, 1983.
- Trahar, W. J. "A Rational Interpretation of the Role of Particle Size in Flotation", Int. J. Miner. Proces., vol. 8, pp. 289-327, 1981.
- Wright, P. A. Extractive Metallurgy of Tin, 2nd Ed. Elsevier Scientific Publishing Co. pp. 65-77, 1982.