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An evaluation of gravity separators by use of a synthetic ore

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SYNOPSIS

The Tromp distribution curve was used in an evaluation of gravity separators treating a synthetic ore made up of silica and marker minerals of various relative densities. The results pinpoint differences in the concentration of particles of different sizes, and indicate how the grade of a concentrate is likely to vary with particle size.

An increase in the feed rate increased the efficiency of the 'waterless' spiral concentrator that was tested, but the multiport spiral concentrator tested was more efficient. Both spiral concentrators were found to be more efficient than the pinched sluice and the shaking table that were tested.

The method was found to be a useful way of evaluating gravity concentrators, but, for ease of analysis, care must be taken in the choice of marker minerals.

SAMEVATTING

Die Tromp-verdelingskromme is gebruik vir 'n evaluering van die doeltreffendheid van gravitasieskeiers vir die behandeling van 'n sintetiese erts bestaande uit silika en merkerminerale met verskillende relatiewe digthede. Die resultate wys verskille in die konsentrasie van partikels met verskillende groottes uit, en toon ook hoe die graad van die konsentraat waarskynlik volgens die partikelgrootte sal wissel.

Verhoging van die toevoertempo het die doeltreffendheid van die 'waterlose' spiraalkonsentreerder wat getoets is, verbeter maar die meerpoortspiraalkonsentreerder wat getoets is, was meer doeltreffend. Die het ook geblyk dat albei spiraalkonsentreerders meer doeltreffend is as die toegeknypte sluis en skudtafel wat getoets is.

Die metode was 'n nuttige manier om gravitasiekonsentreerders te evalueer, maar vir maklike ontleding moet daar sorg gedra word met die keuse van merkerminerale.

Introduction

Insufficient information is available on the efficiency of gravity separators, and on the effects of variables such as the particle-size distribution and relative densities of the various minerals comprising the feed material, the feed rates, and the pulp densities. Also, not much attention has been given to the assessment of various gravity separators that treat similar feed material.

A number of methods can be used in the evaluation of the efficiency of gravity separators. The standard method involves feeding of an ore to the separator, analysis of the products for the valuable mineral, and plotting of a curve of grade versus recovery. This method would be adequate if the valuable mineral was in the form of particles of a set relative density, but this is seldom the case, and the values also occur in unliberated particles with a range of densities.

Another method for the assessment of gravity separators is being used by the Diamond Research Laboratory in Johannesburg. They have manufactured a range of plastic markers, using a different colour for each relative density, and have extruded these markers into pellets. These markers are being used very successfully in the assessment of heavy-medium separation. However, it is unlikely that markers fine enough to be useful in the gravity separation of fine material on spiral concentrators and shaking tables could be produced. Even if they could be manufactured, their recovery for re-use and for the identification of the various colours would be difficult.

In the coal, andalusite, and other industries where heavy-medium separation is employed, the Tromp distribution curve is used as a test of the efficiency of separation. This curve can be called an error curve because, the steeper the curve, the more efficient is the separation that it represents. Heavy-liquid separations are made at various densities on the two products from a dense-medium separator, and the results are used in the calculation of the proportion of material of specific density that reports to the concentrate. This proportion, sometimes referred to as the distribution number, is plotted against the mean of that density interval to give the Tromp curve (Fig. 1).

The density of separation (d_{50}) is that density at which 50 per cent of the material reports to the concentrate. The probable error, E_p, is defined as half the difference between the relative densities at 75 and 25 per cent, and is therefore a measure of the average slope of the mid-section of the Tromp curve. This shows that, the steeper the curve, the sharper is the separation, an ideal separation being represented by a vertical line.

In gravity or heavy-medium separation, the materials of the highest and lowest densities generally have no trouble in finding their way into the concentrate and tailing respectively. However, some of the material of a density close to the density of separation (or 'near-density' material) is likely to be misplaced. This material, which probably contains the unliberated particles, is represented by the mid-section of the distribution curve and by the probable error.

The imperfection, I, is the probable error divided by the density of separation minus 1, and is sometimes preferred to describe the performance of a gravity separator.

The use of heavy-liquid separation for the determina-

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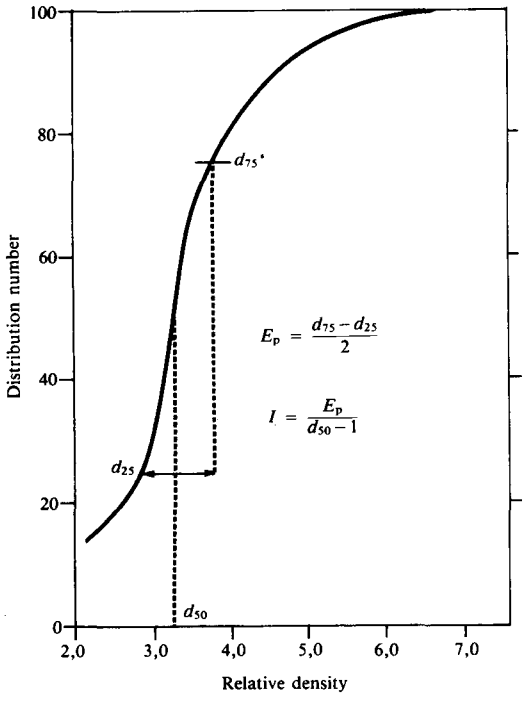

Fig. 1—A Tromp distribution curve

tion of the Tromp curve is laborious and, at relative densities above 3,3, becomes unpractical.

The Council for Mineral Technology (Mintek) has for some time been interested in the use of marker minerals of known relative densities and chemical composition to determine the efficiency of gravity concentration, and in 1975 Overbeek[1] reported the results of an investigation on the efficiency of a Dryflo concentrator by the use of such marker minerals.

The method used is as follows. Marker minerals of known relative density and composition are mixed in set proportions with a low-density material such a silica. This synthetic mixture is concentrated by gravity separation, and the products are analysed for the marker components. The proportion of the marker mineral reporting to the concentrate is calculated and plotted against the density of the marker to yield the Tromp distribution curve in the same way as heavy-liquid density fractions are used.

The object of this paper is to present the results obtained in an investigation of the efficiency of two spiral concentrators, a laboratory shaking table, and a pinched sluice by the use of a synthetic ore consisting of mixtures of various minerals and materials. As the marker minerals used in the work on spiral concentrators were different from those used on the laboratory shaking table and pinched sluice, the work on each type of concentrator is described separately.

The choice of the marker minerals is important in that no marker should contain elements that are being analysed for in another marker. It is also necessary for markers with a wide range of relative densities and of known chemical composition to be used.

In his testwork on the Dryflo concentrator, Overbeek[1] used an artificially made high-density slag or 'lead glass' (of about 5,5 density) as the marker. The manufacturer of that lead glass is no longer able to make it, and many hours were spent in the present investigation in an attempt to reproduce this material. A homogeneous mixture could not be obtained, and small particles of metallic lead were found to precipitate out of the molten solution. Finally, the idea that this material should be used as a marker was abandoned.

Spiral Concentrators

The marker minerals used in the tests on spiral concentrators and their origin, composition, and relative densities, which range from 3,18 for apatite to 6,78 for cassiterite, are shown in Table I.

TABLE I
COMPOSITION OF THE SYNTHETIC ORE

Marker mineral	Origin	Analysis of marker mineral	Amount added %	Relative density
Cassiterite	Uis	74,33% Sn	1	6,78
Baddeleyite	Palabora	77,71% ZrO_2	1	5,33
Chromite	Rustenburg	44,93% Cr_2O_3	4	4,46
Magnesite	Sea water	96,01% MgO	4	3,54
Apatite	Foskor	35,78% P_2O_5	5	3,18
Silica	Pietersburg	-	85	2,65

An attempt was made to prepare the markers in such a way that their particle-size distributions would be similar to that of the main gangue mineral, quartz. However, the apatite, chromite, and baddeleyite were obtained in a comminuted form, and were found to contain very few fine particles. This is shown in Table II, which gives the particle-size distributions of the various components of the synthetic ore.

The magnesite was calcined sea-water magnesia and, as such, not a naturally formed mineral. When crushed, this material had an unusual size distribution, with more coarse particles and more fine particles than the other markers. The particles of this material also differed in shape from the particles of the other markers, being predominantly flat or splinter-shaped.

Test Procedures

Of the two spiral concentrators used, one (referred to here as the 'waterless' spiral concentrator) had a single concentrate cutter and used no wash water (Fig. 2); the other (the multiport spiral concentrator) had multiple concentrate ports and used wash water (Fig. 3).

Each spiral concentrator was tested at three feed rates of solids and at various pulp densities. So that the grade, feed rate, and pulp density could be kept constant, a closed-circuit rig was not used; instead, the dry material and the water were fed separately to a sump in the correct amounts. The feeder used to introduce the solids was a slow-moving variable-speed conveyor belt with a steady-head feed hopper that was kept filled to a given mark during the tests.

Originally, the material was pumped to the head of the spiral concentrator, but surging in the pump caused irregularities in the feed rate. The feed was therefore introduced to the concentrator by gravity through a 50 mm pipe into which a static mixer had been inserted to ensure good mixing and wetting of the solids.

TABLE II
SIZE ANALYSIS OF COMPONENTS OF SYNTHETIC ORE

Sieve size, μm	Feed	Silica	Apatite	Magnesite	Chromite	Baddeleyite	Cassiterite
>425	6,9	5,2	8,9	20,1	23,6	1,7	10,1
<425 >300	26,2	25,0	17,0	20,1	16,2	12,7	16,6
<300 >212	25,2	27,4	25,7	14,7	22,8	32,0	18,8
<212 >150	16,9	17,0	27,2	9,2	28,4	29,7	15,4
<150 >106	9,8	9,9	16,3	7,1	8,4	10,4	13,1
<106 >75	5,4	5,6	5,0	7,2	0,4	2,2	9,8
<75 >53	3,6	3,8	1,5	7,5	0,1	0,3	8,0
<53 >38	1,7	1,8	0,3	3,6	tr	tr	3,8
<38	4,3	5,2	0,1	10,5	0,1	tr	4,4

tr = trace

TABLE III
DATA FROM TEST A OF THE REPRODUCIBILITY TESTS

		Concentrate (9,6% by mass)			Tailing (90,4% by mass)				
			Marker mineral			Marker mineral		Total mass of marker mineral	Distn no.
Relative density	Marker component	Assay *a* %	Content *b* %	Mass *c* %	%	Content %	Mass *d* %	%	%
6,78	Sn	7,1	74,33	0,917	0,01	74,33	0,012	0,929	98,7
5,33	ZrO_2	7,0	77,71	0,865	0,026	77,71	0,030	0,895	96,6
4,46	Cr_2O_3	16,5	44,93	3,525	0,30	44,93	0,604	4,129	85,4
3,54	MgO	10.9	96,01	1,090	3,6	96,01	3,390	4,480	24,3
3,18	P_2O_5	4,80	35,78	1,288	1,42	35,98	3,588	4,876	28,8
2,65	SiO_2	—	—	1,915	—	—	82,776	84,691	2,3

*Total marker mineral that reports to the concentrate

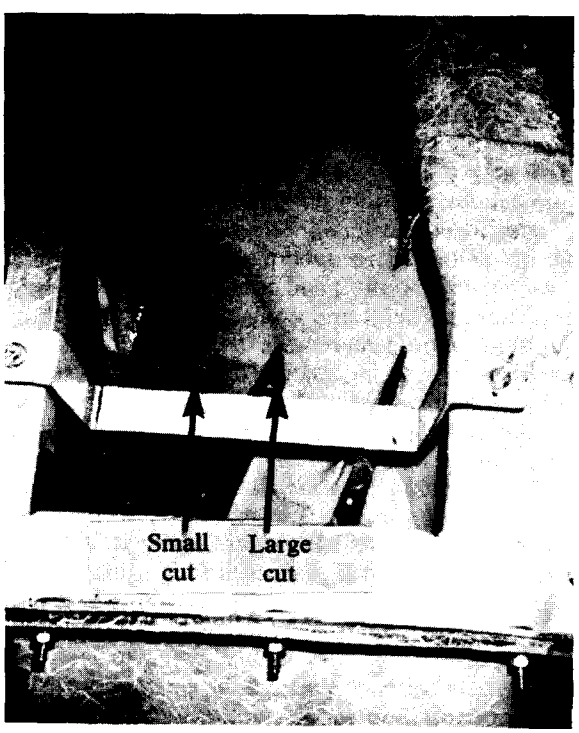

Fig. 2—Waterless spiral concentrator with single concentrate cutter

The three feed rates at which the spiral concentrators were tested were 750, 1000, and 1250 kg/h, and the feed rate and pulp density at the time of sampling were calculated. If the calculated feed rate differed from the set feed rate by more than 5 per cent, the results of the test were rejected. The value of 5 per cent was chosen because, in the series of tests used as a check on the reproducibility of the procedure, the standard deviation was 5 per cent of the mean.

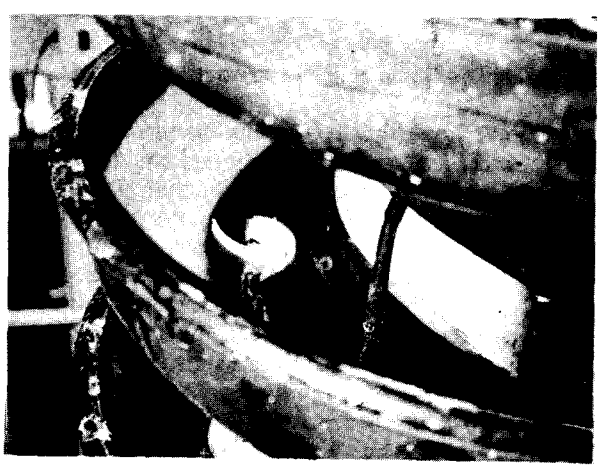

Fig. 3—Spiral concentrator with multiple concentrate ports

Tests on Reproducibility

In ten tests on the reproducibility of the procedure, the waterless spiral concentrator was used at the same settings. The tests were conducted over a period of five days by two separate operators, and the results were compared by use of Tromp curves. The method used in the calculations for the Tromp curves is given in Table III.

The analysis of each marker component in the concentrate (*a*), the analysis of the component in the marker mineral (*b*), and the mass of the concentrate (9,6 per cent) were used in the calculation of the mass percentage of the marker mineral present in the concentrate (*c*). A similar calculation was done for the tailing (*d*), the total amount of marker mineral present (*e*) being the sum of *c* and *d*. The distribution number is the total percentage of marker mineral that reports to the concentrate, $c/e \times 100$, and was plotted against the density to give the Tromp distribution curve.

The Tromp curve for test A is shown in Fig. 4. The distribution number at a density of 3,54 (magnesite) is obviously out of position, resulting in a Tromp curve that is not typical. It was found that the shape of the particles and the large amount of fines in the magnesite were responsible for its poor recovery. This point was therefore omitted, and a more acceptable curve (Fig. 5) was obtained.

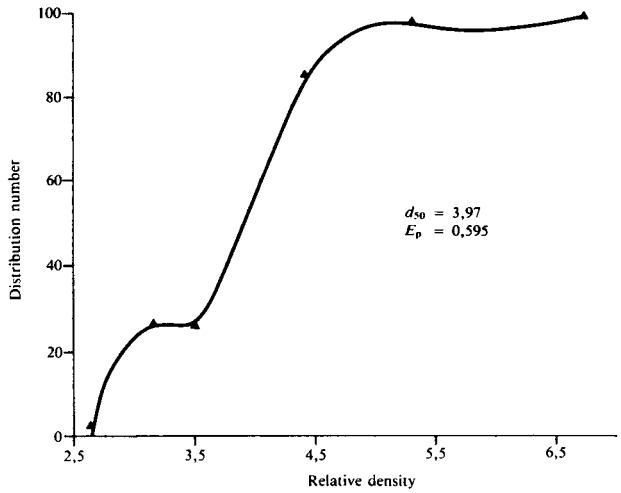

Fig. 4—Tromp curve for reproducibility test A

Table IV shows that the results for tests B, H, and J were rejected because the reconstituted feed rate fell outside the 5 per cent range. The results that fell within the acceptable feed rate were analysed statistically, and the E_p values that fell outside one standard deviation of the mean were rejected. Of the seven tests with acceptable feed rates, only five were statistically acceptable.

The reproducibility of the values in all the columns for the seven accepted test results was also tested, only the values lying within one standard deviation on either side of the mean being accepted. All the tailing values for ZrO_2, Cr_2O_3, and MgO were accepted, but several values for the other constituents were rejected.

The marker minerals were checked for whether any of their constituents occurred in the other markers. The bad-

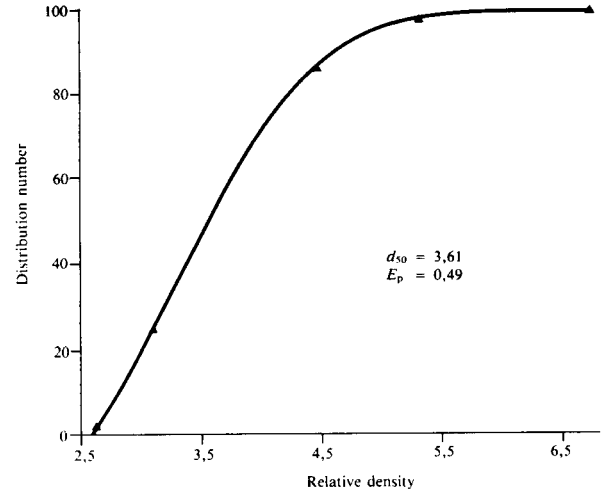

Fig. 5—Adjusted Tromp curve for test A

deleyite was found to contain a trace of phosphorus, but this was in the form of liberated apatite and would therefore follow the P_2O_5 marker.

Magnesium was found to be present in various marker minerals as shown in Table V. Investigation showed that the bulk of the magnesium in the chromite was in the chromite crystals and would therefore report to the concentrate. Therefore, the magnesium in the chromite was not the cause of the poor recoveries of magnesium but would enhance the recovery.

The reproducibility of the method is poor, and various tests were carried out on whether changes in the operating conditions would still be detectable from the d_{50} and E_p values.

Recovery of Different Size Fractions

The products from a test on the waterless spiral concentrator were sized, and the individual size fractions were analysed for their content of marker mineral. The Tromp curves for the fine fractions showed variations from the true Tromp shape, mainly because of the low recovery of the baddeleyite, and possibly of the chromite, in the fine sizes. This is probably because the presence of trace amounts of these two minerals in the fine sizes (Table II) made the separation, and even the analysis of the products, more difficult and open to error.

From the Tromp curves, the density of separation and probable error were calculated for each size fraction, and these results are shown in Table VI. The recovery of chromite in each fraction of the concentrate is included for purposes of comparison.

Table VI gives a good indication of the behaviour of the different size fractions when a feed with a range of particle sizes is treated. It can be seen that, for coarser particles, the d_{50} and E_p values are high, indicating the loss of medium- to high-density particles (chromite) to the tailing, but at an increased density of separation (or grade of concentrate). Probably owing to their larger physical size, the coarse particles were carried further into the part of the pulp stream with a high flowrate.

As the particle size decreases to 53 μm, the E_p values show a distinct decrease, indicating that the efficiency of the spiral concentrator is increasing. However, the den-

TABLE IV
RESULTS OF REPRODUCIBILITY TESTS

Test	Mass, %		Concentrate assay, %				Tailing assay, %				Pulp density of solids %	Feed rate t/h	d_{50}	E_p	Remarks
	Conct.	Tail.	Sn	ZrO_2	Cr_2O_3	MgO	P_2O_5	Sn	ZrO_2	Cr_2O_3	MgO	P_2O_5					
A	9,6	90,4	7,1	7,0	16,5	10,9	4,80	0,01	0,026	0,30	3,6	1,42	25,1	0,95	3,61	0,49	Accept
B	9.5	90,5	6,6	7,0	16,0	10,7	4,80	0,01	0,024	0,35	3,4	1,30	31,7	0,94	3,61	0,54	Reject on feed rate
C	7,3	92,7	8,4	8,0	19,0	10,9	4,43	0,01	0,024	0,30	3,4	1,23	26,2	1,05	3,68	0,48	Accept
D	7,9	92,1	8,6	8,5	19,8	10,7	4,22	0,01	0,025	0,30	3,5	1,36	25,5	0,99	3,68	0,45	Reject statistically
E	8,8	91,2	7,1	7,2	17,2	10,7	4,65	0,01	0,025	0,31	3,3	1,24	23,6	0,96	3,61	0,50	Accept
F	8,8	91,2	7,6	7,6	17,2	10,4	4,80	0,01	0,024	0,32	3,2	1,18	22,8	0,97	3,58	0,52	Accept
G	8,8	91,2	7,7	7,6	18,1	10,9	5,00	0,01	0,024	0,32	3,2	1,18	24,4	0,97	3,58	0,54	Reject statistically
H	8,6	91,4	7,6	7,6	17,0	10,6	4,80	0,02	0,027	0,31	3,17	1,14	23,5	1,08	3,58	0,52	Reject on feed rate
I	9,2	90,8	7,2	7,4	17,7	10,1	4,73	0,02	0,026	0,30	3,6	1,30	23,5	1,00	3,58	0,48	Accept
J	8,3	91,7	6,7	6,7	16,5	10,0	4,95	0,03	0,026	0,30	3,4	1,23	24,2	1,06	3,61	0,52	Reject on feed rate
Mean*	8,6	91,4	7,7	7,6	17,9	10,7	4,66	0,01	0,025	0,31	3,4	1,27	24,4	0,98	3,62	0,49	
s*	0,8	0,8	0,6	0,5	1,1	0,3	0,26	0,004	0,001	0,01	0,2	0,09	1,2	0,03	0,05	0,03	
Number accepted	5	5	5	5	5	6	5	6	7	7	7	6	5	6	5	5	

Conct. = Concentrate
Tail. = Tailing
*The mean and the standard deviation (s) were calculated only for the tests that were accepted as having correct feed rates.
The mean and the s of the feed rate in all 10 tests are 1,00 and 0,05 respectively.

TABLE V
CONSTITUENTS OF THE MARKER MINERALS

Mineral	Marker element	MgO %
Cassiterite	74,33% Sn	0,1
Baddeleyite	77,7% ZnO_2	2,1
Chromite	44,93% Cr_2O_3	6,0
Magnesite	96,01% MgO	96,01
Apatite	35,78% P_2O_5	0,8

TABLE VI
RESULTS OF TESTS ON DIFFERENT SIZE FRACTIONS

Size, μm	d_{50}	E_p	Recovery of Cr_2O_3, %
>300	3,05	0,402	78,5
<300 >212	3,33	0,440	93,3
<212 >150	3,18	0,415	94,1
<150 >106	2,93	0,327	95,8
<106 >75	2,78	0,238	94,9
<75 >53	2,73	0,188	91,3
<53 >38	2,88	0,515	77,8
<38	3,76	1,853	50,7

sity of separation decreases because more particles of a lower density report to the concentrate.

For the particles finer than 53 μm, the d_{50} and E_p values show an increase, which is accompanied by a corresponding decrease in the recovery of chromite. This is because the gravitational force is not sufficiently strong to cause the finer particles to sink fast enough, and consequently they are caught up in the faster-flowing pulp. The increase in the d_{50} values indicates that fewer particles of medium density are recovered; this is confirmed by the decrease in the recovery of Cr_2O_3.

Comparison of the Two Spiral Concentrators

Comparative tests were carried out on the efficiency of the two spiral concentrators. In the first three groups of tests on the waterless spiral concentrator (Fig. 2), a relatively small amount of concentrate was taken (about 5 per cent). The second three groups of tests were conducted under the same conditions, but a larger amount of concentrate (about 20 per cent) was taken because there was no clear demarcation between the concentrate and the tailing. (Such a demarcation is visible with multiport spiral concentrators using wash water.) The concentrate was flung out in waves, and the larger concentrate cut was intended to collect this displaced concentrate (Fig. 2).

In the individual tests on the two spiral concentrators at the same feed rate but at different pulp densities, the E_p values followed no pattern (Table VII). However for the waterless spiral concentrator, an increase in the feed rate tended to be accompanied by a decrease in the E_p values, i.e. in an improvement in the efficiency of separation. This tendency was not apparent for the multiport spiral concentator.

A comparison of the results of the tests for the waterless spiral concentrator on large and small concentrate cuts (Table VII) shows that, for a small amount of concentrate, the d_{50} and the E_p both increase, resulting

TABLE VII
COMPARISON OF THE RESULTS FOR THE TWO SPIRAL CONCENTRATORS

Spiral concentractor	Mass of conct. %	Feed rate t/h	Pulp density % solids	d_{50}	E_p	Mean	Standard deviation (s)
Waterless, small amount of conct.	5,7	0,72	18	4,23	0,603		
	6,3	0,74	29	4,08	0,515		
	4,8	0,77	39	4,40	0,590	0,575	0,035
	6,0	0,74	41	4,31	0,578		
	5,2	0,78	55	4,28	0,590		
	6,8	0,99	19	4,15	0,552		
	4,9	0,97	29	4,40	0,477	0,492	0,062
	5,8	1,02	36	4,21	0,527		
	4,8	1,00	48	3,93	0,412		
	7,9	1,28	20	3,53	0,490		
	5,9	1,25	30	3,81	0,425	0,486	0,047
	6,1	1,32	37	3,93	0,490		
	5,4	1,28	48	4,10	0,540		
Waterless, large amount of conct.	17,6	0,74	18	3,08	0,402		
	23,6	0,72	29	3,03	0,365		
	20,8	0,79	39	3,13	0,415	0,399	0,024
	21,2	0,73	41	3,10	0,415		
	16,9	1,05	19	3,10	0,415		
	14,6	1,01	29	3,31	0,477	0,427	0,045
	23,4	1,05	36	3,08	0,390		
	17,7	1,31	30	3,08	0,315		
	24,2	1,24	37	3,03	0,290	0,312	0,021
	24,6	1,24	41	3,03	0,302		
	23,4	1,28	48	3,05	0,340		
Multiport	8,1	0,94	21	3,61	0,387		
	7,6	0,86	28	3,53	0,377	0,371	0,019
	7,6	0,93	44	3,63	0,350		
	7,1	1,07	20	3,61	0,400		
	8,1	0,98	31	3,56	0,390	0,384	0,020
	7,3	1,06	40	3,63	0,362		
	6,7	1,30	20	3,71	0,387		
	7,2	1,21	27	3,56	0,362	0,375	0,014
	6,5	1,31	36	3,76	0,387		
	6,5	1,31	45	3,73	0,362		

TABLE VIII
HEAD ASSAYS OF SYNTHETIC ORE (IN PERCENTAGES)

Marker component	Calculated head	ICP assay	XRF assay	Back-calculated head		
				Mean	Standard deviation	Range
Sn	0,74	0,86	0,72	0,79	0,18	0,60 to 1,56
ZrO_2	0,78	0,53	0,72	0,78	0,14	0,56 to 1,11
Cr_2O_3	1,80	1,89	2,05	1,71	0,23	0,97 to 2,36
MgO	3,84	4,26	4,08	4,02	0,36	2,98 to 5,38
P_2O_5	1,79	1,72	1,75	1,87	0,14	1,40 to 2,43

XRF = X-ray fluorescence
ICP = Inductively coupled plasma

in a shift of the Tromp curve to the left. The large amount of concentrate contains more materials of lower density (which reduces the d_{50}) but includes the part of the high-density material that is being flung outwards into the pulp stream (Fig. 2). The results of the tests on the different size fractions (Table VI) showed that more of the extremely coarse and extremely fine particles are recovered in a large amount of concentrate; the efficiency is therefore better but at a lower density of separation (or grade of concentrate).

A comparison of the test results for the multiport spiral and the waterless spiral concentrators on a small amount of concentrate (Table VII) reveals that the E_p values for the multiport spiral concentrator are lower, and hence that its efficiency is higher, possibly because it has wash water to wash away the gangue particles. However, this increased efficiency should lead to an increase in the density of separation, which it does not. The only explanation for this anomaly is that the multiport spiral concentrator was cutting out a concentrate of higher mass than the waterless spiral concentrator, thus decreasing the d_{50}.

Another way in which such test results can be compared is in graphs showing chromite grade versus recovery. The results for cassiterite and baddeleyite were not plotted because the range of recoveries was too small. The grade–recovery plot for chromite (Fig. 6) shows that markedly better results were obtained on the spiral concentrator with multiports and wash water than on the waterless spiral concentrator.

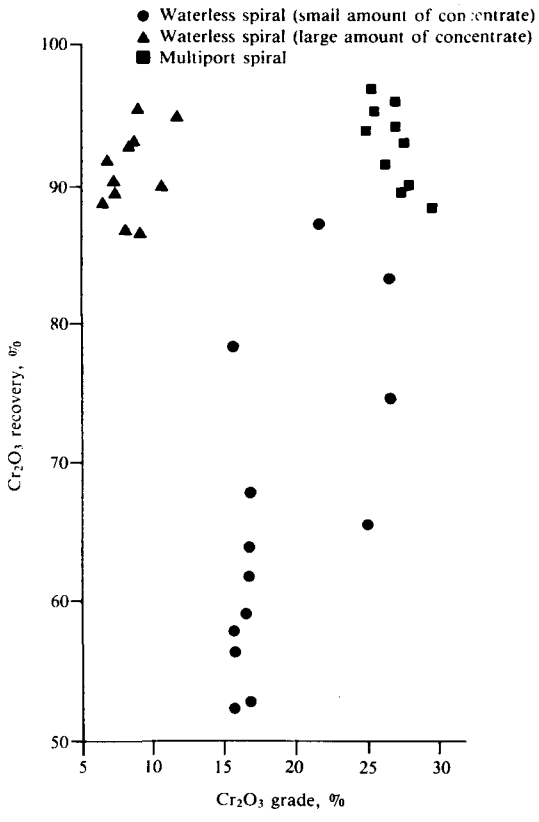

Fig. 6—Chromite grade versus recovery

Analytical Problems

The various marker minerals were added to the silica in the precise proportions listed in Table I, and, from these and the compositions of the marker minerals, head assays were calculated for the synthetic ore. Different methods were used for the analysis of the synthetic ore, and Table VIII shows that the results cover a fairly wide range.

If X-ray methods are used, there is interference between the spectral lines of some of the elements, which can be partially corrected for by the use of correction factors. Wet-chemistry methods can be used, but the separation techniques are lengthy and complicated. Analysis by methods using inductively coupled plasma require dissolution of the sample, during which zirconium phosphate precipitates out, causing incorrect values to be obtained.

The suite of marker minerals used is clearly unsuitable, and a different combination of marker minerals was used in the tests on the pinched sluice and shaking table.

Pinched Sluice and Shaking Table

As only a few tests were conducted on these separators, fairly small amounts of synthetic ore were required. A mixture of silica and chromite was used as the basic ore, to which small amounts of marker minerals were added. The various components used in this phase of the investigation are listed in Table IX.

TABLE IX
COMPONENTS USED IN TESTS ON PINCHED SLUICE AND SHAKING TABLE

Marker mineral	Origin	Component determined	Relative density
Galena	U.S.A.	Pb	7,25
Scheelite	Zimbabwe	WO_3	5,89
Magnetite	Phalaborwa	-	4,72
Chromite	Rustenburg	Cr_2O_3	4,46
Chalcopyrite	Messina	Cu	4,14
Apatite	Foskor	P_2O_5	3,20
Fluorspar	Naboomspruit	CaF_2	3,11
Silica	Pietersburg	-	2,65

Two separate mixtures were made up, one of which was slightly coarser than the other. For the coarse mixture, fluorspar was used as the marker mineral with a density of about 3; the fine mixture contained apatite instead of fluorspar. The size range of the fine mixture was narrower than that of the coarse mixture (Table X). The coarse mixture had a silica content of 80 per cent, whereas extra chromite was added to the fine mixture to give a silica content of 75 per cent. Magnetite was recovered from the products by magnetic separation and did not require analysis.

Because the size distribution of the marker minerals and the basic ore should be fairly similar, each marker mineral was screened into size fractions, and these size fractions were then combined in the correct proportions to give a size distribution similar to that of the silica.

The pinched sluice used in the experiments was constructed of wood and was about 1 m long, narrowing from 250 mm at the feed end to 25 mm at the discharge end. The concentrate was removed through a single slot at the constricted end of the sluice.

As the pulp flows down the sluice, the particles stratify as in any flowing-film concentrator, but this stratification is accentuated by the gradual reduction in width due

TABLE X
SIZE ANALYSIS OF COMPONENTS FOR TESTS ON PINCHED SLUICE AND SHAKING TABLE

Size μm	Coarse mixture, % by mass		Fine mixture, % by mass	
	Silica	Chromite	Silica	Chromite
>300	16,1	21,0		
<300 >212	21,2	27,1		
<212 >150	15,7	17,6		
<150 >106	13,2	12,6	18,5	16,5
<106 >75	8,3	6,4	27,4	27,0
<75 >53	6,4	4,6	22,7	26,6
<53 >38	7,3	3,2	26,6	28,1
<38	11,8	7,4	4,7	1,8

to the pinching action of the sluice walls. Separations on a pinched sluice are somewhat impaired by the development of cross-currents caused by the constricting effects of the side walls.

Because shaking tables are still considered to be among the most effective gravity-separation devices, a small laboratory table was used so that the results obtained could be compared with those for the pinched sluice.

Comparative Tests

Four tests were conducted on each separator, i.e. two tests on each synthetic mixture (Table XI). Even though more experiments would have been desirable, some trends can be observed from the limited conditions investigated.

TABLE XII
EFFICIENCY OF SEPARATION AT VARIOUS PARTICLE SIZES

Separator	Size, μm	d_{50}	E_p	I
Pinched sluice	<150 >106	3,65	1,200	0,451
	<106 >75	3,90	1,315	0,455
	<75 >53	4,10	1,375	0,442
	<53 >38	4,20	1,925	0,600
Shaking table	<150 >106	3,700	0,445	0,165
	<106 >75	3,725	0,450	0,166
	<75 >53	3,775	0,475	0,172
	<53 >38	3,820	0,525	0,186

TABLE XIII
RECOMMENDED MINERALS FOR USE AS MARKERS

Marker mineral	Approximate relative density	Component determined
Silica	2,65	
Apatite	3,2	P_2O_5
Sphalerite	4,0	Zn
Chalcopyrite	4,2	Cu
Ilmenite	4,7	TiO_2
Magnetite	4,7	(Mag. sepn)
Pyrolusite	4,9	MnO_2
Chromite	5,1	Cr_2O_3
Arsenopyrite	6,0	As
Cassiterite	6,9	SnO_2
Galena	7,5	Pb

TABLE XI
COMPARISON OF RESULTS FOR PINCHED SLUICE AND SHAKING TABLE

Synthetic feed	Slope	Slot width mm	Feed rate kg/h	Pulp density solids %	Conct. mass %	d_{50}	E_p	I
Pinched sluice								
Coarse	14°20′	1	900	53	45,5	3,15	1,500	0,698
Coarse	14°20′	1	846	53	37,1	3,55	1,475	0,578
Fine	11°35′	5	774	54	37,4	4,0	1,210	0,404
Fine	11°35′	5	702	54	43,3	3,5	1,025	0,410
Shaking table								
Coarse	6°	17	32	–	21,5	3,53	0,590	0,234
Coarse	6°	17	29	–	25,5	3,45	0,637	0,261
Fine	6°	17	29	–	27,7	3,80	0,490	0,175
Fine	6°	17	22	–	27,5	3,75	0,475	0,173

A lower feed rate gave rise to increased efficiency of separation by the pinched sluice, and a fine feed with a narrow spread of sizes was beneficial. The shaking table was found to be more efficient than the pinched sluice in all the tests. Minor changes in the feed to the table caused only minor changes in the probable error and the imperfection, but a narrow range of particle sizes in the feed was found to be beneficial.

Efficiency at Different Particle Sizes

The products of the tests conducted on the fine synthetic mixture on both separators at an increased feed rate were screened into four size fractions. These fractions were analysed individually, and Tromp efficiency curves were drawn. The results (Table XII) show that, as the particle size decreased from 150 to 38 μm, the efficiency of separation decreased but the d_{50} increased.

Conclusions

In the tests on the spiral concentrators, the probable error showed that the multiport spiral concentrator is a more efficient separator than the waterless spiral concentrator. It should be stressed, however, that only one make of waterless spiral concentrator was tested, and that the results apply only to that particular concentrator.

An increase in the feed rate to the waterless spiral concentrator improved the efficiency of separation (within the range tested), and the collection of a large amount of concentrate increased the efficiency but at a lower grade.

The efficiency of the separation of coarse particles was low but improved as the particle size decreased to 53 μm. For finer sizes, the efficiency again decreased. Also, a decrease in the efficiency of separation was accompanied

by an increase in the density of separation (or grade of concentrate).

It is of interest that the spiral concentrators gave a lower probable error (E_p) than the shaking table, although on different synthetic mixtures and not necessarily at optimum conditions.

Another point of interest is that the efficiency of the pinched sluice was much lower than that of the spiral concentrators. This is confirmed in industry, where a slurry has to be passed over pinched sluices or cone concentrators several times for an efficient *primary* separation to be obtained, whereas a single pass on a spiral concentrator will often suffice.

The choice of the correct marker minerals is essential in this type of investigation. Table XIII gives a list of marker minerals that can be analysed successfully by X-ray methods. The products of separation should be dried with care at low temperatures to prevent the oxidation of any sulphide minerals, especially if they are to be blended and re-used in future tests.

It is suggested that the tests should be carried out in open circuit with well-blended feed material to ensure a constant feed to the separators. Analytical methods may not be accurate enough for the analysis of low-grade tailings products, and concentrate and feed values may be needed for the calculation of the Tromp distribution curve.

Finally, the method is an interesting tool for the evaluation of gravity separators, and could possibly be used in the establishment of optimum conditions for the treatment of a conventional ore, and to provide operator confidence in the correct operation of a concentrator.

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Reference

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Automation in mining and mineral processing

The International Federation of Automatic Control (IFAC) is to hold its 5th Symposium on Automation in Mining, Mineral and Metal Processing in Tokyo from 24th to 29th August, 1986.

Organized by The Society of Instrument and Control Engineers, Japan, the Symposium has as its scope the latest development of factory automation in the mining, metal- and mineral-processing industries.

Papers are invited on the following or related topics that will promise new contributions to the field of automation applications and development.

- **Mining and Mineral Processing**
  Prospecting; material handling; transportation; mining equipment and methods; crushing; grinding; classifying; flotation and other concentrating methods for metallic and non-metallic minerals; agglomeration; kiln operation; other beneficiation methods.
- **Metal Processing**
  Sintering; pyrometallurgy; direct reduction; electrolysis; blast furnaces; steel-making; continuous casting; reheat furnaces; rolling mills; annealing and finishing.
- **Ceramic Processing**
  Powder preparation; forming; firing; machining; pulverizing; handling; packaging (for glasses, cements, refractories, and other ceramics).
- **Process Control**
  Control methods; economics of process control; energy and environmental control systems; control system design; case studies of control systems; sequencing and batch control systems; operator interface.
- **Production Control and Management**
  Systems engineering; feasibility studies; information systems; scheduling; production planning; maintenance scheduling and management; hierarchical systems.
- **Systems and Equipment**
  Distributed systems; reliability; computer control hardware and software; simulation; computer-aided design; use of digital control equipment (programmable logic controllers, microprocessors, computers); data-transmission and telemetry systems.
- **Control Theory**
  Identification; modelling; optimization and optimal control. Sensitivity and stability analysis; control algorithms; adaptive systems and self-tuning regulators.
- **Mechanical Automation**
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