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BENEFICIATION OF A CHINESE PHOSPHATE ORE BY DENSE MEDIUM SEPARATION - FINAL REPORT

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BENEFICIATION OF A CHINESE PHOSPHATE ORE BY

DENSE MEDIUM SEPARATION

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SUMMARY

This report presents the results of testwork carried out on a Chinese sedimentary phosphate ore which is reported to be a low grade phosphate (15-25% P_2O_5) with a gangue of mostly dolomite (4-9% MgO).

Data from heavy liquid analysis of the sized feed, processed with the WSL predictive computer software, has shown that the target recovery of 78% of the phosphate in the ore, at specification with respect to MgO (<1.5%) and Al_2O_3 + Fe_2O_3 (<3%) can be achieved by dense media pre-concentration using the TRI-FLO separator.

Pilot plant testwork using a TRI-FLO 250 dense media separator has shown that it is possible to recover 83.1% of the phosphate in the ore (89.5% on dense media feed) at a grade of 33.9% P_2O_5 and within specification to a combined sinks product.

64.5% of the ore weight was rejected to dense media float and -0.5 mm fines products. This result was achieved using a media of ferrosilicon cyclone 40 and magnetite, in which the magnetite content was 33.5%, fed to the TRI-FLO at a density of 2.56 g ml^{-1} and media input pressure of 15 psi (103 KN/m^2).

When the magnetite content of the media is increased to 50.8% it is possible to recover 81.2% of the phosphate in the ore to a combined sinks product assaying 34.5% P_2O_5 and within specification with respect to MgO, Al_2O_3 and Fe_2O_3. This is associated with a total weight rejection of 63. to dense media floats and -0.5 mm fines products. This result was achieved with the media fed to the TRI-FLO at a density of 2.76 g ml^{-1} and media input pressure of 15 psi.

Separation efficiency as represented by the Ep values taken from the Tromp Curves was 0.032 for the test with 33.5% magnetite in the media and 0.036 for the test with 50.8% magnetite media.

Using 50.8% magnetite in the media it should be possible to further improve recovery of P_2O_5 to a combined sinks product by lowering feed media density and thus the separation d_{50}.

It has not been possible to quantify media losses from the testwork carried out. However, a properly designed plant, with adequate screening area for product washing, will exhibit media losses in the range 80-400 grammes of media per tonne of ore processed through the plant. This figure can rise to 1000 grammes per tonne treated with a poorly designed and badly operated plant.

CONTENTS

1. INTRODUCTION

This report details the work carried out on UNIDO contract 88/74/HQ (Development of new technologies for phosphate achievement for the fertilizer industry). This testwork aimed to establish the amenability of a Chinese sedimentary phosphate ore to dense medium pre-concentration. The ore was a low grade phosphate (15-25% P_2O_5) with a gangue mostly of dolomite (4-9% MgO). Preliminary studies in China indicated that dense media preconcentration could potentially give a product assaying more than 33% P_2O_5 and less than 1.5% MgO by this process alone. A further specification limit was that the combined Al_2O_3 and Fe_2O_3 content should be less than 3.0% and target recovery is 78% of the phosphate.

The prime objectives of the work programme were:

1. Laboratory heavy liquid testwork and analysis to confirm that the targets indicated by the Chinese testwork are attainable and to determine the optimum separation density, medium composition and predicted plant performance.

2. To assess the optimum media composition, in terms of magnetite to ferrosilicon content, through settling tests and viscosity measurements.

3. Pilot plant testwork using a 250 mm TRI-FLO dense medium separator to prove that the predicted plant performance is attainable at plant scale.

4. Computer analysis of the data derived from the pilot plant testwork to establish the sensitivity of the process performance to small changes in the separating density.

5. The training of two Chinese engineers in the operation of a pilot plant TRI-FLO and in the associated heavy liquid analytical techniques.

6. To make recommendations on

   (i) scale up from pilot scale to full scale operation
   (ii) the beneficiation of the <0.5 mm raw material
   (iii) treatment of waste water

2. SAMPLE

A 21 tonne sample of sedimentary phosphate ore was received from China. This sample was all -15 mm in size, therefore no size reduction prior to testwork was necessary. The sample was contained in 527 bags each of which contained 40 kg. Each of these bags was sampled to give an accumulated head sample of 630 kg. This head sample was further sampled by cone and quartering to give 52 kg for the initial heavy liquid analysis and 31.5 kg for a head analysis sample. The head sample was reduced to -1.7 mm before a final 1 kg was taken for analysis.

The results of the head analysis are given below:

P	Mg	Al	Fe	Ca	Insol	SiO_2	F
9.71	2.04	1.03	1.13	25.10	21.5	14.5	2.05

P_2O_5	MgO	Al_2O_3	Fe_2O_3	CaO
22.24	3.38	1.95	1.62	35.12

Note: This -15 mm sample was supplied by the Chinese Institute as an alternative to the material requested in the original proposal. Since material in the size range 15-50 mm was not available, prediction of its dense media potential was not possible (heavy liquid analysis is necessary in order to predict).

The mineralogical analysis of the ore was effected exclusively through heavy liquid fractionation and assay of the fractionated products. Classical microscope techniques fail to provide the data necessary for the subsequent performance prediction; serving only as a guide to the choice of fractionation densities and as a rapid but crude quality control check on the fractionation itself.

3. LABORATORY TESTWORK

Heavy Liquid Analysis

Heavy liquid analysis of the Chinese Phosphate ore consisted of separating 52 kg of the ore into five size fractions, -15 +10 mm, -10 +5 mm, -5 +2 mm, -2 +0.5 mm and -0.5 mm. Each of the four +0.5 mm fractions was subjected to density fractionation at 2.70, 2.80, 2.90, 2.95, 3.00, 3.05 and 3.10 g ml^{-1} using tetrabromoethane and di-iodomethane (methylene iodide) based solutions and using tri-ethyl phosphate as the solution diluent. Each size/density fraction and the -0.5 mm material were analysed for P, Al, Fe and Mg.

The purpose of this procedure is to determine the grades and distributions of the elements in the chosen size/density fractions and to use this information to predict the likely separation performance in the TRI-FLO separator.

The four sizes chosen are the sizes which past experience has shown to be necessary and generally sufficient to quantify the variation in separating efficiencies with particle size (ie varying Tromp curves) for materials of a similar top size to that considered here. It has also been observed, that for ores containing coarser material, the Tromp curves for the +15 mm fraction generally closely follows that for the -15 +10 mm fraction.

The -0.5 mm particles have been excluded from the heavy liquid analysis because this size is not normally suitable for dense media separation owing to the fact that the separation efficiency at this size is poor (Ep typically >0.1 g ml^{-1}) and also because the presence of ore fines in the separator will adversely affect the required media conditions by increasing its viscosity which reduces separation efficiency at the coarser sizes.

The results of the size analysis are given in Table 1 and the metallurgical balance for the heavy liquid analysis in Table 2. The densities determined for the <2.7 g ml^{-1} and >3.3 g ml^{-1} products are given in parenthesis, the results of the head analysis are reported as Head (meas).

From the size/heavy liquid data shown in Table 2 it can be seen that 92.9% of the phosphate in the ore is contained in the +0.5 mm fraction of the ore and that 82.5% of it is in the >2.9 g ml^{-1} density fraction. Further examination shows that 65% of the Al, 65% of the Fe and 88.6% of the Mg are contained in the <2.9 g ml^{-1} density fraction of the +0.5 mm part of the ore. This analysis indicates that the liberation characteristics of the ore are satisfactory and that there is good potential to preconcentrate the Chinese phosphate using the Dense Medium Separation process.

TABLE 1. - Head Size Analysis

Size mm	Wt %
-15 +10	17.27
-10 + 5	39.53
-5 + 2	23.14
-2 + 0.5	9.88
-0.5	10.18

Total sample wt = 51.606 kg

3.1 Dense Media Prediction

The stage/density distributions of weight, P, Al, Fe and Mg reported in Table 2 for each size range, were processed using the WSL computer simulation programs to predict distributions to float and sink products which would be achieved at separation densities of 2.8, 2.84, 2.88, 2.92, 2.96, 3.00 and 3.04 g ml^{-1}.

The computer contains models of the Tromp curves for the four particle size ranges examined. These models were derived from experience and data collected from an industrial Tri-Flo operation and from previous dense medium pilot plant testwork carried out at Warren Spring Laboratory. The computer prediction is effected by compounding the model Tromp curves with the measured SG distributions. For a prediction, both must be available: For the prediction to be extended to sizes coarser than 15 mm, the -15 +10 mm model curve can be taken as a good first approximation to the real Tromp curve. Prediction for this coarser material was not possible with the samples supplied from China as they had been pre-crushed to -15 mm which precluded the necessary SG data from being collected.

The predicted dense medium performance is determined as follows: consider for example the -15 +10 mm size fraction in Table 2 and in particular its stage distribution of weight to the density fractions. This information processed by the computer models gave the results shown in Table 2A. For each chosen separation D50 the computer calculates a % to sink product and a % to float product. For a separation density D50 of 2.8 g ml^{-1}, 60.61% of the weight of the -15 +10 mm size fraction is predicted to report to the sink product. For a 2.84 g ml^{-1} separation density 55.72% of the weight of the -15 +10 mm fraction would report to the sink product. These values are tabulated in Table 3 for all the separation densities chosen for all size ranges examined. The predicted distributions given in Table 3 are further processed to give the weight distributions of say the -15 +10 mm size fraction to the sink product based on total ore (Head) and again based on +0.5 mm feed to the dense medium separator (DMS). This process is repeated for each size fraction. The overall distribution to the sink product based on the whole ore (Head) is obtained by summing the individual % to sink values (Head) for each size fraction.

Similarly the predicted distributions of P, Mg, Al and Fe have been determined and are reported in Table 3. Only the predicted distributions of weight in the -15 +10 mm size fraction have been given in Table 2A to show how Table 3 has been developed.

Also reported in Table 3 are grades of products which would result at the separation densities chosen. Since the important consideration is the production of a phosphate concentrate only the distributions to the sink product are reported in Table 3.

It should be noted that Table 3 presents recoveries relative to the total ore (Head) and also relative to what would be the dense medium feed (DMS) which excludes the fraction of material which is <0.5 mm.

The required specification for a phosphate concentrate is that the product should be greater than 33% P_2O_5, contain 1.5% MgO or less and that the combined Al_2O_3 and Fe_2O_3 should be 3.0% or less.

Examination of the predicted P_2O_5 and MgO grades in Table 3 shows that at a separation density of 2.88 g ml^{-1} a phosphate product assaying 34.03% P_2O_5 containing 1.35% MgO could be achieved. This is associated with a combined Al_2O_3 and Fe_2O_3 grade of 1.89% (well within specification) and the separation would result in 52.53% of the weight of DMS feed reporting to the sink. The recovery of phosphate to the sink product at this density is 77.75%, just slightly below the 78% target. Since the MgO level is 1.35% at a separation density of 2.88 g ml^{-1} and 1.93% at 2.84 it should be possible to achieve the target recovery of 78% with a slightly lower separation of 2.87 g ml^{-1}.

TABLE 2A. - Heavy Liquid Analysis

Chinese Phopshate Ore -15 +10 mm Weight Distributions

Calibration Curve A

D50 Separation Density 2.8

Heavy Liquid Analysis		Prediction			
Average Density	% Weight	Density -D50	Efficiency Factor	% Sinks	% Floats
2.58	26.54	-0.22	0.3	0.07	26.47
2.75	12.97	-0.05	21.0	2.72	10.25
2.85	11.30	0.05	84.5	9.55	1.75
2.93	6.83	0.13	95.9	6.55	0.28
2.98	12.22	0.18	97.8	11.95	0.27
3.03	19.17	0.23	98.7	18.91	0.26
3.08	10.32	0.28	99.0	10.21	0.10
3.17	0.65	0.38	99.4	0.65	0.00
	100.00			60.61	39.38

TABLE 2A. - Continued

D50 Separation Density 2.84

2.58	26.54	-0.26	0.1	0.03	26.51
2.75	12.97	-0.09	8.5	1.10	11.87
2.85	11.30	0.01	61.0	6.89	4.41
2.93	6.83	0.09	92.4	6.31	0.52
2.98	12.22	0.14	96.4	11.77	0.45
3.03	19.17	0.19	98.0	18.79	0.38
3.08	10.32	0.24	98.8	10.18	0.13
3.17	0.65	0.34	99.3	0.65	0.00
	100.00			55.72	44.27

D50 Separation Density 2.88

2.58	26.54	-0.30	0.1	0.02	26.52
2.75	12.97	-0.13	3.0	0.39	12.58
2.85	11.30	-0.03	32.0	3.62	7.68
2.93	6.83	0.05	84.5	5.77	1.06
2.98	12.22	0.10	93.5	11.43	0.79
3.03	19.17	0.14	96.8	18.56	0.61
3.08	10.32	0.19	98.2	10.12	0.19
3.17	0.65	0.29	99.1	0.65	0.01
	100.00			50.54	49.45

D50 Separtion Density 2.92

2.58	26.54	-0.34	0.0	0.01	26.53
2.75	12.97	-0.17	0.9	0.12	12.85
2.85	11.30	-0.07	13.5	1.53	9.77
2.93	6.83	0.01	61.0	4.17	2.66
2.98	12.22	0.06	87.1	10.64	1.58
3.03	19.17	0.11	94.5	18.12	1.05
3.08	10.32	0.15	97.2	10.02	0.29
3.17	0.65	0.25	98.9	0.64	0.01
	100.00			45.24	54.75

D50 Separation Density 2.96

2.58	26.54	-0.38	0.0	0.01	26.53
2.75	12.97	-0.21	0.3	0.04	12.93
2.85	11.30	-0.11	5.0	0.56	10.74
2.93	6.83	0.03	32.0	2.19	4.64
2.98	12.22	0.02	70.0	8.55	3.67
3.03	19.17	0.07	89.2	17.10	2.07
3.08	10.32	0.12	95.2	9.82	0.49
3.17	0.65	0.21	98.6	0.64	0.01
	100.00			38.91	61.08

TABLE 2A. - Continued

D50 Separation Density 3.00

2.58	26.54	-0.42	0.0	0.01	26.53
2.75	12.97	-0.25	0.1	0.02	12.95
2.85	11.30	-0.15	1.8	0.20	11.10
2.93	6.83	-0.07	13.5	0.92	5.91
2.98	12.22	-0.02	37.0	4.52	7.70
3.03	19.17	0.03	76.0	14.57	4.60
3.08	10.32	0.08	91.0	9.38	0.93
3.17	0.65	0.17	97.8	0.64	0.01
	100.00			30.26	69.73

D50 Separation Density 3.04

2.58	26.54	-0.46	0.0	0.01	26.53
2.75	12.97	-0.29	0.1	0.01	12.96
2.85	11.30	-0.19	0.5	0.06	11.24
2.93	6.83	-0.11	5.0	0.34	6.49
2.98	12.22	-0.06	17.0	2.08	10.14
3.03	19.17	-0.01	44.0	8.43	10.74
3.08	10.32	0.04	81.0	8.35	1.96
3.17	0.65	0.13	96.3	0.63	0.02
	100.00			19.91	80.08

3.2 Medium Characteristics

Two factors are considered when choosing medium for use with heavy medium cyclone separators:

1. the medium must be capable of achieving good efficiency at the required density of separation

2. the basic cost of the medium.

For metalliferous ores and industrial mineral separations, ferrosilicon is most frequently used, either on its own or in combination with magnetite. Ferrosilicon is often used with magnetite to reduce costs and also to condition the medium to reduce large density differentials between the sink and float medium streams which can occur at low medium viscosities when attempting to achieve lower separation densities.

Ferrosilicon is available in two forms; (1) Ground ferrosilicon, obtained by cooling liquid ferrosilicon, followed by crushing and grinding to the desired specification and (2) Atomised ferrosilicon obtained by cooling the molten product in a stream of air or steam with the resulting particles quenched in water, dried and sized.

Although more expensive than the ground product, atomised ferrosilicon has improved corrosion properties, allows higher volume concentrates to be used at lower viscosity and, because the rounded surfaces of the atomised variety minimises adhesion between medium and the ore, a significant reduction in medium losses can be achieved.

Ferrosilicon cyclone 40, cyclone 60 (supplied by Hoechst) and magnetite cover all the dense medium requirements of the Laboratory. Magnetite alone is used for separation D50's of less than 2.4 g ml^{-1}, ferrosilicon cyclone 40 and magnetite for separation D50's in the range 2.4 to 3.0 g ml^{-1}, cyclone 40 alone for separation D50's 3.0-3.3 g ml^{-1} and cyclone 60 for separation D50's 3.3-3.6 g ml^{-1}.

Heavy liquid analysis of the Chinese phosphate ore and predicted performance (Table 3) indicates a separation density D50 of 2.88 g ml^{-1} is required to achieve the required specification target. Therefore a combination of ferrosilicon cyclone 40 and magnetite is likely to be the most suitable medium to use.

The size distributions of ferrosilicon cyclone 40 and magnetite are given in Appendix A.

Settlement Tests on Medium Suspensions

To assess the settlement characteristics of the media being used for the testwork a series of cylinder settlement tests were carried out at two media densities covering the range of separation densities expected to be used on the processing of the ore (2.8 and 3.0 g ml^{-1}). The tests were carried out with a range of magnetite/ferrosilicon ratios.

From the curves obtained a set of mudline initial settlement rates were established. These are given in Table 4.

TABLE 4. - Initial Settlement Rates of Mudlines for Various Media

% Magnetite	Mudline Settlement cm/min		% Solids by volume	
	SG 2.8	SG 3.0	SG 2.8	SG 3.0
0	1.50	1.01	31.1	34.5
12.5	0.49	0.32	32.7	36.4
25	0.24	0.16	34.6	38.3
50	0.14	0.05	39.2	43.4
75	0.03	-	41.4	-

In previous studies[1] the muds had shown greater stability at lower volume concentrations but had not been as stable at the higher volume concentrations. This made it difficult to judge the appropriate mud to use for the separation, although mudline settlement rates of >1 cm/min were previously considered to be too high and those below 0.1 cm/min were always associated with highly viscous conditions. On this basis some magnetite would be recommended for this separation but it is likely that 50% magnetite will be the limit and then only at the lower operating densities.

[1] Separation Efficiency in dense media cyclones D N Collins et al. Trans IMM Section C. Vol 92 Mar 1983.

Attempts to measure the viscosity of the media were not successful due to segregation effects. However, without knowing the appropriate shear rates prevalent in the TRI-FLO, viscosity measurement is meaningless. In addition mudline settlement or density differential (P_u - P_0) within the separator are good guides to both media instability and high viscosity problems. For the phosphate beneficiation it would appear that volume concentrations of 33-40% (approx) are suitable. Further discussion of medium characterisation based on the testwork is given in Section 4.5.

Storage of Medium

The Warren Spring dense medium pilot plant incorporates a spiral classifier into the dense medium circuit. This has two functions (1) it is used to return medium recovered from the washing section to the main medium cone during operation and (2) it is used to store the bulk of the medium during shutdown. Whenever the dense medium plant is shutdown for more than a day the medium must be stored at pH greater than 11 in order to minimise corrosion. If storage longer than 3-4 weeks is anticipated then it is recommended that the medium is removed from the circuit and dried.

4. PILOT PLANT TESTWORK

4.1 TRI-FLO Circuit Details

The DMS separator used for the pilot plant testwork was a Tri-Flo 250 which has an upper feed size limit of 18 mm and is capable of processing ore at a rate of 15 tonnes per hour. Feedrate of the pilot plant testwork carried out at Warren Spring Laboratory on the Chinese Phosphate ore was limited to 3-4 tonnes per hour because the associated peripheral equipment had a lower throughput capability than the Tri-Flo itself. This was because the Tri-Flo separator had been installed into an existing dense medium cyclone circuit designed to handle ore at the rate of 4-5 tonnes per hour maximum. The important rate limiting factors are the areas of the primary, medium drainage and product washing screens.

Total medium requirement is 80 cubic metres per hour with 50% of this to the inlet of the first section of the separator, 45% to the second section and 5% of the medium directed to the ore input chute.

The separator is fitted with sink outlet reducers which vary the back pressure at the sink outlet and thus can vary the sink product quality. Sink outlet reducers may be required to optimise product quality during commissioning but are not always required. The greater part of the testwork carried out at Warren Spring Laboratory was conducted without reducers fitted with a limited amount of testwork with them in to demonstrate their effect.

The circuit used in the pilot plant testwork is shown in Fig 1. The incoming ore was passed to a 0.5 mm flat bed vibrating screen onto which powerful water sprays were directed. This ensured that most of the -0.5 mm material was removed from the feed before it was introduced to the separator. From the primary screen the +0.5 mm was conveyed to the Tri-Flo inlet chute by bucket elevator. This was necessary because of the physical constraints of the Warren Spring installation. Normally the circuit should be designed to allow direct entry from the primary screen.

In the first section of the separator the main separation of the heavier minerals takes place according to the medium density chosen. This results in a "sinks 1" product. The remaining particles and medium then proceed to the second section of the separator where a second separation takes place which effectively is a scavenging stage giving a product "sinks 2". The remaining particles and medium leave the separator and are called the "float" product of the process.

All the products leaving the separator are associated with large amounts of medium. This was removed by passing all three products (separately) over an inclined 0.5 mm wedge bar screen. Here the medium from both sinks products was returned to the storage cone while the medium associated with the float product (at a density lower than the medium set density) was pumped to a densifying cyclone. The thickened under-flow from this cyclone was returned to the main medium storage cone, whilst the over-flow passed to a low intensity magnetic drum separator and then to waste. Magnetics recovered from the drum separator were returned to the main medium cone via a spiral classifier. After removal of medium by the inclined screen, the three separate products passed to a three compartment product washing screen which was a flat bed vibrating 0.5 mm wedge bar screen similar to the primary washing screen. This screen was divided into two areas. The first area (about 30 cm in length) allowed medium, which had failed to pass through the inclined screen, to be returned to the main storage cone. The remainder of the screen area was fitted with powerful wash sprays which removed medium adhering to the products. The products then passed to separate storage bins. At the point of product discharge a sampling device was fitted. This consisted of three separate bins mounted on a wheeled platform which could be rolled into the path of all three products simultaneously to sample the entire streams.

The medium removed from the products on the washing screen, together with any residual -0.5 mm ore particles, was passed to a low intensity magnetic drum separator. Here the recovered medium was passed through a demagnetising coil and returned to the main storage cone via a spiral classifier.

The non-magnetic fraction of the washings from the product screen (0.5 mm) was returned to -0.5 mm material from the primary washing screen to give a total -0.5 mm fraction from the ore.

4.2 Medium Handling and Control

The medium circuit consisted of a 45 Kw motor driving a 75 mm centrifugal pump at 1100 RPM. A Kay-Ray gamma density meter was fitted on the pipe between the pump outlet and Tri-Flo inlets 1 and 2 and the by-pass return to the storage cone (see Fig 1). The gamma density meter was linked to an electro-pneumatic control system to adjust water addition for the control of medium feed density.

Start up consisted of the following procedure (refer to Fig 1). Valves A and B were closed and Valve C fully opened. The cone was filled to approximately one third of its volume with water and circulated around the by-pass line to the cone. Medium of chosen composition (ferrosilicon-magnetite ratio) was then added until a slurry density of approximately 2.6 g ml^{-1} was obtained. Filling of the cone continued adding water and medium to maintain the density at 2.6 g ml^{-1} until the cone was full. Valves A and B were then slowly opened to pass the medium to the rest of the circuit. This involved partially closing Valve C and adjustment of Valves A and B until the required pressures at the Tri-Flo medium inlets were achieved. The circuit required a large volume of medium resulting in a low level in the cone. More water and medium were added tc ensure that when all the medium circuit was full the storage cone also had a large volume of medium available. This procedure was necessary to ensure that when ore passed through the circuit, which would remove medium to the washing and recovery section, there was sufficient volume of medium to cope with circuit requirements until the medium removed by the ore was returned via the spiral classifier.

4.3 Testwork Rationale

(A) TRI-FLO Characteristics with Respect to Media and Separating Density

Establish a relationship between media density at the point of entry to the TRI-FLO (ie media feed density or input density) and the actual density of separation for solids within the TRI-FLO (ie the separating density). Assess the effects of - back pressure on the products
- input pressure on the products
- media composition with respect to overflow and underflow differentials.

(B) Main Pilot Plant Run

Having established separating density - feed density, and media composition relationships, run the pilot plant in order to process several tonnes of material. The products from these tests would then be examined in detail using size and density (heavy liquid analysis) to establish TRI-FLO characteristics in terms of the efficiency factor (E_p) and effective density of separation (d_{50}). The data could then be used to calibrate computer simulation programs which could then be used to more accurately predict the results of density changes. Detailed accounts are given below.

Testwork Details

4.3.1 (i) TRI-FLO Characteristics with Respect to Media and Feed Material

Because the TRI-FLO is a centrifugal dense media separator it was necessary to establish relationships between the density of the incoming media (input density) and the separation density within the TRI-FLO.

In principle very little segregation of the media should occur within the TRI-FLO. This can be monitored by measuring the media inlet density, the overflow density (O/F) and the densities of the two underflows sinks 1 and sinks 2. The differentials between the overflow and sinks 1 and the overflow and sinks 2 densities should be a minimum. During the testwork and based on past experience, a stable media condition was assumed for differentials less than 0.7 g ml^{-1}.

Inlet, overflow, sinks 1 and sinks 2 densities were initially measured using a Marcy pulp density can. Although the Marcy can is a relatively crude way of measuring pulp density: ie the results are subject to errors of approximately ±0.03 g ml^{-1}, it has the advantage of giving an almost instantaneous result which can show trends. It was thus considered accurate enough to detect effects such as changes in media input pressure and back pressure.

An initial relationship was established between media input density and the effective density of separation by running batch lots of ore through the circuit and collecting the products over a fixed time once equilibrium had been established. Table 5 records the results. Plotting the measured input density and, separately, the predicted separating density from the simulation against the weight proportion of products to the total sinks (Figure 2), gives an indication of the relationship between the media input density and the separation density for a particular inlet pressure and back pressure.

It should be noted that the prediction indicates an area in which to work. In practice better (or worse) results might be achieved. Data from the later pilot plant studies were used to determine the final operating conditions. The effects of the various circuit changes are detailed below.

a) Changing Media Feed Density at Constant Media Composition (ie Constant Magnetite to Ferrosilicon Ratio)

Tests DMS 1, 2 and 3 (Table 5) show that with a magnetite-ferrosilicon media containing 22.5% magnetite, decreasing the media feed density in order to achieve a separating density of 2.88 g ml^{-1} (52.5% wt of ore to sinks product) results in a widening of the differential between the overflow and underflow media densities. Tests DMS 5 and 6 also exhibit the same effect for media with higher magnetite content (26%).

b) Changing the Proportion of Magnetite to Ferrosilicon in the Media

Within the range of media input densities 2.65 to 2.81 g ml^{-1} for magnetite contents of 22.5% to 31.5%, the differentials of the overflow density to the underflow densities remained relatively constant at 0.50-0.70 g ml^{-1}. Later pilot plant runs, DMS 19-22 were conducted at 33.5% and 50.8% magnetite where the differential at the higher content was significantly lowered. The effects on the efficiency and the density of separation are discussed more fully in later sections dealing with the pilot plant runs DMS 19-22.

c) Increasing the Media Inlet Pressure to the TRI-FLO

DMS 8 and its nearest equivalent, DMS 6 show the effects of increasing the media inlet pressure to the TRI-FLO. The overflow to underflow density differentials are seen to widen quite dramatically. The effect on the separating density can be seen by referring to Figure 2 for a media feed density of 2.78 g ml^{-1} where the proportion of ore reporting to the total sinks product would be expected to be 31%. The DMS 8 test run gave a marginally higher proportion of 34.8% which is considered relatively insignificant considering the errors in pulp density measurement.

d) Increasing the Back Pressure on the Media

Within the TRI-FLO the back pressure on the media can be altered by increasing the standpipe height, on the outlets to underflows, through the addition of cylindrical pipe extenders. DMS 10 and 11 (Table 5) show the effects on underflow and overflow differentials and on the proportion of ore reporting to the sink products.

Adding two extenders to the two sink standpipes appears to reduce the differential densities between the underflow and overflow. However the separating density was also found to increase for a fixed feed density, as the ore reporting to the sinks product is dramatically decreased.

4.3.1 (ii) Intermediate Pilot Plant Run

As has been mentioned earlier, part (i) of the experimental studies were based on batch lots of ore going through the TRI-FLO. In order to run the pilot plant on a continuous basis very careful control of the media density is necessary, requiring the system to replenish ferrosilicon-magnetite or water to maintain a steady, constant media density, through a rapid response control system. The control system was itself ultimately controlled by a gamma gauge. Unfortunately mid-way through the testwork the gamma gauge became faulty and had to be replaced with a more modern, direct reading unit.

The calibration procedure for the new gamma density meter was as follows:

The medium density was adjusted to a high value (approx 3.0 g ml^{-1}) using a Marcy type density balance, noting the count readings of the gamma meter and taking a large sample of medium from the circuit for exact medium density measurement. Water additions, sample collections and count recording continued until five measurements had been collected with the lowest density approximately 2.4 g ml^{-1}. Data from this calibration was entered into the gamma density meter which then displayed the measured density in SG units.

Control of medium density thereafter was simply adjustment of the water addition pneumatic controller set point until the required medium density was indicated. An important point to note is that whenever the medium composition is changed (ferrosilicon-magnetite ratio) the mass of the solids through which the gamma rays must pass changes for a particular volume concentration and therefore a different attenuation of the gamma rays occurs. Recalibration is therefore necessary for each medium composition used.

The calibrated gamma gauge highlighted inadequacies in Marcy type density readings, with Marcy can densities being more variable and slightly in error of the stable calibrated gamma gauge reading. Marcy can readings could only be used to follow trends where slight differences in density were not so critical. Thus the density differentials between the two underflows and the overflow of the Tri-Flo were still valid as trend-setting indicators.

Before running the main pilot plant run therefore it was necessary to run several tests (DMS 13 to 17) to test the calibrated new gamma gauge against the weight proportions of ore reporting to the products. These tests also served to test the "rapid response system" for maintaining media inlet density and also to find the exact media inlet density required for the continuous pilot plant run.

A similar graph to that in Figure 2 was plotted from the results. This graph, Figure 3, additionally included predicted phosphate and magnesia recoveries. From these results conditions were set for the continuous pilot plant run.

4.3.2 Continuous Pilot Plant Runs DMS 19-22

DMS runs 13-17 established that, in order to meet specification the continuous pilot runs should operate at a media feed density of 2.565 g ml^{-1}. Two tests were then run at the media inlet density of 2.56 g ml^{-1} (DMS 19) and at a higher density of 2.61 g ml^{-1} (DMS 20). The higher density aimed to produce a much reduced magnesia product such that mixing the dense media heavy products with the <0.5 mm fines could potentially still meet the overall specification with respect to magnesia whilst increasing the overall phosphate recovery.

A sampling campaign, at 15 minute intervals was also conducted. Sampling points are indicated in Figure 1. These results were balanced using WSL computer software. The results are presented in Table 6.

DMS 19 and 20 were conducted with a magnetite to ferrosilicon content of 33.5%:66.5%.

Two further continuous runs, DMS 21 and 22, were conducted in which the magnetite to ferrosilicon content was increased from 33.5% to 50.8%. From Table 4 it can be seen that this led to relatively low values for the density differentials between the overflow and the two underflows, ie due to apparently increased stability of the media. Because the stability had increased, the established relationship between media inlet density and separating density no longer held good. The optimum separating density was therefore assessed by aiming for 50-60% weight to the sink products and the corresponding media feed density measured and kept stable during a full run. At the suggestion of the TRI-FLO manufacturers, Inpromin, a further test DMS 22 was conducted where the media feed density was further increased to a value -0.10 g ml^{-1} less than the target separating density 2.88 g ml^{-1}. This resulted in a further reduction in the underflow - overflow differential (Table 5).

4.4 Results for DMS 19-22

Phosphate, magnesia, iron (Fe_2O_3) and alumina analyses and recoveries for the four continuous test runs, DMS 19-22 are presented in Tables 7(a), 8, 9(a), and 10. A comparative summary is also presented in Table 11 of the total sinks products. CaO, SiO_2 and F analyses are also presented in Tables 7(b) and 9(b) for the two best runs, namely test DMS 19 and 21.

Allowing for 10% by weight of <0.5 mm material, it can be seen that phosphate recovery (>78%) and specification grades, were easily met in tests DMS 19 and 21. The resulting products from DMS 19 and 21 were heavy liquid analysed on a size basis along with comparative sample DMS 22 (Tables 12, 13 and 14) to ascertain the separation characteristics size by size. Tromp distribution curves were then plotted (Figures 4(a), (b), 5(a), (b), 6(a), (b)).

4.5 Discussion

Figures 4(a), 5(a) and 6(a) illustrate the TRI-FLO characteristic Tromp curves, namely that the overall Tromp curve for the two sections of the TRI-FLO is steeper than the two individual sections making the overall efficiency of separation (E_p) better. The separation density, defined as the d_{50} value, is also seen to shift to a lower value.

As expected, Figures 4(b), 5(b) and 6(b) show that the d_{50} values move to higher densities as grain size gets smaller, and the resulting efficiencies of separation get worse, ie the E_p values get larger, particularly for the 2-0.5 mm size range. DMS 19 and 21 not only gave products within specification, but also gave the best E_p values. Although the differentials between the overflows and underflow media densities were lowest with DMS 22, the resulting increased viscosity has a marked detrimental effect on the separation of the finer sizes. (In practice the ferrosilicon content of the media would have to be increased relative to magnetite to operate at the higher separating density in order to lessen viscosity effects).

The evidence of the testwork based on tests 19 and 21 would suggest that, for separation densities in the 2.86-2.88 g ml^{-1} region on a magnetite/cyclone 40 ferrosilicon mix, a 33-50% magnetite loading is satisfactory. However the separation efficiency was affected to some degree at the higher magnetite loading (see Figs 4B and 5B). The effect of increasing magnetite loading is to reduce the differential between sink and float product densities (Table 5) indicating better medium stability. From test results however the viscosity is obviously becoming too high. Normally separation density lies between the sink and overflow densities for Tri-Flo separation although it is nearer feed density when the differential is low (tests 21 and 22).

Use of a coarser medium (say 80% passing 60 micron rather than 40 micron) should permit a higher magnetite content to be used - in fact more magnetite would be necessary to stabilise the coarser ferrosilicon for a given solids volume concentration. There would however be an optimum magnetite content (higher than that achieved with cyclone 40 ferrosilicon) from the point of view of medium viscosity. At what density this optimum occurs will be dependent upon both the magnetite loading, the volume concentration, and consequently the cyclone differential. It is difficult to predict whether this density will be in the desired range for this ore but in general more practical problems can be associated with the use of coarser medium particularly in relation to steel wear.

5. COMPUTER ANALYSIS OF THE PLANT TESTWORK TO ESTABLISH THE SENSITIVITY OF THE PROCESS PERFORMANCE TO SMALL CHANGES IN SEPARATING DENSITY

Data collected from test DMS 19 was used to calibrate the WSL general TRI-FLO model for the phosphate ore by setting model parameters based on the operating conditions, and the size and density distributions of the feed and products. Once set, the model was used to predict the effects of operating at different separating densities for media of composition 33.5% magnetite, 66.5% ferrosilicon. The resulting variations are plotted in Figures 7a and 7b. Predicted versus measured results are given below in Table 15.

Although the predicted values and measured values are not in absolute agreement being relatively small for phosphate, but significant for magnesia (in view of its low value), the model can be used to predict behavioural trends. It is apparent from Figure 7a that decreasing the separating density below 2.86 g ml^{-1} will result in a relatively small increase in phosphate recovery, whilst phosphate grade (Figure 7b) decreases relatively sharply (Figure 7b). Magnesia recovery and grade are sharply rising at less than 2.87 g ml^{-1} separating density. The separating density of 2.86 g ml^{-1}, which was the actual value used on the plant, would thus appear to give the optimum grade and recovery of both phosphate and magnesia. Figures 7a and 7b highlight the earlier comment that it is crucial to have tight control of the media density in order to achieve the optimum results.

6. RECOMMENDATIONS FOR SCALE-UP TO FULL SIZE PLANT

The TRI-FLO unit to be used in China will be the TFS 250 ie the same as used in the laboratory. According to the manufactures this will treat materials up to a top size of 18 mm at 15 tonnes per hour.

As noted in the text, for successful continuous running of the TRI-FLO it is essential to be able to control the density of the media at a constant value and have a system of classification/densification water addition linked to the density gauge that will respond rapidly enough to maintain the constant density value.

Media losses in the Warren Spring Laboratory TRI-FLO circuit are higher than those expected on a commercial plant, mainly because the circuit is an adaption of an existing simpler dense media circuit. Thus the ability to wash all the media off the products on the product screen is impaired. The WSL screen size is 2.30 m long by 1 m wide. The width is divided into three sections being 0.5 m for the float product and 0.25 m for each of the two sink products. For a commercial plant a much larger screen area would be envisaged although the exact measurements would need to be assessed through plant trials and consultation with appropriate manufacturers.

Ferrosilicon/magnetite losses through inefficient magnetic separation is another problem that would need to be addressed on a commercial plant. At WSL recovery is performed by a one stage drum magnetic separator. On a commercial plant a two stage device is envisaged with the second drum magnet scavenging the "non magnetic" fraction from the first stage or scavenging a thickened first stage "non magnetics" fraction.

According to the manufacturer, media losses should be in the range 80-400 grammes per tonne of ore fed to the plant. They also state, however, that poorly designed plants report figures of up to 1,000 g/t when treating fine ores (M Porter, Inpromin, Personal Communication).

In a commercial plant, water used in the plant should be close-circuited being returned to the plant as clarified water after suspended solids have been taken out by flocculation and thickening.

7. SUGGESTIONS FOR TREATING THE -0.5 MM MATERIAL

Heavy liquid analysis of the fine (<0.5 mm) material from the phosphate ore is presented in Table 16. From this analysis it is apparent that the magnesia grades increase markedly below 0.125 mm whilst the phosphate grades fall. Magnesia in all the size fractions is above the 1.5% level of the specification requirements. Treatment routes for this material must therefore aim to reduce the magnesia level to <1.5% or, to a level slightly above this such that blending with the dense media concentrate would give an overall product <1.5% MgO and with a higher phosphate recovery than could be achieved by dense media alone.

Initial laboratory testwork has shown that desliming the <0.5 mm material at 0.02 mm followed by flotation of the dolomite from the phosphate, using the fatty acid collector FA2 in acid conditions (pH 4.5-5.5, controlled with sulphuric acid), produced a phosphate rich tailings of 23.6% P_2O_5 and 1.7% MgO at 75% phosphate recovery. Increased phosphate recovery, at a lower magnesia grade, could only be achieved by increasing the liberation of the phosphate which would involve milling to finer sizes which is likely to be less cost effective.

An alternative to a flotation only process could be size classification at 0.125 mm using a spiral classifier, or a hydrosizer, followed by sands upgrading with a spiral. The <0.125 mm material in this circuit could be treated by the reverse froth flotation technique described above. Computer simulation studies of the classifier - spiral circuit, using WSL computer models, indicated that the resulting 0.125-0.5 mm product would be of the order 26% P_2O_5 and 2.6% MgO for a 50% phosphate recovery (relative to the total 0.5 mm material). The simulation indicated that further reductions in phosphate recovery only reduced the magnesia content by 0.1-0.2%

8. CONCLUSIONS

The Inpromin TRI-FLO 250 has successfully been used to beneficiate the >0.5 mm fraction of a low grade Chinese phosphate ore to produce a product well within specification.

The behavioural characteristics of the TRI-FLO were examined during the study. It would appear that the optimum medium composition is 33-40% magnetite to 60-67% ferrosilicon at a media TRI-FLC inlet density of 2.56 g ml^{-1}. This would result in a separating density of 2.86 g ml^{-1}.

Computer modelling using the ore fed to, and the products produced from the optimum experimental run (DMS 19) indicate that further reductions in separating density, for increased phosphate recovery are likely to grossly affect the magnesia grade, leading to a product out of specification.

Further phosphate recovery could be achieved by classifying the <0.5 mm ore at approximately 0.125 mm, treating the sands on a spiral and the fines by flotation.

TABLE 2: PHOSPHATE ORE - FEED HEAVY LIQUID ANALYSIS

SIZE	DENSITY	% Weight Stage	% Weight Head	F % Grade	F Distribution Stage	F Distribution Head	Al % Grade	Al Distribution Stage	Al Distribution Head	Fe % Grade	Fe Distribution Stage	Fe Distribution Head	Mg % Grade	Mg Distribution Stage	Mg Distribution Head
-15+10 mm	<2.7 (2.582)	26.54	4.58	1.04	3.14	0.49	1.60	54.60	9.27	2.13	54.76	8.59	0.46	5.43	1.02
	2.7-2.8	12.97	2.24	2.42	3.57	0.56	0.56	9.34	1.59	0.78	10.16	1.54	8.90	51.38	9.60
	2.8-2.9	11.30	1.95	6.11	7.86	1.23	0.45	6.54	1.11	0.53	6.01	0.91	6.80	34.20	6.39
	2.90-2.95	6.83	1.18	13.30	10.35	1.62	0.71	6.24	1.06	0.79	5.42	0.82	0.99	3.01	0.56
	2.95-3.00	12.22	2.11	15.00	20.87	3.26	0.64	10.06	1.71	0.59	7.24	1.10	0.42	2.28	0.43
	3.00-3.05	19.17	3.31	15.70	34.26	5.35	0.39	9.61	1.63	0.45	8.66	1.31	0.30	2.56	0.48
	3.05-3.10	10.31	1.78	16.10	18.90	2.95	0.24	3.18	0.54	0.43	4.45	0.67	0.23	1.06	0.20
	>3.10(3.175)	0.65	0.11	14.20	1.05	0.16	0.51	0.43	0.07	1.99	1.30	0.20	0.28	0.08	0.02
	ALL(CALC)	100.00	17.27	(8.78)	100.00	15.62	(0.78)	100.00	16.97	(1.00)	100.00	15.14	(2.25)	100.00	18.68
-10+5 mm	<2.70(2.639)	20.35	8.04	1.11	2.23	0.92	1.60	48.17	16.26	2.16	48.25	15.29	0.54	5.45	2.09
	2.70-2.80	8.55	3.38	2.75	2.32	0.96	0.55	6.96	2.35	0.84	7.88	2.50	8.40	35.60	13.47
	2.80-2.90	13.99	5.53	5.99	8.27	3.41	0.44	9.11	3.07	0.62	9.52	3.02	6.90	47.86	18.37
	2.90-2.95	6.91	2.73	13.60	9.27	3.82	0.68	6.95	2.35	0.70	5.31	1.68	0.80	3.01	1.16
	2.95-3.00	14.19	5.61	15.10	21.13	8.72	0.59	12.38	4.18	0.58	9.03	2.86	0.45	3.16	1.22
	3.00-3.05	22.32	8.82	15.90	35.00	14.45	0.36	11.89	4.01	0.45	11.03	3.49	0.29	3.21	1.23
	3.05-3.10	12.95	5.12	16.20	20.69	8.54	0.22	4.21	1.42	0.43	6.11	1.94	0.25	1.60	0.62
	>3.10(3.175)	0.75	0.29	15.00	1.10	0.46	0.30	0.33	0.11	3.51	2.87	0.91	0.27	0.10	0.04
	ALL(CALC)	100.00	39.53	(10.14)	100.00	41.27	(0.60)	100.00	33.76	(0.91)	100.00	31.69	(2.02)	100.00	[illegible]
-5+2 mm	<2.70(2.631)	21.20	4.91	1.12	2.24	0.57	1.70	49.17	10.54	2.15	47.73	9.28	0.68	8.29	1.61
	2.70-2.80	6.23	1.44	3.33	1.96	0.49	0.66	5.61	1.20	1.07	6.90	1.36	7.90	28.30	5.48
	2.80-2.90	10.21	2.36	4.37	4.21	1.06	0.45	6.27	1.34	0.61	6.52	1.27	8.50	49.90	9.67
	2.90-2.95	6.20	1.43	13.30	7.78	1.96	0.84	7.10	1.52	0.78	5.06	0.98	1.08	3.85	0.75
	2.95-3.00	13.60	3.15	15.10	19.38	4.89	0.67	12.43	2.66	0.58	8.26	1.61	0.46	3.60	0.70
	3.00-3.05	23.52	5.44	15.70	35.30	8.91	0.40	12.83	2.75	0.45	11.08	2.16	0.26	3.52	0.68
	3.05-3.10	17.83	4.13	16.30	27.43	6.93	0.25	6.08	1.30	0.43	8.03	1.56	0.23	2.36	0.46
	>3.10(3.194)	1.22	0.28	14.70	1.70	0.43	0.31	0.52	0.11	4.95	6.34	1.23	0.27	0.19	0.04
	ALL(CALC)	100.00	23.14	(10.59)	100.00	25.25	(0.73)	100.00	21.44	(0.95)	100.00	19.45	(1.74)	100.00	19.30
-2+0.5 mm	<2.70(2.592)	22.99	2.27	1.03	2.24	0.24	1.94	57.45	5.57	2.24	45.35	4.48	0.86	12.52	0.94
	2.70-2.80	3.06	0.30	3.32	0.96	0.10	1.05	4.14	0.40	1.85	4.99	0.49	5.90	11.43	0.86
	2.80-2.90	11.02	1.09	3.49	3.65	0.39	0.46	6.53	0.63	0.73	7.09	0.70	9.00	62.82	4.72
	2.90-2.95	3.86	0.38	12.40	4.54	0.49	0.02	4.08	0.40	1.01	3.44	0.34	1.28	3.13	0.24
	2.95-3.00	9.00	0.89	14.80	12.63	1.36	0.62	7.19	0.70	0.69	5.47	0.54	0.49	2.79	0.21
	3.00-3.05	19.50	1.93	15.80	29.20	3.13	0.43	10.89	1.05	0.51	8.76	0.86	0.27	3.33	0.25
	3.05-3.10	27.74	2.74	16.30	42.86	4.60	0.24	8.57	0.83	0.49	11.97	1.18	0.20	3.51	0.26
	>3.10(3.195)	2.92	0.29	14.60	3.91	0.42	0.34	1.24	0.12	5.20	12.94	1.28	0.26	0.46	0.03
	ALL(CALC)	100.00	9.88	(10.55)	100.00	10.73	(0.78)	100.00	9.69	(1.14)	100.00	9.07	(1.58)	100.00	7.51
-0.5 mm	ALL	100.00	10.18	6.79	100.00	7.12	1.41	100.00	18.15	2.66	100.00	23.84	3.27	100.00	16.04
HEAD(CALC)				(9.71)			(0.79)			(1.14)			(2.08)		
(MEAS)				9.71			1.03			1.13			2.04		

ABL

PREDICTED DISTRIBUTIONS FOR ORE 2260 CHINESE PHOSPHATE

SEP'N	-15+10mm			-10+5mm			-5+2mm			-2+0.5mm			Overall % TO SINKS			
S.G.	% to sinks			% to sinks			% to Sinks			% to sinks			ON	ON		
	Stage	Head	DMS	Stage	Head	DMS	Stage	Head	DMS	Stage	Head	DMS	HEAD	DMS FEED		
							WEIGHT DISTRIBUTIONS									
		(17.27)	(19.23)		(39.53)	(44.01)		(23.14)	(25.76)		(9.88)	(11.00)				
2.80	60.61	10.47	11.66	68.85	27.22	30.30	64.26	14.87	16.55	44.09	4.36	4.85	56.92	63.36		
2.84	55.72	9.63	10.71	63.75	25.20	28.06	59.23	13.71	15.26	38.82	3.84	4.27	52.38	58.30		
2.88	50.54	8.73	9.72	57.73	22.82	25.11	53.30	12.33	13.73	33.38	3.30	3.67	47.18	52.53		
2.92	45.24	7.81	8.70	51.29	20.27	22.57	46.05	10.66	11.86	27.57	2.72	3.03	41.46	46.16		
2.96	38.91	6.72	7.48	44.08	17.42	19.40	37.31	8.63	9.61	21.67	2.14	2.38	34.91	38.87		
3.00	30.26	5.23	5.82	33.84	13.38	14.89	27.42	6.34	7.06	16.27	1.61	1.79	26.56	29.56		
3.04	19.91	3.44	3.83	22.26	8.80	9.80	17.95	4.15	4.62	11.45	1.13	1.26	17.52	19.51		
							PHOSPHATE DISTRIBUTIONS								% GRADE	
		(15.62)	(16.82)		(41.27)	(44.14)		(23.25)	(27.19)		(10.73)	(11.55)			P	P2O5
2.80	91.29	14.26	15.35	92.35	38.11	41.04	86.70	21.89	23.57	61.45	6.59	7.10	80.85	87.06	13.79	31.58
2.84	88.04	13.75	14.81	89.06	36.76	39.58	82.29	20.78	22.37	55.10	5.91	6.36	77.20	83.12	14.31	32.77
2.88	83.65	13.07	14.07	84.29	34.79	37.45	76.07	19.21	20.68	48.01	5.15	5.55	72.21	77.75	14.86	34.03
2.92	77.37	12.09	13.01	77.61	32.03	34.19	67.09	16.94	18.24	40.07	4.30	4.63	65.36	70.37	15.31	35.06
2.96	67.91	10.61	11.12	67.69	27.94	30.08	55.07	13.91	14.97	31.69	3.40	3.66	55.86	60.13	15.54	35.59
3.00	53.53	8.36	9.00	52.61	21.71	23.38	40.79	10.30	11.09	23.83	2.56	2.75	42.93	46.22	15.69	35.93
3.04	35.51	5.55	5.97	34.80	14.36	15.47	26.77	6.76	7.28	16.73	1.80	1.93	28.47	30.65	15.78	36.14
							MAGNESIUM DISTRIBUTIONS								% GRADE	
		(18.68)	(22.25)		(38.39)	(45.75)		(19.38)	(23.08)		(7.51)	(8.95)			Mg	MgO
2.80	48.47	9.05	10.78	55.98	21.19	25.59	46.05	8.72	10.63	26.27	1.97	2.35	41.43	49.35	1.48	2.46
2.84	33.85	6.32	7.53	40.47	15.54	18.50	33.16	6.43	7.65	19.15	1.44	1.71	29.73	35.36	1.16	1.93
2.88	20.76	3.88	4.62	24.69	9.48	11.29	21.90	4.24	5.05	13.66	1.03	1.22	18.63	22.18	0.81	1.35
2.92	12.43	2.32	2.77	13.31	5.11	6.09	13.81	2.68	3.19	9.36	0.70	0.84	10.81	12.89	0.53	0.88
2.96	7.82	1.46	1.74	9.10	3.49	4.16	8.80	1.71	2.03	6.30	0.47	0.56	7.13	8.49	0.42	0.70
3.00	4.93	0.92	1.10	5.82	2.23	2.66	5.52	1.07	1.27	4.29	0.32	0.38	4.54	5.41	0.35	0.58
3.04	2.82	0.53	0.63	3.53	1.36	1.61	3.38	0.66	0.78	2.85	0.21	0.26	2.76	3.28	0.32	0.53
							ALUMINIUM DISTRIBUTIONS								% GRADE	
		(16.97)	(20.73)		(33.76)	(41.24)		(21.11)	(26.19)		(9.67)	(11.84)			AL	AL2O3
2.80	36.51	6.20	7.57	43.88	14.81	18.19	46.10	9.60	10.50	22.30	2.16	2.64	31.77	38.81	0.58	1.10
2.84	33.29	5.65	6.90	40.10	13.51	16.54	36.29	7.78	9.50	19.11	1.85	2.26	28.82	35.20	0.57	1.08
2.88	29.94	5.08	6.21	35.77	12.08	14.75	31.78	7.31	8.32	16.07	1.56	1.90	25.53	31.18	0.56	1.06
2.92	26.16	4.44	5.42	30.83	10.41	12.71	25.38	5.66	6.71	13.01	1.26	1.54	21.77	26.58	0.54	1.02
2.96	21.44	3.64	4.44	25.69	8.47	10.25	20.18	4.33	5.29	10.02	0.97	1.19	17.41	21.27	0.51	0.96
3.00	15.33	2.60	3.13	17.60	5.71	7.26	13.95	2.99	3.65	7.11	0.72	0.88	12.25	14.97	0.48	0.91
3.04	9.30	1.58	1.93	10.59	3.59	4.37	8.61	1.85	2.25	5.23	0.51	0.62	7.52	9.17	0.44	0.83
							IRON DISTRIBUTIONS								%GRADE	
															Fe	Fe_2O_3
2.80	33.88	5.13	6.74	43.11	13.66	17.94	41.01	7.98	10.47	31.67	3.13	4.10	29.90	39.25	0.59	0.84
2.84	30.75	4.66	6.11	39.25	12.44	16.31	37.11	7.28	9.55	28.33	2.80	3.67	27.18	35.67	0.59	0.84
2.88	27.65	4.19	5.50	35.04	11.19	14.55	33.51	6.52	8.56	24.99	2.47	3.24	24.28	31.88	0.58	0.83
2.92	24.34	3.69	4.84	30.64	9.71	12.75	29.06	5.65	7.42	21.48	2.12	2.78	21.17	27.79	0.58	0.83
2.96	20.40	3.09	4.06	25.98	8.23	10.81	23.99	4.67	6.13	17.90	1.77	2.32	17.76	23.32	0.57	0.82
3.00	15.46	2.34	3.07	19.35	6.27	8.26	19.61	3.62	4.75	14.55	1.44	1.89	13.69	17.97	0.58	0.83
3.04	10.23	1.55	2.03	13.58	4.30	5.65	13.59	2.64	3.47	11.37	1.12	1.47	9.61	12.62	0.62	0.89

TABLE 5: TRI-FLO OPERATING CONDITIONS AND WEIGHTS TO SINKS PRODUCTS

Test	Sinks 1 Wt %	Sinks 2 Wt %	Total Sinks Wt %	% Magnetite in Medium	Media Feed SG	Sinks 1 SG	Sinks 2 SG	O/F SG	Differentials S1	Differentials S2	Pressures S1	Pressures S2	Back Pressure Adjusters S1	Back Pressure Adjusters S2
DMS 1	-	-	4.87	22.5	* 2.95	3.14	3.12	2.72	0.42	0.40	15	15	-	
DMS 2	-	-	7.91	22.5	* 2.90	3.15	3.20	2.66	0.49	0.54	15	15	-	-
DMS 3	-	-	25.83	22.5	* 2.81	3.17	3.21	2.5	0.67	0.71	15	15	-	-
DMS 4	-	-	17.08	24.5	* 2.84	3.18	3.19	2.51	0.67	0.68	15	15	-	-
DMS 5	36.6	10.7	47.3	26.0	* 2.69	3.00	3.13	2.44	0.56	0.69	15	15	-	-
DMS 6	31.5	10.0	41.5	26.0	* 2.75	3.03	3.10	2.53	0.50	0.57	15	15	-	-
DMS 8	28.5	6.3	34.8	26.0	* 2.78	3.16	3.24	2.37	0.79	0.87	20	20	-	-
DMS 9	32.3	11.4	45.7	26.0	* 2.71	3.03	3.11	2.42	0.61	0.69	15	15	-	-
DMS 10	50.5	7.7	58.2	31.5	* 2.65	3.03	3.09	2.35	0.68	0.74	15	15	-	-
DMS 11	18.8	8.4	27.2	31.5	* 2.65	3.05	3.15	2.48	0.57	0.67	15	15	?	?
DMS 12	38.1	9.5	47.6	31.5	* 2.67	3.10	3.11	2.41	0.69	0.70	15	15	-	-
DMS 13	41.9	10.1	52.0	33.5	* 2.65	-	-	-	-	-	15	15	-	-
DMS 14	44.2	6.7	50.9	33.5	2.65	-	-	-	-	-	15	15	-	-
DMS 15	45.9	8.7	54.6	33.5	2.56	-	-	-	-	-	15	15	-	-
DMS 16	55.7	7.6	63.3	33.5	2.45	-	-	-	-	-	15	15	-	-
DMS 17	51.8	8.2	60.0	33.5	2.50	-	-	-	-	-	15	15	-	-
DMS 19	51.0	9.5	60.5	33.5	2.56	-	-	-	-	-	15	15	-	-
DMS 20	45.9	9.8	55.7	33.5	2.61						15	15	-	
DMS 21	48.3	10.5	58.8	50.8	2.76	2.93	2.95	2.58	0.35	0.37	15	15	-	-
DMS 22	37.3	7.5	44.8	50.8	2.85	2.94	-	2.74	0.20	-	15	15	-	-

* Media Feed Densities measured initially by Marcy can until DMS 13 after which a new calibrated Gamma Density Meter was used.

TABLE 6: SOLIDS BALANCE IN TRI-FLO CIRCUIT FOR TESTS DMS 19 AND 20

	DMS 19		DMS 20	
	Measured t/hr	Balanced t/hr	Measured t/hr	Balanced t/hr
Dry Feed Rate	3.04	3.14	3.04	2.38
-0.5mm Fines from Feed	0.06	0.06	0.09	0.09
Product- Sinks 1	1.58	1.55	1.24	1.27
Product- Sinks 2	0.29	0.29	0.27	0.27
Product- Floats	1.22	1.21	1.20	1.22
-0.5mm Fines from product screen	0.03	0.03	0.04	0.04

TABLE 7(a): RESULTS OF TEST DMS 19

TEST	PRODUCT	% Weight		P2O5			MgO			FE2O3			AL2O3		
				% Grade	Distribution		% Grade	Distribution		% Grade	Distribution		% Grade	Distribution	
		Stage	Head		Stage	Head		Stage	Head		Stage	Head		Stage	Head
DMS 19															
	FLOAT	39.50	39.50	6.11	10.52	10.52	6.15	76.96	76.96	2.50	64.39	64.39	2.46	65.66	65.66
	SINK 1	51.00	51.00	34.58	76.87	76.87	0.91	14.70	14.70	0.89	29.60	29.60	0.81	27.92	27.92
	SINK 2	9.50	9.50	30.46	12.61	12.61	2.77	8.34	8.34	0.97	6.01	6.01	1.00	6.42	6.42
	ALL(CALC)	100.00	100.00	(22.94)	100.00	100.00	(3.16)	100.00	100.00	(1.53)	100.00	100.00	(1.48)	100.00	100.00
HEAD(CALC)				(22.94)			(3.16)			(1.53)			(1.48)		

TRI-FLO DENSE MEDIUM RUN

TEST	PRODUCT	% Weight		P2O5			MgO			FE2O3			AL2O3		
				% Grade	Distribution		% Grade	Distribution		% Grade	Distribution		% Grade	Distribution	
		Stage	Head		Stage	Head		Stage	Head		Stage	Head		Stage	Head
DMS 19															
	FLOAT	39.50	39.50	6.11	10.52	10.52	6.15	76.96	76.96	2.50	64.39	64.39	2.46	65.66	65.66
	SINK(TOTAL)	60.50	60.50	33.93	89.48	89.48	1.20	23.04	23.04	0.90	35.61	35.61	0.84	34.34	34.34
	ALL(CALC)	100.00	100.00	(22.94)	100.00	100.00	(3.16)	100.00	100.00	(1.53)	100.00	100.00	(1.48)	100.00	100.00
HEAD(CALC)				(22.94)			(3.16)			(1.53)			(1.48)		

TABLE 7(b): CONTINUATION OF RESULTS FROM TEST DMS 19

TEST	PRODUCT	% Weight		CaO			SiO2			F		
				% Grade	Distribution		% Grade	Distribution		% Grade	Distribution	
		Stage	Head		Stage	Head		Stage	Head		Stage	Head
DMS 19												
	FLOAT	39.50	39.50	15.95	17.59	17.59	30.20	82.89	82.89	0.56	10.45	10.45
	SINK 1	51.00	51.00	49.53	70.51	70.51	3.82	13.54	13.54	3.19	76.89	76.89
	SINK 2	9.50	9.50	44.91	11.91	11.91	5.42	3.58	3.58	2.82	12.66	12.66
	ALL(CALC)	100.00	100.00	(35.83)	100.00	100.00	(14.39)	100.00	100.00	(2.12)	100.00	100.00
HEAD(CALC)				(35.83)			(14.39)			(2.12)		

TRI-FLO DENSE MEDIA RUN

TEST	PRODUCT	% Weight		CaO			SiO2			F		
				% Grade	Distribution		% Grade	Distribution		% Grade	Distribution	
		Stage	Head		Stage	Head		Stage	Head		Stage	Head
DMS 19												
	FLOAT	39.50	39.50	15.95	17.59	17.59	30.20	82.89	82.89	0.56	10.45	10.45
	SINK (TOTAL)	60.50	60.50	48.80	82.41	82.41	4.07	17.11	17.11	3.13	89.55	89.55
	ALL(CALC)	100.00	100.00	(35.83)	100.00	100.00	(14.39)	100.00	100.00	(2.12)	100.00	100.00
HEAD(CALC)				(35.83)			(14.39)			(2.12)		

TABLE 8: RESULTS OF TEST DMS 20

TEST	PRODUCT	% Weight		P205			MnO			FE203			AL203		
				% Grade	Distribution		% Grade	Distribution		% Grade	Distribution		% Grade	Distribution	
		Stage	Head		Stage	Head		Stage	Head		Stage	Head		Stage	Head
DMS 20															
	FLOAT	44.27	44.27	7.72	15.04	15.04	6.38	85.80	85.80	2.24	66.19	66.19	2.25	68.12	68.12
	SINK 1	45.90	45.90	35.04	70.79	70.79	0.71	9.90	9.90	0.93	28.49	28.49	0.81	25.43	25.43
	SINK 2	9.83	9.83	32.75	14.16	14.16	1.44	4.30	4.30	0.81	5.31	5.31	0.96	6.45	6.45
	ALL(CALC)	100.00	100.00	(22.72)	100.00	100.00	(3.29)	100.00	100.00	(1.50)	100.00	100.00	(1.46)	100.00	100.00
HEAD(CALC)				(22.72)			(3.29)			(1.50)			(1.46)		

TRI-FLO DENSE MEDIUM RUN

TEST	PRODUCT	% Weight		P205			MnO			FE203			AL203		
				% Grade	Distribution		% Grade	Distribution		% Grade	Distribution		% Grade	Distribution	
		Stage	Head		Stage	Head		Stage	Head		Stage	Head		Stage	Head
DMS 20															
	FLOAT	44.27	44.27	7.72	15.04	15.04	6.38	85.80	85.80	2.24	66.19	66.19	2.25	68.12	68.12
	SINK(TOTAL)	55.73	55.73	34.64	84.96	84.96	0.84	14.20	14.20	0.91	33.81	33.81	0.84	31.88	31.88
	ALL(CALC)	100.00	100.00	(22.72)	100.00	100.00	(3.29)	100.00	100.00	(1.50)	100.00	100.00	(1.46)	100.00	100.00
HEAD(CALC)				(22.72)			(3.29)			(1.50)			(1.46)		

TABLE 9(a): RESULTS OF TEST DMS 21

TEST	PRODUCT	% Weight		P2O5			MgO			FE2O3			AL2O3		
				% Grade	Distribution		% Grade	Distribution		% Grade	Distribution		% Grade	Distribution	
		Stage	Head		Stage	Head		Stage	Head		Stage	Head		Stage	Head
DMS 21															
	FLOAT	41.19	41.19	7.10	12.60	12.60	5.89	81.66	81.66	2.37	63.03	63.03	2.32	66.80	66.80
	SINK 1	48.29	48.29	35.04	72.88	72.88	0.71	11.54	11.54	0.99	30.86	30.86	0.77	25.99	25.99
	SINK 2	10.52	10.52	32.06	14.52	14.52	1.92	6.80	6.80	0.90	6.11	6.11	0.98	7.20	7.20
	ALL(CALC)	100.00	100.00	(23.22)	100.00	100.00	(2.97)	100.00	100.00	(1.55)	100.00	100.00	(1.43)	100.00	100.00
HEAD(CALC)				(23.22)			(2.97)			(1.55)			(1.43)		

TRI-FLO DENSE MEDIUM RUN

TEST	PRODUCT	% Weight		P2O5			MgO			FE2O3			AL2O3		
				% Grade	Distribution		% Grade	Distribution		% Grade	Distribution		% Grade	Distribution	
		Stage	Head		Stage	Head		Stage	Head		Stage	Head		Stage	Head
DMS 21															
	FLOAT	41.19	41.19	7.10	12.60	12.60	5.89	81.66	81.66	2.37	63.03	63.03	2.32	66.80	66.80
	SINK(TOTAL)	58.81	58.81	34.51	87.40	87.40	0.93	18.34	18.34	0.97	36.97	36.97	0.81	33.20	33.20
	ALL(CALC)	100.00	100.00	(23.22)	100.00	100.00	(2.97)	100.00	100.00	(1.55)	100.00	100.00	(1.43)	100.00	100.00
HEAD(CALC)				(23.22)			(2.97)			(1.55)			(1.43)		

TABLE 9(b): CONTINUATION OF RESULTS OF TEST DMS 21

TEST	PRODUCT	% Weight		CaO			SiO2			F		
				% Grade	Distribution		% Grade	Distribution		% Grade	Distribution	
		Stage	Head		Stage	Head		Stage	Head		Stage	Head
DMS 21												
	FLOAT	41.19	41.19	17.77	20.82	20.82	28.90	84.24	84.24	0.62	11.97	11.97
	SINK 1	48.29	48.29	47.57	65.36	65.36	3.41	11.76	11.76	3.25	73.58	73.58
	SINK 2	10.52	10.52	46.17	13.82	13.82	5.38	4.01	4.01	2.93	14.45	14.45
	ALL(CALC)	100.00	100.00	(35.15)	100.00	100.00	(14.13)	100.00	100.00	(2.13)	100.00	100.00
HEAD(CALC)				(35.15)			(14.13)			(2.13)		

tri-flo dense media run

TEST	PRODUCT	% Weight		CaO			SiO2			F		
				% Grade	Distribution		% Grade	Distribution		% Grade	Distribution	
		Stage	Head		Stage	Head		Stage	Head		Stage	Head
DMS 21												
	FLOAT	41.19	41.19	17.77	20.82	20.82	28.90	84.24	84.24	0.62	11.97	11.97
	SINK (TOTAL)	58.81	58.81	47.32	79.18	79.18	3.79	15.76	15.76	3.19	88.03	88.03
	ALL(CALC)	100.00	100.00	(35.15)	100.00	100.00	(14.13)	100.00	100.00	(2.13)	100.00	100.00
HEAD(CALC)				(35.15)			(14.13)			(2.13)		

TABLE 10: RESULTS OF TEST DMS 22

TEST	PRODUCT	% Weight Stage	% Weight Head	P2O5 % Grade	P2O5 Distribution Stage	P2O5 Distribution Head	MgO % Grade	MgO Distribution Stage	MgO Distribution Head	FE2O3 % Grade	FE2O3 Distribution Stage	FE2O3 Distribution Head	AL2O3 % Grade	AL2O3 Distribution Stage	AL2O3 Distribution Head
DMS 22															
	FLOAT	55.18	55.18	12.60	30.57	30.57	5.27	91.34	91.34	2.40	73.75	73.75	2.12	77.72	77.72
	SINK 1	37.35	37.35	35.50	58.30	58.30	0.53	6.22	6.22	1.10	22.88	22.88	0.72	17.87	17.87
	SINK 2	7.47	7.47	33.89	11.13	11.13	1.04	2.44	2.44	0.81	3.37	3.37	0.89	4.42	4.42
	ALL(CALC)	100.00	100.00	(22.74)	100.00	100.00	(3.18)	100.00	100.00	(1.80)	100.00	100.00	(1.51)	100.00	100.00
HEAD(CALC)				(22.74)			(3.18)			(1.80)			(1.51)		

TRI-FLO DENSE MEDIUM RUN

TEST	PRODUCT	% Weight Stage	% Weight Head	P2O5 % Grade	P2O5 Distribution Stage	P2O5 Distribution Head	MgO % Grade	MgO Distribution Stage	MgO Distribution Head	FE2O3 % Grade	FE2O3 Distribution Stage	FE2O3 Distribution Head	AL2O3 % Grade	AL2O3 Distribution Stage	AL2O3 Distribution Head
DMS 22															
	FLOAT	55.18	55.18	12.60	30.57	30.57	5.27	91.34	91.34	2.40	73.75	73.75	2.12	77.72	77.72
	SINK(TOTAL)	44.82	44.82	35.[illegible]	69.43	69.43	0.62	8.66	8.66	1.05	26.25	26.25	0.75	22.28	22.28
	ALL(CALC)	100.00	100.00	(22.74)	100.00	100.00	(3.18)	100.00	100.00	(1.80)	100.00	100.00	(1.51)	100.00	100.00
HEAD(CALC)				(22.74)			(3.18)			(1.80)			(1.51)		

TABLE 11: COMPARATIVE SUMMARY OF PRODUCTS (SINKS 1 AND SINKS 2) OF TESTS DMS 19 - 22 FOR >0.5 mm ORE

TEST	Wt %	% P2O5	RECOVERY P2O5 %	% MgO	% Fe2O3	% Al2O3
DMS 19	60.50	33.93	89.48	1.20	0.90	0.84
DMS 20	55.73	34.64	84.96	0.84	0.91	0.84
DMS 21	58.81	34.51	87.40	0.93	0.97	0.81
DMS 22	[illegible]	35.23	69.43	0.62	1.05	[illegible]

TABLE 12: HEAVY LIQUID ANALYSIS ON TEST DMS 19 PRODUCTS

PRODUCT	DENSITY	% Weight Stage	% Weight Head (DMS feed)	% to Sink 1 / % to Sink 2	% to Total Sink
FLOAT					
	<2.7	53.77	21.24		
	2.7-2.8	17.39	6.87		
	2.8-2.85	18.91	7.47		
	2.85-2.88	2.58	1.02		
	2.88-2.90	1.34	0.53		
	2.90-2.95	1.82	0.72		
	2.95-3.00	1.70	0.67		
	3.00-3.05	1.57	0.62		
	3.05-3.10	0.86	0.34		
	>3.10	0.05	0.02		
	ALL(CALC)	100.00	39.50		
SINK 1				% to Sink 1	% to Total Sink
	<2.7	0.02	0.01	0.05	0.33
	2.7-2.8	0.29	0.15	2.07	5.11
	2.8-2.85	2.00	1.02	10.77	21.12
	2.85-2.88	1.27	0.65	29.68	53.42
	2.88-2.90	1.78	0.91	32.00	69.71
	2.90-2.95	7.37	3.76	68.99	86.79
	2.95-3.00	17.60	8.89	77.02	94.25
	3.00-3.05	40.78	20.80	85.60	97.45
	3.05-3.10	26.24	13.38	87.97	97.7[illegible]
	>3.10	2.63	1.34	95.04	98.53
	ALL(CALC)	100.00	51.00		
SINK 2				% to Sink 2	
	<2.7	0.74	0.07	0.33	
	2.7-2.8	2.32	0.22	3.10	
	2.8-2.85	10.32	0.98	11.60	
	2.85-2.88	5.47	0.52	33.77	
	2.88-2.90	3.26	0.31	36.90	
	2.90-2.95	10.21	0.97	57.40	
	2.95-3.00	21.16	2.01	75.00	
	3.00-3.05	30.32	2.88	82.29	
	3.05-3.10	15.68	1.49	81.42	
	>3.10	0.53	0.05	71.43	
	ALL(CALC)	100.00	9.50		
HEAD(CALC)			100.00		

TABLE 13: HEAVY LIQUID ANALYSIS ON TEST DMS 21 PRODUCTS

PRODUCT	DENSITY	% Weight Stage	% Weight Head (DMS feed)		
FLOAT					
	<2.7	49.78	20.51		
	2.7-2.8	15.36	6.33		
	2.8-2.85	20.46	8.43		
	2.85-2.88	2.91	1.20		
	2.88-2.90	1.72	0.71		
	2.90-2.95	2.89	1.19		
	2.95-3.00	2.74	1.13		
	3.00-3.05	2.50	1.03		
	3.05-3.10	1.53	0.63		
	>3.10	0.10	0.04		
	ALL(CALC)	100.00	41.20		
SINK 1				% to sink 1	% to total sink
	<2.7	0.02	0.01	0.05	0.19
	2.7-2.8	0.06	0.03	0.46	2.91
	2.8-2.85	0.91	0.44	4.54	13.01
	2.85-2.88	0.95	0.46	22.12	42.31
	2.88-2.90	1.10	0.53	32.12	56.97
	2.90-2.95	7.02	3.39	57.65	79.76
	2.95-3.00	17.27	8.34	70.80	90.41
	3.00-3.05	41.39	19.99	82.30	95.73
	3.05-3.10	30.58	14.77	85.28	[illegible]
	>3.10	0.70	0.34	80.95	[illegible]
	ALL(CALC)	100.00	48.30		
SINK 2				% to sink 2	
	<2.7	0.29	0.03	0.15	
	2.7-2.8	1.52	0.16	2.47	
	2.8-2.85	7.90	0.83	8.96	
	2.85-2.88	4.00	0.42	25.93	
	2.88-2.90	3.90	0.41	36.67	
	2.90-2.95	12.38	1.30	52.10	
	2.95-3.00	22.00	2.31	67.15	
	3.00-3.05	29.33	3.08	74.94	
	3.05-3.10	18.29	1.92	75.29	
	>3.10	0.38	0.04	50.00	
	ALL(CALC)	100.00	10.50		
HEAD(CALC)			100.00		

TABLE 14: HEAVY LIQUID ANALYSIS ON TEST DMS 22 PRODUCTS

PRODUCT	DENSITY	% Weight Stage	% Weight Head (DMS feed)	% to Sink 1	% to Total Sink
FLOAT					
	<2.70	39.36	21.66		
	2.70-2.80	14.22	7.85		
	2.80-2.85	15.16	8.37		
	2.85-2.90	5.97	3.30		
	2.90-3.00	14.72	8.13		
	3.00-3.10	10.33	5.70		
	>3.10	0.34	0.19		
	ALL(CALC)	100.00	55.20		
SINK 1				% to Sink 1	% to Total Sink
	<2.70	0.02	<0.01	0.03	0.04
	2.70-2.80	0.01	<0.01	0.04	0.42
	2.80-2.85	0.15	0.06	0.71	2.22
	2.85-2.90	0.80	0.30	7.79	14.29
	2.90-3.00	14.98	5.59	33.47	51.32
	3.00-3.10	81.32	30.33	75.77	85.76
	>3.10	2.72	1.01	77.10	85.50
	ALL(CALC)	100.00	37.30		
SINK 2				% to Sink 2	
	<2.70	0.10	0.01	<0.01	
	2.70-2.80	0.39	0.03	0.38	
	2.80-2.85	1.71	0.13	1.33	
	2.85-2.90	3.29	0.25	7.04	
	2.90-3.00	39.84	2.98	26.82	
	3.00-3.10	53.29	4.00	41.24	
	>3.10	1.49	0.11	36.67	
	ALL(CALC)	100.00	7.50		
HEAD(CALC)			100.00		

TABLE 15. - Measured v Computer Predicted Results for Media of Composition 33.5% magnetite 66.5% ferrosilicon

			Wt Yield %	Phosphate (P_2O_5)		Magnesia (MgO)	
				% Recovery	% Grade	% Recovery	% Grade
DMS 19							
Separating D_{50}	= 2.86	Predicted	57.50	84.99	34.10	17.5	0.91
Feed Density	= 2.56	Measured	60.50	89.48	33.93	23.04	1.20

TABLE 16: HEAVY LIQUID ANALYSIS ON PHOSPHATE FEED <0.5mm

size	density	% Weight		P205			MgO		
				% Grade	Distribution		% Grade	Distribution	
		Stage	Head		Stage	Head		Stage	Head
-500+250									
	<2.7	18.27	4.53	1.21	0.86	0.34	0.43	3.92	0.35
	2.7-3.0	20.45	5.07	14.79	11.76	4.69	8.31	84.02	7.44
	3.0-3.3	60.71	15.05	36.98	87.26	34.84	0.36	10.91	0.98
	>3.3	0.56	0.14	5.40	0.12	0.05	1.23	0.35	0.03
	ALL(CALC)	100.00	24.79	(25.73)	100.00	39.93	(2.00)	100.00	9.01
-250+125									
	<2.7	20.43	3.29	1.17	1.02	0.24	0.45	3.20	0.27
	2.7-3.0	24.47	3.94	11.75	12.22	2.90	10.61	90.22	7.58
	3.0-3.3	54.22	8.73	37.56	86.56	20.53	0.33	6.22	0.52
	>3.3	0.87	0.14	5.40	0.20	0.05	1.23	0.37	0.03
	ALL(CALC)	100.00	16.10	(23.53)	100.00	23.71	(2.88)	100.00	8.40
-125+75									
	<2.7	21.21	2.03	1.19	1.48	0.15	0.46	1.54	0.17
	2.7-3.0	38.66	3.70	6.34	14.38	1.47	15.75	95.80	10.57
	3.0-3.3	37.83	3.62	37.33	82.86	8.46	0.40	2.38	0.26
	>3.3	2.40	0.23	9.05	1.28	0.13	0.75	0.28	0.03
	ALL(CALC)	100.10	9.58	(17.04)	100.00	10.21	(6.36)	100.00	11.03
-75+53									
	<2.7	22.75	1.59	1.26	2.39	0.13	1.08	2.62	0.31
	2.7-3.0	50.50	3.53	4.85	20.46	1.07	17.58	94.55	[illegible]
	3.0-3.3	25.18	1.76	36.18	76.08	3.99	0.99	2.45	0.32
	>3.3	3.00	0.21	4.26	1.07	0.06	0.56	0.18	0.02
	ALL(CALC)	101.43	7.09	(11.97)	100.00	5.24	(9.39)	100.00	[illegible]
-53									
	ALL	100.00	42.55	7.85	100.00	20.91	7.73	100.00	57.65
HEAD(CALC)				(15.97)			(5.51)		

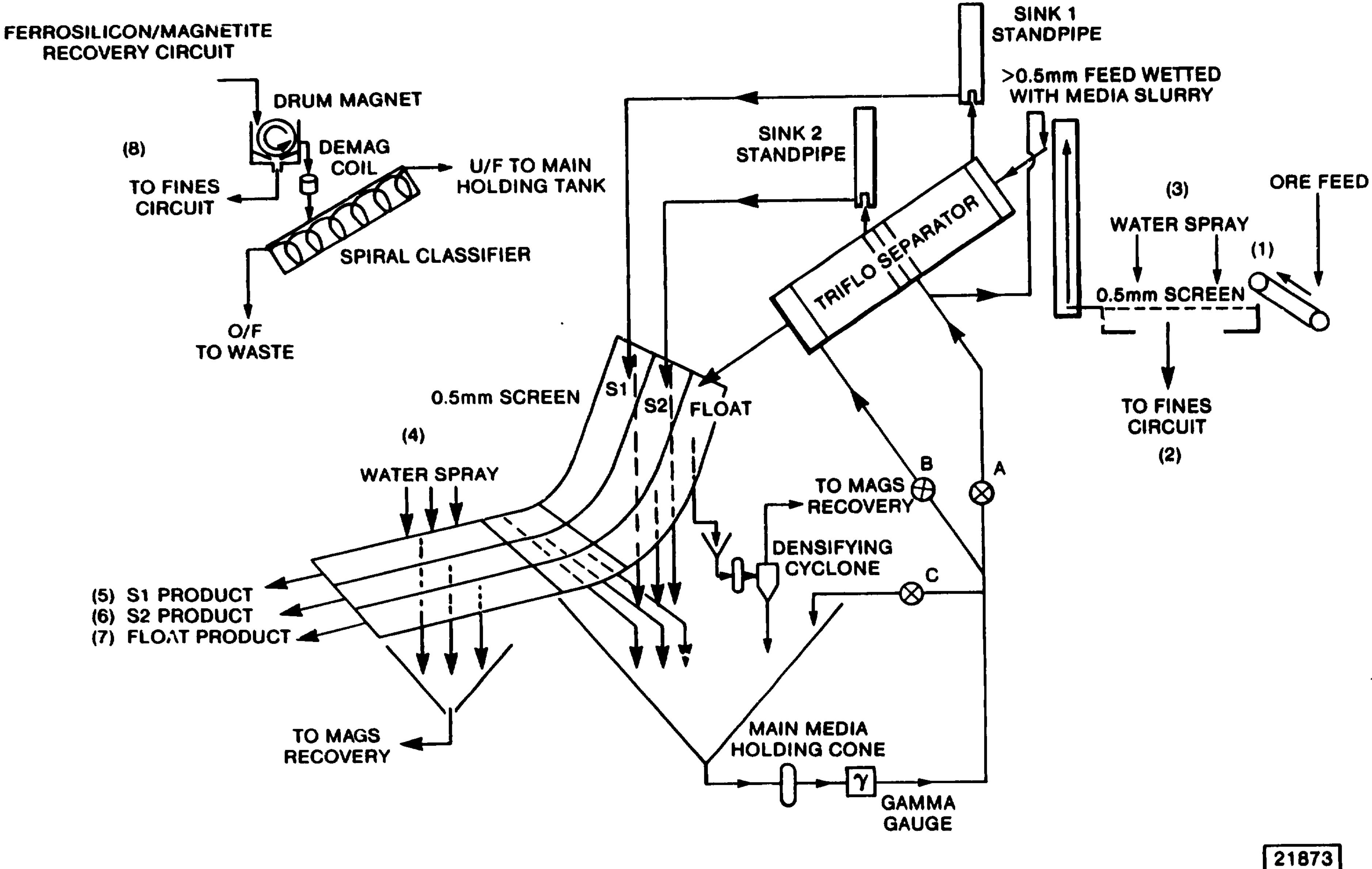


FIG. 1 TRI-FLO SEPARATOR CIRCUIT

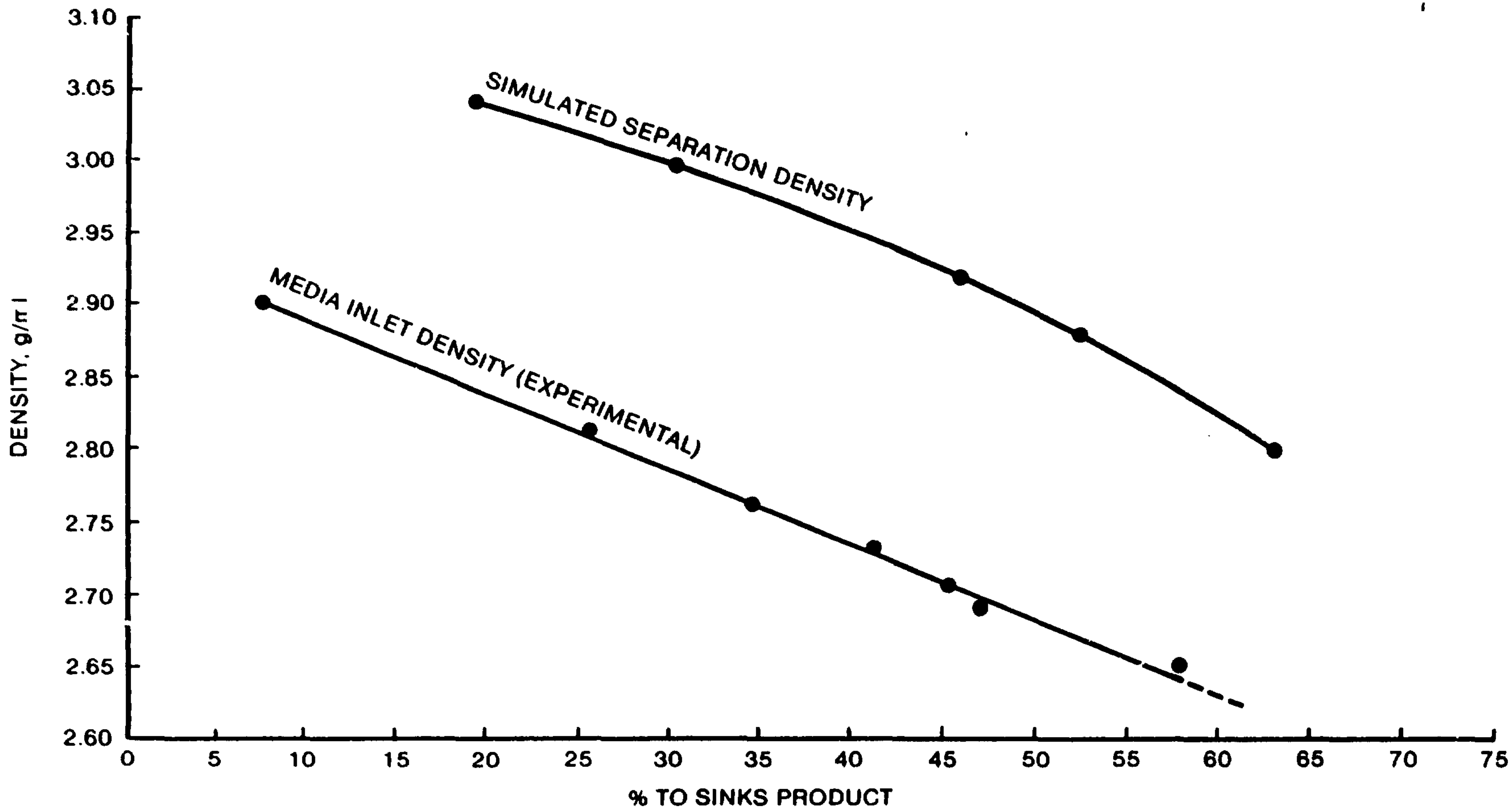


FIG. 2 RELATIONSHIP BETWEEN THE MEASURED MEDIA INLET DENSITIES, THE SIMULATED SEPARATION DENSITIES AND THE PROPORTION OF ORE REPORTING TO THE SINKS PRODUCTS. DMS 1-12

21499

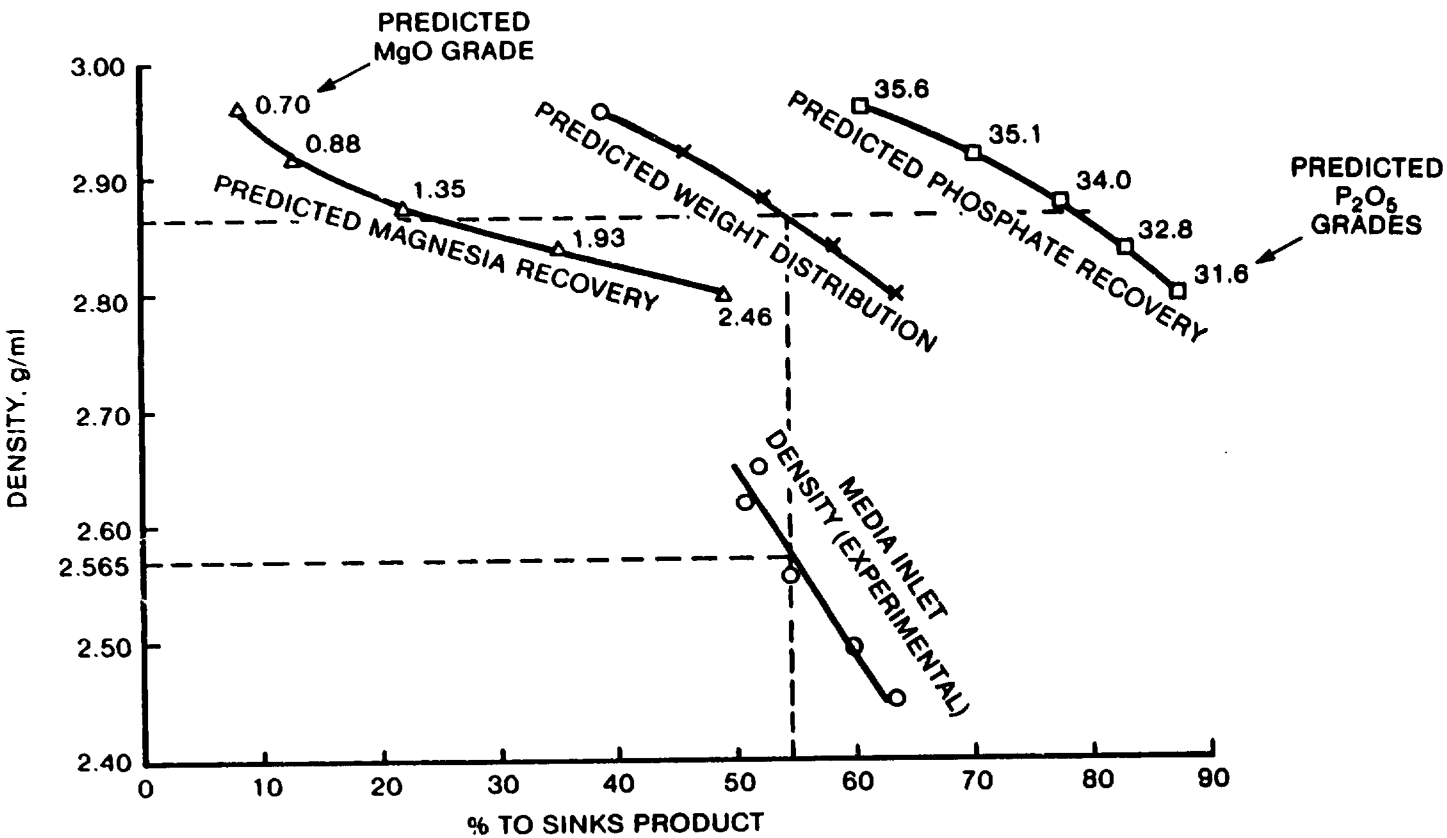
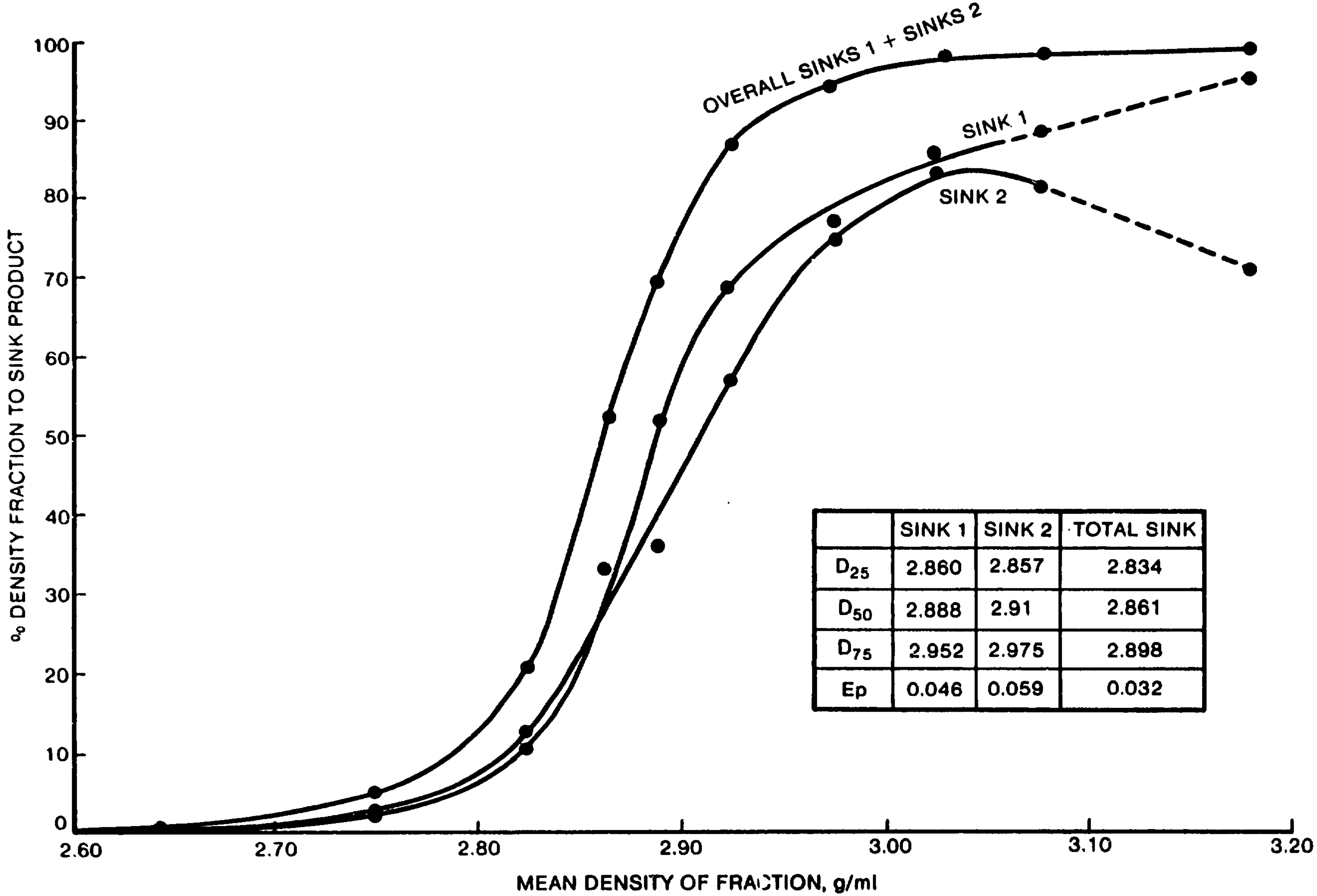


FIG. 3 RELATIONSHIP BETWEEN THE MEASURED MEDIA INLET DENSITIES (γ-GUAGE), THE SIMULATED SEPARATION DENSITIES. PHOSPHATE RECOVERIES AND MAGNESIA RECOVERIES AND THE PROPORTION OF ORE REPORTING TO THE SINKS PRODUCTS DMS 13-17

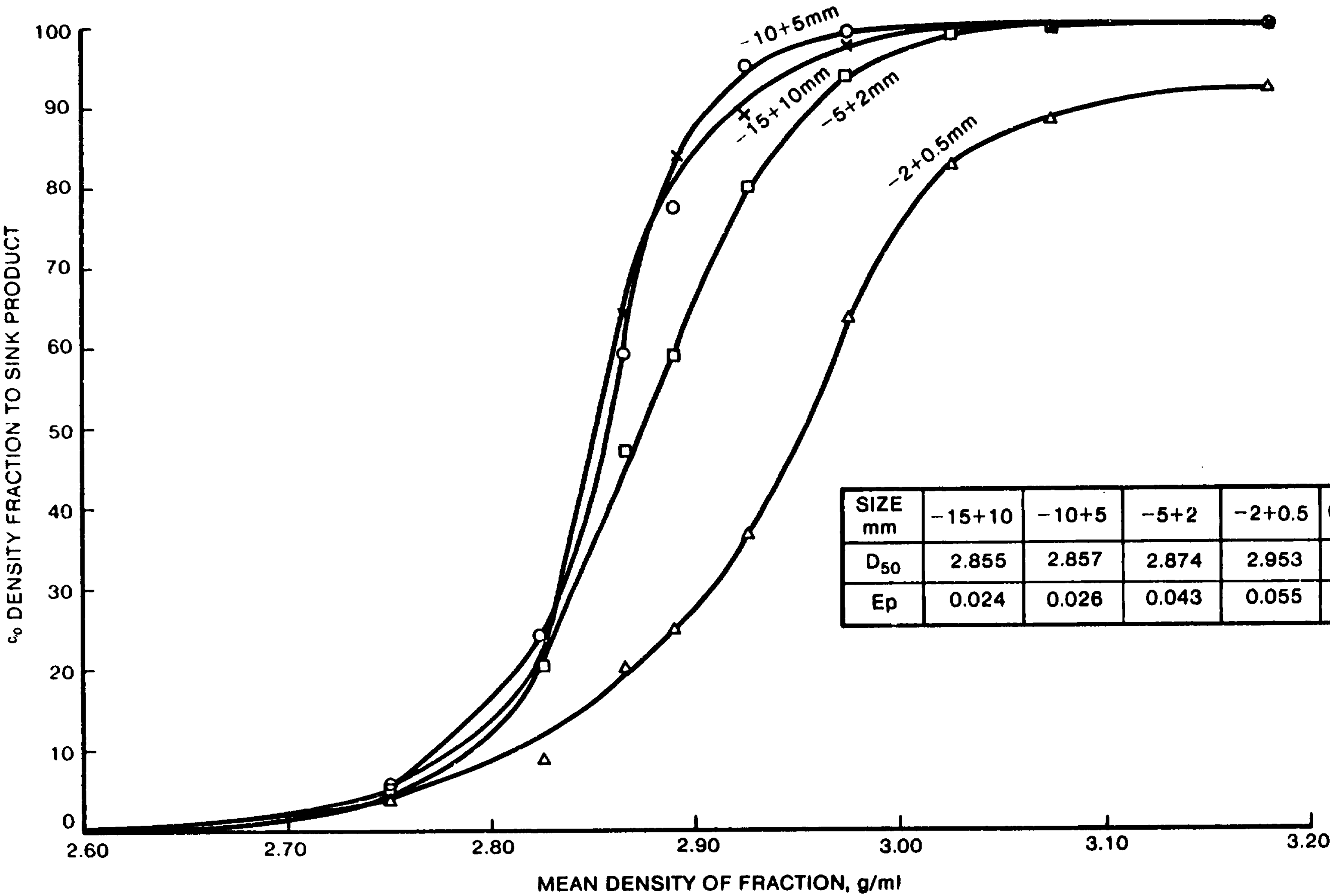
21500



	SINK 1	SINK 2	TOTAL SINK
D_{25}	2.860	2.857	2.834
D_{50}	2.888	2.91	2.861
D_{75}	2.952	2.975	2.898
Ep	0.046	0.059	0.032

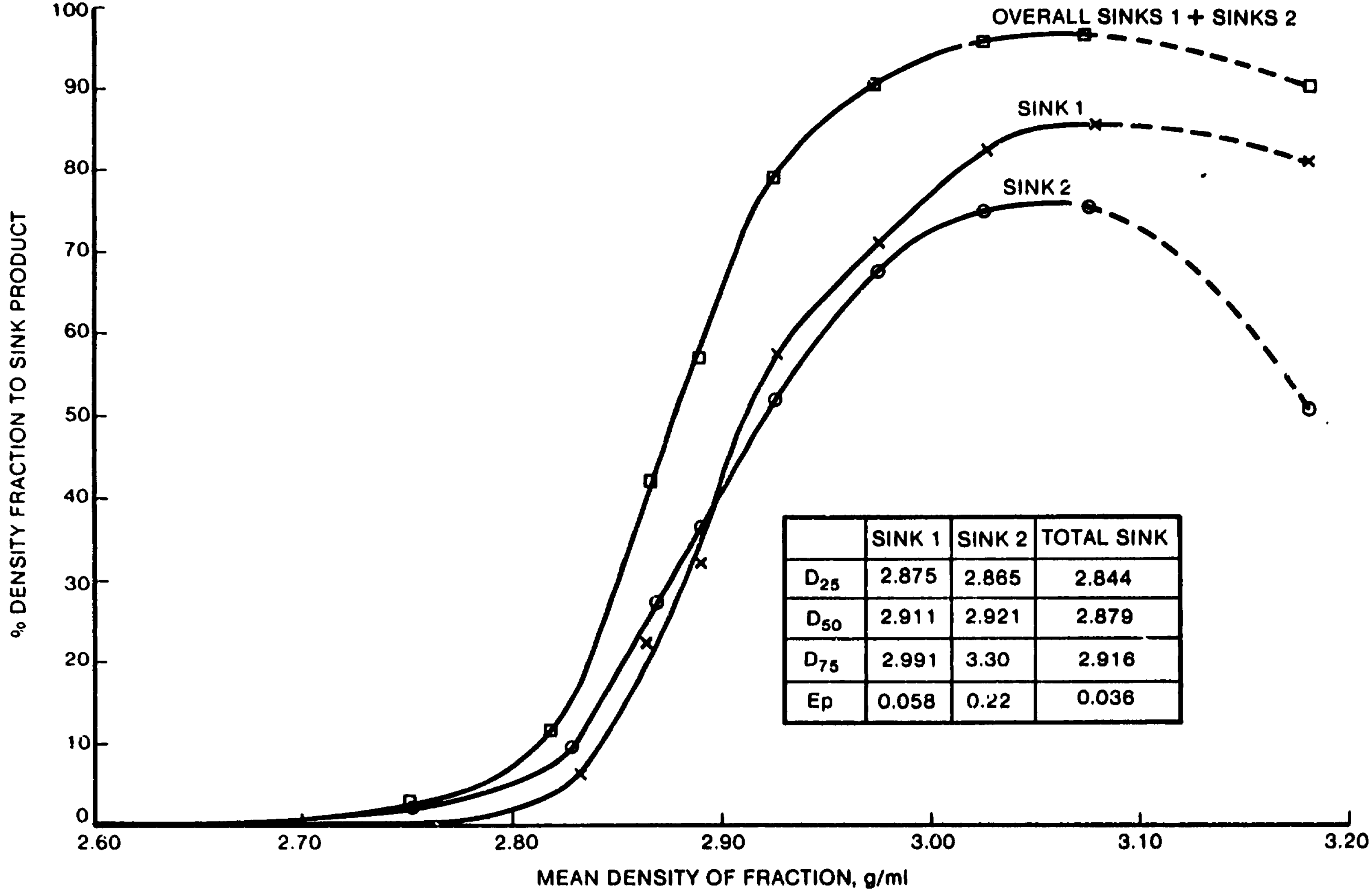
FIG. 4a PARTITION CURVES FOR INDIVIDUAL SECTIONS AND THE COMBINED TWO SECTIONS FOR DMS 19

21501



SIZE mm	−15+10	−10+5	−5+2	−2+0.5	OVERALL
D_{50}	2.855	2.857	2.874	2.953	2.861
Ep	0.024	0.026	0.043	0.055	0.032

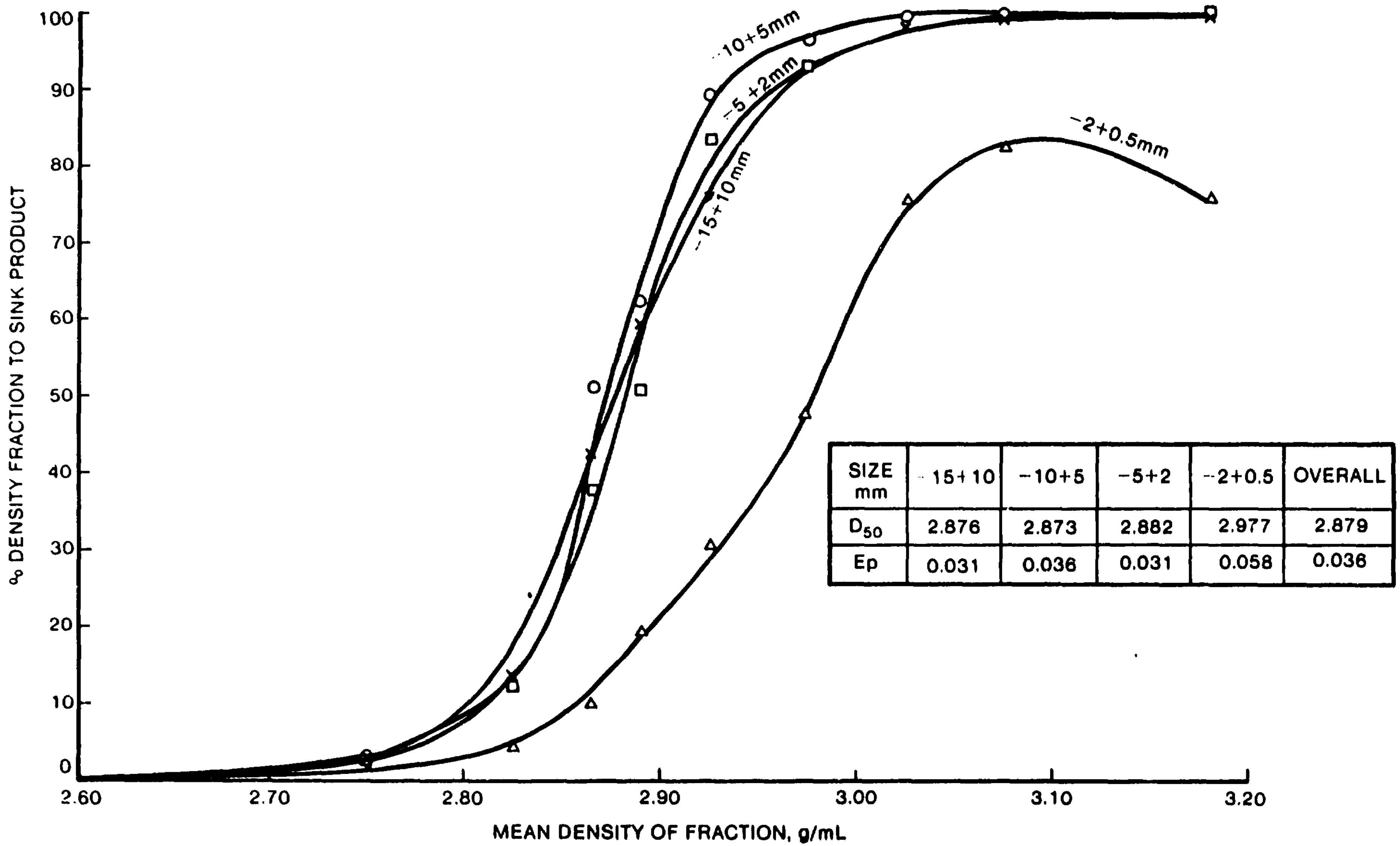
FIG. 4b COMBINED SINK PRODUCT PARTITION CURVES FOR INDIVIDUAL SIZES FOR DMS 19 21502



	SINK 1	SINK 2	TOTAL SINK
D_{25}	2.875	2.865	2.844
D_{50}	2.911	2.921	2.879
D_{75}	2.991	3.30	2.916
Ep	0.058	0.22	0.036

FIG. 5a PARTITION CURVES FOR THE INDIVIDUAL SECTIONS AND THE COMBINED TWO SECTONS FOR DMS 21

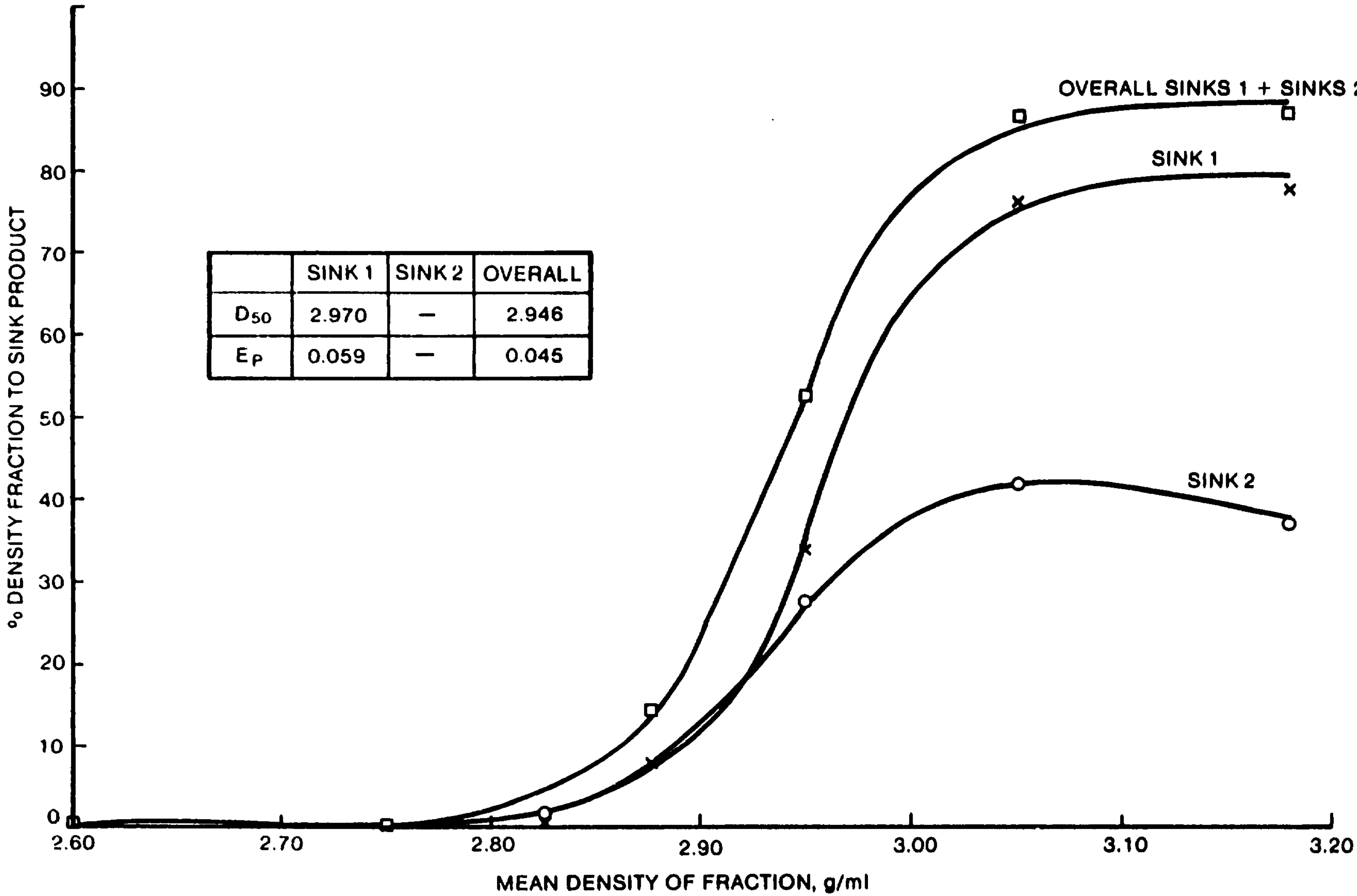
21503



SIZE mm	-15+10	-10+5	-5+2	-2+0.5	OVERALL
D_{50}	2.876	2.873	2.882	2.977	2.879
Ep	0.031	0.036	0.031	0.058	0.036

FIG. 5b COMBINED SINK PRODUCT CURVES FOR INDIVIDUAL SIZES FOR DMS 21

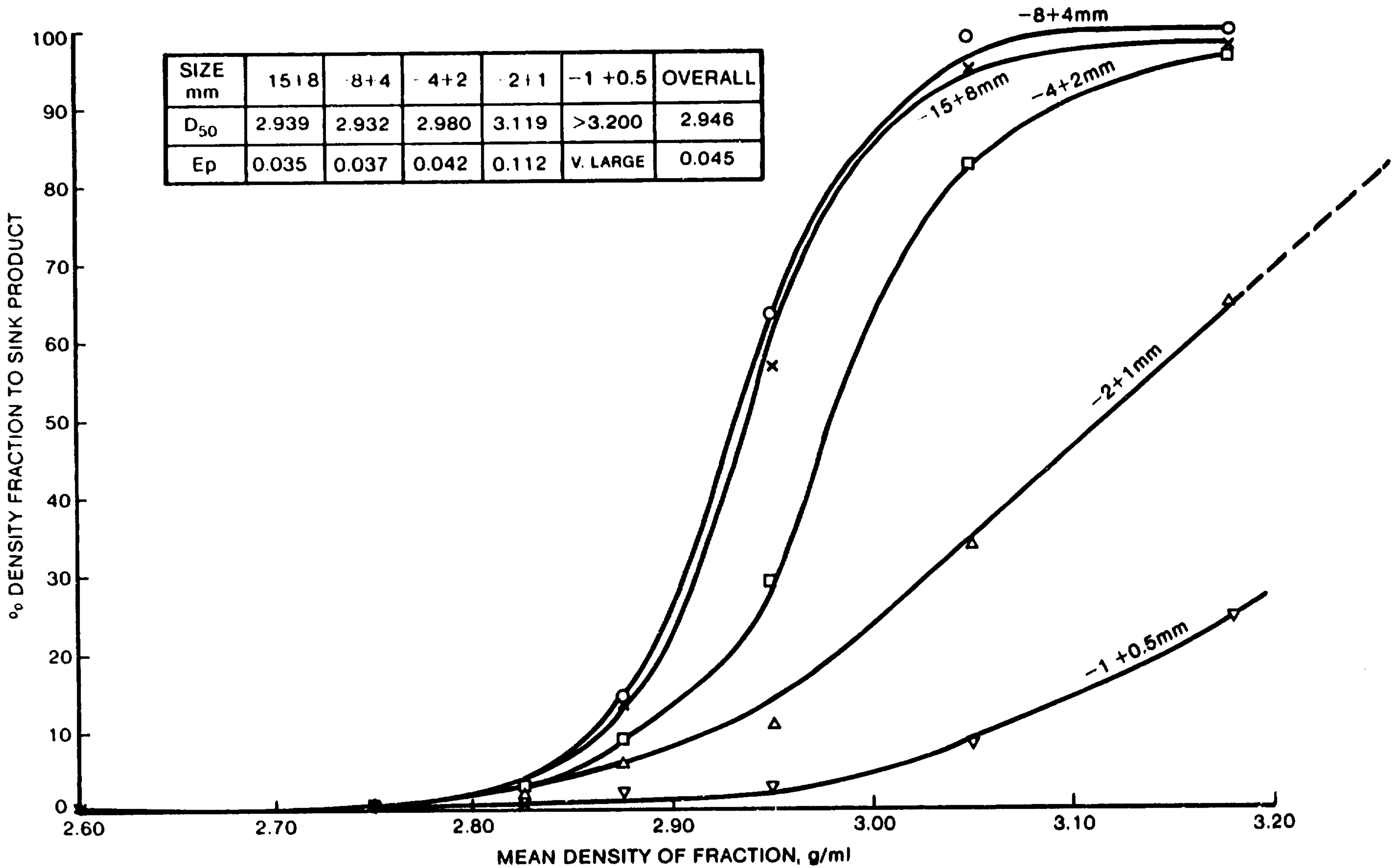
21504



	SINK 1	SINK 2	OVERALL
D_{50}	2.970	—	2.946
E_P	0.059	—	0.045

FIG. 6a PARTITION FOR INDIVIDUAL SECTION AND THE COMBINED SECTIONS FOR DMS 22

21505



SIZE mm	−15+8	−8+4	−4+2	−2+1	−1 +0.5	OVERALL
D_{50}	2.939	2.932	2.980	3.119	>3.200	2.946
Ep	0.035	0.037	0.042	0.112	V. LARGE	0.045

FIG. 6b PARTITON CURVES FOR THE INDIVIDUAL SIZES FOR DMS 22

21506

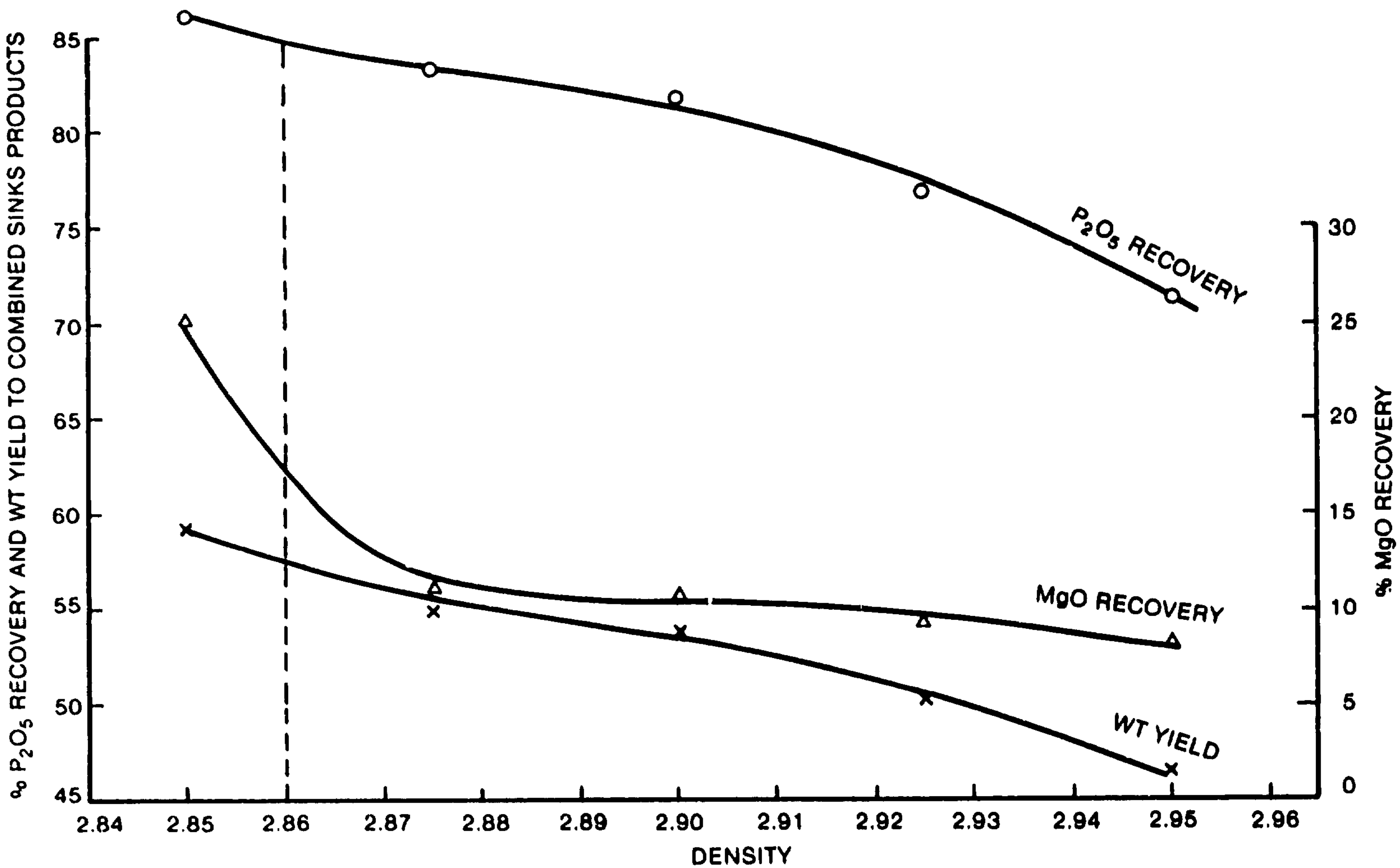
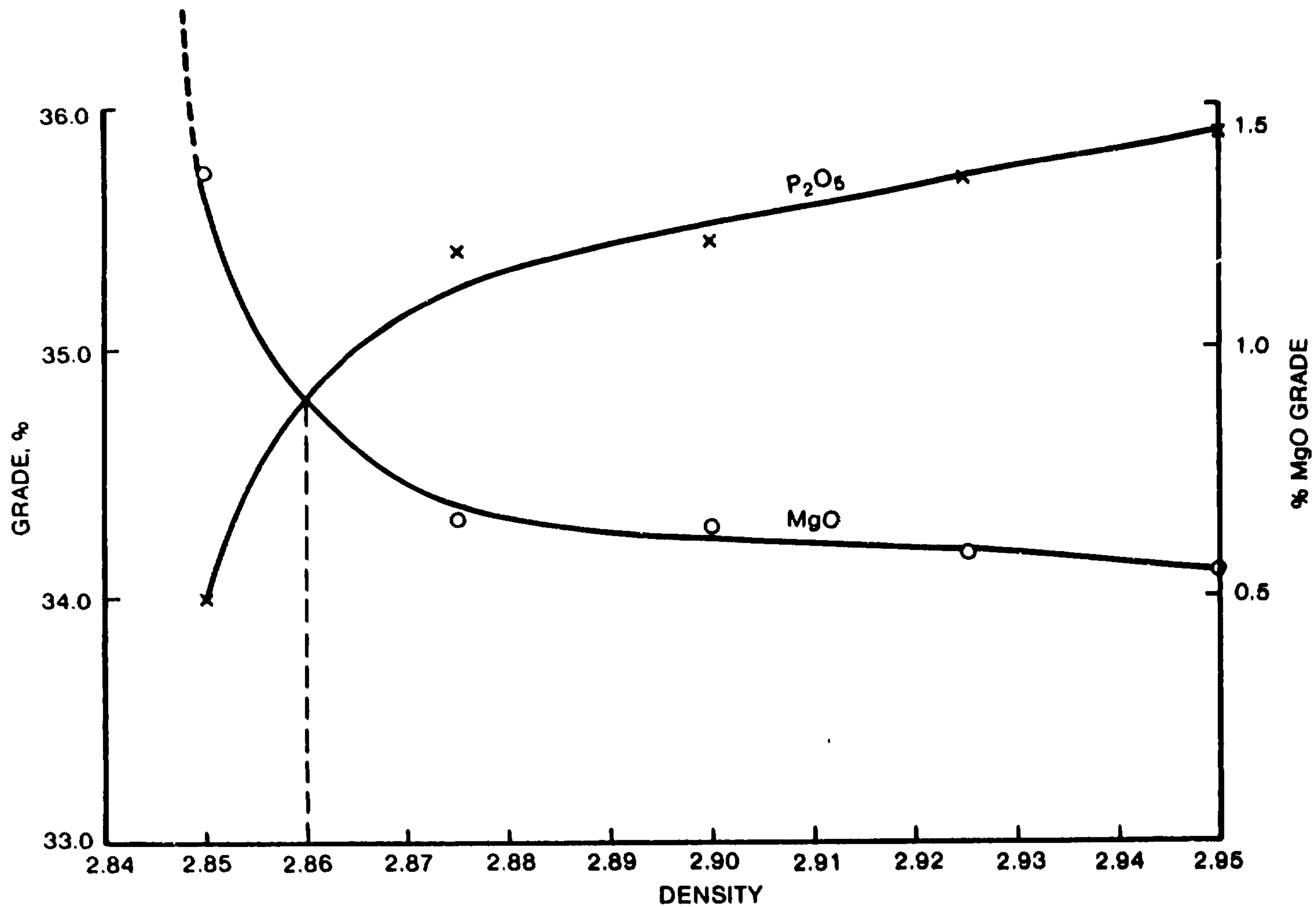


FIG. 7a MODEL PREDICTED EFFECTS OF VARYING THE MEDIA SEPARATING DENSITY (D_{50}) WITH PHOSPHATE AND MAGNESIA RECOVERY AT 33.5% MAGNETITE, 66.5% FERRO SILICON

21507



21508

FIG. 7b MODEL PREDICTED EFFECTS OF VARYING THE MEDIA SEPARATING DENSITY (D_{50}) WITH PHOSPHATE AND MAGNESIA GRADE AT 33.5% MAGNETITE, 66.5% FERRO SILICON

APPENDIX A
to WSL Report
CR 3209 (MM)

MEDIA SIZE DISTRIBUTIONS

	Cyclone 40	Magnetite
Size microns	Wt %	Wt %
118.4 - 54.9	7.8	4.0
54.9 - 33.7	11.0	13.3
33.7 - 23.7	10.0	10.5
23.7 - 17.7	10.6	11.8
17.7 - 13.6	10.6	8.4
13.6 - 10.5	10.6	7.5
10.5 - 8.2	11.6	6.8
8.2 - 6.4	10.4	7.2
6.4 - 5.0	6.8	3.9
5.0 - 3.9	4.1	4.1
3.9 - 3.0	2.3	6.2
3.0 - 2.4	1.1	3.2
2.4 - 1.9	0.4	0.7
1.9 - 1.5	0.3	0.7
1.5 - 1.2	0.8	3.2
1.2 - 0.0	1.5	8.5

Report No. CR 3209 (MM)

Date: March 1990

Report written by:	M Pearl, W N Lewis
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Under the supervision of:	P Tucker, D N Collins
Approved by:	D S Flett
Illustration reference numbers:	Fig 1 - 21873
	Fig 2 - 21499
	Fig 3 - 21500
	Fig 4a - 21501
	Fig 4b - 21502
	Fig 5a - 21503
	Fig 5b - 21504
	Fig 6a - 21505
	Fig 6b - 21506
	Fig 7a - 21507
	Fig 7b - 21508